TECHNICAL REPORT

FEASIBILITY STUDY OF THE HOSCO DEPOSIT - JOANNA GOLD PROJECT

for

Aurizon Mines Ltd.

NI 43-101 Technical Report

Effective Date: June 5th, 2012

Report Issued July 2012

PREPARED BY :

IN COOPERATION WITH :

DATE AND SIGNATURE PAGE

This report is effective as of the 5th day of June 2012.

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3. This certificate accompanies the report titled "Technical Report - Feasibility Study of the Hosco Deposit – Joanna Gold Project" having an effective date of June 5, 2012.

4. I graduated from McGill University of Montreal with a B. Eng in Metallurgy in 1981, and M. Eng. in 1988.

5. I am in good standing as a member of the Order of Engineers of Québec (#38342).

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

7. I have personally visited the property on July 19-20, 2011.

8. I am responsible for the coordination, consolidation and review of this NI 43-101 Technical Report. I have also authored and am responsible for sections: 1, 2, 3, 13, 17, 18, 19 and 21 to 27.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101. I have authored a previous NI 43-101 report related to the Joanna gold deposit development project.

10. I have practiced my profession continuously since my graduation in 1981. My relevant experience includes technical and operations management and project management in iron ore and gold projects.

11. I have read National Instrument 43-101, Form 43-101F1 and the Technical Report has been prepared in compliance with this Instrument.

12. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report and the parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Prepared in Montréal, Québec, July 12, 2012

Angelo Grandillo

Signed
Angelo Grandillo, Eng. M.Eng.

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4. I graduated from Laval University (Quebec) with a B. Sc. in Mining in 1976.
5. I am in good standing as a member of the Order of Engineers of Québec (#38991).
6. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I have visited the property on June 16 and 17, 2009..
8. I have authored and am responsible for sections: 15 and 16.
9. I am independent of the issuer as described in Section 1.5 of NI 43-101. I have authored previous NI 43-101 reports related to the Joanna gold deposit development project.
10. I have practiced my profession continuously since my graduation in 1976. My relevant experience includes technical and operations management and project management in iron ore and gold projects.
11. I have read National Instrument 43-101, Form 43-101F1 and the Technical Report has been prepared in compliance with this Instrument.
12. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report and the parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Prepared in Montréal, Québec, July 12, 2012

Signed
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4. I graduated from Laval University in 1981, in Forest Engineering and obtain a Master Degree in Forest Management in 1984.

5. I am a specialist in mining environment since 20 years;

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

7. I have not personally visited the property.

8. I am responsible for the coordination, consolidation and review of this NI 43-101 Technical Report. I have also authored and am responsible for section 20.

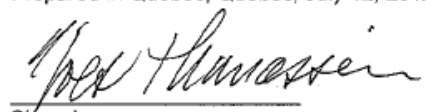
9. I am independent of the issuer as described in Section 1.5 of NI 43-101. I have authored a previous NI 43-101 report related to the Joanna gold deposit development project.

10. My relevant experience includes environmental baseline study, environmental and social impact assessment and rehabilitation plan for mining projects.

11. I have read National Instrument 43-101, Form 43-101F1 and the Technical Report has been prepared in compliance with this Instrument.

12. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report and the parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Prepared in Québec, Québec, July 12, 2012



Signed
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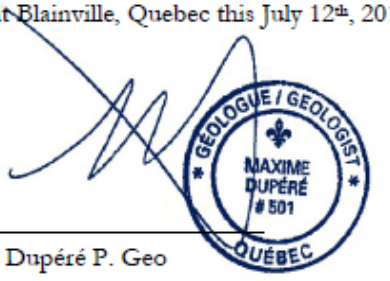
- This certificate applies to the report entitled: "Technical report - Feasibility Study of the Hosco Deposit - Joanna Gold Project for Aurizon Mines Ltd." Effective date of June 5th, 2012 (The "Technical Report")
- I am a graduate of the Université de Montréal, Quebec in 1999 with a B.Sc. in geology and I have practiced my profession continuously since 2001. I am a registered member of the Ordre des Géologues du Québec (#501). My relevant experience includes 11 years experience in mining exploration in diamonds, gold, silver, base metals, and Iron Ore. Experience also includes 5 years of consulting in mineral resource estimations for different exploration projects at different stages of exploration. I am a "qualified person" within the meaning of National Instrument 43-101 – Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators.
- I visited the Joanna Property from August 24th to August 26th, 2011.
- I am responsible for the preparation of sections 4 to 12, 14 and 23 of the technical Report
- I am independent of Aurizon Mines Limited as defined by Section 1.5 of the Instrument.
- I have authored a previous NI 43-101 report related to the Joanna gold deposit resource estimation dated March 14th, 2012.
- I have read the National Instrument 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed at Blainville, Quebec this July 12th, 2012



Maxime Dupéré P. Geo
Geologist
SGS Canada Inc. – Geostat

LIST OF ABBREVIATIONS

BWI	Bond Ball Mill Work Index
CAPEX	Capital Cost
CIL	Carbon in Leach
CIM	Canadian Institute of Mining, Metallurgy and Petrolium
CIP	Carbon in Pulp
CNA	Cyanide Amenability Test
CWI	Crusher Work Index
DWI	Drop Weight Test
FS	Feasibility Study
GML	Geographical Markup Language
ICP	Inductively Coupled Plasma
LCT	Locked Cycle Testing
MDDEP	Ministère du Développement Durable, de l'Environnement et des Parcs du Québec
MMI	Mobil Metal Ion
MRNF	Ministère des Ressources Naturelles et Faune
MTM	Modified Transverse Mercator
NSR	Net Smelter Return
NTS	National Topographic Service
OPEX	Operating Cost
OREAS	Ore Research & Exploration Pty Ltd Assay Standards
PFS	Prefeasibility Study
POX	Pressure Oxidation Process
QA/QC	Quality Assurance/Quality Control
RWI	Rod Mill Work Index
SAG	Semi Autogenous Grinding
SG	Specific Gravity
SMC	SAG Mill Comminution Test
SPI	SAG Power Index
TSX	Toronto Stock Exchange
U/G	Underground

UNITS OF MEASURE

ft	Foot
in	Inches
$	Dollar
$/t	Dollar per tonne
°	Degree
°C	Degree Celsius
µm	Micron
A	Ampere
cm	Centimeter
CND	Canadian Dollars
F_{80}	Feed - Screen size at which 80% of mass passes
fpm	Feet per minute
g	Gram
g/cc, g/cm^3	Gram per cubic centimeter
GPM	Gallons per minute
GWh	Gigawatt hour
ha	Hectare
hp	Horsepower
kg	Kilogram
km	Kilometer
km^2	Square kilometer
kt	Kilotonne
kV	Kilovolt
kW	Kilowatt
kWh/t	Kilowatt-hours per tonne
lb/h	Pounds per hour
IRR	Internal Rate of Return
m	Meter
M	Million
M t/y	Million tonnes per year
m^3/h	Cubic metre per hour
Ma	Megaanum
masl	Meters Above Sea Level
mi	Mile
mm	Millimeter
Mt	Million tonnes
mt/h	Metric tonnes per hour

MVA	Mega Volt Ampere
MW	Mega Watt
NPV	Net Present Value
P_{80}	Product - Screen size at which 80% of mass passes
scfm	Standard cubic feet per minute
t	Tonnes
t/h	tonnes per hour
t/m^3	tonnes per cubic meter
t/y	tonnes per year
tonnes or t	metric tons
tons	short tons
USD	United States Dollar

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1 SUMMARY

1.1 Introduction

Aurizon Mines retained the services of BBA Inc. to undertake the current Feasibility Study for the development of the Joanna Gold Project. Aurizon also retained the services of Roche Inc. to cover all environmental aspects of the project including tailings management.

The geological block model and resource estimate for the current Feasibility Study was prepared by SGS Canada Inc. (Geostat) and has been previously posted on Sedar in the following document:

"NI 43-101 Technical Report Mineral Resource Estimation Joanna Gold Project Rouyn-Noranda, Quebec Aurizon Mines Ltd. September 2011 Update". The Report was prepared by SGS Canada Inc. (Geostat) and authored by M. Dupéré, P. Live, E. Palumbo, A. Grandillo, M Magnan. The effective date of the report is December 31, 2011.

This current Technical Report entitled "Technical Report - Feasibility Study for the Development of Joanna Gold Project" was prepared by a Qualified Person following the guidelines of the "Canadian Securities Administrators" National Instrument 43-101 (effective June 30, 2011), and in conformity with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standard on Mineral Resources and Reserves. This Report provides a summary of the Feasibility Study. The Feasibility Study considers only the Mineral Reserves associated with the Hosco deposit. It should be understood that the mineral resources which are not mineral reserves do not have demonstrated economic viability.

This Report is effective as of June 5, 2012 and is considered as the Current Report for the development of the Joanna Property.

1.2 Property Description and Location

The Joanna Property is situated 20 km east of Rouyn-Noranda, Province of Québec. The Property is accessible via a gravel road connecting to Highway 117, which is the main road between Rouyn-Noranda and Val d'Or. The nearest city, Rouyn-Noranda, with a population

exceeding 40,000, is a regional center with significant mining history, infrastructures and a workforce to support a mining operation.

The Joanna Property consists of three (3) separate blocks without inclusions located in the Joannes and Bousquet townships totalling 156 claims covering 4,294 ha. The project area extends east-west for more than 12 km and reaches 3 km in the north-south direction. The Joannes Township claim block comprises a total of 118 claims composed of the original 67 claims acquired in 2006, 25 claims acquired in 2007, 19 claims optioned in 2008, 6 claims acquired in 2009, and another claim acquired in March 2010. The claim blocks located in the Bousquet Township comprise 30 claims for the West Block and 8 claims for the East Block all acquired by staking in 2009. The Joanna Property is subject to four (4) separate royalties.

1.3 Geology and Mineralization

The Joanna Property is situated on the famous Cadillac structural break. The structure generally strikes east-west and dips northward. It is characterized by a layer of schistose, chloritic rocks cross cutting a uniform package of metamorphosed, fine-grain sediments. Gold mineralization is composed of disseminated sulphides (pyrite, arsenopyrite and pyrrhotite) in deformed and biotite-rich zones along the Cadillac Fault. Small quartz veins ranging from a few centimetres to a metre wide are encountered in these zones. The Au disseminated mineralization is found in strongly altered and deformed corridors with an azimuth of N260˚ dipping 55° to the north-west. These mineralized corridors average a horizontal width of 20 metres and extend along the strike over 3,000 metres, reaching a depth of 400 m.

1.4 Exploration and Drilling

The Joanna Property has been explored throughout the last seventy-five years by various owners and optionees. Aurizon optioned the Property in 2006 and initiated an extensive exploration program in 2007. The following number of holes and meters of drilling pertain only to the Hosco sector of the Property. The 2007 program involved systematically re-sampling 118 historical holes and drilling 62 holes totaling 20,647 m. Following this, the first NI 43-101 compliant resources estimate was completed by SGS Geostat. In 2008, Aurizon drilled 354 holes totalling 86,320 m, and completed a Preliminary Assessment Study for an open-pit

operation conducted by BBA Inc. In 2009, 55 exploration and infill holes were drilled totaling 13,536 m. SGS Geostat then completed the updated NI 43-101 compliant resource estimate. BBA completed a positive Pre-Feasibility Study for an open pit operation in the Hosco sector. In 2010, Aurizon drilled 349 exploration and infill holes totaling 71,917 m including 43 condemnation holes that were drilled in the northern part of the Property. In 2011, Aurizon drilled 115 exploration and infill holes totaling 30,055 m. Following this, a last resources estimate update was completed by SGS Geostat with the effective date of December 31, 2011. In 2012, prior to this Feasibility Study, nine infill holes totaling 3,002 m were drilled.

As part of the independent verification program. SGS Geostat validated Aurizon's exploration methodology including core sampling, analytical procedures, and their QA/QC protocol. The QA/QC practice implemented by Aurizon consists of the insertion of reference material in the sample stream (certified and composite analytical standards and blanks). The author considers the samples representative and of good quality, and is confident that the system is appropriate for the collection of data suitable for the estimation of a NI 43-101 compliant mineral resource estimate.

1.5 Mineral Processing and Metallurgical Testwork

Metallurgical testwork has been performed at the various study stages of the Joanna Project. Testwork results have been analyzed and interpreted and used to define the Process Flowsheet, mass balances, gold recovery and other process design parameters and equipment sizing.

1.5.1 Comminution

1.5.1.1 Comminution Testwork

Seven (7) samples were prepared and were subjected to the SMC test, the Bond rod and ball mill tests, and the bond abrasion test. In addition, one (1) of these samples was also analyzed using the standard JKTech drop weight impact test and Bond low-energy impact test, in addition to the tests listed above. The results of these tests are shown in Table 1.1.

Table 1.1: Grindability Test Statistics

Statistics	Relative	SMC Results			Work Indices (kWh/t)			AI
	Density	A x b	t_a	DWI	CWI	RWI	BWI	(g)
Average	**2.75**	**31.1**	**0.29**	**8.85**	**-**	**16.5**	**14.1**	**0.360**
Std. Dev.	0.01	1.9	0.02	0.57	-	0.8	1.0	0.066
Rel. Std. Dev.	0	6	6	6	-	5	7	18
Minimum	2.73	32.1	0.30	8.62	16.2	15.7	12.8	0.268
Median	**2.75**	**31.2**	**0.29**	**8.86**	**16.2**	**16.1**	**14.2**	**0.382**
Maximum	2.76	27.5	0.26	9.93	16.2	17.6	15.7	0.436

Values obtained of A x b, DWI and CWI were generally considered to be hard with respect to the SGS database, while RWI and BWI values were considered moderately hard, and AI values were considered moderately abrasive.

1.5.1.2 Crusher Selection

The crusher selection for the FS was based on the CWI result obtained as part of the comminution tests performed. The CWI value used for the crusher sizing was 16.2 kWh/t. The crusher selected is a jaw-type crusher with a feed opening measuring 1,250 mm x 950 mm, and an installed power of 200 kW. It is fed ROM ore having F_{80} of 600 mm, and generates product having P_{80} of 110 mm. In sizing the crusher, the crusher availability was assumed to be in the order of 65% with a nominal capacity of 545 t/h. This would provide the nominal daily throughput of 8,500 t/d.

1.5.1.3 Grinding Circuit Modelling and Mill Selection

Analysis of the comminution testwork data and comminution modelling were carried out in order to determine optimum SAG and ball mill sizes for various grinding scenarios, whereby ore hardness, throughput and final grind size are varied within a design envelope. Based on the simulations conducted, the SAG mill and ball mill were sized as shown in Table 1.2. This sizing was based on design ore (A x b of 30.4, BWI of 15.7), nominal throughput of 381 t/h nominal grind of 90 µm SAG mill power of 85% of maximum and Ball mill power of 82.5% of maximum.

Table 1.2: Summary of Mill Selections

Mill	Diameter	Effective Grinding Length (EGL)	Motor size	Specific Energy at Motor (kWh/t)
SAG	30 ft (9.14m)	9.5 ft (2.90m)	5,040 kW (6,760 hp)	11.3
Ball	18 ft (5.49m)	28 ft (8.53m)	5,040 kW (6,760 hp)	10.9

1.5.2 Gravity Separation of Gravity Recoverable Gold

Five (5) Composite samples were subjected to testwork with the goal of measuring their gravity recoverable gold (GRG) content. Modeling was then performed using the results of this testwork to predict the gold recovery that may be expected from this ore in a gravity concentration circuit. Based on the results of the modeling it was decided that two (2) Knelson XD-48 units would be used, having the capacity to process a total of 600 t/h. Furthermore, considering the marked effect of grind size on GRG recovery, a grind having a P_{80} of 90 microns was the selected grind. This grind was also found to provide good flotation recoveries. It was estimated that gold recovery in gravity would be in the order of 34.8%. This value was used in the process flowsheet mass balance.

1.5.3 Flotation

In addition to flotation testing undertaken for PFS, a series of batch and locked cycle tests were performed in order to determine optimum conditions for a flotation pilot plant. Grind size, reagents and circuit configuration were investigated.

1.5.3.1 Pilot Plant for the Production of Albion Feed

A flotation pilot plant was carried out on approximately 8.6 tonnes of ore from the Joanna deposit. The objectives of the pilot plant were to see if results previously obtained at a laboratory scale could be replicated on a larger scale, and to produce a relatively large quantity of representative concentrate for feed to pilot scale Albion testwork. The average Au recovery

achieved in the flotation pilot plant was 90.7% with a weight recovery of 4.33% at a grind of 77 µm. The results show that the final concentrate %S was well controlled at the target of 15%.

1.5.3.2 Pilot Plant for the Production of POX Feed

Following the decision to adopt the POX process for oxidation, a second pilot scale production run was undertaken to produce concentrate for subsequent POX testwork. The flowsheet used for this POX flotation pilot plant was similar to the flowsheet used for the Albion flotation pilot plant, with the exception that in the latter, the second cleaner was removed.

The ore for the production of the concentrate for POX development was derived from drill core material (DC) and from reverse circulation material (RC). These were processed separately.The average combined gold recovery for gravity and flotation for the DC material was 88.8%, while that of the RC was 92.3%.

1.5.3.3 Flotation Circuit Design

Data from the pilot testwork as well as from the batch flotation locked cycle tests was used to develop the design criteria for the flotation circuit. The flotation circuit design consists of a rougher stage comprised of mechanical tank cells, a cleaner column and a cleaner scavenger column with residence times of 37 minutes, 13 minutes and 8 minutes respectively.

1.5.4 Sulphide Oxidation by POX

During the course of the Feasibility Study, it was decided that the Albion Process, previously adopted in the PFS, should be replaced by the POX process for ore oxidation. Aurizon Mines mandated Sherritt Technologies to conduct the testwork and develop the POX area of the processing plant.

The concentrates generated from the flotation pilot plant treating the RC and DC material were treated separately in the POX pilot plant. Gold extraction for the RC concentrate was typically between 88 to 90%, whereas for the DC concentrate, gold extraction was between 93 to 94%. In both cases, lower gold extractions were obtained in cyanide leaching of the solids from continuous test solids than those obtained in the batch test program. Based on the results from this testwork, it was concluded that plant design should be based on a 75-minute retention time and 220ºC autoclave operating temperature. For the purpose of the FS, the autoclave was designed to treat up to 3.85t/h of sulphur, which corresponds to the nominal feed flow to the autoclave at the 80th percentile sulphur head grade.

The oxygen required for the POX process will be generated on-site using a vacuum pressure swing adsorption (VPSA) plant at a purity of 92.5%. The autoclave is designed to operate at 3 MPa and 220ºC, and have a residence time of 75 minutes.

1.5.5 Cyanidation

The design of the cyanide leaching circuit was based on a CIP-type circuit as no “preg-robbing” behavior was exhibited in the leaching tests on the pilot plant samples. The residence time in the cyanide leaching portion of the circuit was selected to be 48 hours. The Pumpcell technology by Kemix was selected for use as the activated carbon adsorption portion of the circuit. This technology allows the circuit to be operated in a carousel mode.

1.5.6 Thickening

Five (5) thickeners are included in the design: one (1) each for the flotation concentrate and tailings and three (3) for the POX area, (one (1) in neutralization, and two (2) for counter current

decantation). The sizing of the thickeners was based on settling testwork performed on samples of each of the materials to be thickened.

1.5.7 Cyanide Destruction

Various cyanide destruction options, including Caro's acid, SO_2/Air and CombinOx processes were investigated on a laboratory scale for treatment of slurry effluent samples from the Joanna Project. Performance and reagent consumption data for each process were compared using a weak acid dissociable cyanide (CN_{WAD}) treatment target of 20 ppm or less, while also recognizing the 20 ppm total cyanide (CN_T) limit. OPEX and CAPEX estimates were prepared by Cyanco for the three (3) processes based on the results of the testwork. The SO_2/Air process was recommended by Cyanco because, although it was the least economical in terms of OPEX, it had a much lower CAPEX than the other processes. For plant design, the SO_2/Air system was designed with a residence time of 4 hrs.

1.5.8 Gold Recovery

By consolidating the gold recoveries for the individual process areas, the overall gold recovery at nominal conditions, and an average LOM gold head grade of 1.25 g/t was estimated to be in the order of 87.5%.

1.6 Mineral Resources

This Feasibility Study only considers resources classified as reserves for the Hosco deposit. It should be understood that mineral resources which are not mineral reserves do not have demonstrated economic viability.

The updated mineral resources for the Hosco deposit are shown in Table 1.3.

Table 1.3: September 2011 Updated Mineral Resources for the Hosco Deposit (In-Pit)

Grade Cut-off (Au g/t)	Category	Tonnage* (t)	Grade (g/t)	Au metal** (oz)
0.33 (Base Case)	Measured (M)	30,690,000	1.33	1,311,000
	Indicated (I)	27,150,000	1.18	1,033,000
	Total (M+I)	57,840,000	1.26	2,344,000
	Inferred	7,050,000	1.18	267,000
0.5	Measured (M)	29,460,000	1.37	1,295,000
	Indicated (I)	26,410,000	1.21	1,024,000
	Total (M+I)	55,870,000	1.29	2,319,000
	Inferred	6,950,000	1.19	266,000
1.0	Measured (M)	20,370,000	1.62	1,058,000
	Indicated (I)	15,130,000	1.51	736,000
	Total (M+I)	35,500,000	1.57	1,794,000
	Inferred	4,520,000	1.39	201,000

Updated: September 22, 2011- *Rounded to nearest 10k- **Rounded to 1k
Relative density used: 2.75t/m3
CIM Definitions were followed for mineral resources
Mineral resources which are not mineral reserves do not have demonstrated economic viablilty
Historical production of 9,704 oz has not been removed from mineral resources

1.7 Mineral Reserves

The reserves for the detailed pit design have been calculated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM Standards on Mineral Resources and Reserves) in August 2000. The mineral reserves (with dilution and ore loss) contained within the Main pit and West pit are shown in Table 1.4.

Table 1.4: Mineral Reserve Estimate (Cut-Off 0.5 g/t Au)

	Main Pit		West Pit		TOTAL			
Material	Material (k tonnes)	Grade (g/t Au)	Material (k tonnes)	Grade (g/t Au)	Material (k tonnes)	Grade (g/t Au)	Au koz (in-situ)	Au koz (recovered)
Proven Reserves	27 419	1.29	817	1.71	28 236	1.30	1 185	1 040
Probable Reserves	11 356	1.11	1 496	1.47	12 851	1.15	476	414
TOTAL	**38 775**	**1.24**	**2 313**	**1.56**	**41 087**	**1.26**	**1 660**	**1 454**
Waste	150 587		11 733		162 320			
Overburden	18 338		3 973		22 311			
TOTAL	**168 925**		**15 706**		**184 631**			
Strip Ratio	**4.36**		**6.79**		**4.49**			

1.8 Mining Methods

The ore of the Joanna Hosco deposit will be mined using conventional open-pit mining methods based on a truck/shovel operation. The open-pit operation will be designed to support an average daily production rate of 8,500 tonnes, seven (7) days a week, 365 days per year. The ore will be trucked to surface using the ramp, crushed and conveyed to the processing plant.

Waste rock consists of material containing 0.3 g/t Au or less. The waste rock will be disposed of within two (2) separate stockpiles given environmental considerations. The majority of the waste material, approximately 90%, will be stockpiled in the main waste dump and does not require any special environmental considerations. The remaining 10% consists of waste material which could leach arsenic, and requires environmental protection measures. Overburden material will be removed during the first four (4) years of operation, and will be stored in a separate area.

The mining equipment fleet consists of up to 10 (ten), 100-short ton class haulage trucks, two (2) drill rigs, two (2) 10 m^3 hydraulic excavators, one (1) front-end loader, two (2) dozers, one (1) motor grader, one (1) backhoe excavator and auxiliary equipment. All equipment is assumed to be purchased as new by Aurizon Mines.

1.9 Recovery Methods and Processing Plant Design

The metallurgical testwork for the Hosco deposit performed during this Study allowed for the development of the process mass and water balances. These were used to develop the process plant design. The plant is required to process a nominal flow of 8,500 tonnes per calendar day or 3.1 Mt/y. The hourly flow rates for the plant were evaluated based on an overall plant utilization of 85%, with the exception of the crusher, which was assumed to have a 65% utilization rate. The main process areas are the following:

- Crushing;
- Grinding (including gravity gold concentration);
- Flotation;
- Pressure oxidation;
- Cyanide leaching;
- CIP (activated carbon adsorption);
- Carbon stripping and regeneration;
- Electrowinning and Casting.

For the purposes of the FS, the weight recovery in the flotation stage was assumed to be 5%, representing the amount of material to be treated in the POX and cyanide leaching stages. Flotation tails represent 95% of the ROM and are sent directly to tails.

Based on metallurgical testing, a gold recovery of 87.5% was assumed for the purposes of the FS. Based on an average head grade of 1.26 g/t, the annual gold production will be 109,237 oz/y.

Based on this information, the process design criteria and mechanical equipment list were developed and used as the basis of the capital and operating cost estimates.

1.10 Environment

1.10.1 Jurisdictions and Applicable Laws and Regulations

The evaluation process and review of environmental impacts will apply. For the Joanna Project, it is assumed that public hearings will be required given the relative proximity of a biodiversity reserve and residential areas, and the fact that environmental groups have raised concerns about an open-pit operation in the mining industry. It should be noted that Aurizon Mines Ltd. has already initiated active and participative discussions with local and regional stakeholders and that such an approach will be continuing as the project progresses. This approach aims at establishing and maintaining a sustainable dialogue with stakeholders in order to identify specific issues associated with the Project. A period of about 20 months is anticipated for the preparation of the Environmental Impact Assessment and obtaining the government decision. Within the context of the current Study, fish habitat was not considered a trigger for the federal environmental review procedure considering project-specific characteristics and their potential impacts on the receiving environment.

1.10.2 Environmental Permitting

Once the Project will have undergone Environmental Impact Assessment and be authorized by the Government pursuant to Section 31.5 of the Act, it would still be subject to Section 22 of the EQA and must therefore obtain a general certificate of authorization. The issuance of that authorization, however, should only be a formality as the certificate issued pursuant to Section 31.5 of the Act binds the Minister as to where he exercises the powers provided in Section 22. In addition to the certificate of authorization required under Section 22 of the *Environment Quality Act*, the proponent must obtain various permits, authorizations, approvals, certificates and leases required from the appropriate authorities. The authorization application and permitting process is expected to take one year. Applications may be filed concurrently with the construction work and should not therefore impact the critical project schedule.

1.10.3 Environmental Baseline Studies

Several comprehensive assessments were performed from 2007 to now in order to define, in compliance with all applicable guidelines, policies, regulations and laws in Quebec and Canada, the state of reference of the Joanna Project receiving environment. A review of all existing

information on local topography, regional hydrography (watershed) and geomorphology was done in order to precisely define the physical context in which the Joanna Project will be implemented.

In order to characterize the groundwater flow regime, detailed geotechnical and hydrogeological studies were conducted as part of the project in the pit area, and at the sites where the development of associated mining infrastructure is foreseen. The modeling results indicated that a maximum rate of 8,100 m³/d should be maintained for operation of the mine. The radius of influence of the dewatering activities will potentially affect up to approximately 1 km from the pit limits. According to the results of the hydrogeology study performed in 2010 and 2012, the operation of the Joanna mine should not impact residential water wells in the area, in terms of either water supply or quality. Moreover, groundwater quality and level will be monitored to ensure residential wells are not affected by mining operations.

A study area covering the entire Property was established in order to assess biological conditions prevailing prior to the implementation of the Project. A vast review of all existing information on wildlife in the study area was also completed. Additional field surveys were done to precisely map all fish habitats within the Property limits.

A comprehensive Fisheries and Fish Habitats Assessment Program was performed to precisely locate all sections of the Stitchman Creek and its affluents, which are to be considered as fish habitats according to Canada's *Fisheries Act*. The objective of such work was to limit the ecological footprint of the Project to areas located out of those habitats. None of the Stitchman Creek and its affluents hosts fish species of importance to any ongoing productivity of commercial, recreational or Aboriginal fisheries.

1.10.4 Tailings Characteristics and Management

Ore processing includes the flotation of sulphides associated with gold and aims to produce a sulphide concentrate. It also involves treating sulphide concentrates by way of cyanidation to recover gold. For the Joanna Project, the objective is to incorporate environmental protection measures in a context of sustainable development as soon as the initial phase of the Project is complete. It has therefore been decided to manage tailings separately. The goal is to produce

desulphurized tailings through step flotation of sulphides (~95% of tailings). The other type of tailings (~5% of tailings) is leachable and requires protective measures specified under Directive 019. Consequently, two tailings management facilities (TMF) will be constructed on site.

1.10.5 Waste Rock, Overburden and Top Soil Management

According to the waste rock geochemical characterization results, the wacke, conglomerate and mafic intrusions are low-risk under Directive 019 and account for about 85-90% of all waste rocks, whereas the amphibolites and the Cadillac Fault rocks are considered leachable for arsenic and represent about 10-15% of all the waste rocks. Waste rocks will therefore be stockpiled into separate piles. The area where the leachable waste rock is stockpiled will be protected with a geo-membrane to prevent groundwater contamination. A low-grade ore stockpile will be set up near the crusher site and will be protected similarly to the leachable waste rock stockpile. Topsoil and overburden that is removed to allow for pit operations will be stockpiled into two separate piles and used for site restoration.

1.10.6 Site Geotechnical Assessment

Site Geotechnical Assessment works identified the presence of materials such as surface organic soil; cohesive soils; granular soils; and bedrock (from 0 to almost 30m deep). The northwestern extremity of the Hosco area is preferable for the construction of buildings with superficial foundations as the presence of rock close to the surface would allow for heavy loads such as those associated with mill, crusher and other similar buildings.

Preliminary analysis at the Hosco and West pits indicates that excavation slopes (in overburden) should be within 1.0V: 4.0H and 1.0V: 5.5H depending on their location. A more specific analysis will be required to further specify the pit slopes. Many areas were considered to locate the TMFs: the western part of the Property was selected for the Flotation TMF and an area located directly to the north of the Hydro-Quebec power lines for the POX TMF.

1.10.7 Water Management Plan

All mine infrastructures are located within only one sub-watershed, namely the Stitchman Creek. No water discharge is located outside the Stitchman Sub-Watershed. The effluent of the Process Water Polishing Pond is considered as the final effluent following Québec's

Directive 019. The final effluent will comply with all provincial (Directive 019) and federal (CCME, MMER) criteria/guidelines. Two (2) water treatment units will be implemented – one at the outlet of the POX TMF and another at the outlet of the Process Water Pond – to make sure their effluent complies with all applicable criteria/guidelines.

Three other water discharges were also considered (Topsoil, Overburden and Main Waste Rock piles). It is planned to collect water running off of the As-Ni Waste Rock Pile, the Ore Stockpile and the Mill Area by pumping them from a collecting basin towards the Process Water Pond. Mine water will be pumped to the Process Water Pond. Water from the Flotation TMF Pond will be recirculated to the mill. All water accumulated in the Process Water Pond and that will not have been recirculated to the mill will be discharged with the final effluent following treatment.

1.10.8 Rehabilitation and Mine Closure Plan

The main objective of mine site rehabilitation is to restore the site to a satisfactory condition by: a) eliminating unacceptable health hazards and ensuring public safety; b) limiting the production and circulation of substances that could damage the receiving environment and, in the long-term, trying to eliminate maintenance and monitoring; c) restoring the site to a condition in which it is visually acceptable to the community; and, d) reclaiming the areas where infrastructures are located (excluding the accumulation areas) for future use. The mine site rehabilitation plan focuses on land reclamation, reclamation of waste dumps, tailings areas and water basins, and surface drainage patterns to prevent erosion. The successful completion of a rehabilitation plan will ensure that the project results in a minimum of disturbance. Site inspections and monitoring (physical stability, geotechnical, environmental, agronomic) will be carried out before the property is retroceded to the Government. The rehabilitation concept for the current project complies with the requirements set out in the *Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements* and current legislation.

1.11 Capital and Operating Costs

The total initial capital cost, including direct costs, indirect costs and contingency has been estimated to be in the order of $422.2M, as indicated in Table 1.5. Mining costs include costs of pre-stripping of waste and overburden, required as part of the mine construction phase, which have been estimated at $21.8M. To support continued operations, sustaining capital required over the life of the operation has been estimated to be in the vicinity of $70.2M. An additional amount estimated by Roche in the order of $27.1M is expected to be put in escrow for site closure and restoration related to the permanent closure of waste rock stockpiles and tailings ponds. Of this amount, $20.3M will be deposited during the initial period. The total of Sustaining Capital plus Closure and Rehabilitation Costs is in the order of $97.3M.

Table 1.5: Total Estimated Initial Capital Costs (M$ CAD)

Category	Initial	Sustaining
Mining	39.0	24.6
Concentrator and Site Infrastructure	301.6	1.0
Environmental and Tailings Management	81.5	44.6
TOTAL	**422.2**	**70.2**
Closure and Restoration	0	27.1

Table 1.6 presents a summary of total estimated average operating costs presented in Canadian Dollars (CAD$) per tonne of ore milled, and per ounce of gold produced. Operating costs were estimated based on the average over the life of the mine. Operating costs exclude pre-stripping costs, which have been capitalized. The total estimated operating costs are in the order of $25.32 per tonne of ore milled, and $716 per ounce of gold produced. Transportation and refining costs, as well as Royalties paid, are presented separately.

Table 1.6: Total Estimated Average Operating Cost

Category	$/t Milled	$/oz Gold
Mining	11.92	337
Concentrator and Infrastructure	12.37	349
Sales, General & Administration	0.59	17
Environmental and Tailings Management	0.45	13
TOTAL	**25.32**	**716**
Transportation and Refining	0.11	3
Royalties (varies according to gold price)	0.95	27

1.12 Economic Analysis

The economic evaluation of the Joanna Gold Project was performed by BBA using the discounted cash flow model on a before-tax basis. The Net Present Value (NPV), based on a discounting rate of 0% and 5% (base case) resulting from the net cash flow generated by the Project, was calculated. The payback period is also indicated as a financial measure. A discount rate is applied to the cash flow to arrive at a NPV and payback period. The results of the analysis are presented in Table 1.7. It was found that the pre-tax net NPV at a 5% discounting rate is 112M$, and the pre-tax IRR is 8.7%.

The following assumptions were made for the financial analysis:

- A production life of **13.4 years** for the Hosco deposit, based on the resource estimate;
- A gold price of **$1350/oz** based on a 36 month moving average to the end of May 2012;
- Production startup 44 months after initiation of environmental permitting process;
- No escalation or inflation factor has been taken into account (constant 2012 $);
- US Dollar at par with Canadian Dollar.

Table 1.7: Base Case Financial Analysis Results

IRR		8.66%
Discount Rate	**NPV (M$)**	**Payback (Years)**
0%	358.5$	8.2
5%	111.6$	10.3

A +/- 20% sensitivity analysis was done on parameters that are deemed to have the biggest impact on project financial performance: CAPEX, OPEX and gold price. The NPV and IRR of the project were found to be more sensitive to gold price than to CAPEX and OPEX.

1.13 Conclusions

Based on the overall results obtained in this Feasibility Study, the following conclusions can be drawn:

- The Joanna Project Hosco deposit consists of a low grade, arsenic bearing refractory gold mineralization.
- Mining at a rate of 8,500 t/d classifies this operation as a low tonnage operation.
- Ore processing requires crushing, grinding, gravity, flotation, pressure oxidation, cyanide leaching and CIP gold extraction.
- The arsenic content of the ore has significant repercussions on environmental management, namely in the design of the higher grade arsenic bearing waste rock and tailings. This also adds to the complexity of managing water and effluent on site. Despite incorporating diligent design techniques to minimize environmental risks, the mere presence of arsenic bearing waste rock and tailings on site may affect the Environmental Permitting process.
- The pre-production (initial) capital cost for the Project was estimated at $422.2M. Site closure and restoration costs were estimated at $27.1M. Sustaining capital required over the life of the operation has been estimated to be in the order of $70.2M.
- The average operating cash cost is estimated at US$716 per ounce of gold and $25.32 per tonne milled. This excludes transportation, refining and royalties.

- The financial analysis, using a price of gold of US$1,350 per ounce indicates a pre-tax Net Present Value ("NPV") (using a 5% discount rate) of $112 million with a pre-tax internal rate of return ("IRR") of 8.7%. The undiscounted payback period is 8.2 years.

The results of the Feasibility Study indicate that, although Project economics are positive (based on the financial assumptions used), the potential financial returns are however marginal and small fluctuations in gold price (and to a lesser extent capital and operating costs) can significantly impact the economic viability of the Project.

At this time, given the exploration potential elsewhere on the property, Aurizon Mines may want to consider further development of the Heva deposit and evaluate if a joint Hosco/Heva mining operation would improve project economics. Furthermore, considering that the Environmental Permitting process for this Project may be more complex and longer than planned, Aurizon Mines may want to consider proceeding with the obtention of permits and authorizations for the Project so that, if and when project economics become more favorable, they can proceed quickly with its execution.

2 INTRODUCTION

2.1 Scope of Study

The following Technical Report (the Report) presents a summary of the results of the Feasibility Study (FS) for the development of the Aurizon Mines, Joanna Gold Property Project, situated in proximity of Rouyn-Noranda in North-Western Quebec. This FS has been based solely on the development of the Hosco deposit. Aurizon Mines commissioned the engineering consulting group BBA Inc. to perform this Study. This Report was prepared at the request of Mr. Martin Bergeron, Vice President of Operations of Aurizon Mines, a Canadian publicly traded company listed on the TSX under the symbol ARZ, with its head office situated at:

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This Technical Report titled “Technical Report - Feasibility Study for the Development of Joanna Gold Project” was prepared by a Qualified Person following the guidelines of the “Canadian Securities Administrators” National Instrument 43-101 (effective June 30, 2011), and in conformity with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standard on Mineral Resources and Reserves.

This Report is considered effective as of June 5, 2012 and is considered the current report for the development of the Joanna Property.

2.2 Sources of Information

This Report is based in part on, internal company technical reports, maps, published government reports, company letters and memoranda, and public information, as listed in Section 27 "References” of this Report. Sections from reports authored by other consultants

may have been directly quoted or summarized in this Report, and are so indicated where appropriate.

It should be noted that the authors have relied upon selected portions or excerpts from material contained in the following NI 43-101 compliant Technical Report. This Report is publicly available on SEDAR (www.sedar.com):

"NI 43-101 Technical Report Mineral Resource Estimation Joanna Gold Project Rouyn-Noranda, Quebec Aurizon Mines Ltd. September 2011 Update". The report was prepared by SGS Canada Inc. (Geostat) and authored by M. Dupéré, P.Live, E. Palumbo, A. Grandillo, and M. Magnan. The effective date of the report is December 31, 2011.

This Feasibility Study, NI 43-101 compliant report, has been completed using the previously mentioned Technical Report as well as available information contained in, but not limited to the following reports and documents:

- Block model provided by SGS (Geostat) updating only the mineral resource for the Hosco deposit. The mineral resource estimates for the Heva and Alexandria deposits referred to in this current document were last updated in an NI 43-101 report dated November 9, 2009. The mineral resources associated with the Heva and Alexandria deposits were not considered for this Feasibility Study;
- SGS Minerals Services testwork results;
- Sherritt Technologies testwork results;
- Internal and commercially available databases and cost models;
- Various reports produced by Roche for environmental studies and considerations for permitting, site hydrology, hydrogeology and geotechnical, tailings management and site closure plan.

2.3 Terms of Reference

Unless otherwise stated:

- All units of measurement in the Report are in the metric system;

- All costs, revenues and values are expressed in terms of Canadian (CDN) Dollars;
- All metal prices are expressed in terms of CDN dollars unless otherwise stated;
- An exchange rate of $1.00US = $1.00CDN was used.

Grid coordinates for the block model are given in a local coordinate system established by Aurizon and called GML; map coordinates are either in the MTM system or in the latitude / longitude system.

2.4 Site Visit

A site visit was conducted on July 19 and 20, 2011 by Angelo Grandillo of BBA. The purpose of the visit was to provide all key project team members with an overview of the Joanna Project and to review project development milestones and planning. Aurizon geology and mining personnel were available to discuss general geological conditions and to provide a tour of the core storage facility with a presentation of select bedrock core material. BBA performed a visual examination of selected drill cores used to compose the composite samples for metallurgical testwork. To provide an overview of the Property terrain, a tour of the site was performed.

Mr. Maxime Dupéré visited the Property on August 24-26, 2011 for a review of exploration methodology, sampling procedures, and quality control procedures and to conduct an independent check sampling of mineralized drill core intervals selected form recent holes from the Hosco sector.

3 RELIANCE ON OTHER EXPERTS

BBA prepared this Report using reports and documents as noted in Section 27 “References” of this Report. BBA has not verified the legal titles to the Property nor any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties, but has relied on Aurizon Mines to have conducted the proper legal due diligence. Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Report.

It should be understood that the mineral resources which are not mineral reserves do not have demonstrated economic viability. The mineral resources presented in this Technical Report are based on available sampling and on assumptions and parameters currently available to the authors.

BBA had the responsibility for assuring that this Technical Report meets the guidelines and standards stipulated. Certain sections of this Report however, were contributed by SGS (Geostat) and Roche. Table 3.1 outlines responsibility for the various sections of the Report.

Table 3.1: Technical Report Section List of Responsibility

Section Number	Section Title	Responsibility	Comments and Exceptions
1	SUMMARY	BBA	
2	INTRODUCTION	BBA	
3	RELIANCE ON OTHER EXPERTS	BBA	
4	PROPERTY DESCRIPTION AND LOCATION	SGS Geostat	
5	ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY	SGS Geostat	
6	HISTORY	SGS Geostat	
7	GEOLOGICAL SETTING AND MINERALIZATION	SGS Geostat	
8	DEPOSIT TYPE	SGS Geostat	
9	EXPLORATION	SGS Geostat	
10	DRILLING	SGS Geostat	
11	SAMPLE PREPARATION, ASSAYING AND SECURITY	SGS Geostat	
12	DATA VERIFICATION	SGS Geostat	
13	MINERAL PROCESSING AND METALLURGICAL TESTING	BBA	
14	MINERAL RESOURCE ESTIMATE	SGS Geostat	
15	MINERAL RESERVE ESTIMATE	BBA	
16	MINING METHODS	BBA	Pit slope and Geomechanics by Golder Associates, Overburden Geotech by LEQ, Hydrogeology By

Section Number	Section Title	Responsibility	Comments and Exceptions
			Roche
17	RECOVERY METHODS	BBA	
18	PROJECT INFRASTRUCTURE	BBA	
19	MARKET STUDIES AND CONTRACTS	BBA	
20	ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT	Roche	Community and First Nations relations by Aurizon
21	CAPITAL AND OPERATING COSTS	BBA/Roche	Roche provided CAPEX and OPEX for environmental, tailings management and site closure plan
22	ECONOMIC ANALYSIS	BBA	
23	ADJACENT PROPERTIES	SGS Geostat	
24	OTHER RELEVANT DATA AND INFORMATION	BBA	
25	INTERPRETATION AND CONCLUSIONS	BBA	
26	RECOMMENDATIONS	BBA	
27	REFERENCES	BBA	

The following Qualified Persons (QP) have contributed to the writing of this Report and have provided QP certificates included in this Report. The information contained in the certificates outlines the sections in this Report that each of the QPs are responsible for.

- Angelo Grandillo, Eng. BBA
- Patrice Live, Eng. BBA
- Maxime Dupéré SGS (Geostat)
- Yves Thomassin, Roche

BBA has not relied on any other experts for legal or technical matters.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Joanna property is situated in the Joannes Township, Province of Québec, 20 km east of Rouyn-Noranda and five (5) kilometres northeast of the municipal airport, proximal to Highway 117 between Rouyn-Noranda and Val d'Or (Figure 4.1). The Property is accessible via a gravel road to the covered shaft collar of the old Hosco mine. Some concrete foundations remain as evidence of the previous mining activities. The north-south gravel road crosses a railway line that runs east-west. The railway line does not pass over any previously identified gold mineralization. The Project is located on the NTS sheet 32D02.

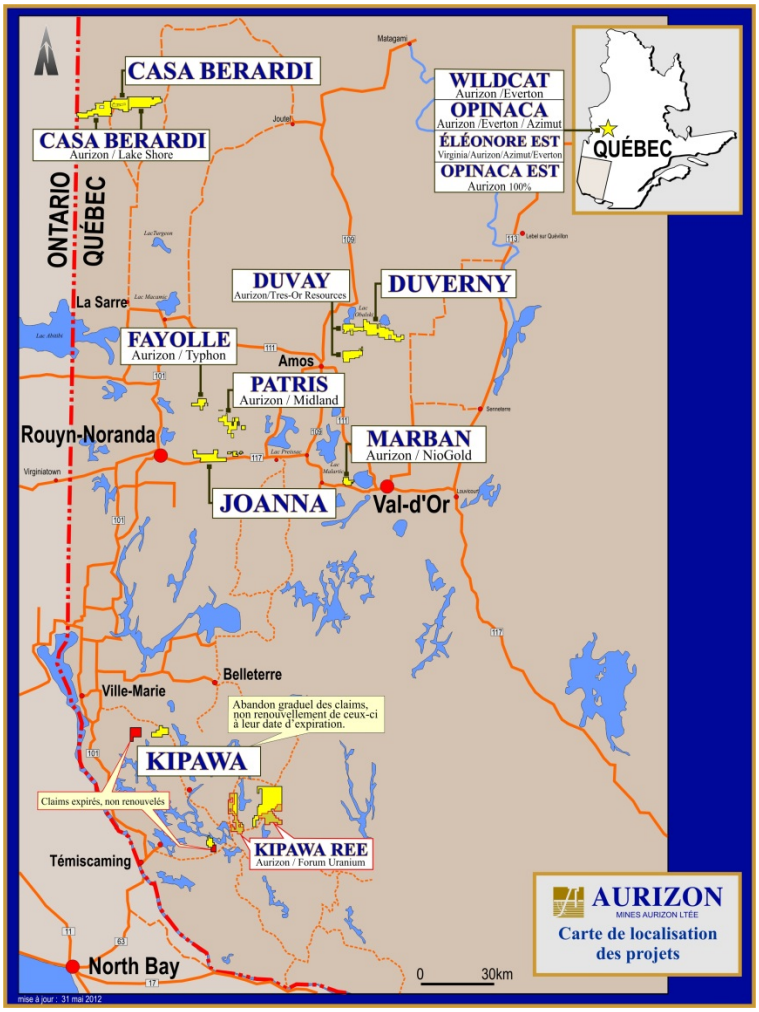

Figure 4.1 : General Property Location Map

4.2 Property Description, Ownership and Agreements

As of July 2012, the Property consists of three separate blocks without inclusions, totalling 156 claims covering 4,294 ha. The main claim block, which hosts the Hosco, Heva, and Alexandria (Figure 4.2) deposits, is located in the Joannes Township. The two (2) other claim blocks are situated in the Bousquet Township, east of the main block, in the so-called Bousquet sector (Figure 4.3). The project area extends east-west for more than 12 km and reaches 3 km in the north-south direction.

The Joannes Township claim block comprises a total of 118 claims covering 3,690 ha. The claim block is composed as follows: The original 67 claims (with the old Hosco mine and parts of the old Heva mine) were optioned in 2006; three (3) claims from a private owner, the 20 Henriksen claims, the two (2) Vantex claims (with the rest of the old Heva mine) acquired in 2007; and the 19 Alexandria claims optioned in 2008. In May 2009, Aurizon acquired another six (6) claims from a private owner. Finally, one (1) claim was acquired from a private owner in March 2010.

The claim blocks located in the Bousquet Township are composed of 30 claims covering 480 ha for the West Block and eight (8) claims covering 124 ha for the East Block. The Bousquet Township claims were acquired by staking in May 2009.

The Property is adjacent to a biodiversity reserve located just south of the Joannes Township claim block, which has been excluded for exploration and mining activities. Gescad Inc. manages the mining titles for Aurizon and generates an update on the titles each month. All statutory work and tax payments are in good standing. A detailed listing of the Joanna property claims is included in Table 4.1.

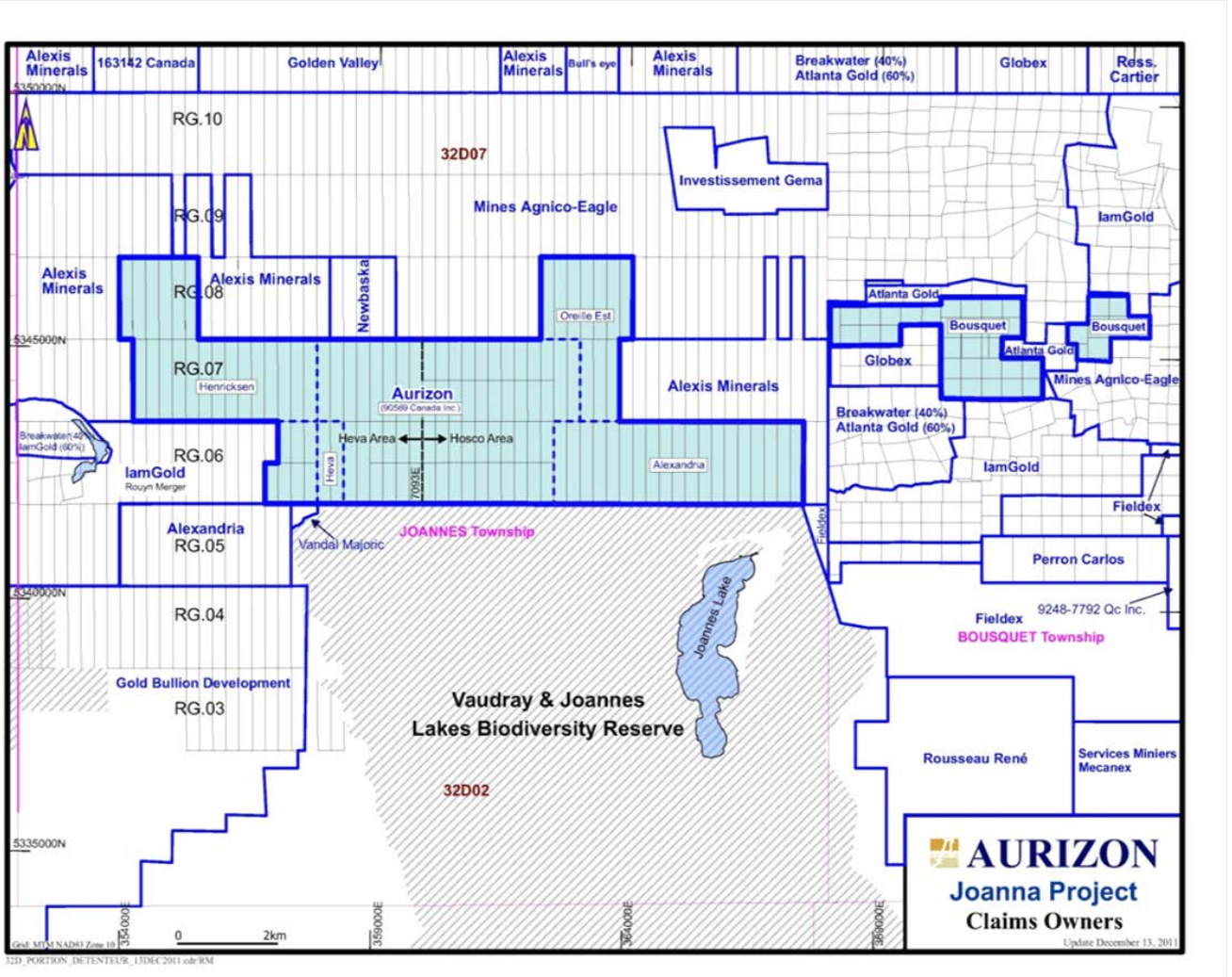

Figure 4.2 : General Claims Location Map

Table 4.1 : Detailed Listing of Property Claims

NTS	Township	RANG	LOT	Title	Entry Date	Expiry Date	Renewal Date	Area (Ha)	Credits	Mining Right	Work Required	Renewals
32D02	Joannes	0006	0042	CDC 1094818	27/05/2002	26/05/2012	26/03/2012	42.63	$ 13,574.78	$ 53.00	$ 1,800.00	4
32D02	Joannes	0006	0043	CDC 1094819	27/05/2002	26/05/2012	26/03/2012	42.65	$ 17,601.28	$ 53.00	$ 1,800.00	4
32D02	Joannes	0006	0044	CDC 1094820	27/05/2002	26/05/2012	26/03/2012	42.66	$ -	$ 53.00	$ 1,800.00	4
32D02	Joannes	0006	0055	CDC 1121309	08/04/2003	07/04/2013	05/02/2013	42.81	$ -	$ 53.00	$ 1,800.00	4
32D02	Joannes	0006	0056	CDC 1121310	08/04/2003	07/04/2013	05/02/2013	42.82	$ -	$ 53.00	$ 1,800.00	4
32D02	Joannes	0006	0057	CDC 1121311	08/04/2003	07/04/2013	05/02/2013	42.82	$ 35,416.70	$ 53.00	$ 1,800.00	4
32D02	Joannes	0006	0058	CDC 1121312	08/04/2003	07/04/2013	05/02/2013	42.84	$ -	$ 53.00	$ 1,800.00	4
32D02	Joannes	0006	0059	CDC 1121313	08/04/2003	07/04/2013	05/02/2013	42.86	$ -	$ 53.00	$ 1,800.00	4
32D02	Joannes	0006	0060	CDC 1121314	08/04/2003	07/04/2013	05/02/2013	42.88	$ -	$ 53.00	$ 1,800.00	4
32D02	Joannes	0008	0009	CDC 2037886	07/12/2006	06/12/2012	06/10/2012	42.57	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0010	CDC 2037887	07/12/2006	06/12/2012	06/10/2012	42.56	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0011	CDC 2037888	07/12/2006	06/12/2012	06/10/2012	42.56	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0012	CDC 2037889	07/12/2006	06/12/2012	06/10/2012	42.56	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0013	CDC 2037890	07/12/2006	06/12/2012	06/10/2012	42.55	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0014	CDC 2037891	07/12/2006	06/12/2012	06/10/2012	42.56	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0042	CDC 2130556	17/10/2007	16/10/2013	16/08/2013	42.55	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0044	CDC 2130557	17/10/2007	16/10/2013	16/08/2013	42.52	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0046	CDC 2130558	17/10/2007	16/10/2013	16/08/2013	42.54	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0041	CDC 2130559	17/10/2007	16/10/2013	16/08/2013	42.52	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0043	CDC 2130560	17/10/2007	16/10/2013	16/08/2013	42.52	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0045	CDC 2130561	17/10/2007	16/10/2013	16/08/2013	42.49	$ -	$ 53.00	$ 1,200.00	2
32D02	Joannes	0008	0047	CDC 2208070	26/02/2010	25/02/2014	26/12/2013	42.49	$ -	$ 53.00	$ 1,200.00	1
32D02	Joannes	0007	0044	CDC 50082	14/01/2005	13/01/2013	13/11/2012	42.6	$ -	$ 53.00	$ 1,800.00	3
32D02	Joannes	0007	0045	CDC 50083	14/01/2005	13/01/2013	13/11/2012	42.6	$ -	$ 53.00	$ 1,800.00	3
32D02	Joannes	0007	0046	CDC 50084	14/01/2005	13/01/2013	13/11/2012	42.6	$ -	$ 53.00	$ 1,800.00	3
32D02	Joannes	0006	0038	CL 3181861	16/11/1971	29/10/2012	29/08/2012	20	$ 34,162.56	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0037	CL 3181862	16/11/1971	29/10/2012	29/08/2012	20	$ 43,118.96	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0036	CL 3181863	16/11/1971	29/10/2012	29/08/2012	20	$ 43,118.96	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0039	CL 3206811	16/11/1971	29/10/2012	29/08/2012	20	$ 28,291.47	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0039	CL 3206961	16/11/1971	30/10/2012	30/08/2012	20	$ 41,523.15	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0038	CL 3206962	16/11/1971	30/10/2012	30/08/2012	20	$ 41,523.15	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0037	CL 3206963	16/11/1971	30/10/2012	30/08/2012	20	$ 41,523.15	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0036	CL 3206964	16/11/1971	30/10/2012	30/08/2012	20	$ 34,323.15	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0026	CL 3209271	08/02/1972	22/01/2013	22/11/2012	40	$ 164,103.98	$ 53.00	$ 2,500.00	11
32D02	Joannes	0006	0027	CL 3209272	08/02/1972	22/01/2013	22/11/2012	40	$ 164,103.98	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0040	CL 3540201	05/01/1976	28/11/2012	28/09/2012	20	$ 41,678.03	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0041	CL 3540202	05/01/1976	28/11/2012	28/09/2012	20	$ 44,078.03	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0040	CL 3540203	05/01/1976	28/11/2012	28/09/2012	20	$ 22,193.42	$ 27.00	$ 1,000.00	11

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32D02	Joannes	0006	0041	CL 3540204	05/01/1976	28/11/2012	28/09/2012	20	$ 45,278.03	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0028	CL 3705201	01/11/1977	15/10/2013	15/08/2013	20	$ 18,409.47	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0029	CL 3705202	01/11/1977	15/10/2013	15/08/2013	20	$ 18,409.47	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0030	CL 3705203	01/11/1977	15/10/2013	15/08/2013	20	$ 18,409.47	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0031	CL 3705204	01/11/1977	15/10/2013	15/08/2013	20	$ 18,409.47	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0032	CL 3705211	01/11/1977	15/10/2013	15/08/2013	20	$ 18,532.96	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0033	CL 3705212	01/11/1977	15/10/2013	15/08/2013	20	$ 18,532.96	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0034	CL 3705213	01/11/1977	15/10/2013	15/08/2013	20	$ 18,532.96	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0035	CL 3705221	01/11/1977	16/10/2013	16/08/2013	20	$ 18,498.72	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0036	CL 3705222	01/11/1977	16/10/2013	16/08/2013	20	$ 18,498.72	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0037	CL 3705223	01/11/1977	16/10/2013	16/08/2013	20	$ 18,498.72	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0038	CL 3705224	01/11/1977	16/10/2013	16/08/2013	20	$ 18,498.72	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0039	CL 3705231	01/11/1977	16/10/2013	16/08/2013	20	$ 18,095.97	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0040	CL 3705232	01/11/1977	16/10/2013	16/08/2013	20	$ 18,495.97	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0041	CL 3705233	01/11/1977	16/10/2013	16/08/2013	20	$ 18,495.97	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0026	CL 3707441	15/02/1978	27/01/2013	27/11/2012	40	$ 5,639.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0027	CL 3707442	15/02/1978	27/01/2013	27/11/2012	40	$ 5,639.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0030	CL 3707443	15/02/1978	27/01/2013	27/11/2012	40	$ 5,639.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0031	CL 3707444	15/02/1978	27/01/2013	27/11/2012	40	$ 6,339.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0024	CL 3707591	15/02/1978	27/01/2013	27/11/2012	40	$ 3,239.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0025	CL 3707592	15/02/1978	27/01/2013	27/11/2012	40	$ 5,639.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0028	CL 3707593	15/02/1978	27/01/2013	27/11/2012	40	$ 5,639.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0029	CL 3707594	15/02/1978	27/01/2013	27/11/2012	40	$ 5,639.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0006	0032	CL 3707621	15/02/1978	29/01/2013	29/11/2012	20	$ 22,839.82	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0033	CL 3707622	15/02/1978	29/01/2013	29/11/2012	20	$ 22,839.82	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0034	CL 3707623	15/02/1978	29/01/2013	29/11/2012	20	$ 22,839.82	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0035	CL 3707624	15/02/1978	29/01/2013	29/11/2012	20	$ 10,339.82	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0028	CL 3707651	15/02/1978	29/01/2013	29/11/2012	20	$ 64,814.99	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0029	CL 3707652	15/02/1978	29/01/2013	29/11/2012	20	$ 67,214.99	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0030	CL 3707653	15/02/1978	29/01/2013	29/11/2012	20	$ 67,214.99	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0031	CL 3707654	15/02/1978	29/01/2013	29/11/2012	20	$ 65,608.33	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0020	CL 3708001	15/02/1978	27/01/2013	27/11/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0021	CL 3708002	15/02/1978	27/01/2013	27/11/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0022	CL 3708003	15/02/1978	27/01/2013	27/11/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0023	CL 3708004	15/02/1978	27/01/2013	27/11/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0032	CL 3708141	15/02/1978	28/01/2013	28/11/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0033	CL 3708142	15/02/1978	28/01/2013	28/11/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0034	CL 3708143	15/02/1978	28/01/2013	28/11/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11

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32D02	Joannes	0007	0035	CL 3708144	15/02/1978	28/01/2013	28/11/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0023	CL 3717651	12/06/1978	13/05/2013	13/03/2013	20	$ 90,230.26	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0022	CL 3717652	12/06/1978	13/05/2013	13/03/2013	20	$ 97,430.26	$ 27.00	$ 1,000.00	11
32D02	Joannes	0006	0021	CL 3717653	12/06/1978	13/05/2013	13/03/2013	20	$ 77,630.26	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0032	CL 3887691	17/11/1980	28/10/2012	28/08/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0033	CL 3887692	17/11/1980	28/10/2012	28/08/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0034	CL 3887693	17/11/1980	28/10/2012	28/08/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0035	CL 3887694	17/11/1980	28/10/2012	28/08/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0036	CL 3887701	17/11/1980	28/10/2012	28/08/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0037	CL 3887702	17/11/1980	28/10/2012	28/08/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0038	CL 3887703	17/11/1980	28/10/2012	28/08/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0039	CL 3887704	17/11/1980	28/10/2012	28/08/2012	20	$ 18,389.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0040	CL 3951391	17/11/1980	29/10/2012	29/08/2012	20	$ 18,039.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0041	CL 3951392	17/11/1980	29/10/2012	29/08/2012	20	$ 18,039.85	$ 27.00	$ 1,000.00	11
32D02	Joannes	0007	0042	CL 3951393	17/11/1980	29/10/2012	29/08/2012	40	$ 8,439.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0007	0043	CL 3951394	17/11/1980	29/10/2012	29/08/2012	40	$ 8,439.85	$ 53.00	$ 2,500.00	11
32D02	Joannes	0006	0052	CL 5109718	25/08/1995	24/08/2013	24/06/2013	40	$ -	$ 53.00	$ 2,500.00	8
32D02	Joannes	0006	0054	CL 5109719	25/08/1995	24/08/2013	24/06/2013	40	$ -	$ 53.00	$ 2,500.00	8
32D02	Joannes	0006	0053	CL 5109720	25/08/1995	24/08/2013	24/06/2013	40	$ -	$ 53.00	$ 2,500.00	8
32D02	Joannes	0006	0045	CL 5110884	18/09/1993	17/09/2013	18/07/2013	40	$ -	$ 53.00	$ 2,500.00	9
32D02	Joannes	0006	0046	CL 5110885	18/09/1993	17/09/2013	18/07/2013	40	$ -	$ 53.00	$ 2,500.00	9
32D02	Joannes	0006	0047	CL 5110886	18/09/1993	17/09/2013	18/07/2013	40	$ -	$ 53.00	$ 2,500.00	9
32D02	Joannes	0006	0048	CL 5111706	18/09/1993	17/09/2013	18/07/2013	40	$ -	$ 53.00	$ 2,500.00	9
32D02	Joannes	0006	0049	CL 5111708	18/09/1993	17/09/2013	18/07/2013	40	$ -	$ 53.00	$ 2,500.00	9
32D02	Joannes	0006	0050	CL 5111709	18/09/1993	17/09/2013	18/07/2013	40	$ -	$ 53.00	$ 2,500.00	9
32D02	Joannes	0006	0051	CL 5111710	20/07/1995	19/07/2013	19/05/2013	40	$ -	$ 53.00	$ 2,500.00	8
32D02	Joannes	6	25	CL 5215033	06/12/1999	01/07/2012	01/05/2012	42.55	$ 154,928.30	$ 53.00	$ 1,800.00	4
32D02	Joannes	6	24	CL 5215038	06/12/1999	01/07/2012	01/05/2012	42.6	$ 125,661.29	$ 53.00	$ 1,800.00	4
32D02	Joannes	0007	0023	CL 5217916	30/04/1998	29/04/2012	28/02/2012	40	$ 58,749.11	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0017	CL 5217917	25/06/1999	24/06/2013	24/04/2013	40	$ 8,312.99	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0010	CL 5217918	30/04/1998	29/04/2012	28/02/2012	40	$ -	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0011	CL 5217919	30/04/1998	29/04/2012	28/02/2012	40	$ -	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0018	CL 5219048	30/04/1998	29/04/2012	28/02/2012	40	$ -	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0019	CL 5219049	30/04/1998	29/04/2012	28/02/2012	40	$ -	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0020	CL 5219050	30/04/1998	29/04/2012	28/02/2012	40	$ -	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0021	CL 5219051	30/04/1998	29/04/2012	28/02/2012	40	$ 5,098.31	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0022	CL 5219052	30/04/1998	29/04/2012	28/02/2012	40	$ 5,098.30	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0016	CL 5226673	25/06/1999	24/06/2013	24/04/2013	40	$ 40,177.48	$ 53.00	$ 2,500.00	6

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32D02	Joannes	0007	0015	CL 5226674	21/09/1999	20/09/2013	21/07/2013	40	$ 25,978.07	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0014	CL 5226675	21/09/1999	20/09/2013	21/07/2013	40	$ 27,767.38	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0013	CL 5226676	21/09/1999	20/09/2013	21/07/2013	40	$ 1,302.22	$ 53.00	$ 2,500.00	6
32D02	Joannes	0007	0012	CL 5226677	21/09/1999	20/09/2013	21/07/2013	40	$ 11,481.07	$ 53.00	$ 2,500.00	6
32D02	Bousquet	0029	0013	CL 5271785	04/03/2009	03/03/2013	01/01/2013	16	$ 3,323.48	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0014	CL 5271786	04/03/2009	03/03/2013	01/01/2013	16	$ 2,839.10	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0015	CL 5271787	04/03/2009	03/03/2013	01/01/2013	16	$ 2,977.49	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0014	CL 5271788	04/03/2009	03/03/2013	01/01/2013	16	$ 4,430.66	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0015	CL 5271789	04/03/2009	03/03/2013	01/01/2013	16	$ 3,531.09	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0016	CL 5271790	04/03/2009	03/03/2013	01/01/2013	16	$ 5,468.65	$ 27.00	$ 500.00	1
32D02	Bousquet	0028	0015	CL 5271791	04/03/2009	03/03/2013	01/01/2013	16	$ 3,392.69	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0006	CL 5275542	04/03/2009	03/03/2013	01/01/2013	16	$ 2,908.29	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0005	CL 5275543	04/03/2009	03/03/2013	01/01/2013	16	$ 2,700.71	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0004	CL 5275544	04/03/2009	03/03/2013	01/01/2013	16	$ 2,631.50	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0004	CL 5275545	04/03/2009	03/03/2013	01/01/2013	16	$ 2,977.50	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0003	CL 5275546	04/03/2009	03/03/2013	01/01/2013	16	$ 2,562.30	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0003	CL 5275547	04/03/2009	03/03/2013	01/01/2013	16	$ 2,562.30	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0002	CL 5275548	04/03/2009	03/03/2013	01/01/2013	16	$ 2,562.30	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0002	CL 5275549	04/03/2009	03/03/2013	01/01/2013	16	$ 3,115.89	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0001	CL 5275550	04/03/2009	03/03/2013	01/01/2013	16	$ 2,977.50	$ 27.00	$ 500.00	1
32D02	Bousquet	0028	0014	CL 5276061	04/03/2009	03/03/2013	01/01/2013	11.2	$ 2,404.79	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0007	CL 5276062	04/03/2009	03/03/2013	01/01/2013	16	$ 3,600.27	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0007	CL 5276063	04/03/2009	03/03/2013	01/01/2013	16	$ 3,254.28	$ 27.00	$ 500.00	1
32D02	Bousquet	0028	0007	CL 5276064	04/03/2009	03/03/2013	01/01/2013	16	$ 3,115.89	$ 27.00	$ 500.00	1
32D02	Bousquet	0027	0007	CL 5276065	04/03/2009	03/03/2013	01/01/2013	16	$ 3,185.08	$ 27.00	$ 500.00	1
32D02	Bousquet	0026	0007	CL 5276066	04/03/2009	03/03/2013	01/01/2013	16	$ 3,185.08	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0008	CL 5276067	04/03/2009	03/03/2013	01/01/2013	16	$ 3,531.08	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0008	CL 5276068	04/03/2009	03/03/2013	01/01/2013	16	$ 4,015.45	$ 27.00	$ 500.00	1
32D02	Bousquet	0028	0008	CL 5276069	04/03/2009	03/03/2013	01/01/2013	16	$ 3,669.46	$ 27.00	$ 500.00	1
32D02	Bousquet	0027	0008	CL 5276070	04/03/2009	03/03/2013	01/01/2013	16	$ 3,185.07	$ 27.00	$ 500.00	1
32D02	Bousquet	0026	0008	CL 5276071	04/03/2009	03/03/2013	01/01/2013	16	$ 2,908.28	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0009	CL 5276072	04/03/2009	03/03/2013	01/01/2013	16	$ 3,600.27	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0009	CL 5276073	04/03/2009	03/03/2013	01/01/2013	16	$ 3,185.07	$ 27.00	$ 500.00	1
32D02	Bousquet	0028	0009	CL 5276074	04/03/2009	03/03/2013	01/01/2013	16	$ 3,046.68	$ 27.00	$ 500.00	1
32D02	Bousquet	0027	0009	CL 5276075	04/03/2009	03/03/2013	01/01/2013	16	$ 2,631.49	$ 27.00	$ 500.00	1
32D02	Bousquet	0026	0009	CL 5276076	04/03/2009	03/03/2013	01/01/2013	16	$ 3,115.88	$ 27.00	$ 500.00	1
32D02	Bousquet	0030	0010	CL 5276078	04/03/2009	03/03/2013	01/01/2013	16	$ 2,977.49	$ 27.00	$ 500.00	1
32D02	Bousquet	0029	0010	CL 5276079	04/03/2009	03/03/2013	01/01/2013	16	$ 2,977.49	$ 27.00	$ 500.00	1

NTS	Township	RANG	LOT	Title	Entry Date	Expiry Date	Renewal Date	Area (Ha)	Credits	Mining Right	Work Required	Renewals
32D02	Bousquet	0027	0010	CL 5276080	04/03/2009	03/03/2013	01/01/2013	16	$ 2,631.49	$ 27.00	$ 500.00	1
32D02	Bousquet	0026	0010	CL 5276081	04/03/2009	03/03/2013	01/01/2013	16	$ 3,046.68	$ 27.00	$ 500.00	1
32D02	Bousquet	0027	0011	CL 5276082	04/03/2009	03/03/2013	01/01/2013	16	$ 2,839.09	$ 27.00	$ 500.00	1
32D02	Bousquet	0026	0011	CL 5276083	04/03/2009	03/03/2013	01/01/2013	16	$ 3,046.68	$ 27.00	$ 500.00	1

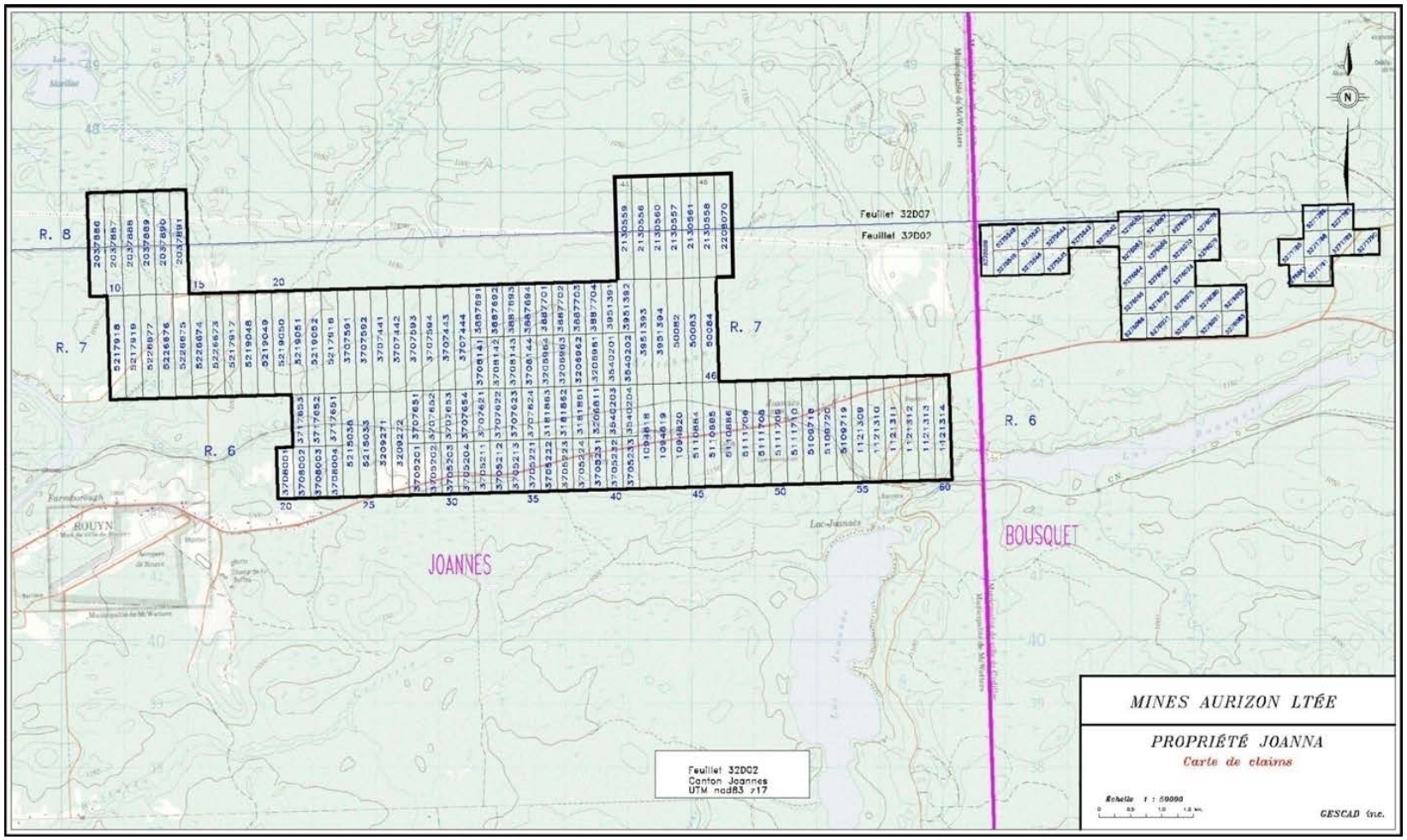


Figure 4.3: Joannes and Bousquet Township Claims Map

Aurizon owns all the claims composing the Property. Details of Aurizon's interests and commitments are summarised in Table 4.2.

Table 4.2 : Details of Aurizon's Interests and Commitments

Claims	Ownership/Title	Royalties	Remaining Payments to Exercise	Remaining Work Commitments
Original Joanna (67 Claims) [1]	100%	2% NSR[2]	—	—
Private Owner (10 claims)	100%	—	—	—
Henriksen (20 Claims)	100%	2% NSR	--	—
Vantex (Heva) (2 Claims)	100%	2.5% NSR[3]	—	—
Alexandria (19 claims)	100%	2% NSR[4]		--
Bousquet (38 claims)	100%	—	—	—

Notes:

1) Forty-two (42) of these claims comprise the Hosco block and 25 of these claims form part of the Heva block.
2) An advance royalty of $500,000 is payable upon completion of a final Feasibility Study and a receipt of a decision by a financial institution to finance the development of the property to commercial production.
3) Aurizon has an option to purchase 0.75% of NSR for $500,000. Advance royalties of $0.7 million are payable upon completion of a final Feasibility Study and $0.8 million upon achieving commercial production.
4) Also a 2% gross overriding receipts royalty on diamonds. Aurizon has an option to purchase 1% of the NSR for $2 million.

Details of the legal agreements are available upon request at Aurizon Mines office. As per conversation with management, all payments and obligations of Aurizon to third parties are in good standing.

4.3 Royalties Obligations

As described in Section 4.2, the Property is subject to four separate royalties. The first concerns the 67 original Joanna claims where a 2% NSR is retained by 90569 Canada Inc. for 1.5% and 0.5% to Iamgold Corporation. The mineral reserves associated with the Hosco deposit are located within this 67 claim block. The second royalty related to the 20 Henriksen claims where Crus Tal Exploration has rights to 2% NSR. The third royalty is a 2.5% NSR retained by Resources Vantex for 1.5% and private owners for 1% on the two (2) Vantex claims where 0.75% of the Vantex Royalty can be repurchased by Aurizon for $500,000. The last royalty concerns the 19 Alexandria claims where private owners have rights to 2% NSR in addition to a 2% gross overriding receipts royalty on diamonds (on 13 claims). Aurizon has an option to buy back 1% of the NSR for $2 million.

4.4 Permits and Environmental Liabilities

Aurizon is conducting exploration work under valid intervention permits delivered by the Ministère des Ressources Naturelles et Faune (“MRNF”) which are renewed by the MRNF before each exploration campaign. Aurizon received a Certificate of Authorization from the Ministère du Développement Durable, de l’Environnement et des Parcs du Québec (“MDDEP”) for the construction of access roads in poorly drained areas which represent 75% of the property surface.

According to MRNF, there are no environmental liabilities related to the historical mining operations at Hosco and Heva due to the fact that there was no mineral processing and/or waste disposal facilities at the Property. There are no other environmental liabilities pertaining to the Property according to Aurizon management.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Physiography

The topography of the area is generally flat with a gentle downward slope toward the north. The land drainage patterns flow westward by small creeks. Outcrops occur north and mainly south of the main mineralization corridor observed on the Property. The overburden consists of glacial-fluvial till and lacustrine deposits and is generally less than 15 m thick. Elevation in the area averages 310 m above sea level.

Vegetation of the surrounding area is characterized by grassy swamps in topographic lows, mainly to the north and trembling aspens and balsam poplars in forested areas. Formations of balsam fir trees in pure settlements or associated white spruce and, to a lesser extent, black spruce are also present on the Property. Most of the mineralized zones do not outcrop and are situated under swampy terrain.

5.2 Accessibility

The Joanna project is located in the Joannes and Bousquet Townships in the Province of Québec. It is situated 20 km east of the city of Rouyn-Noranda, just north of Highway 117. In the Joannes Township, the Property is accessible via a gravel road to the covered old shaft collar of the Hosco mine. In the Bousquet Township, the side road to St Norbert du Mont-Brun and the Doyon mine crosses the Property. The Property is well located regarding access (air, road, and train), electricity, water and manpower.

5.3 Climate

The climatic data used to characterize the sector under study comes from the meteorological station of Val-d'Or, Québec. These observations were carried out during 1961-1991. Exploration work in the area can typically be carried out year-round but soft ground in the areas covered by wetlands creates difficult working conditions from late spring until early winter.

In the Val-d'Or region, the average daily temperature is slightly above the freezing point, i.e. 1.6°C. The average temperature during July reaches 17°C, while the temperature in January falls to -16°C.

Precipitation averages 928 mm of water annually in the area. Average monthly precipitation ranges from 48 mm in February to 103 mm in September. Snow falls from October to April, but is much more significant from November to March. The average for these five months is 26 mm using a snow to water conversion factor. The pH of the precipitations measured at the Joutel station in 1991 varies from 4.30 in November to 4.78 in June.

The anemometric data collected in Val d'Or between 1961 and 1991 indicates that from June to January, the southwest winds are dominant, whereas from February to May, the northwest winds are more frequent. In this sector, the winds have an average velocity varying between 11 and 14 km/h.

5.4 Local Resources and Infrastructures

The regional resources regarding labour force, supplies and equipment, are sufficient, the area being well served by geological and mining service firms. The city of Rouyn-Noranda, with more than 40,000 citizens, is a regional center for the Western Abitibi region and has the necessary infrastructures and workforce to support a mining operation. While there is currently a general shortage of qualified personnel in the mining and exploration sector, the location of the Project is favourable in that regard. The area is traditionally a mining area with several operating mines and active exploration companies. All major services are available in Rouyn-Noranda and Val d'Or.

The existing infrastructures on the Joanna property consist mainly in an access gravel road from Highway 117 which leads to the site near the old shaft collar of the Hosco mine. Some concrete foundations remain as witness of the past mining operations. The existing inclined shaft and ramp could be used to perform underground exploration activity although there is no indication at this stage that their integrity would allow it. The north-south gravel road crosses an east-west railway which is not located on the identified gold mineralization. A recently refurbished local electrical distribution is available at the former Hosco shaft from the power line on the nearby Highway 117. A 120 kV power line passes 2.7 km north of the Hosco deposit.

6 HISTORY

Reference to the mineral resource and mineral reserves reported in this section is provided only to comply with the requirement in NI 43-101 to provide such historical information on the Property and such estimates should not be relied upon. Readers are cautioned that these estimates pre-date NI 43-101 and are not comparable to estimates prepared in accordance with the mineral resource and mineral reserves categories required by NI 43-101. Neither Aurizon nor the authors of this Report have verified any of the data upon which such estimates were based. Original numbers stated in imperial units have been converted to the metric system for comparative purpose.

6.1 Hosco

The following is a summary of the exploration work history compiled for the Hosco sector of the Property.

The principal source of the historical information for the Hosco sector is Descarreaux (1985). The Hosco sector of the Property has been explored over the last sixty years by various owners and optionees. A summary of the historical exploration work is as follows:

1944-1945: Hosco Gold Mines did a magnetic survey and 81 diamond drillholes totalling approximately 20,000 m.

1946: Sinking of an inclined shaft (55°) to a vertical depth of 131 m. Three underground levels opened, i.e. 40 m, 80 m and 120 m.

1946-1947: Drifting and crosscutting on the three levels, i.e. 95 m on level 40, 1042 m on level 80 and 1262 m on level 120 for a total of 2399 m.

1948-1949: Production at a rate of 100 tpd. A total of 45,872 t grading 6.58 g/t Au was extracted from nice (9) shrinkage stopes and milled at the nearby mill of McWatters Gold Mines Ltd. Mill head grade was 4.90 g/t Au.

1949: Production stopped. Remaining proven ore reserves of 75,344 t grading 9.60 g/t Au.

1972: The property was staked by Yvon and Gaston Vezina.

1973: Ore reserves calculations by Derry, Michener and Booth: probable and possible reserves amount to 954,556 t grading 5.14 g/t Au. SGS Geostat and Maxime Dupéré, P. Geo., Qualified Person have not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves and do not consider, and Aurizon is not treating, this historical estimate as current mineral resources or mineral reserves. The above historical estimates should be considered for reference purposes only.

1979-1983: Soquem acquired the mining rights. The surface geology was mapped and the base line resurveyed. Seven (7) holes totalling 1,128 m were drilled.

1983: The Property is transferred to Cambior.

1984: SASU Investments Inc. acquired an option on the Hosco Property and drilled ten (10) holes totalling 2,988 m. New ore reserves calculations by Louvicourt Mining Management Company Ltd.

1985: Jean Descarreaux and Associates Ltd. completed an economic assessment and review of the SOQUEM and Louvicourt Mining Management reserve estimates.

1986-1987: Louvicourt Mining Management Ltd., agent for Eastern Mines Ltd. and Silver Sceptre Resources Ltd. completed a $4.1 M exploration program consisting of 9,798 m of diamond drilling to depths of 100 m, 389 m of overburden drilling, 484 m of exploration ramping to a vertical depth of 107 m, 392 m of crosscutting, 238 m of drifting, 20 m of raising, and extraction of approximately 21,555 tonnes at 2.65 g/t Au of mineralization stockpiled on surface. This stockpile remains in place on surface. Metallurgical testing of four (4) 10 kg drill core samples by the

Canmet and the Centre de Recherche Minerales in Sainte-Foy, Quebec and metallurgical testing of one composite core sample by Lakefield Research was also completed. A sample of the Lakefield flotation concentrate was sent to Hydrochem Developments for an evaluation of the NITROX process on the Hosco gold mineralization.

1997: The access ramp was blocked and the portal was filled in at the request of the Minister of the Environment of Quebec for security purposes.

1998-2004: 1149127 Ontario Inc. acquired 100% interest in the mining claims. There are no outstanding obligations on these claims except for a 1% net smelter royalty held by Cambior (Vallée 2004). 1149127 Ontario Inc. property transferred to 90569 Canada Inc. in a name change.

2006: Aurizon Mines Ltd. optioned the Property in June from 90569 Canada Inc. A 2% net smelter royalty is held by 90569 Canada Inc. and Iamgold-Qc (formerly Cambior).

6.2 Heva

The following is a summary of the exploration history compiled for the Heva sector of the Property. The principal source of the following historical information is Mr. Chris Davis M.Sc., P.Geo's report of August 15th, 2004 "Review and Assessment of the Hosco-Heva Gold Property". The Heva sector has been explored over the last sixty years by various owners and optionees. A summary of the historical exploration work is as follows:

1944-1945: Heva Cadillac Gold Mines Ltd. Drilled 56 diamond drillholes totalling 9,960 m. A-50 degrees N inclined three compartment shaft was excavated to a vertical depth of 122 m. Underground work consisted of drifting and crosscutting on 84 m and 122 m levels.

1946: Company name change to Heva Gold Mines Ltd.

1947: The shaft was deepened to a vertical depth of 177 m and the 160 m level was excavated.

1948: Mines and Resources Canada completed an investigation of the Heva gold mineralization. The report indicated that the 99% of the gold could be recovered by direct cyanidation. Seven (7) underground drillholes were completed from the 122 m level. Nine surface drillholes were completed. Operations are suspended and the mine is allowed to fill with water.

1951-1953: The mine is dewatered and underground operations are resumed. A total of 47,475 tonnes of ore grading 6.86 g/tonne Au was produced before the operations are stopped due to a shortage of mine labour. A total of 960 m of drifting and crosscutting was completed from 1946 to 1953.

1975: Yvon and Gaston Vezina acquire the mining rights.

1978-1983: SOQUEM Exploration options the Property. The surface geology is mapped and the base line resurveyed. A total of 6,920 m of surface diamond drilling, detailed mineralogical study and humus geochemical survey are completed.

1984: SASU Investments Inc. acquires an option to the Property.

1985: New Goldcore Ventures and Amberquest Resources Ltd option the Heva Block.

Norgold Management Ltd. agent for New Goldcore Ventures and Amberquest

Resources Ltd hired Ovaltex Inc. to complete twenty eight (28) surface diamond drillholes totalling 7,967 m.

1986-July 1987: Louvicourt Mining Management, agent for Eastern Mines Ltd. And Silver Sceptre Resources Ltd. completed 11,126 m of surface diamond drilling and installation of the following surface infrastructure; access road, powder storage, sedimentation basin, septic system, new power line, service building, dewatering facilities, hoist and a head frame in preparation for completion of an advanced underground exploration program. The program was suspended on July 28th, 1987 due to surface overburden caving into stope 2-15 shortly after dewatering had started.

August 1987-1988: Starting on August 1, 1987, Louvicourt Mining Management Ltd., agent for Eastern Mines Ltd. And Silver Sceptre Resources Ltd. completed a revised advanced exploration program. It consisted of 7,614 m of surface diamond drilling, and an evaluation of the 2-15 cave in area, sinking of vertical three compartment 7 m by 3 m shaft to a depth of 227 m, 260 m of cross cutting, 122 m of drifting on the new 200 m level and 148 m of underground bazooka drilling. The 2-15 stope assessment consisted of a seismic survey, overburden drilling, thirty two (32) diamond drillholes and an overburden analysis. Shaft stations were excavated to correspond to the same levels within the old workings and a 40 m pillar was established above the new 200 m level. A loading pocket was established on the 200 m level. A total of 1,386 tonnes of mineralization was extracted and piled on surface.

1999: T. P. O'Connor acquired Lots 24 and 25 in Range VI of Joannes Township

1998-2004: 1149127 Ontario Inc. acquires 100% interest in its mining claims. There are no outstanding obligations on these claims except for a 1% net smelter return royalty held by Cambior (Vallée 2004).

2004: Minerals right are transferred to Vantex Resources Limited from O'Connor and Gauthier.

2005: Stellar Pacific Ventures signs an agreement to carry out exploration work and acquire 25% of the property.

2007: Aurizon acquired an option on 100% of the 75% ownership interest of Vantex Resources Limited in the two Heva claims.

2009: On September 28, 2009 Aurizon purchased the remaining 25% interest in the Heva claims owned by Stellar Pacific Ventures.

6.3 Alexandria

1935: James Thompson and Associates carried out stripping and trenching on silicified mineralization in greywacke.

1937: Clericy Consolidated Mines Limited prospected range VI, lots 52-62 + range VII, lots 56-62. Five (5) diamond drillholes were drilled for a total of 448.7 m.

1940-1943: Belleterre Quebec Mining Limited drilled two diamond drillholes on range VI, lots 42, 43.

1944-1946: Hosco Gold Mines Limited drilled thirty eight (38) holes for a total of 6,094.73 metres.

1945-1946: Bouzan Gold Mines Limited drilled eighteen (18) diamond drillhole (B1 to B20) for a total of 2,884.93 metres.

1946: Belleterre Quebec Mines Limited drilled four (4) diamond drillholes.

1966-1969: W.L. Landgridge Jr. completed some trenching, pit digging and sampling on range VI and VII, lots 42-55.

1974-1979: Darius Gold Mines Limited dug nine (9) trenches and drilled one hole and conducted geological geophysical and geochemistry surveys on Range VI, N1/2 of lots 42-47.

1981: J. Beaulieu drilled a 225 ft vertical hole.

1981-1983: Sulpetro Minerals Limited carried-out line-cutting, geophysical survey and geological work.

2003: Alexandria Minerals Corporation signed an option with Coyle and Tremblay for a 100% interest on thirteen (13) claims directly east of the Hosco sector.

2004: Alexandria Minerals Corporation signed an option with Salmasi and Greisbach for a 100% interest on six (6) claims directly east of the thirteen (13) above.

2007: Alexandria Minerals Corporation drilled three holes to test the Cadillac Break and Sheean shear, for a total of 634 metres.

2008: Aurizon signs an option to acquire 100% of the Alexandria property.

6.4 Henriksen

The history of the Henriksen sector was taken, translated, and summarised from the rapport Sommaire des travaux d'exploration 2007-2009 and from the 2010 Annual information form (AIF) (see references).

1924-1937: Some geological surveys included mapping, trenching and pitting by different operators. Eleven (11) exploration holes were drilled during 1937 by Joannes-Davidson M.L.

1938-1952: Teck Exploration Co. Ltd. acquired rights on the Property and drilled eight (8) holes (1200 m). A shaft was sunk on a gold bearing structure east of the Davidson Creek Fault.

1952-1974: Joannes-Davidson acquired 100% of the property rights. One hole was drilled. Geological Mapping and geophysical surveys were conducted.

1977-1990: Gold Fields Inc./Darius Gold Mine Inc. conducted a geophysical survey and five (5) holes were drilled.

1992-1996: Agnico Eagle staked the northern part of the Property and drilled one (1) hole. Geological and geophysical surveys were done including mapping, IP and magnetic surveys.

1998-2007: Current claims block is staked by Gordon Henriksen. Geological and geophysical surveys were done including ground magnetic IP and VLF surveys. Ressources minières Coleraine drilled three (3) holes for 423 m.

2007: Aurizon signed an option with Gordon Henriksen to acquire 100% of the claims group.

6.5 Bousquet

The Claims acquired by Aurizon in the Bousquet sector (formerly La Pause sector) of the Bousquet Township have been explored over the last sixty (60) years by various owners and optionees. Information regarding the history of operations on the Bousquet sector was taken

and translated from the internal report: Rapport de travaux statutaires 2009-2010 and from Boudreault (2009) (see references).

1947: Geological survey by The Consolidated Mining and Smelting Company of Canada.

1964: Geophysical and geological work on the NE block by East Sullivan Mines Ltd.

1966: Falconbridge Nickel Mines Ltd. performed a magnetometer survey covering the southern part of the property. It is likely that mapping and sampling occurred, however no information regarding this can be found.

1971: Ministry of Natural Resources of Quebec performs an aerial survey of the Malartic area.

1986: La Pause Resources Inc. completed a VLF electromagnetic survey and a magnetic survey measuring the total field and the vertical gradient on 4 claims close to highway 117.

1986-1987: Relevés Géophysiques Inc. processed the geophysical data from 1986.

1988: IP and EM surveys were completed on the property now belonging to Ecudor Mining Society Inc. Geological surveying and sampling was completed by Platinor Mining Resources Inc. The study area covered most of the East Block.

1989: Mr. Chouinard owns the western half of the property and produced a VLF EM and magnetic survey. Glenn Griesbach was hired to perform mapping, trenching, sampling, and compiling a report.

1989: Ecudor Mining Society Inc. appointed Dominique Beaudry to evaluate the gold potential of the Bousquet property. The exploration program consisted of sampling mineralized areas and lithological studies.

1995: Platinor Mining Resources Inc. completed an IP survey on what is now the East Block.

1998: Randon Ferderber carried out a Beep Mat survey and mapping on a section of the Labour property (West Block).

2003: The Department of Natural Resources, Wildlife and Parks produced geological maps based on more recent surveys.

From what precedes, it looks like no drilling has ever taken place on the Aurizon claims Bousquet sector in the Bousquet Township. From 1978 to 1989, Long Lac Exploration conducted some geological, geophysical and geochemical work and did some drilling on a property to the north with a south extremity crossing the Bousquet sector claims. Only some line cutting as well as a magnetometric survey was done on the southern extremity crossing the Bousquet sector claims.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting

The following information is based on observations by Aurizon geologists and contract geologists involved in field work and petrographic studies, which were supervised by Martin Demers P.Geo, Exploration Manager of Aurizon and Qualified Person on the project.

7.1.1 Regional Geology

The Joanna property (formerly Hosco-Heva-Alexandria) is situated in Joannes Township in the south central portion of the Abitibi Greenstone Belt, within the Superior structural province of the Canadian Shield. All rocks are Archean in age except for the late crosscutting Proterozoic diabase dykes. A wide variety of syn-volcanic to late tectonic intrusive rocks of intermediate to felsic composition occur throughout the region, forming large batholiths or minor intrusions controlled by regional faults.

The Rouyn-Noranda mining district is well known for its polymetallic volcanogenic massive sulphides deposits associated with the Blake River Group but several gold-only deposits in the immediate vicinity of the Cadillac Break have been defined over the years. Thick sequences of Archean lavas of the Blake River Group yield ages between 2703 and 2696 Ma (Mortensen, 1993; Lafrance et al., 2005). The recent isotopic data coupled with stratigraphic relationships suggest that the various formations composing the Blake River Group do not represent a simple stratigraphic pile, but rather different geographically isolated, synchronous volcanic complexes (Lafrance et al., 2005). Volcanism in this area tends to be bimodal, demonstrated by successions of mafic and felsic lavas with affinities varying between tholeiitic and calcalkaline. Pyroclastic equivalents of mafic to felsic lavas occur intercalated with massive flows.

Younger flysh-type sediments, such as the Cadillac and Kewagama group, are about 10 million years younger than the youngest volcanic units (Gouthier. 2007). They sit on top of volcanic units and form continuous units, generally in an East-West direction. Locally, younger fluviatile sedimentary sequences dated between 2679 and 2673 Ma, such as the Temiskaming group, are developed within basins closely associated to the regional faults (Mueller et al, 1996).
Volcanic and sedimentary assemblages are tightly folded and cut by major E-W and NW-SE faults developing a regional losangic or lenticular shaped pattern with mostly sub-vertical north

or south dip. Regional deformation is generally expressed by an east-west, steeply dipping schistosity, sub-parallel to the main folds axial plan. On a regional scale, fold axis tend to be sub-horizontal but can also be more vertical within the major fault corridors (Daigneault et al, 2002).

The main structural feature of the region is the Cadillac or Cadillac-Larder Break. It is a large-scale, regional tectonic feature extending for 200 km from Kirkland Lake, Ontario to Val d'Or, Quebec bordering the Abitibi Greenstone Beltto the south. The structure is divided in different branches forming a succession of a few metres to a few hundred metres wide heterogeneous, ductile deformation zones dipping northward with main decollement following volcanic-sediments and ultramafic units contacts inside various stratigraphic units. A strong carbonate and micas alteration track the deformation intensity. Late brittle-ductile expression of the fault corresponds to talc, chlorite-carbonate schist, or tectonic breccias.

Metamorphism varies from sub-greenschist to greenschist facies throughout the region and increases quickly to amphibolites facies immediately south of the Cadillac Break in the Pontiac Sub-Province.

A number of gold showings occur in the region typically associated with the Cadillac Break. The previously mined McWatters Mine and O'Brien Mine located respectively 10 km west and 50 km east of the Joanna property are previously known as the most significant deposits of the area with past production of approximately 1 million ounces of gold. These mineralized occurrences which are classified as orogenic, are characterized by narrow, discontinuous, syn-tectonic quartz veins mostly conformable with structural grain. Gold bearing alteration halos are prominent with strong replacement by biotite, albite and locally tourmaline. Disseminated arsenopyrite is the main gold carrier in this environment.

7.1.2 Property Geology

The east-west striking zone favourable for gold mineralization is closely related to the Cadillac fault, which dips 55° to the north in this area. According to traditional nomenclature it is underlain from south to north by rocks of the Pontiac, Temiskaming and Cadillac groups.

The complete sedimentary sequence cut by exploration works is composed of poorly graded greywacke interbedded with a main matrix-supported polymict conglomerate unit historically related to the Temiskaming Group depositional environment.

The east-west striking zone favourable for gold mineralization is closely related to the Cadillac Fault, which dips 55 degrees to the north in this area. The main 10 to 20 metres-wide brittle structure, composed essentially of chloritic schist, is included in a much broader ductile deformation zone that goes from about 100 metres north of the fault to more than 500 metres south of it. In general, the principal foliation is conformable to the fault zone and bedding. A tight asymmetric folding pattern with fold hinges gently plunging toward the west is developed in conglomerate units south of the fault.Deformation within the sediments can be difficult to evaluate as a result of re-crystallization of some minerals at superior greenschist to amphibolite facies metamorphism. Some sediment intervals show mineral segregation in a tectonic fabric and sulfides remobilization along foliation planes indicating a strong ductile deformation level. Vein folding and orientation within the different types of zone suggest a gradual decrease in deformation intensity from the Cadillac Fault to the south.

The Cadillac Group, as described in literature by a facies succession of greywacke with mud rock and iron formation units (reef), has yet to be interpreted on the property and its limit is suspected to be located farther north toward the Blake River Group limit. A few diabase dykes also cut the host sedimentary rocks with a kilometric spacing. Figure 7.1 shows the geology in the immediate Property area (taken from Aurizon Mines Ltd., Annual Information Report 2010, see References).

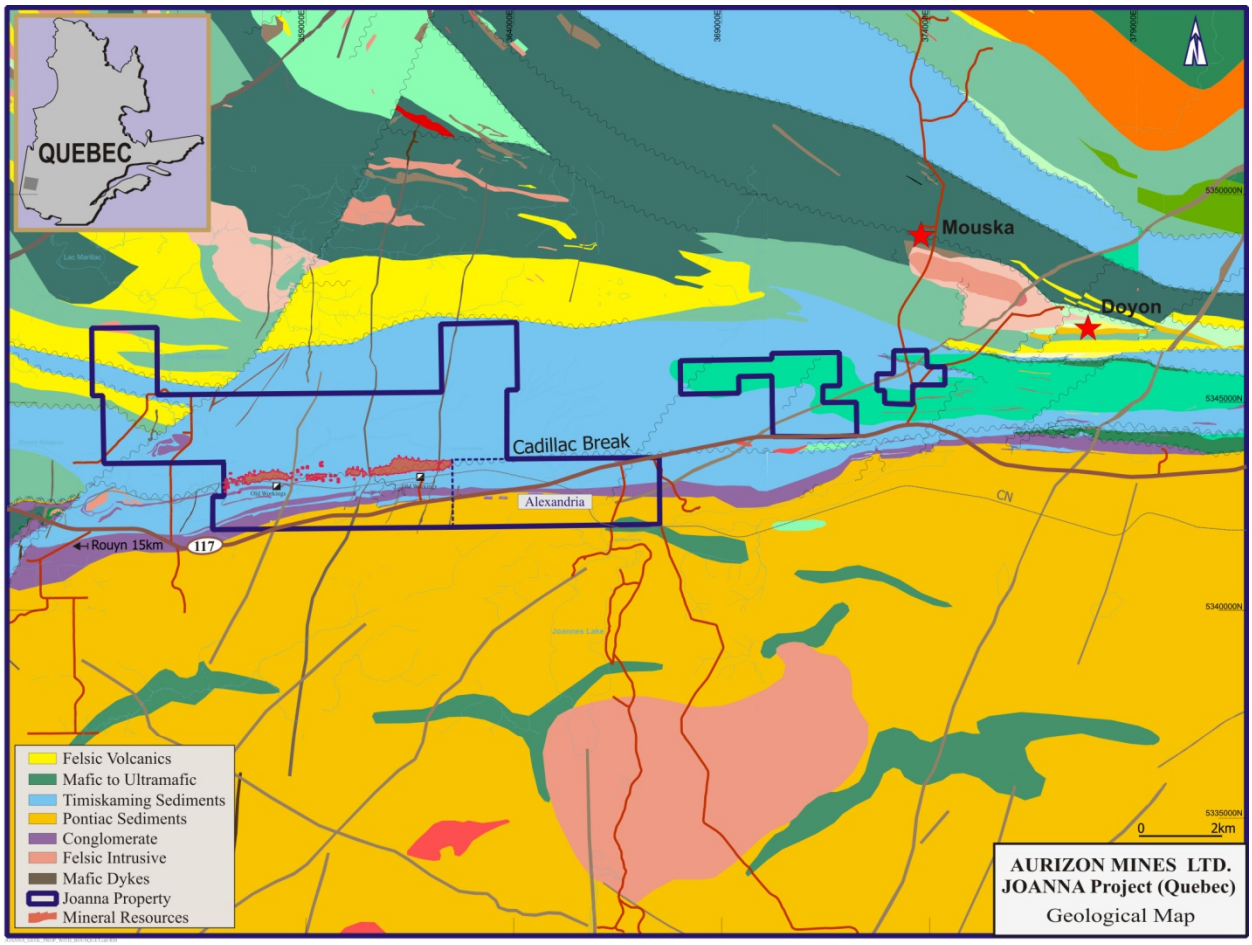


Figure 7.1 : Surface Geology Map Displaying the Joanna Property Boundaries.

7.2 Mineralization

Gold mineralization of the Hosco-Heva area is distributed in multiple lenses on both sides of the Cadillac Fault, making each one a few metres to a few tens-of-metres thick. To date, mineralization has only been identified in the Fault zone. The more continuous and gold-rich zones seem to be associated with a silica-biotite, albite and tourmaline alteration. Mineral assemblage also includes variable concentrations of white mica, chlorite carbonate, garnet and possibly other alumino-silicates. Amphibole and tourmaline rich replacement zones have been locally identified. In the Heva zone, gold mineralization contoured this type of alteration zone. Mineralized lenses are included within a 100 to 190 metre wide lower grade halo of 0.5 to 2.0 grams per tonne.

Generally speaking, most of the zones look alike in terms of structure and mineralization, some differentiate slightly by the content in quartz veins (millimetric to centimetric), arsenopyrite, pyrrhotite, and pyrite but currently they are believed to be all related to the same geological event with variations in the alteration assemblages and degree of deformation. Narrow, widely spaced, quartz veins with higher grades have been intersected on the Heva side and immediately south of the Hosco Zone.

Gold is concentrated in different locations closely related to sulphides grains and is mostly fine grain (<20 micro metres). According to different petrographic and micro-analytical studies performed on rock samples and on a sulphide concentrate sample, the majority of gold weight is under the form of free or attached particles to the surface of sulphides grains. A fraction of gold including sub-microns particles and ionic gold is concentrated in some types of arsenopyrite grains (taken from Aurizon Mines Ltd., Annual Information Report 2010, see References).

7.2.1 Hosco

Gold in the Hosco deposit is closely associated to sulphides and alteration mineral distribution indicating at first view the prime of hydrothermal reaction with hosted sediments. The microscopic in-situ observation of gold particles or concentrations in many alteration contexts and gold enrichment levels suggested that the sulphidation process was the main process involved in the gold precipitation process.

Gold in the deposit is mostly microscopic with a few occurrences of coarser, sub-millimetric particles. Thin section and sulphide concentrates microscopic observation returned a majority of very fine grain particle (average of 8 µm in the concentrate) and about 30% of gold weight is evaluated to be under a colloidal or solution state in sulphides grains. In-situ measurement of gold concentration in individual sulphide grains by SIMS (Secondary Ion Mass Spectrometry) returned gold concentrations at different levels in all groups of sulphides. The maximum average gold grade was obtained in coarse arsenopyrite with 80 g/t. On the other hand, pyrite and pyrrhotite contain about 1 g/t gold (SGS Lakefield 2008).

Apparent zoning of sulphide assemblages across the deposit seems to be related to the fault position. Pyrrhotite, which covers the whole deposit, is mostly concentrated at both edges of the

mineralized system with a strong concentration inside the ductile portion of the fault zone. The main concentration of arsenopyrite is located inside a few ten to hundred meters south of the fault where the most consistent grades are found. Pyrite, usually unnoticeable, is restricted to a narrow portion of the mineralized system immediately south of the fault. According to micro-textures, all sulphides are contemporaneous.

Many factors had probably interacted to explain zoning and gold location from a late tectonic and syn-metamorphic peek metallogeny. For example, alteration, deformation and vein density will influence the porosity and volume of potentially mineralized fluid in circulation. Location of high grade intersections in strongly altered and laminated host sections support this assertion. Also, chemical gradients such as pH, sulphur and arsenic availability in the mineralized environment possibly influenced gold expulsion from sulphide structures and prevented a more complete absorption of gold by the arsenopyrite crystalline structure. Figure 7.2 shows the sulphide distribution across a conceptual profile of the Hosco mineralized zone (Renou, 2009).

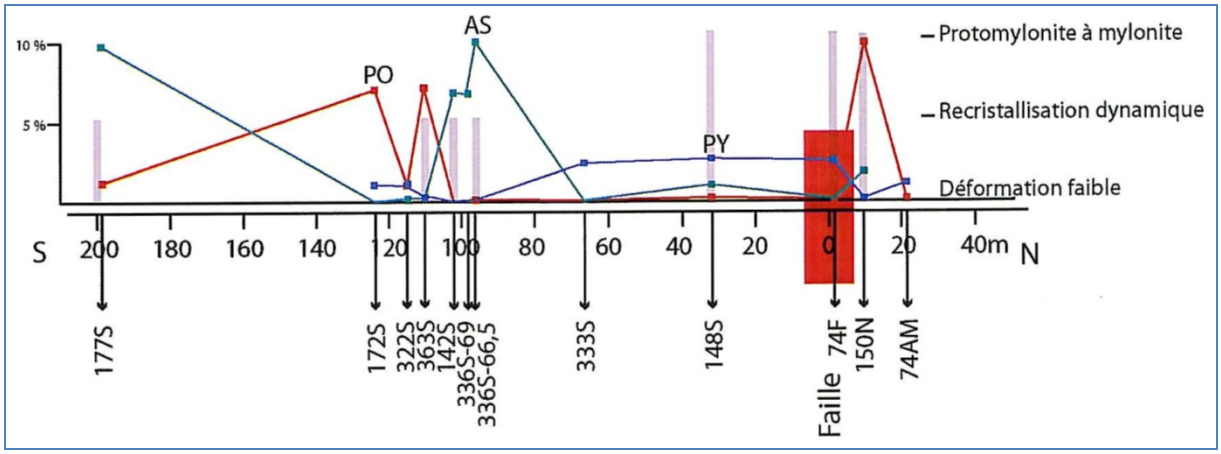


Figure 7.2 : Sulphide Distribution Across a Conceptual Profile of the Hosco Mineralized Zone (Taken from Renou, 2009)

Historical work focused on separating and labelling the narrow but high grade lenses developed immediately south of the fault. These high grade zones are found within the new zones defined by the 2007-2009 Aurizon drilling campaign. The latter was executed with the objective of defining larger mineralized zones and evaluating the open pit potential of lower grade and higher tonnage material. The mineralized corridor extends along a 2,450 m east-west trend and

can be followed down to a depth of 400 m. Mineralized zones within that corridor dip by 50° to 65° to the north with a western plunge. They can be discriminated in 5 types following a combination of features:

1. The North Zone (NZ), a mylonitic unit, occurs just north of the Cadillac fault and is restricted to the western half of the deposit (Hosco block). It is generally represented by a tourmaline-biotite-silica and locally albite-sericite altered greywacke, foliated and injected by up to 20% thin quartz ribbons concordant to tight folding (Figure 7.3). Pyrrhotite is the main sulphide phase with variable arsenopyrite as an accessory; both can represent up to 10% of the material. The zone is often moulded to the north by an amphibolised unit. Occasional porphyroblastic amphiboles can also be seen within the North Zone.

Figure 7.3 : Mineralization in the North Zone (Hole JA-08-200 --Section 8650mE)

2. The South Zone 1 (SZ1) is a minor type of mineralization found just south of the fault and is defined by a high level of deformation (Figure 7.4). It is characterized by thin, grey quartz veins crosscutting thin, competent, tourmaline- rich zones. 5-10% of arsenopyrite can sometimes be observed in semi-massive stringers associated with quartz. Even if some high grade values are associated with this zone, it remains a marginal contributor to the resources of the deposit.

Figure 7.4 : Mineralization in the South Zone 1 (Hole JA-08-200 -- Section 8650mE)

3. South Zone 2 (SZ2) represents the most important gold bearing mineralization. This mineralization can be identified by lower deformation intensity and by its concordant banded aspect caused by the succession of 5-10% of generally sub parallel, millimetric quartz stringers and silica-albite-biotite altered host rock (Figure 7.5). Alteration is also characterised by the presence of sericite, chlorite and carbonate in variable amounts. The colors of these altered bands vary from brown-yellowish to light green. Sulphide content is usually around 5-10% with an assemblage of pyrrhotite-pyrite and arsenopyrite. Arsenopyrite crystals from the SZ2 zone often exhibit a needle shape.

Figure 7.5 : Mineralization in the South Zone 2 (Hole JA-08-200 -- Section 8650mE)

4. The South zone 3 (SZ3) resembles the SZ2 in terms of quartz veining density, level of deformation, alteration, and sulphide content. However, no sericite is present in this zone. Vein contacts are occasionally very diffuse. Even if it remains generally concordant to the deformation zone, veins in SZ3 are often more discordant than those of SZ2. Both zones

show a close spatial association and transitional zones between both are often observed. Figure 7.6 shows visible gold in a quartz vein.

Figure 7.6 : Visible Gold in the South Zone 3 (Hole JA-07-72 -- Section 8975mE)

5. The South Zone 4 (SZ4) is represented by thicker and often more discordant veins composed of white, occasionally smoky, quartz (Figure 7.7). Coarse arsenopyrite in amounts of up to 20% is sometimes associated with these veins. The veining density is highly variable and occasionally takes on a breccia aspect. Some veins sections can reach a metric size. They occur more to the south, isolated from the other zones and in less deformed rocks. They seem to mark the end of the mineralized system in the investigated area. High grade gold samples have been found within this zone but contribution to the bulk of the resources seems marginal.

Figure 7.7 : Mineralization in the South Zone 4 (Hole JA-08-270 -- Section 9150mE)

The different zones do not always respect a distinct, parallel conformable order, and in some sections, SZ2 and SZ3 merge to create a complex series of braid-like lenses parallel to the

Cadillac fault, with variable width and dip extension (please see Figures 7.8 to 7.12) general way, gold bearing envelopes are represented by a single, continuous lens on the north side of the fault and by two or three main lenses on the south side of the fault. The latter is mainly composed of SZ2 with variable amounts of SZ3. Main branches can reach real thickness of up to 50 m. The large envelopes are often interpreted to separate into several thinner branches that can pinch and swell. The largest part of the branches south of the fault form westward plunging ore shoots. The observation on some section of a folded conglomerate layer on both sides of the South Zone may suggest that the distance between mineralized lenses is dependent on a fold opening. Mapping and sampling in the underground openings of the high grade quartz vein zones within the low grade corridors show that those partially mined zones are continuous but with some pinch and swell.

To our knowledge, there is no other significant mineralization on the property which merits modelling and estimation of mineral resources aside from gold. A metal scan on selected control samples from the 2007 data verification program has been done by SGS Geostat to verify the presence of other commodities. Silver values are reported as not significant.

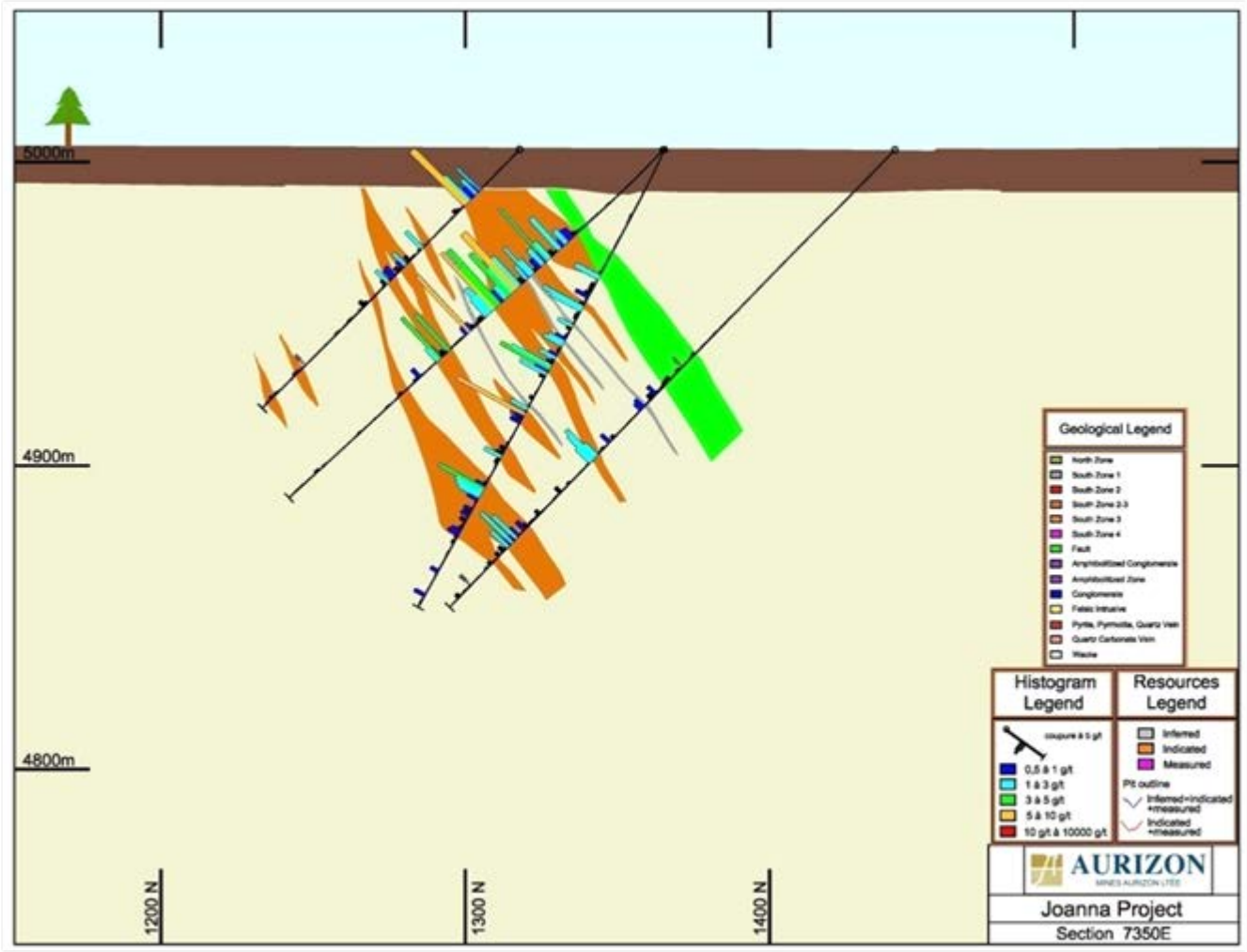


Figure 7.8 : Mineralized Zones at Hosco on Section 7350 mE

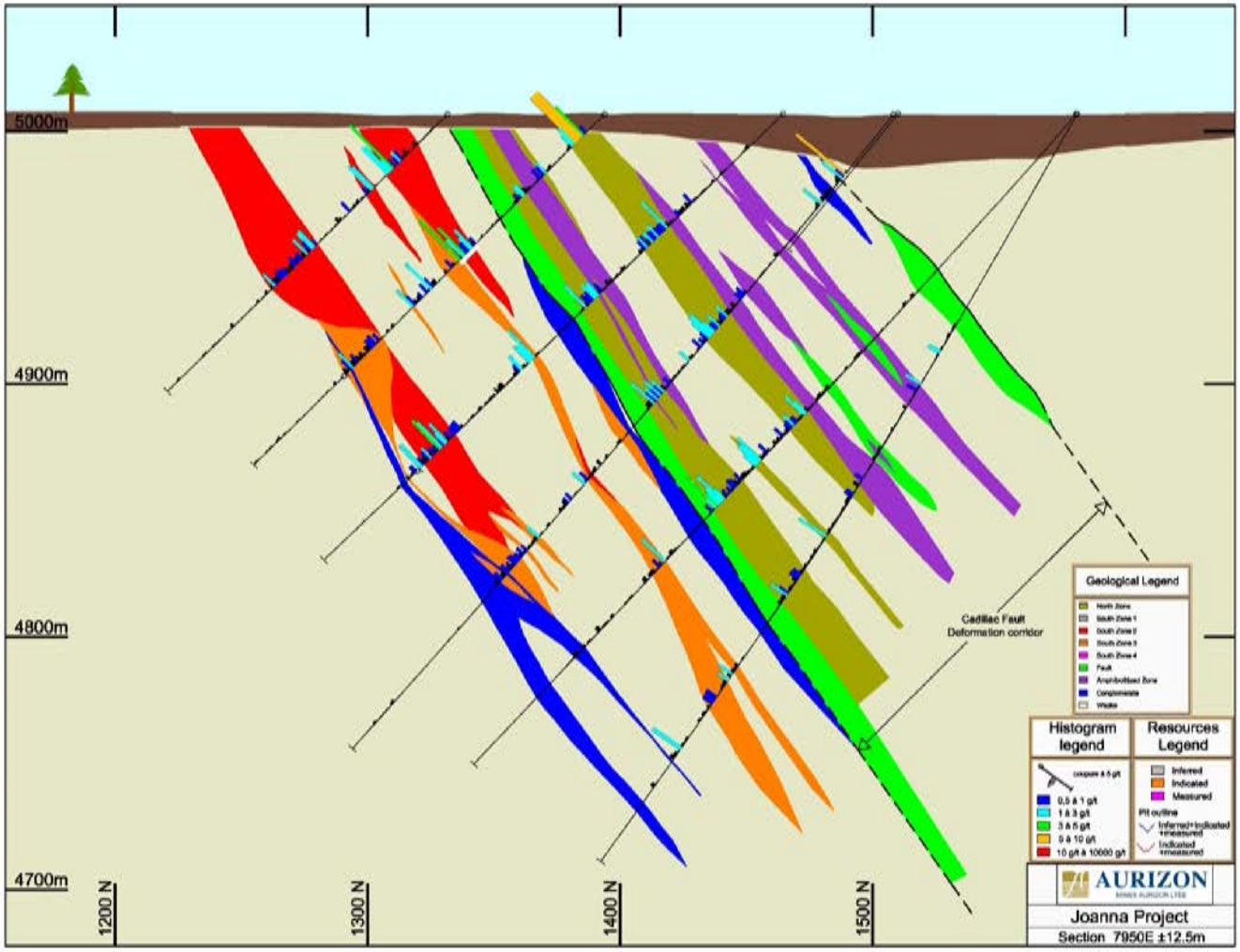


Figure 7.9 : Mineralized Zones at Hosco on Section 7950 mE

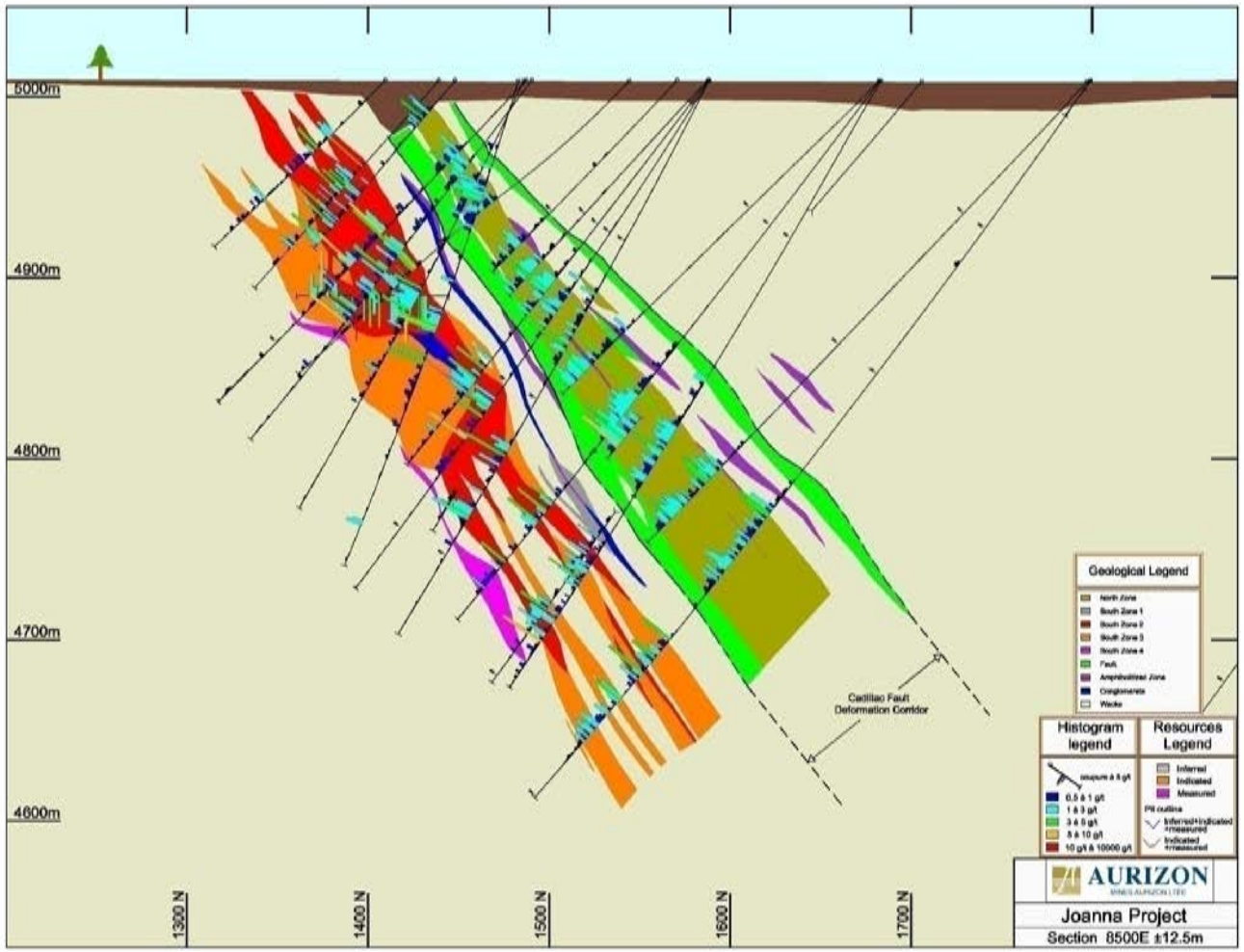


Figure 7.10 : Mineralized Zones at Hosco on Section 8500mE

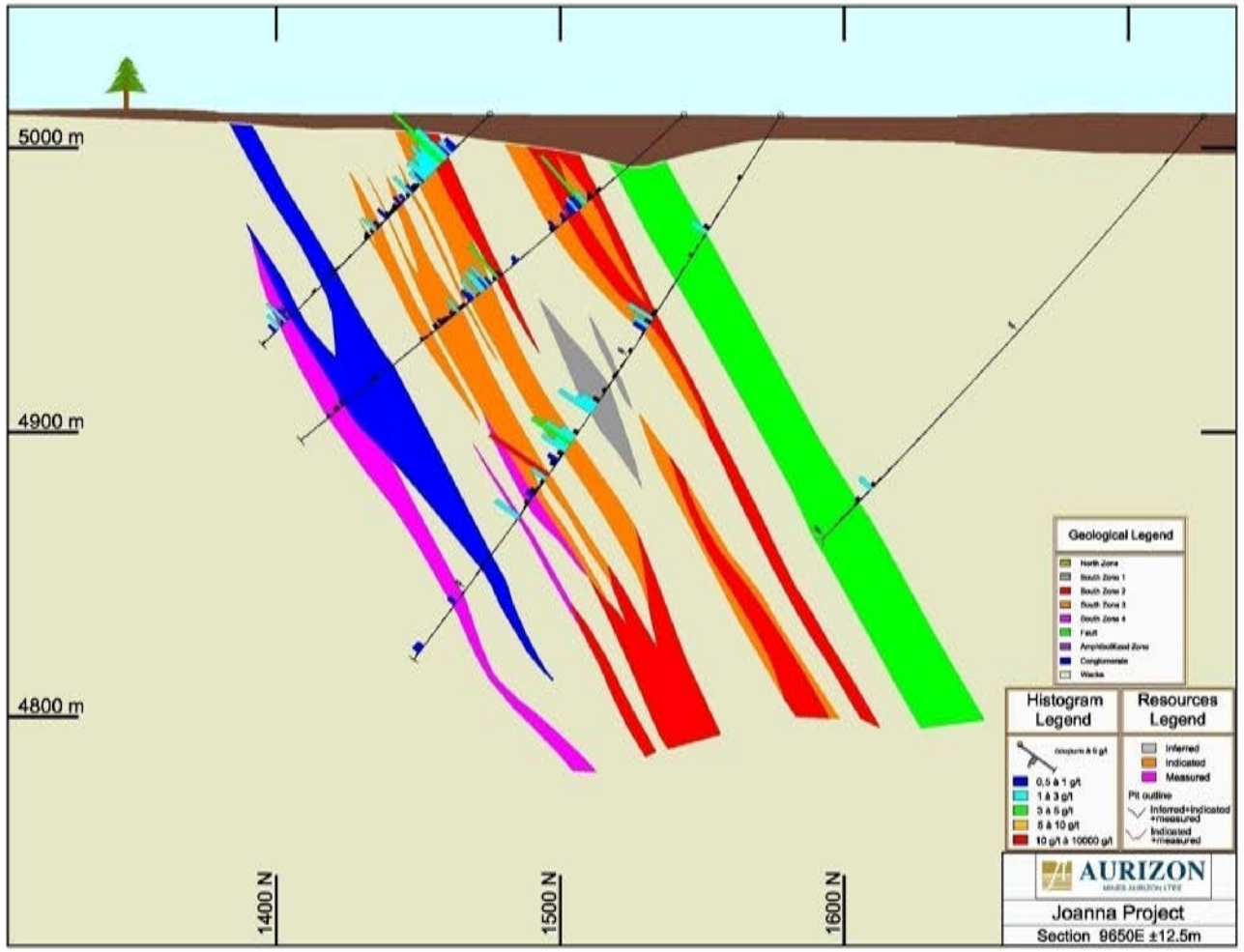


Figure 7.11 : Mineralized Zones at Hosco on Section 9650mE

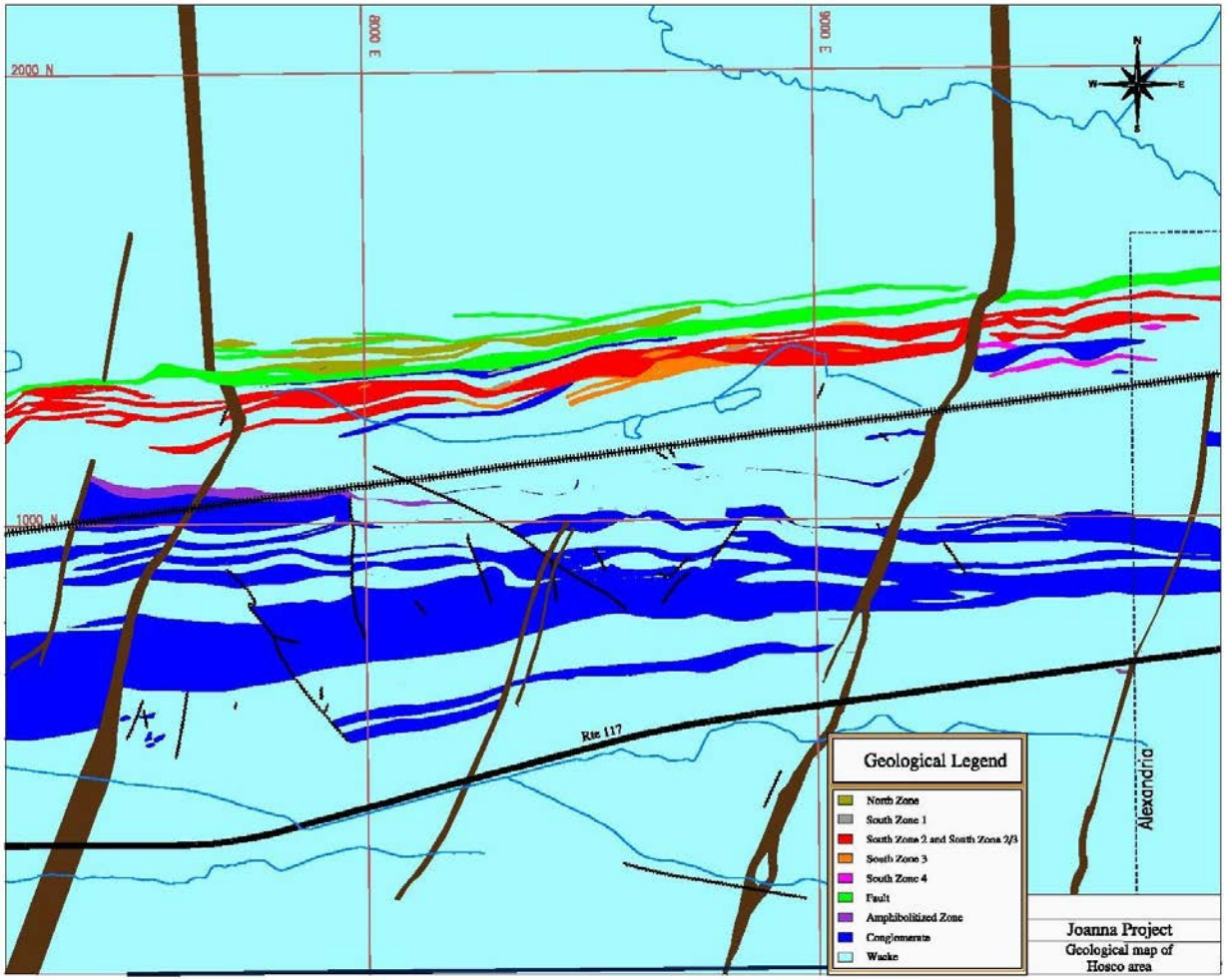


Figure 7.12 : Mineralized Zones at Hosco on Surface

7.2.2 Heva

Gold is located in several lenses in the south side of the Cadillac Break dipping approximately 60 degrees to the north and forming an envelope of thickness between 30 and 60 metres. In the Heva area, main lenses (Zone P) are located on the footwall contact of the Cadillac Fault up to 30 metres from this fault. Mineralization takes the form of a swarm of centimetric to metric grey/white quartz veins within biotite-tourmaline altered wacke and argilite. Pyrrhotite is the main sulphide. Locally, gold is remobilized inside the chlorite schist of the Cadillac Fault (Zone F). Others lenses (Zone S) are developed in amphibolitized conglomerate between 70 to 110 metres from the fault zone footwall. See Figures 7.13 to 7.17 for sections of mineralized intervals.

7.2.2.1 Heva East

Approximately 1 kilometre east of the Heva sector, mineralization has been intersected between 10 to 60 metres from the Cadillac Fault footwall. Gold is associated with strong concentrations (up to 20%) of pyrrhotite inside amphibolitized rocks of unknown origin. Heva East is a new mineralized area, intersected to date along a 200 metre strike and with a possible extension open in all directions.

7.2.2.2 Heva West

High grades have been intersected in this area. Arsenopyrite is mainly associated with quartz/tourmaline veins in silicified, sericitized and biotized rock. These zones are located at least 125 metres from the footwall contact of the Cadillac Fault. Mineralized zones in this area are usually narrower than main lenses (Zone P) located on the footwall contact of Cadillac Fault.

Figure 7.13 : Mineralization in the Heva Main Lens, Zone P (Hole JA-11-888 - Section 5550 mE)

Figure 7.14 : Mineralization at Heva East, (Hole JA-11-920 -- Section 6250 mE)

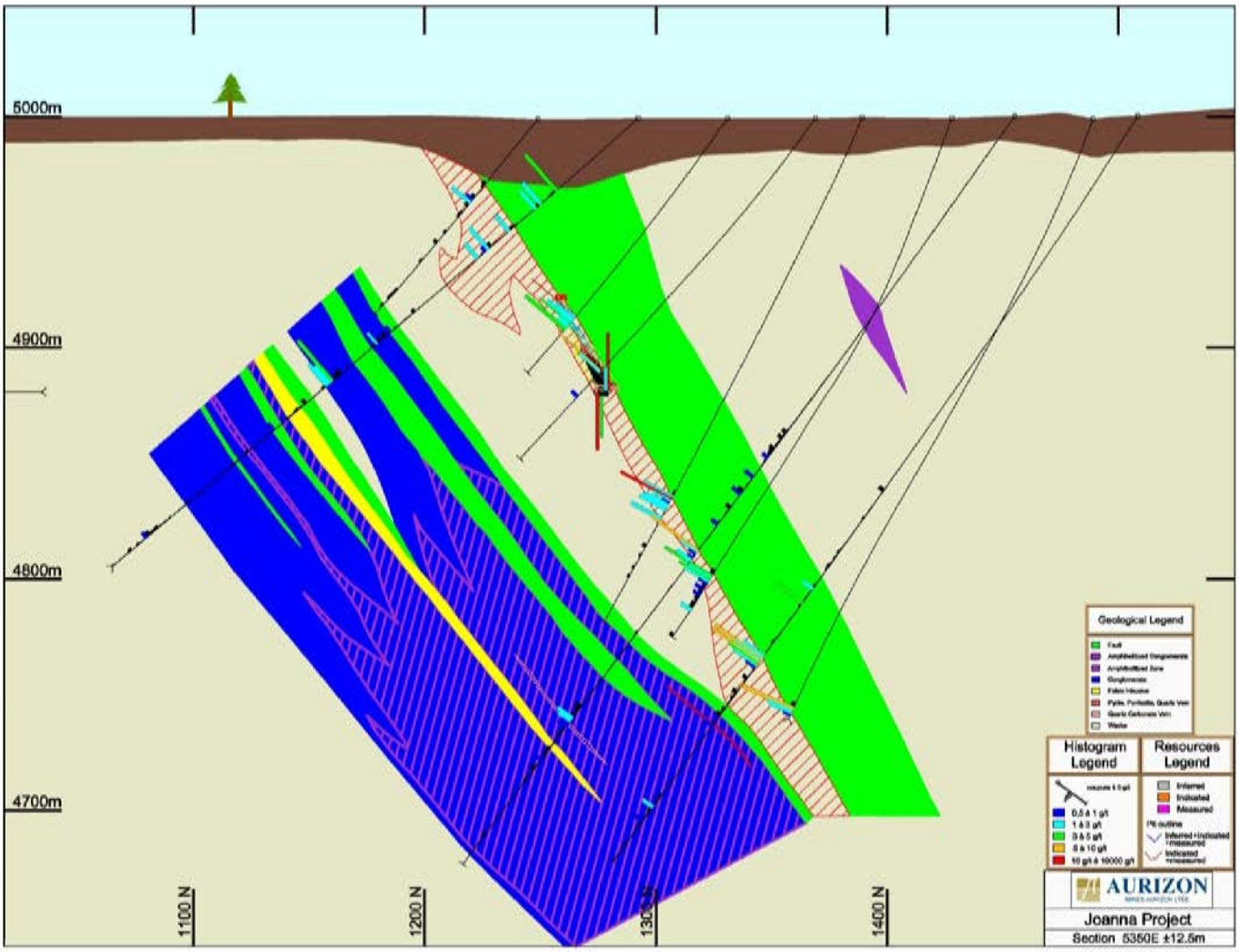


Figure 7.15 : Mineralized Intervals of Heva Section 5350E

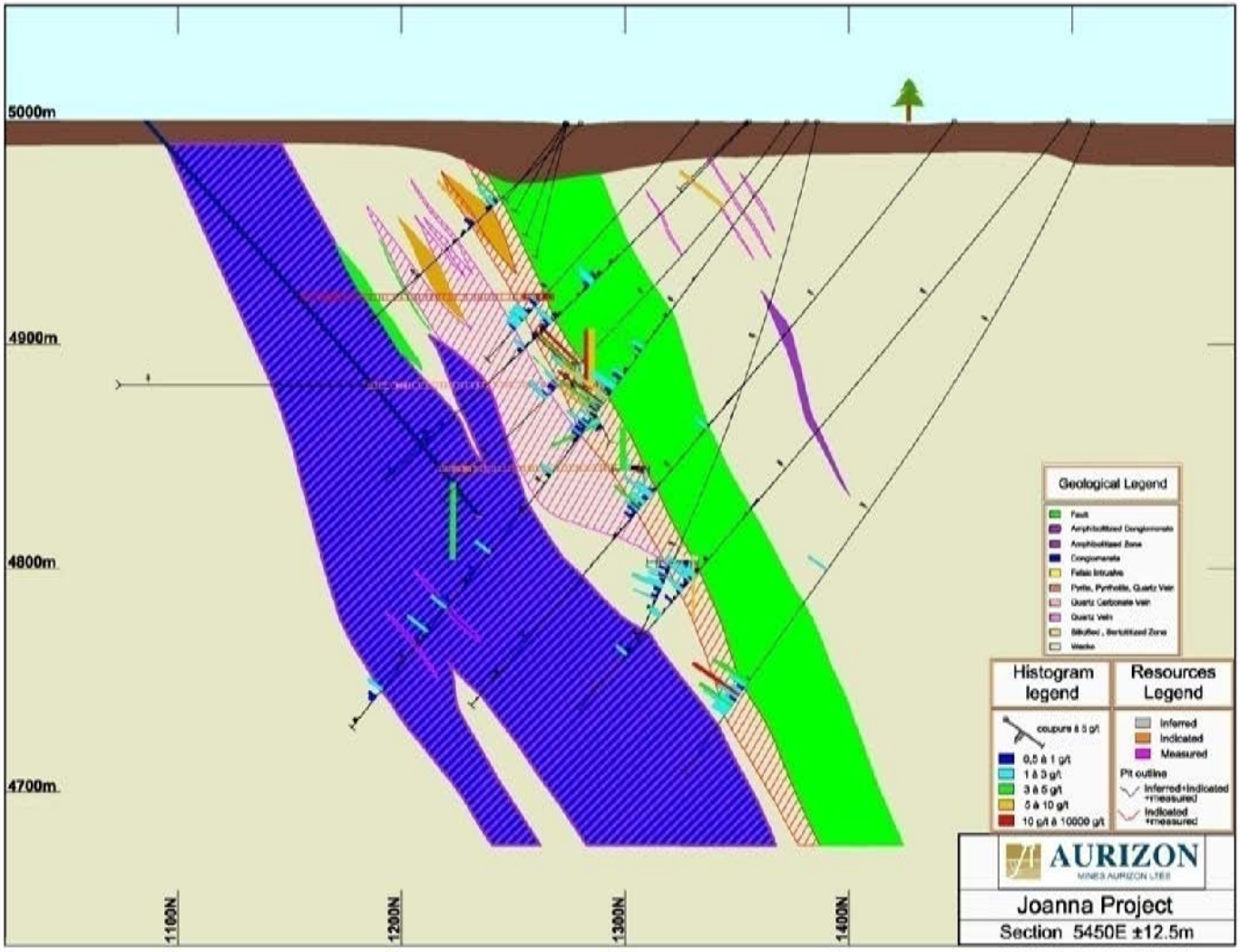


Figure 7.16 : Mineralized Intervals of Heva Section 5450E

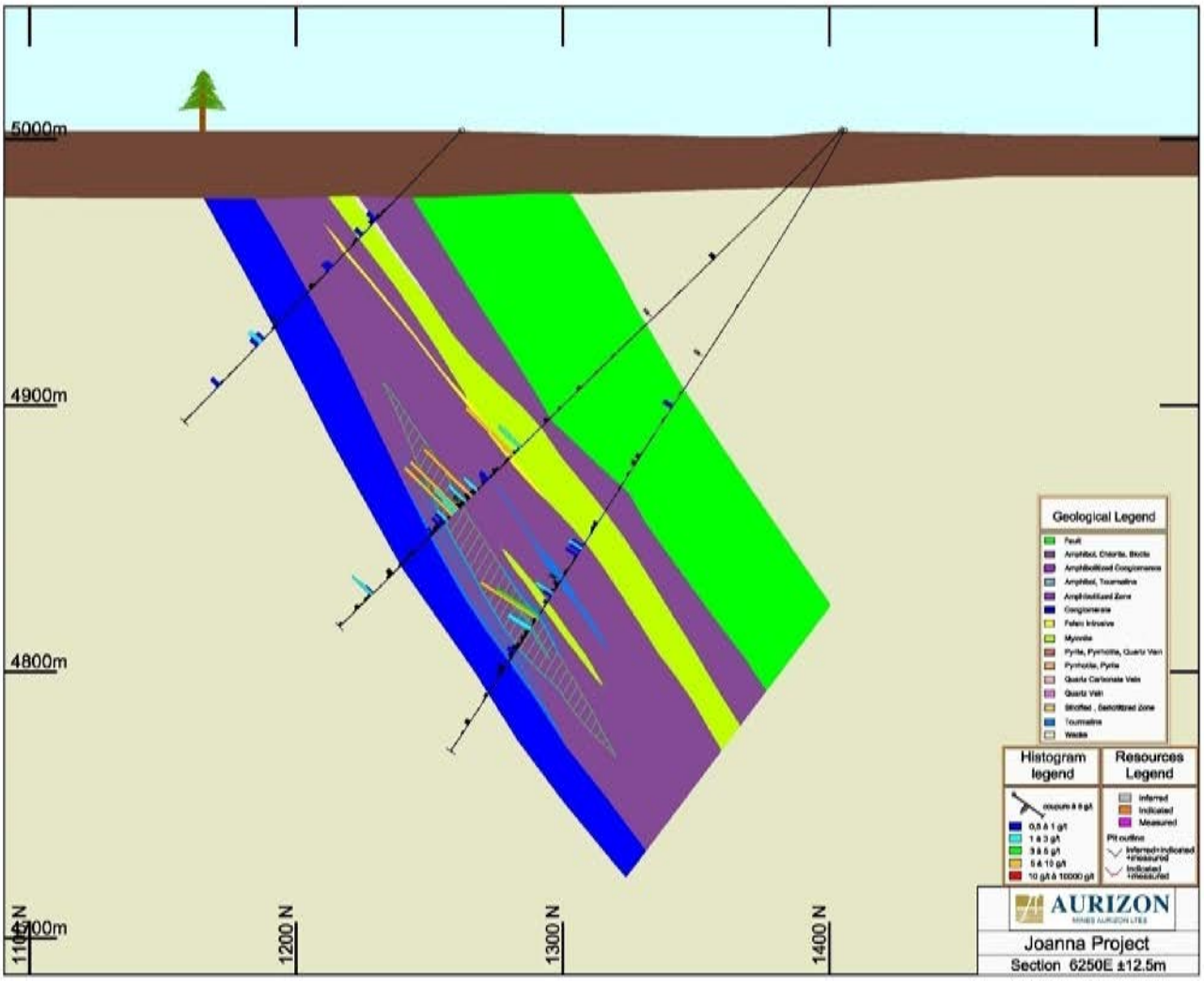


Figure 7.17 : Mineralized Intervals of Heva Section 6250E

8 DEPOSIT TYPES

The following description of the deposit model and associated mineralization is based on information from different technical reports listed in the References section, site visit observations of core, and information provided by Aurizon geologists.

The Joanna gold deposits may be described as sediment hosted orogenic gold deposits related to the structural control of the Cadillac fault. Mineralization is mainly developed in the form of finely disseminated sulphide envelopes (pseudo-lenses) with minor quartz veining in sediments of the Pontiac Group (Consorem, 2010) on both sides of a strong 10 to 100 meter wide ductile to ductile-brittle deformation zone identified as the Cadillac Fault. The ductile-brittle fault is materialized by chlorite-carbonate schist.

The mineralized corridor of the Hosco deposit is described along a 3,000 m east-west trend and can be followed down to a depth of 400 m in the central part of the deposit. The mineralized corridor can be followed in the Alexandria Sector. The Heva, Hosco and Alexandria deposits (related to the structural control of the Cadillac fault) are following the same trend. The east-West extensions of the gold deposits remain open. They will probably be all connected with additional drilling. Mineralized zones within that corridor dip by 50° to 65° to the north with a western plunge. They are usually narrower and less continuous in the Heva sector and wider and more continuous in the Hosco sector. The description of the Heva deposit in this report is not as detailed as for the Hosco sectors of the property. This is because the focus of the exploration and concurrent feasibility study is on the Hosco deposit.

The genesis of the Hosco deposit has not been studied in detail but intensive descriptions of hand specimens and thin sections combined with field investigation of accessible outcrops around the deposits help to develop a robust field model of the deposit that can be outlined by the following points:

- Mineralization is developed in uniform, medium-grained wacke affiliated with the Pontiac Group using Pearce Zr/TiO_2 and Al_2O_3/TiO_2 ratios.
- The gold enrichment corresponds visually to a continuous biotite alteration zone forming an association with fine pyrrhotite, which is Ni bearing. Outside the fault area the ductile

deformation is weak; detrital quartz grains are only slightly dislocated and biotite is moderately oriented (Renou, 2009).

- Quartz veining is common south of the Cadillac Fault and defines probably a sub regional feature recognized more or less continuously along a 7 km strike. As observed near the deposit, increasing vein density may correspond to local folding and decollement following a 15° westerly plunge (Figure 8.1). Increasing deformation is directly related to the development of mica-rich alteration halos. The plunging effect is also materialized by gold grade 3D modelling.

Figure 8.1: A Western Plunge of 15°in the Hosco Area

- Gold bearing quartz veins are mainly conformable to bedding and are contemporaneous to gold bearing alteration halos. Quart micro-textures indicate a late-tectonic emplacement (Renou, 2009).
- Various gold bearing alteration halos are observed across the deposit. North of the fault, wacke are completely replaced by fine grains and a laminated biotite-tourmaline assemblage. Carbonate activity is restricted to the fault area with a weak interaction southward for a few tens of metres. South of the fault in the biotite halos, a strong albite-quartz flooding is observed around mineralized veins. White mica is sporadically observed in the mineralized system and strongly correlates to localized slight increases in deformation.

- A geochemical profile across the deposit using a mass balance approach shows an alternate gain in Na and K while Ca and Mg remain unaffected indicating the low circulation level of CO_2. Figure 8.2 shows the alteration minerals distribution across a conceptual profile of the Hosco mineralized zone.

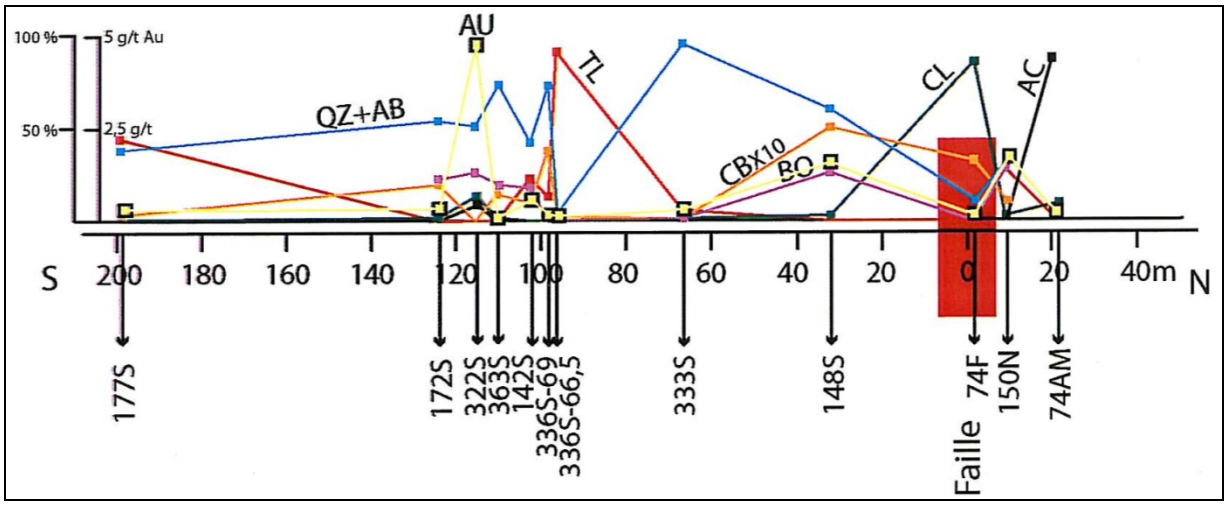


Figure 8.2: Alteration Minerals Distribution Across a Conceptual Profile of the Hosco Mineralized Zone (taken from Renou, 2009)

- Lack of observed chlorite aside from its crystallization as a fringe around biotite flakes may indicate mineralized conditions equilibrate at upper greenschist facies. Syn to late kinematic garnet porphyroblasts support this assessment (Renou, 2009).
- A late, apparently post-kinematic and post-mineralization amphibolitisation event affected host rock on both side of the fault. The tremolite-rich assemblage selectively replaces up to 100% of the host rock.

Figure 8.3 shows geochemical profiles (As, Au, Ca, Cr, Mg, Ni, Sb and W) across the Hosco deposit in two holes, JA-08-379 (section 8400mE and JA-08-427 (section 8900mE).

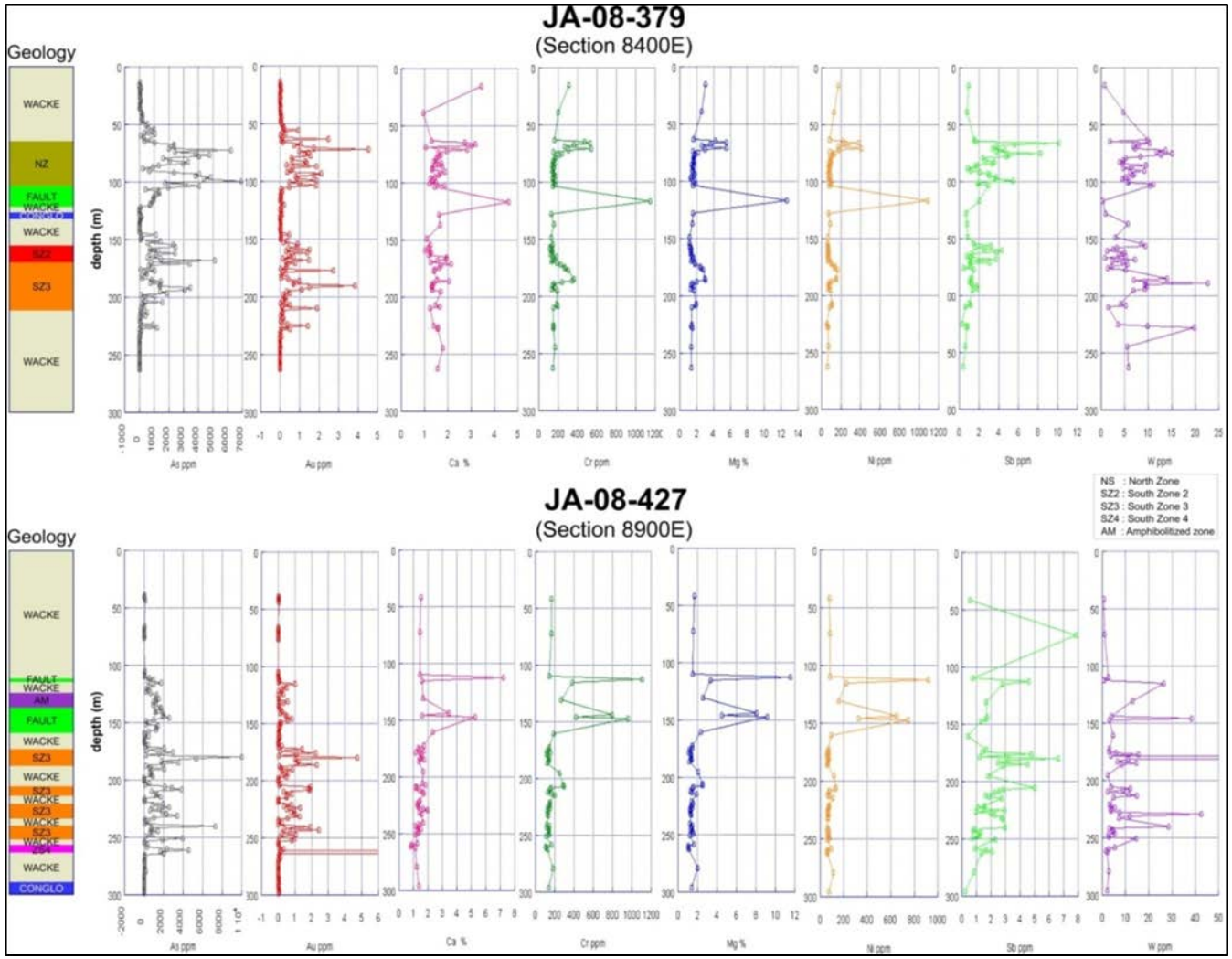


Figure 8.3: Shows Geochemical Profiles (As, Au, Ca, Cr, Mg, Ni, Sb and W) Across the Hosco Deposit in Two Holes, JA-08-379 (section 8400mE) and JA-08-427 (Section 8900mE)

9 EXPLORATION

The exploration history of the property is directly linked to the history of the discovery and development of the Hosco and Heva mine previously discussed in this report. Since 2006, Aurizon has carried out extensive computerization and integration of the historical data. Exploration holes were incorporated into a database in electronic format at the end of 2006. Where possible, the casing of old holes has been located in the field with GPS. A surveyed grid (GML grid) has been established as a unique reference for all information related to the property. Sylvestre, Julien, Leclerc Land Surveyor ("SJL Surveyor") have surveyed 184 old holes. For 170 holes previously surveyed by an unidentified surveyor, we corrected the coordinates in order to integrate them into the GML grid. Easily computerized and reliable underground data was also included in the new database.

In 2007, Aurizon initiated an extensive exploration program by systematically re-sampling 118 (72 Hosco and 46 Heva) historical holes over wide intervals in order to define the extension of the mineralized system according to sulphide disseminations. A magnetic and EM survey has been completed north of the Hosco and Heva sectors. See Section 10. A previous NI 43-101 compliant resources estimate by SGS Geostat was also completed.

In the late summer of 2007, a mapping and sampling campaign was conducted on the Henriksen claims located northeast of the Joanna property. See Section 4. Mineralization indicators as strong alteration and disseminated sulphide occurrences returning anomalous gold values between 0.02 and 0.3 grams per tonne from 233 grab samples on outcrops added to the collection of 1077 soil samples for MMI (Mobil Metal Ion) detection throughout the property and led to the trenching and channel sampling (300 samples) of 6 selected areas. Results in the range of 200 to 1000 ppm for Zn and 100 to 400 ppm for Cu were obtained in the western part of the block located in the Blake River volcanic group. Another signature with gold between 0.002 to 0.01 ppm and arsenic between 0.2 to 1 ppm resulted in the eastern part of the block with a similar gold context as results of the Hosco sector.

In 2008, Aurizon conducted a magnetic survey on lines totalling 27.1 km. A preliminary geometrical, structural, and depositional model based on field observations, including Leapfrog modelling of gold grade and petrographic works, was proposed. Metallurgical testing was

initiated with LTM Laboratory with two (2) composite samples. An IP survey totalling 25.4 km was conducted in the region of the Joanna North sector adjacent to the eastern boundary of the Henriksen sector. An infiniTEM survey (11.0 km) was also conducted in the Henriksen sector. Roche, the consulting company, presented a phase 1 environmental characterization of the property. BBA, the consulting company, completed a preliminary assessment study for an open-pit operation of the Hosco sector.

In 2009, in addition to the drilling (see Section 10) and the completion of the NI 43-101 compliant resource estimates by SGS Geostat, Aurizon completed several field programs. Work included an induced polarization (IP) survey, MMI soil sampling, hammer prospection and mechanical stripping. The results of the IP survey were used to map the properties of resistivity and chargeability of geological formations on the property. In the Bousquet sector, Daniel Gauthier Exploration carried out line cutting work in June 2009 for a total of 31 lines at 200 m spacing. A rock sampling/mapping program was then conducted in the Bousquet sector between September 21, 2009 and October 28, 2009. The team consisted of a consulting geologist (Michael Lacey), a geological engineer, and three (3) technicians from Technominex Services Inc. During this period, a 2.35 km line was covered on the eastern portion of the Bousquet sector. A total of 56 lithological samples were taken. Lakefield Research received five (5) composite samples to perform a sequence of metallurgical testing including grinding, flotation, oxidation and cyanidation. Mineralogical and micro-analytical studies were conducted to explain gold recovery.

In 2010, the mapping and sampling program resumed with two (2) geologists and a Technominex technician. 28 km and 5.25 km of line were surveyed on the west and east blocks respectively from May 16 to July 19th, 2010. A total of 286 samples were taken on the Bousquet sector and analyzed by ALS Laboratory Group of Val d'Or. Electromagnetic surveys were conducted using Beep Mat in an effort to explain certain anomalies of high intensity and considerable lateral extent. A gravimetric survey was conducted on the Henriksen sector in an effort to distinguish amphibolites, wackes, and mineralized conglomerate.

No other exploration work was realized in 2011 and to date, none has been realized in 2012. Only drill holes were completed on the property (see Section 10).

10 DRILLING

10.1 Drilling Methodologies

This section is based on information provided by Aurizon and observations made during the independent verification program conducted at the project work sites by SGS Geostat on December 31, 2011.

Drill holes are planned by a geologist on printed sections showing all geological and grade information from surrounding holes. Corrections to the holes' theoretical traces are made using a determined deviation rate. The planning information is then transmitted to the technical team responsible for checking physical access and conformity to forestry and environmental permits. Holes are physically implemented on surveyed grids attached to the NTS grid. The process has been supervised and certified by Sylvestre, Julien, Leclerc Land Surveyor ("SJL Surveyor"). Drill rig alignment, horizontality and dip, are checked with levels and sticks by the contracted exploration services firm Services Technominex Inc. ("Technominex"). Once holes have been drilled, they are surveyed by SJL Surveyor.

Drillhole deviation (dip and azimuth) is measured by either a Reflex or a Flexit tool. Measurements are made at every 30 m during the drilling operation and at the end of each hole. A continuous survey is also done during the rod pulling operation. An error of 4% is attached to each measurement according to tool technical specifications. Completed holes are secured with a cement plug and closed by a steel cover. The position and orientation of the drill hole casing is surveyed, and the survey values are recorded as the final coordinates and hole orientation in the database.

The exploration drills use NQ-diameter metric coring equipment. During drilling operations, operators place the continuous cored rock in wooden trays and indicate the depth on a wooden block following every 3 m interval drilled. Ground material intervals or missed core is indicated by a specific block. Closed core boxes filled and tagged by the drilling company at the drill rig are carried by truck to the Rouyn-Noranda core shack facility by Services Technominex personnel. The same firm handles all the rock material from length measurement to shipping in an indoor and access-restrained area.

10.2 Historical Drilling

Before Aurizon acquired the Property, other companies had explored, developed and produced on the Property. Table 10.1 summarizes the past exploration and development work conducted on the Property.

Table 10.1: Summary of Historical Drilling Development Work and Production on the Heva and Hosco Sectors

Historical Exploration and Development Work – Joanna Property					
Date	**Surface Drilling (m)**	**Underground Drilling (m)**	**Sinking of Shaft (m)**	**Ramp Development (m)**	**Lateral Development (m)**
1944-1945	20 000	3 000	160		2 400
1980-1981	1 128				
1984	2 988				
1986-1989	9 768			484	630
Total	**33 884**	**3 000**	**160**	**484**	**3 030**

Historical Production – Joanna Property				
Date	**Ore (tonnes)**	**Grade (Au g/t)**	**Processed Grade (Au g/t)**	**Recovery (%)**
1948-1949	45 872	6.58	4.9	74.5
1986-1989	21 555	2.65	Stockpile on site	-

SGS Geostat and Maxime Dupéré P.Geo., the qualified person, have not completed sufficient work to verify this information. The historical tonnage and grade extracted should be considered for reference purposes only.

10.3 Recent Drilling

In 2007, Aurizon drilled 62 surface core holes on the Hosco Sector (JA-07-01 to 25 and 68 to 104) totalling 20,647 m. 42 holes (JA-07-26 to 67) totalling 26,269 m on the Heva sector for a total of 104 holes totalling 46,916 m.

In 2008, 354 holes were drilled (JA-08-105 to164, 166 to 170, 172 to 175, 177 to 182, 184 to 189 and 191 to 463) totalling 86,320 m in the Hosco sector. Five (5) holes were drilled (JA-08-165, 171, 176,183 and 190) totalling 1,254 m on the Heva sector and three (3) holes (JA-08-464 to 466) totalling 561 m on the Henriksen sector for a total of 362 holes totalling 88,135 m drilled on the Property. Most of the holes were in-fill drilling on a 25 m grid, mainly located in the potential open pit modelled in the 2008 preliminary economic assessment study done by BBA.

In 2009, Aurizon drilled 55 holes on the Property totalling 13,536 m. 32 holes were drilled on the Hosco sector (JA-09-467, 472 to 474, and 494 to 499 plus deepening of 22 previous holes) totalling 8,555 m. 15 holes (JA-09-468 to 471 and 483 to 493) totalling 3,415 m were completed along the Cadillac Fault to test the eastern extension of the Hosco mineralization in the Alexandria sector, as well as a new mineralized trend ("Joanna South") about 300 m south of the Cadillac Fault. 6 holes (JA-09- 475 to 480) totalling 1,185 m were drilled to test the mineralized trend ("Joanna North") on the Heva sector about 1 km north of the Cadillac Fault. 2 holes (JA-09-481 to 482) totalling 381 m were also drilled on the Henriksen sector to test the same mineralized trend ("Joanna North") about 1 km north of the Cadillac Fault.

In 2010, Aurizon drilled a total of 349 holes on the Joanna Property including the different sectors, deepened previous holes, exploration and condemnation holes totalling 70,916.5 m. From this total; 56 holes totalling 8,769.6 m were drilled on the Alexandria sector, 14 holes totalling 3,492 m were drilled on the Heva sector, 236 holes were drilled on the Hosco sector totalling 52,081.4 m, and 43 holes totalling 6,573.5 m on the north and north eastern part of the Joanna Property were drilled.

In 2011, Aurizon drilled a total of 115 holes on the Joanna Property including the different sectors, deepened previous holes, exploration and condemnation holes totaling 30,055.5 m.

From this, a total of 61 drillholes totalling 16,591.8 m were made on the Heva sector for resource increase purposes. The area tested covers 1.5 km along a strike with targets between 0 and 200 metres below the surface. All the new drill holes confirm the results from previous holes and the extension of the mineralized zones. Heva East is a new mineralized area,

intersected to date along a 200 metre strike with the possibility of extension in all directions. 20 holes totaling 5,798.3 metres were drilled at 750 to 1,000 metres east of the former Heva workings. In this area, only scattered assays from historical holes indicated the presence of a consistent gold-bearing mineralized system.

52 holes totaling 13,049.7 m were completed on the Hosco sector in order to increase the open pit indicated resources.

Two (2) additional drill holes (414 m) were done on the Alexandria sector corresponding to the eastern limit of the Hosco sector.

In 2012 (May 31st), 73 holes totalling 23,388 m have been completed on the program targeting the Heva area and western extension of the Hosco deposit between the surface and a depth of 200 meters. The objective of the program was to test, with an initial 100 meter spacing, the continuity of the near surface mineralized system westward from the Hosco deposit, which corresponds to a 3.0 kilometer strike length along the Cadillac Fault.

9 holes totalling 3,002 m were realized in the west extension of Hosco in order to test the continuity of higher grades in depth. Initial 50-metre spacing has been used in this area and new drill holes confirm the continuity of higher gold value in depth. Higher gold values are associated with a high level of alteration (biotite-quartz-albite and locally white mica), quartz veins and sulphides (arsenopyrite and pyrrhotite).

64 holes totalling 20,386 m were completed in the Heva area in order to increase inferred resources near the surface in the Heva zone, Heva west and Heva east sectors. Drill holes confirm several gold enrichment trends along the Cadillac Fault.

The program is still in progress with a new objective to define gold potential at 200 m below the surface. The interpretation of the higher grade trends showing a defined plunge has the potential to move the project toward a combination of surface and underground targets. The program supports the hypothesis that higher grade mineralization with continuity and metal

factors may support an underground mining operation along the Joanna portion of the Cadillac Fault.

Figure 10.1 displays all drill holes on the Joanna Property to date. Since 2007, Aurizon has drilled 1058 holes totalling 272,947 metres on the Joanna Property. Table 10.2 summarizes the drilling done on the Property by Aurizon since 2007.

Table 10.2: Drilling Done by Aurizon on the Property Since 2007

Aurizon Drilling on the Joanna Property			
Year	**Surface Drilling (m)**	**Number of Holes**	**Description**
2007	46,916	104	JA-07-01 to 104
2008	88,135	362	JA-08-105 to 466
2009	13,536	55	JA-09-467 to 499 and 22 deepening holes
2010	70,916.5	349	JA-10-500 to 710, JA-10-718 to 829, JA-10-831 to 841, JA-10-847, JA-10-848, JA-10-851 to 855, JA-10-860, JA-10-861, JA-10-880 and 5 deepening holes
2011	30,055.5	115	JA-11-830, JA-11-842 to 846, JA-11-849, JA-11-850, JA-11-856 to 859, JA-11-862 to 879, JA-11-881 to 965
2012 (May)	23,388	73	JA-12-966 to 1038

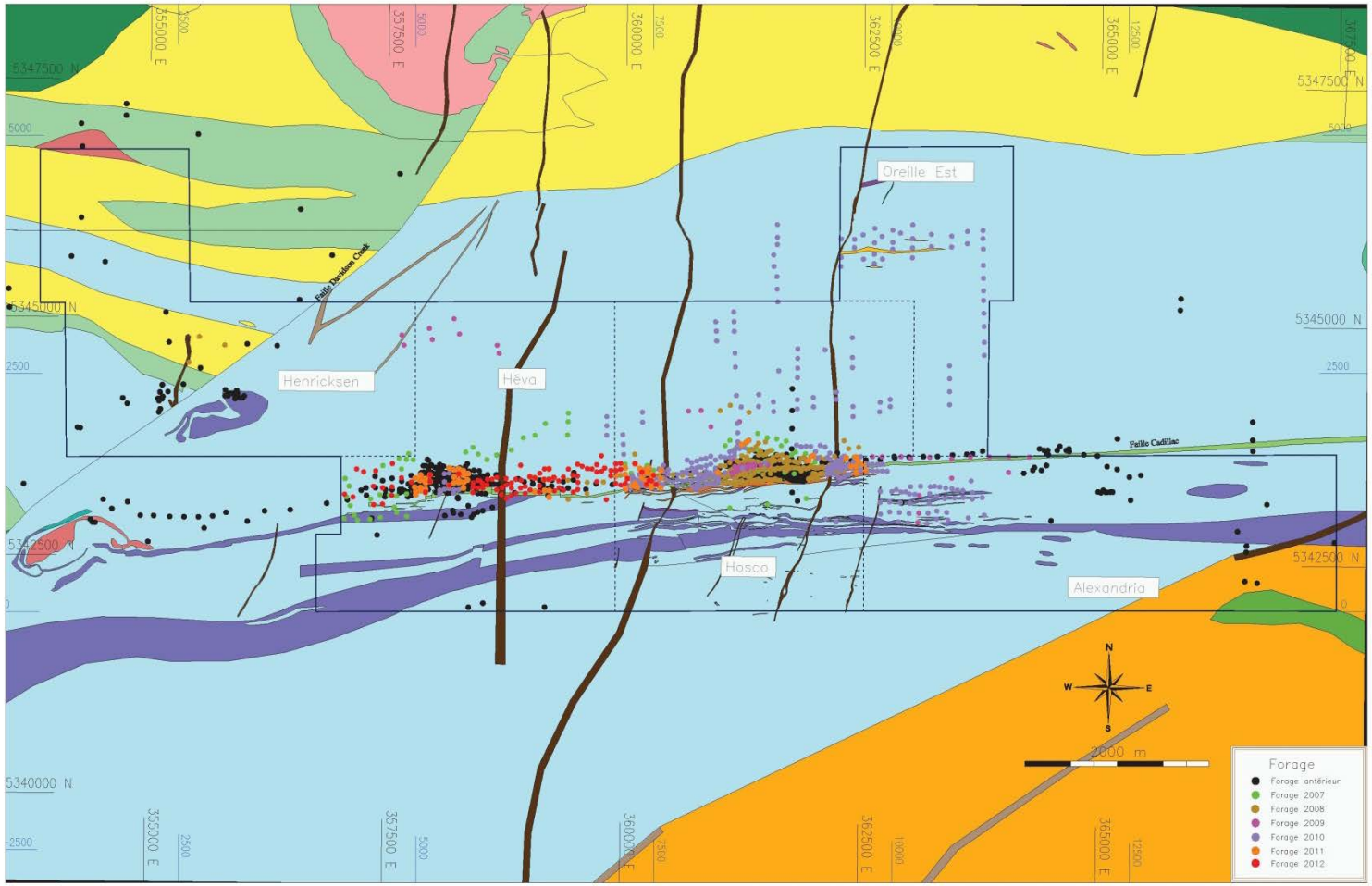


Figure 10.1: Outline of the Joanna Property Displaying all Drillholes to Date

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation and Analyses

From the beginning of exploration on the Joanna gold property ("Property" or "Project") by Aurizon Mines Ltd ("Aurizon" or "Company") in 2007 until early 2008, the drill core samples were sent for analysis at Laboratoire Expert Inc. ("Lab Expert") of Rouyn-Noranda, Quebec. Since then, all drill core samples were and continue to be assayed at ALS Canada Inc. - Chemex laboratories ("ALS Chemex") of Val d'Or, Quebec. As part of Aurizon quality assurance and quality control ("QA/QC") protocol, approximately 10% of crushed reject and pulp duplicates were sent for reanalysis at ALS Chemex for the 2007-early 2008 period and are now sent for reanalysis at Laboratoire d'Analyse Bourlamaque Ltd ("Lab Bourlamaque") of Val d'Or, Quebec.ALS Chemex is a fully accredited laboratory under ISO 9001 and ISO/IEC 17025 standards and Lab Bourlamaque is in the process of obtaining ISO-9001 certification. Lab Expert is not an accredited laboratory.

The handling and transportation to ALS Chemex of the drill core samples is conducted by employees or contractors of Aurizon or Services Technominex Inc. ("Technominex"), the principal consulting group managing the exploration work for the Company.

All samples received at ALS Chemex are digitally inventoried using a bar-code and then weighed. Samples having excess humidity are dried. Sample material is crushed in a jaw and/or roll crusher to 70% passing 9 mesh. The crushed material is split with a rifle splitter to obtain a 250 g sub-sample which is then pulverised to 85% passing 200 mesh using a single component (flying disk) or a two component (ring and puck) ring mill.

The analytical protocol used at ALS Chemex for gold is the trace level 30 g fire assay fusion and atomic absorption finish (code Au-AA23) with a detection limit range of 0.005-10 g/t. For samples returning initial assay value greater than 10 g/t, a reanalysis using ore grade 30 g fire assay fusion with gravimetric finish (code Au-GRA21) is completed.

Lab Bourlamaque uses similar sample preparation and analytical protocols for the reject and pulp duplicates which are 30 g fire assay fusion and atomic absorption finish (code AU020) and 30 g fire assay fusion and gravimetric finish (code AU010) respectively.

11.2 Quality Assurance and Quality Control Procedure

In addition of the standard laboratory QA/QC programs, Aurizon implements its own internal QA/QC protocol consisting of the insertion of certified and custom-made reference materials (analytical standards and blanks) and core duplicates in the Project sample series. Aurizon also sends reject and pulp duplicates for reanalysis at a second laboratory for verification representing approximately 10% of the samples assayed.

11.2.1 Analytical Standards

Twelve certified reference materials having different representative gold grades and two composite reference materials have been used as analytical standards since the beginning of the exploration conducted by the Company on the Project. The analytical standards are inserted in the sample series at a rate of one reference material for every 25 regular samples. Aurizon has defined the tolerance of the different reference materials from their expected grade values to ±10%. When analytical values of reference materials fall outside the determined tolerance range, verification is performed at the different steps -- from sampling at the core logging facilities to the laboratory -- in order to locate and explain the potential source of any difference.

The thirteen certified reference materials OREAS 10Pb, 15Pa, 15Pb, 6Pa&Pc, 7Pb, 61Pa&Pb, 62Pa&Pb, 62c, 65a and 66a are commercial standards from Ore Research & Exploration Pty Ltd, Australia, distributed by Analytical Solutions Inc. of Toronto, Ontario, Canada. Certified set values and expected standard deviation (“Std. Dev.”) values are reported in performance gates statistics for each certified reference material. The two composite reference materials JA-1 and JA-2 are custom-made from rejects of 20 to 30 mineralised samples which are pulverised and homogenised to make 40 to 60 kg batches. The set values for the 2 composite standards are weighted average grades of the selected mineralized drill core intervals. Unfortunately, no expected Std. Dev. values are defined for the composite reference materials. The expected Std. Dev. used in the current QA/QC analysis are derived from the actual reported analytical values for each composite standard and calculated using analytical results having less than a 50% difference relative to the expected value.

Figure 11.1 to Figure 11.15 are graphs showing the variation of the reported analytical results for each certified and composite standard with time. The QA/QC data is taken from the Project

database dated March 18, 2011. For each graph, the QC warning (orange) and QC failure (red) intervals are shown along with the expected mean value (green). The values for the QC warning and QC failure have been defined as 2 times and 3 times the expected Std. Dev. respectively. Table 11.1 shows the results of the statistical analysis for each reference material.

Lab Expert was the main analytical laboratory for the Project from the beginning of exploration in 2007 to April 2008; ALS Chemex was designated as the main laboratory following this. During the Lab Expert period certified standards OREAS 15Pa, 61Pa&Pb and 62 Pa&Pb were the most utilised standards. The Company started inserting the certified standards OREAS 10Pb and 7Pb at the end of 2007 and then decided to include custom-made composite reference material in March 2008 as part of the QA/QC protocol, shortly before changing the laboratory to ALS Chemex. The Company has added five more analytical standards, 6Pa&Pc, 62c, 65a and 66a, to replace depleted certified materials since then.

Table 11.1 : Summary of the Statistical Analysis for Different Reference Materials

Standard	Count	Period (dd/mm/yyyy)		Expected Au (g/t)		Observed Au (g/t)				QC Warning Range		QC Failure Range		Mislabeled
		From	To	Mean	Std. Dev.	Mean	Std. Dev.	Min	Max	Count	%	Count	%	
OREAS 10Pb	900	20/12/2007	31/01/2011	7.15	0.19	7.178	0.308	5.44	8.14	753	84%	845	94%	35
OREAS 15Pa	1217	16/02/2007	01/03/2011	1.02	0.03	1.013	0.054	0.44	1.03	1047	86%	1137	93%	7
OREAS 15Pb	116	20/01/2011	21/07/2011	1.06	0.03	1.086	0.106	0.58	1.64	94	81%	105	91%	0
OREAS 6Pa	29	30/12/2009	22/03/2010	1.65	0.07	1.528	0.041	1.44	1.60	19	66%	28	97%	4
OREAS 6Pc	509	29/12/2009	21/07/2011	1.52	0.06	1.542	0.280	0.02	7.13	443	87%	473	93%	1
OREAS 7Pb	549	19/12/2007	17/12/2009	2.77	0.06	2.724	0.191	1.74	3.43	423	77%	477	87%	4
OREAS 61Pa	238	13/03/2007	03/03/2008	4.46	0.13	4.587	0.193	3.57	5.25	191	80%	226	95%	2
OREAS 61Pb	94	02/03/2007	26/06/2007	4.75	0.13	4.688	0.145	4.45	5.53	89	95%	93	99%	1
OREAS 62Pa	221	16/02/2007	04/12/2007	9.64	0.32	9.715	0.470	8.09	10.83	177	80%	194	88%	8
OREAS 62Pb	51	18/10/2007	10/12/2007	11.33	0.35	10.564	0.374	8.57	10.97	29	57%	47	92%	1
OREAS 62C	173	06/12/2010	21/07/2011	8.79	0.21	8.824	0.798	0	9.81	138	80%	160	92%	0
OREAS 65a	13	21/07/2011	21/07/2011	0.520	0.017	0.034	0.019	0.491	0.561	138	80%	160	92%	0
OREAS 66a	104	20/01/2011	09/05/2011	1.237	0.054	1.214	0.229	0	2.05	90	87%	96	92%	0
Composite JA-1*	600	04/03/2008	30/12/2009	2	0.150	2.226	0.279	1.03	3.23	421	70%	486	81%	3
Composite JA-2*	757	22/01/2009	03/01/2011	1.2	0.079	1.272	0.705	0.82	8.09	326	43%	527	70%	0

* Expected Mean value correspond to weighted average Au grade of drill core intervals composing the standard material.
* Expected Std. Dev. value calculated using actual data contained within 50% difference relative to Expected Mean value

Certified standards OREAS 61Pa&Pb and 62Pa&Pb used in the early months have set values ranging from 4.46 g/t Au to 11.33 g/t Au, which are significantly greater than the average grade of the mineralised samples for the Project (in the order of 1 to 2 g/t Au). Reported results for standard 61Pa returned an average value 2.8% higher in comparison to the expected value but with an acceptable data variance (95% of values within the QC failure range). 61Pb returned an average value 1.5% lower but also with acceptable variance (99% of values within the QC failure range). Standard 62Pa shows observed values comparable with the expected value until roughly August 2008 when they returned consistently higher by more than 8% (on average),

which could be due to confusion with the higher grade 62Pb. Data variance for 62Pa is higher than the 61 series with only 88% of values within the QC failure range although definitely influenced by the higher values observed after August 2008. Standard 62Pb values returned significantly lower values (on average) when compared to the expected value by more than 6.8% but show an acceptable variance with 92% of values within QC failure range.

Certified reference materials OREAS 15Pa, 10Pb and 7Pb were the most used standards since the start of exploration in 2007 and 15Pa is still being inserted in the samples series. 7Pb and 10Pb were discontinued late in 2009 and early in 2010 respectively. Their set Au grades ranging from 1.02 g/t to 7.15 g/t Au are more in line with the Project mineralized samples. Reported values for standard 15Pa display a good correspondence with the expected value with an acceptable data variance (93% of values within the QC failure range) although the pattern of observed values prior to April 2008 show a relatively higher mean Au grade with more data scattering. The same pattern is observed for standard 10Pb prior to April 2008 where mean grade is consistently higher. After that period, reported values for 10Pb show good correspondence with expected value with an acceptable data variance (95% of values within QC failure range). Observed values for standard 7Pb returned a good correspondence with the expected value but are showing relatively more data scattering in comparison to the others standards with numerous low value outliers (overall 87% of values fall with the QC failure range).

Certified analytical standards 6Pa, 6Pc, 62c, and 66a are relatively recent reference materials used in the Project. 6Pa was used during a short period in late 2009. 6Pc, 62c and 66a have been used continuously since late 2009, late 2010 and early 2011 respectively. 6Pa, which has a set grade of 1.65 g/t Au, returned average results significantly lower than the expected value by 7.4% with a fair data variance (97% with QC failure but only 66% with QC warning). Reported values for standard 6Pc show a good correspondence with the expected value with an acceptable data scatter (93% of values within the QC failure range). Reference materials 62c and 66a returned average results in line with expected values with acceptable data variance (92% within QC failure range each). Reported values for composite standards JA-1 and JA-2 are more strongly scattered compared to the certified standard, which is expected for custom-made reference materials. Observed results for JA-1 are consistently higher by 11.3% on

average versus the expected value with a definite bias for the results prior to April 2008. Data variance is significant with only 81% of the values falling within the calculated QC failure range. Numerous outliers are observed with values 30% to 100% higher relative to the expected value. Standard JA-2, which replaced JA-1 in early 2009, also shows significant data scattering (70% of values within the defined QC failure range) but returned values on average 5% lower than the expected value. Several high value outliers are observed with some reaching more than 600% the expected value.

The QA/QC analysis outlined generally acceptable results for the analytical standards with significant improvement noticeable after April 2008 when ALS Chemex became the main laboratory. The results of the certified reference materials show relatively good consistency compared to the expected values and QC ranges. The results for the composite standards returned more scattered values which can be expected for custom-made reference materials. Since We have observed a relatively high amount of QC failure for standards 10Pb, 15Pa, 15Pb, 6Pc since October 2010 but the results for standards 62c and 66a are consistent with the expected values. Aurizon is currently reviewing the data to see if there are any transcription errors. Also, the pulps from the Certificates containing problematic QA/QC results from this period were sent back for re-assay. The results were pending during the writing of this report.

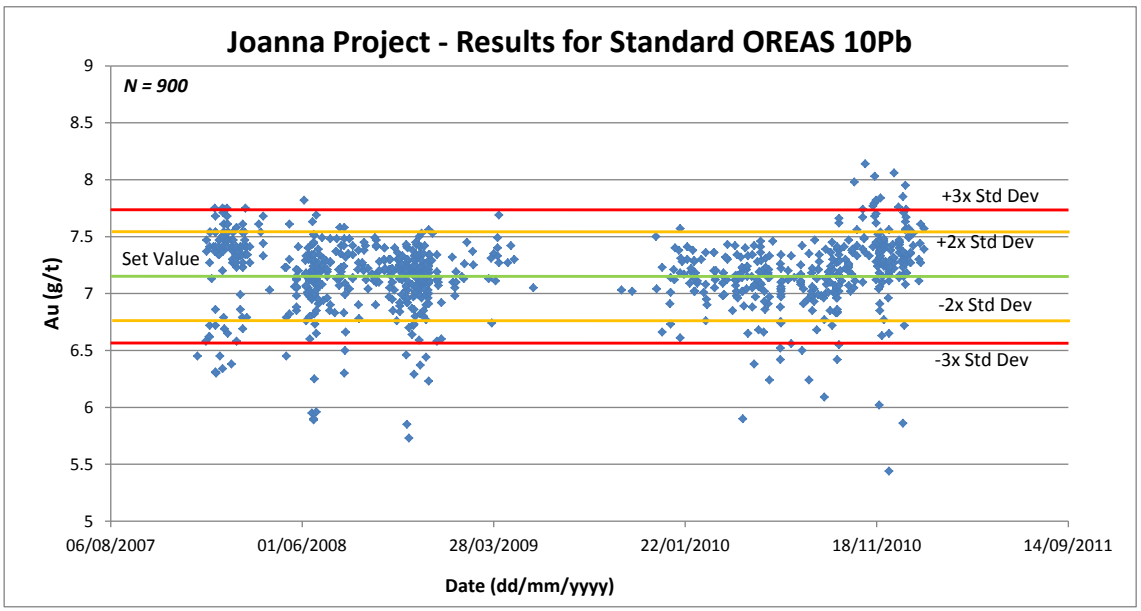


Figure 11.1: Variation of Reported Values with Time for Analytical Standard OREAS 10Pb

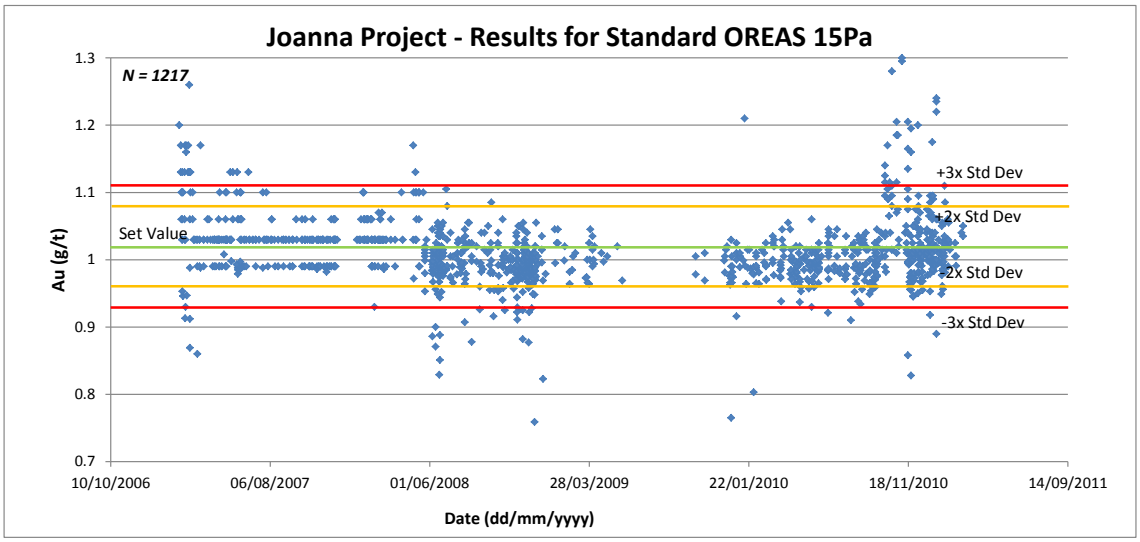


Figure 11.2: Variation of Reported Values with Time for Analytical Standard OREAS 15Pa

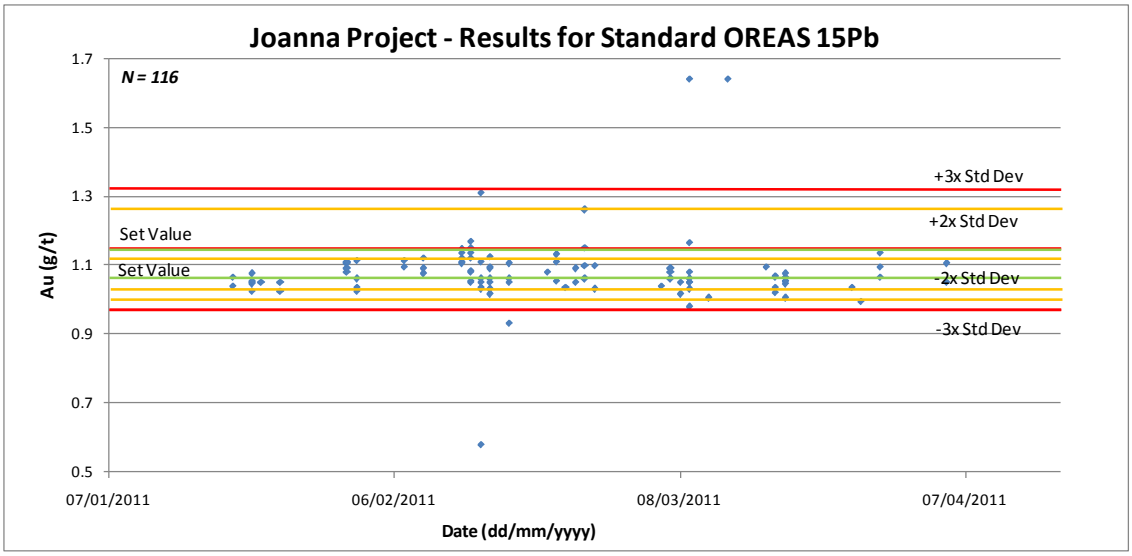


Figure 11.3: Variation of Reported Values with Time for Analytical Standard OREAS 12Pb

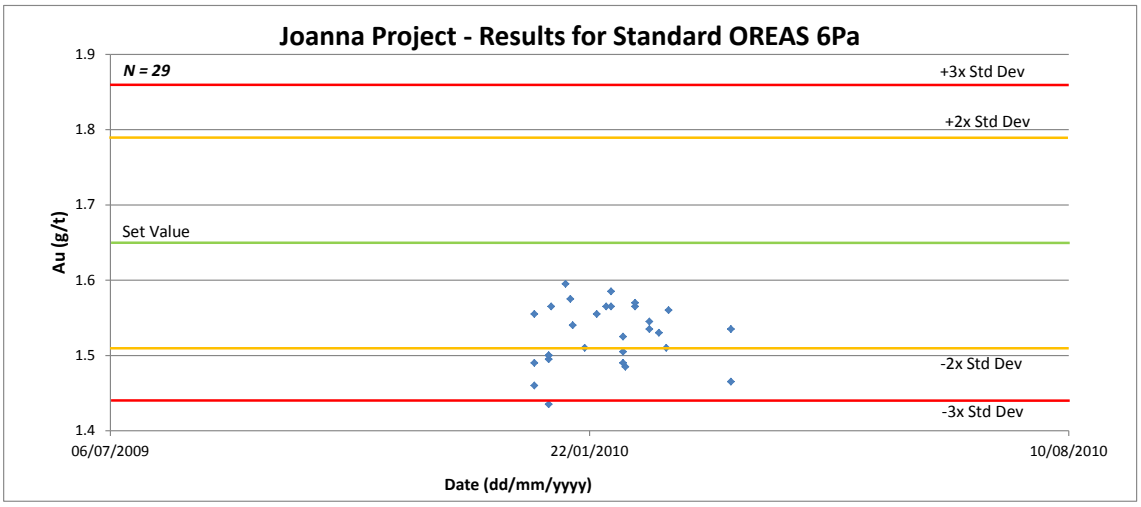


Figure 11.4: Variation of Reported Values with Time for Analytical Standard OREAS 6Pa

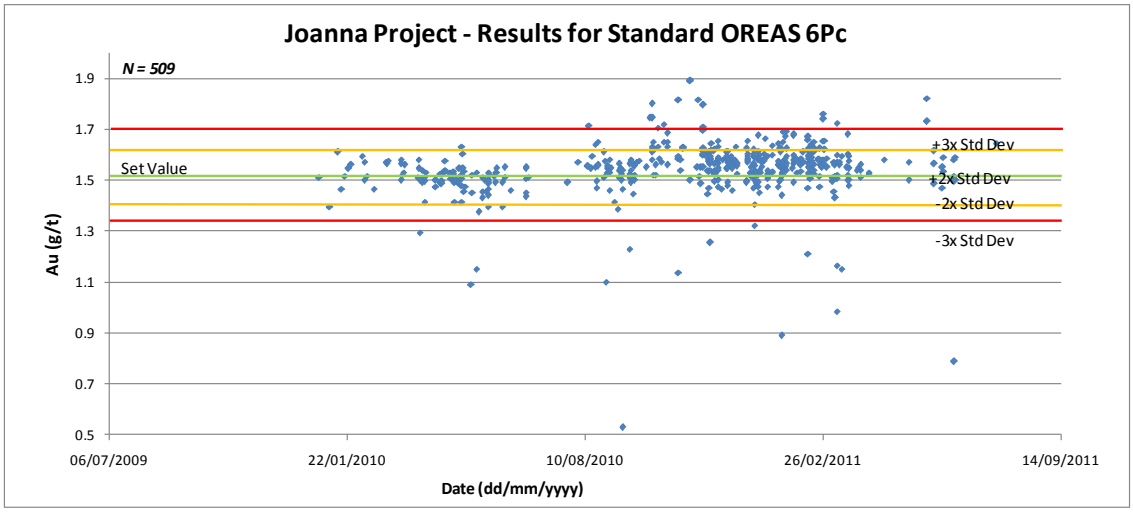


Figure 11.5: Variation of Reported Values with Time for Analytical Standard OREAS 6Pc

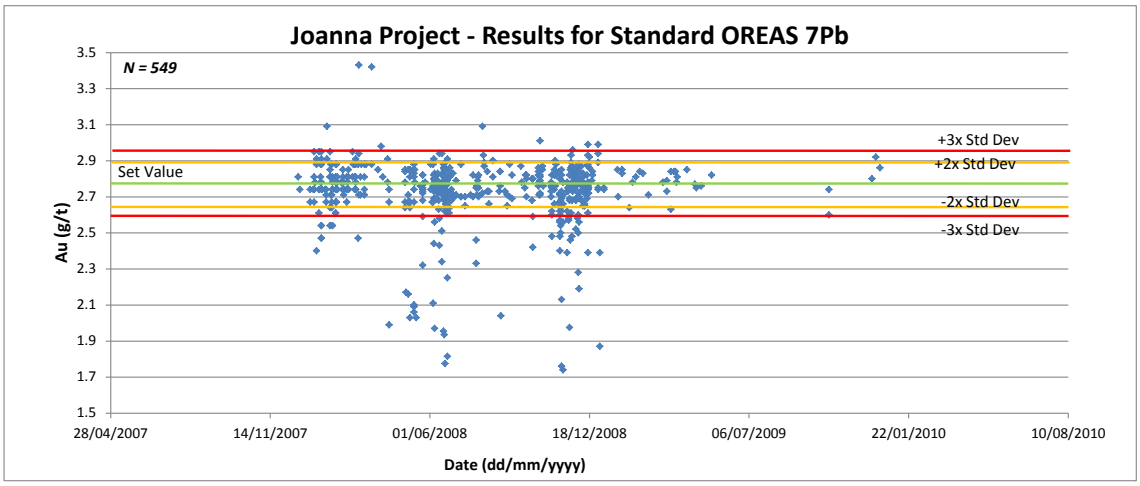


Figure 11.6: Variation of Reported Values with Time for Analytical Standard OREAS 6Pc

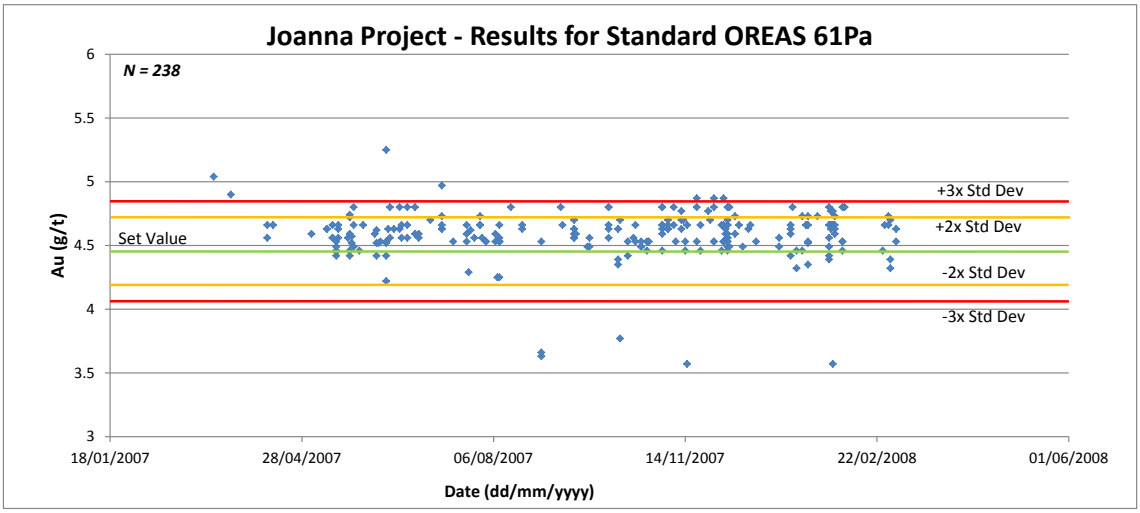


Figure 11.7: Variation of Reported Values with Time for Analytical Standard OREAS 61Pa

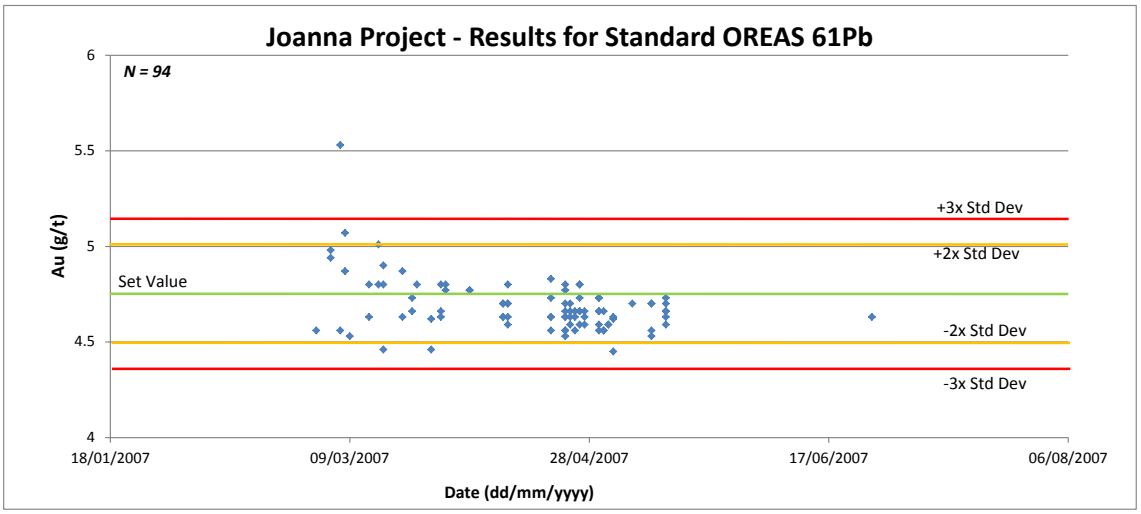


Figure 11.8: Variation of Reported Values with Time for Analytical Standard OREAS 61Pb

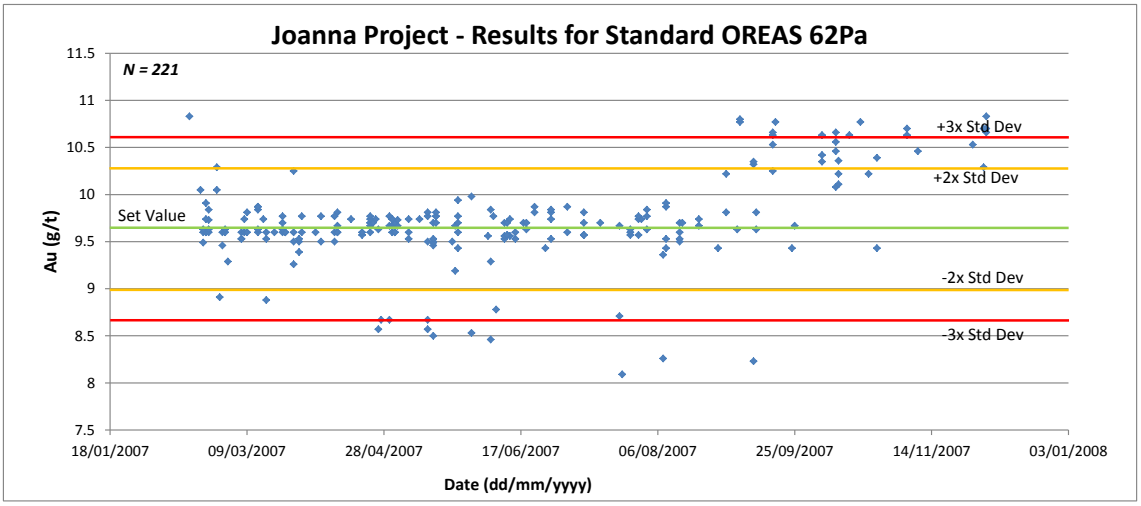


Figure 11.9: Variation of Reported Values with Time for Analytical Standard OREAS 62Pa

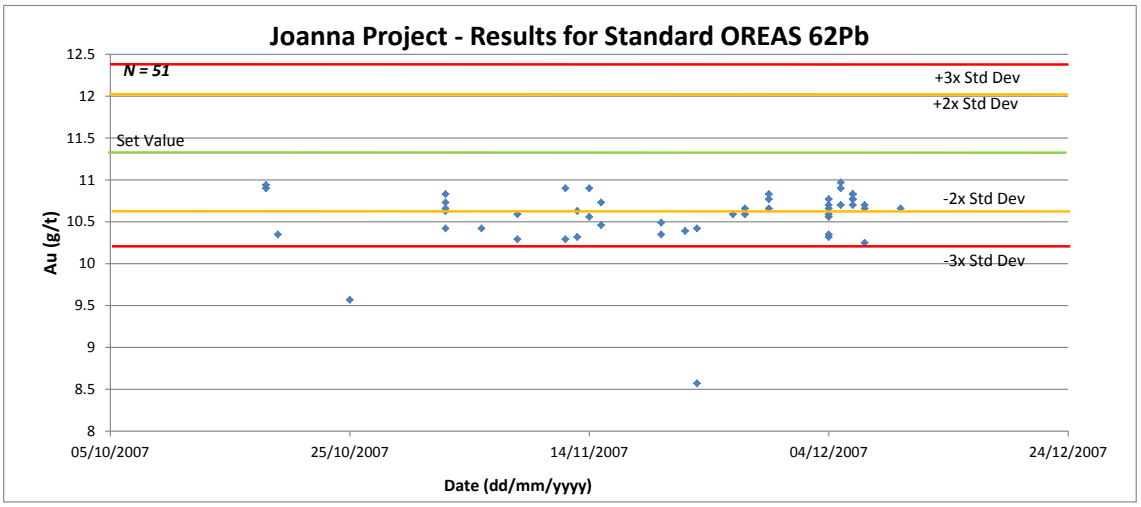


Figure 11.10: Variation of Reported Values with Time for Analytical Standard OREAS 62Pb

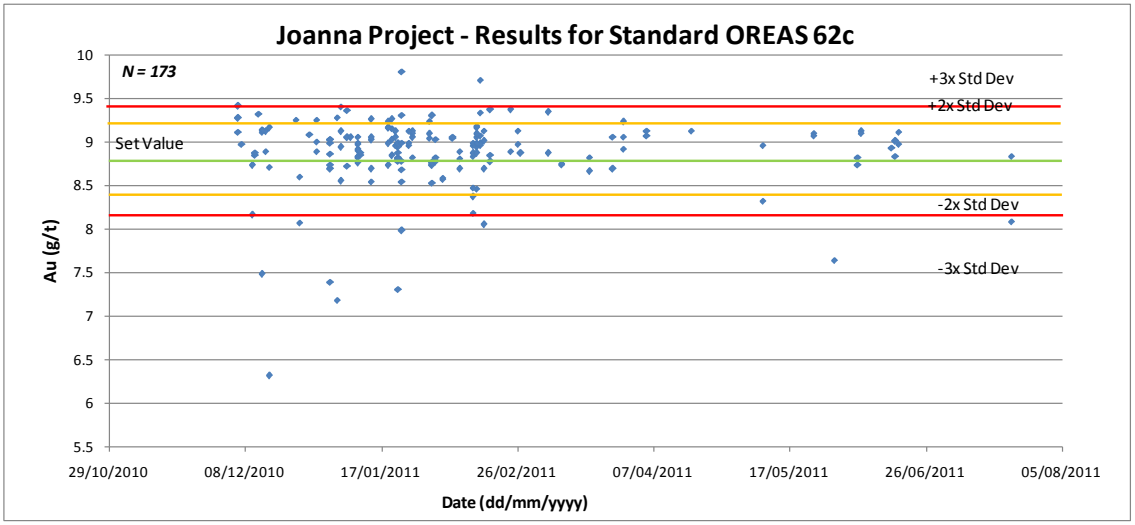


Figure 11.11: Variation of Reported Values with Time for Analytical Standard OREAS 62c

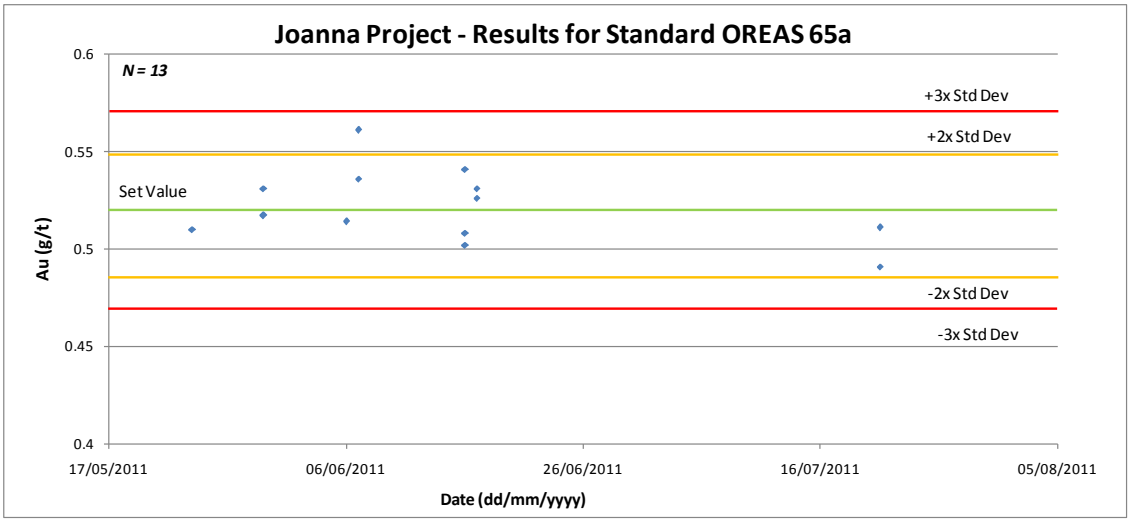


Figure 11.12: Variation of Reported Values with Time for Analytical Standard OREAS 65a

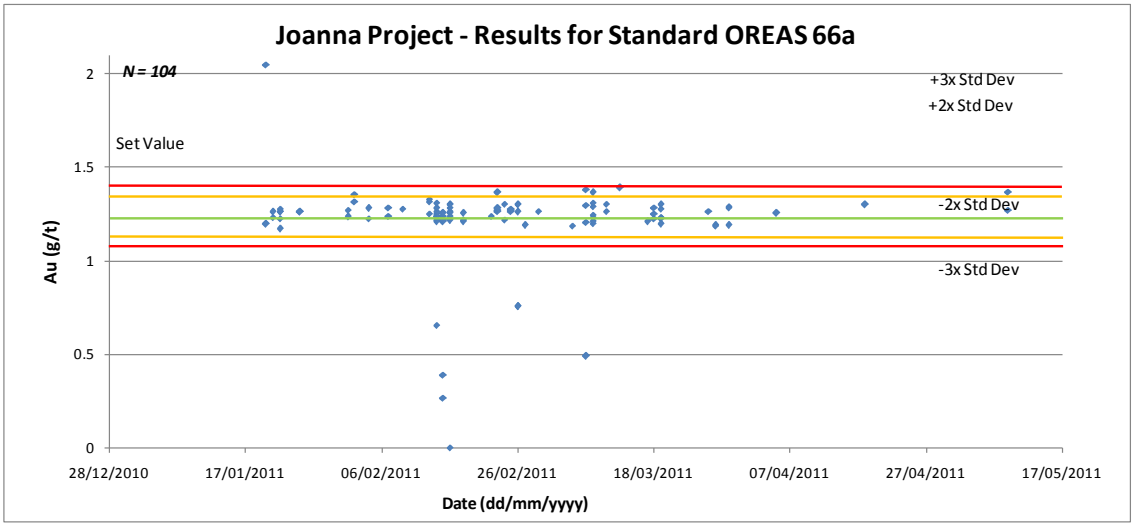


Figure 11.13: Variation of Reported Values with Time for Analytical Standard OREAS 66a

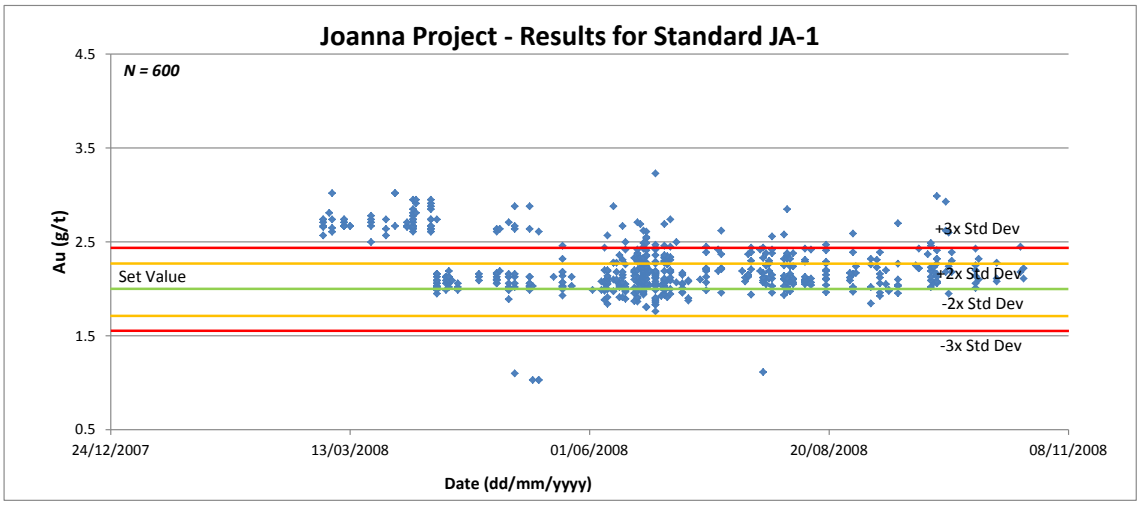


Figure 11.14: Variation of Reported Values with Time for Analytical Standard JA-1R

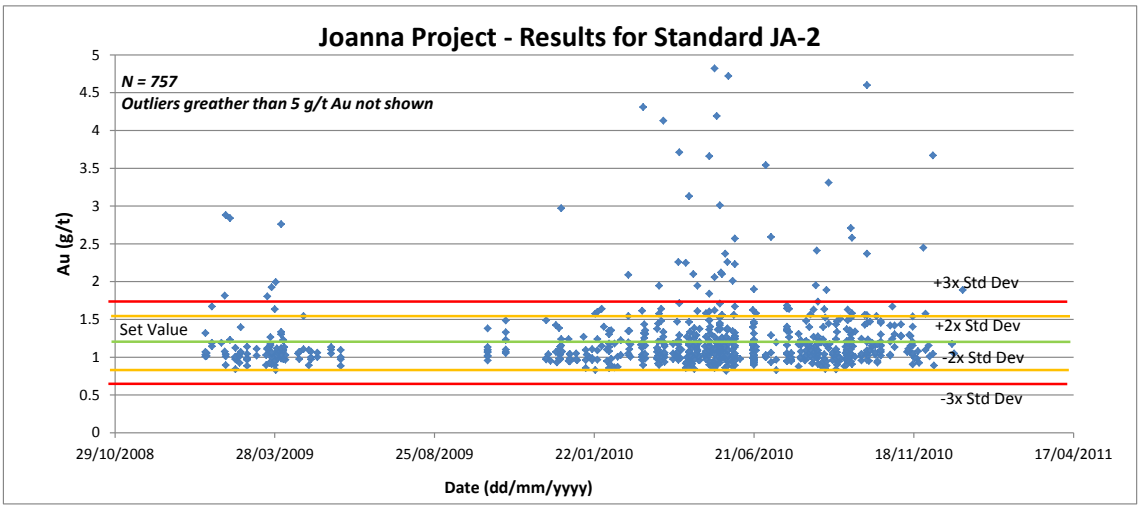


Figure 11.15: Variation of Reported Values with Time for Analytical Standard JA-2

11.2.2 Analytical Blanks

Blank reference material was composed of drill core from local barren rocks until the end of April 2010. Coarse calcareous material used as decorative pebbles (¾" in size) and purchased at a nearby hardware store in Rouyn-Noranda has been used as blank material since then. The blanks are inserted in the sample series at an average rate of 1 per 20 regular samples.

A total of 1,770 blanks were analysed as part of the sample stream since March 26, 2007. From the 1,770 blanks, 95% of them returned less than 0.025 g/t Au, which is 5 times the detection

limit of the most recent analytical method and 99% of the blanks reported values less than 0.1 g/t Au. From the 17 blanks with analytical value greater than 0.1 g/t, only 6 have analytical values greater than 0.5 g/t Au. Figure 11.16 shows the analytical results for blanks over time (results above 0.5 g/t Au are not shown in the graph). It is apparent from the graph that prior to the use of the decorative pebbles in April 2010, a significant number of blanks returned values above 5 times the detection limit. This can be explained by the fact that drill core samples thought to be barren were used as analytical blanks (some may not have been barren of gold).

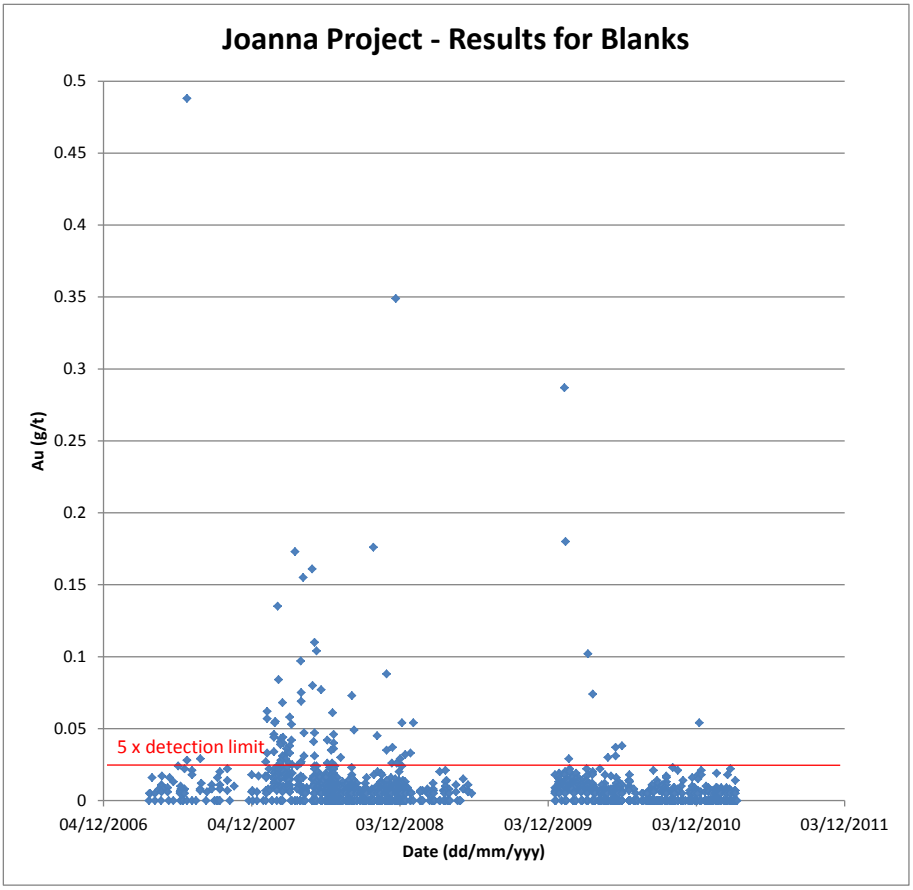


Figure 11.16: Plot of Analytical Results for Blank Reference Material Over Time

11.2.3 Core Duplicates

A total of 156 drill core duplicates were reported in the August 26, 2011 Project database provided by Aurizon. Drill core duplicates consisting of half of the remaining second half of the drill core material (quarter core) were inserted in the sample series at an average rate of 2

quarter core duplicates for every hole. The procedure of regularly inserting a drill core duplicate in the sample series was conducted for the first half of 2008 only. The statistical analysis of the Core duplicates was done only on drillholes done by Aurizon. Assay results from re-analysed older and historical core from previous owners were not included in this statistical analysis. Figure 11.17 shows a correlation plot for the core duplicates and Table 11.2 summarises the results of the statistical analysis.

Of the 156 core duplicates, the reproducibility of 72% of the assays returning values above 0.025 g/t Au was within ±20% and 76% of the assays returning values above 0.5 g/t was within ±20%. The sign test did not highlight any bias with 47% of the core duplicates returning a value greater than the original samples for Au grade above 0.025 g/t and 54% of the duplicates returning a value greater than the original samples for Au grade above 0.5 g/t.

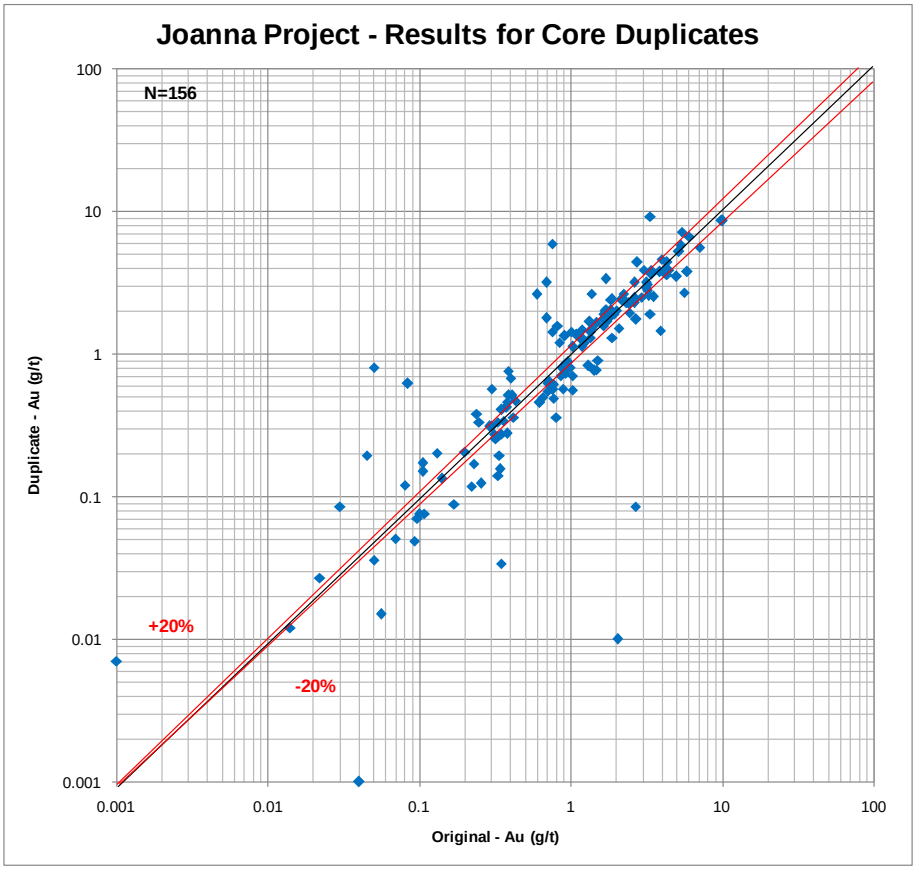


Figure 11.17: Correlation Plot of Analytical Results for Core Duplicates

Table 11.2: Summary of Analytical Results for Core Duplicates

Criteria	Count	Original > Duplicate	Original < Duplicate
All samples	156	84 54%	72 46%
> 5 x d.l. (0.025 g/t	151	80 53%	71 47%
> 0.5 g/t Au	108	55 51%	54 50%
> 1 g/t Au	81	34 42%	47 58%

Criteria	Count	Samples within % Relative Difference		
		± 10%	± 20%	± 50%
All samples	156	69 44%	110 71%	144 92%
> 5 x d.l. (0.025 g/t	151	68 45%	109 72%	143 95%
> 0.5 g/t Au	108	56 52%	82 76%	103 95%
> 1 g/t Au	81	49 60%	66 81%	78 96%

11.2.4 Reject and Pulp Duplicates

As part of the Company QA/QC protocol, preparation (or reject) and pulp duplicates representing approximately 10% of the total samples are sent for reanalysis at a second analytical laboratory. Since the beginning of exploration by Aurizon until April 2008, ALS Chemex served as the check laboratory. Since April 2008, ALS Chemex became the main laboratory and LAB Bourlamaque was selected for the reanalysis of the reject and pulp duplicates. All rejects are re-analysed using fire assay fusion and atomic absorption finish with lower detection limits of 0.005 g/t Au for ALS Chemex and 0.01 g/t Au for Lab Bourlamaque. All pulps are processed using fire assay fusion and gravimetric finish with lower detection limits of 0.05 g/t Au for ALS Chemex and 0.1 g/t Au for Lab Bourlamaque.

Reject Duplicates

A total of 5,810 preparation duplicates were reported in the June 1, 2010 Project database. The analysis of the QA/QC data was not conducted for the reject duplicates between June 1, 2010 and March 18, 2011. Figure 11.18 shows a correlation plot for the reject duplicates and

Table 11.3 summarises the results of the statistical analysis. For the 5,810 reject duplicates, the reproducibility of 79% of the assays returning values above 0.05 g/t Au was within ±20% and 84% of assays with values above 0.5 g/t reproducing within ±20%. The sign test did not highlight any bias; 54% of the reject duplicates returned a value greater than the original samples for Au grades above 0.05 g/t and 46% of the duplicates returned a value greater than the original samples for Au grades above 0.5 g/t.

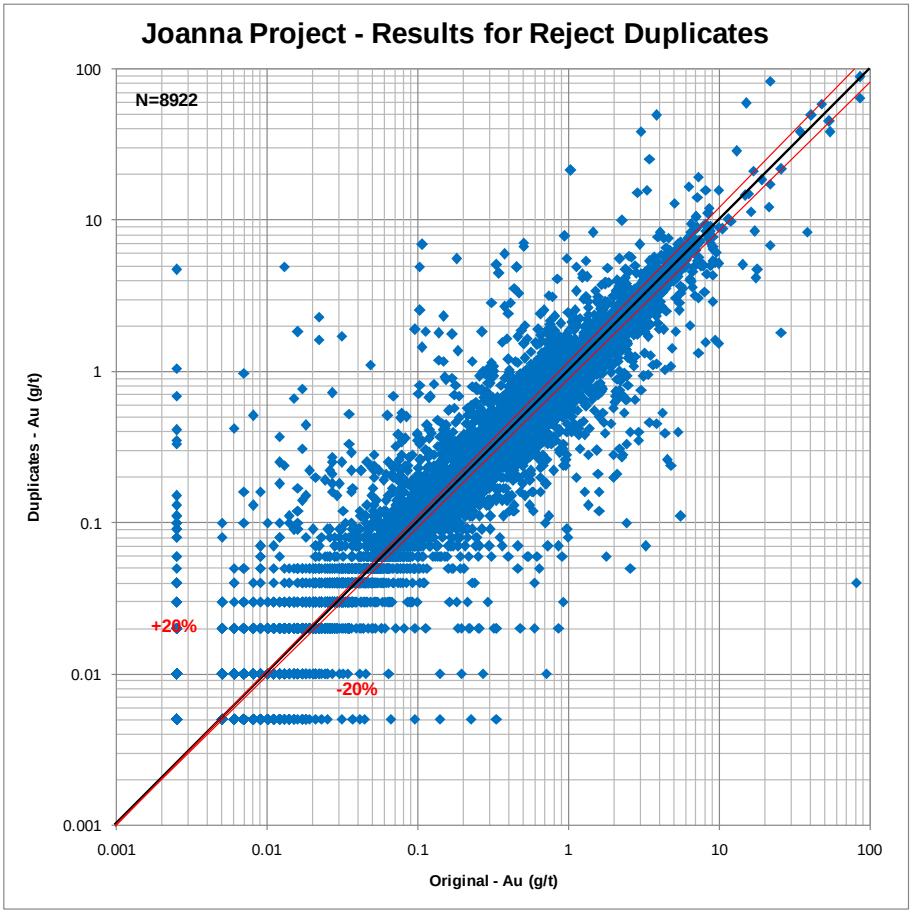


Figure 11.18: Corrrelation Plot of Analytical Results for Reject Duplicates

Table 11.3: Summary of Analytical Results for Reject Duplicates

Criteria	Count	Original>Duplicate	Original<Duplicate	Original=Duplicate
All Samples	8922	3737 42%	4933 55%	252 3%
>5-10xd.l. (0.1 g/t Au)	5937	2960 50%	2908 49%	69 1%
>0.5 g/t Au	2871	1548 54%	1296 45%	27 1%
>1 g/t Au	5137	2616 51%	2467 48%	54 1%
Criteria	**Count**	**Samples within % Relative Difference**		
		±10%	**±20%**	**±50%**
All Samples	8922	6412 72%	4640 52%	2386 27%
>5-10xd.l. (0.1 g/t Au)	5937	3825 64%	2418 41%	845 14%
>0.5 g/t Au	2871	1616 56%	913 32%	259 9%
>1 g/t Au	5137	3218 63%	1979 39%	657 13%

Pulp Duplicates

A total of 5,645 pulp duplicates were reported in the June 1, 2010 Project database. The analysis of the QA/QC data was not conducted for the reject duplicates between June 1, 2010 and March 18, 2011. Figure 11.19 shows a correlation plot for the pulp duplicates and Table 11.4 summarises the results of the statistical analysis. For the 5,645 pulp duplicates, the reproducibility of 78% of the assays returning values above 0.1 g/t Au was within ±20% and 88% of the assays with values above 0.5 g/t reproduced within ±20%. The sign test did not outline any bias; 53% of the pulp duplicates returned a value greater than the original samples for Au grades above 0.1 g/t and 49% of the duplicates returned a value greater than the original samples for Au grades above 0.5 g/t.

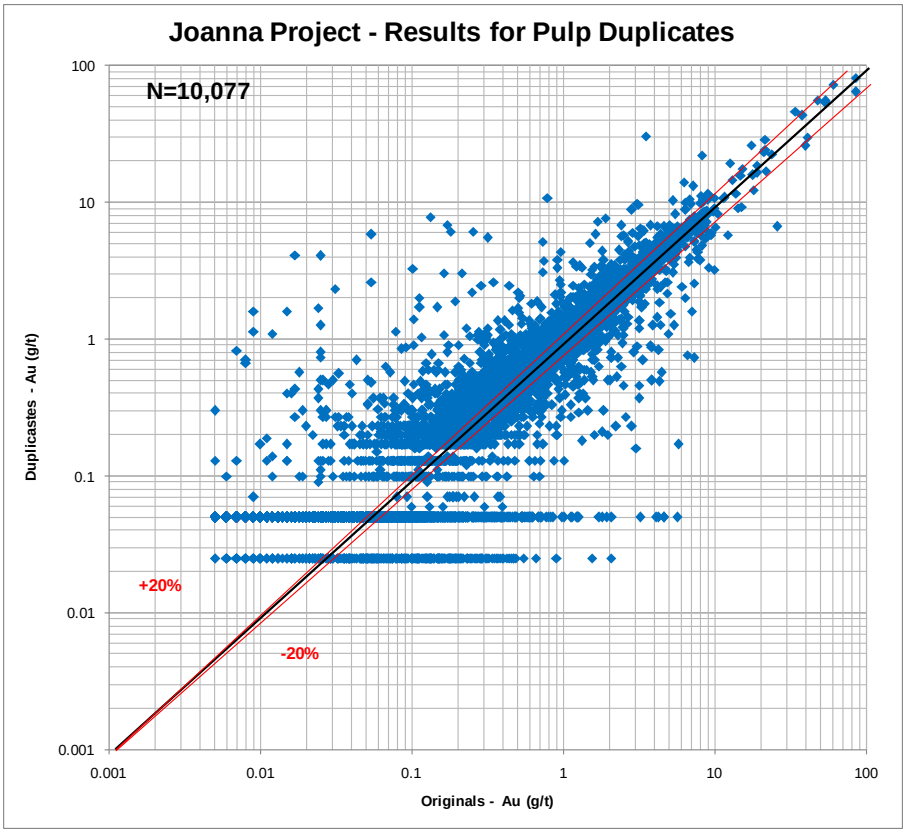


Figure 11.19: Correlation Plot of Analytical Results for Pulp Duplicates

Table 11.4: Summary of Analytical Results for Pulp Duplicates

Criteria	Count	Original>Duplicate	Original<Duplicate	Original=Duplicate
All Samples	10,077	3854 38%	6149 61%	74 1%
>5-10xd.l. (0.1 g/t Au)	5,872	2701 46%	3135 53%	36 1%
>0.5 g/t Au	3,687	1847 50%	1809 49%	31 1%
>1 g/t Au	2,238	1185 53%	1028 46%	25 1%
Criteria	**Count**	**Samples within % Relative Difference**		
		±10%	**±20%**	**±50%**
All Samples	10,077	7534 75%	6014 60%	4021 40%
>5-10xd.l. (0.1 g/t Au)	5,872	3531 60%	2194 37%	748 13%
>0.5 g/t Au	3,687	1802 49%	909 25%	226 6%
>1 g/t Au	2,238	976 44%	443 20%	95 4%

11.2.5 QA/QC Conclusion

Since the start of the exploration work on the Project in 2007, Aurizon implemented an internal QA/QC protocol consisting of the insertion of reference materials in the sample series (certified and composite analytical standards and blanks). The QA/QC program also included the reanalysis of sample reject and pulp duplicates in a second analytical laboratory for verification. Core duplicates were taken as part of the protocol in the first half of 2008.

Reported results for the certified analytical standards since the beginning of the company exploration programs on the Property show a relatively good correlation with expected mean values. The variance of the observed values is relatively high in comparison to the specifications of the certified standards but is considered acceptable with a QC failure rate ranging from 1% to 13%. The variance of the observed values for the custom-made composite analytical standard is significantly higher than the certified standards but this behaviour is expected for custom-made reference materials. Reported results for the blanks are acceptable. Observed results for the drill core, rejects and pulp duplicates show good correlation with the original analytical values and no biases are observed. The variance of the observed values for the duplicates is in line with similar gold projects.

It is the author and SGS Geostat's opinion that Aurizon is operating according to industry standard QA/QC protocol for the insertion of control samples within the sample stream of the Project. The data is considered of sufficient quality to be used for mineral resource estimation.

11.3 Specific Gravity

In 2007, the Company conducted specific gravity ("S.G.") measurements on mineralized drill core samples selected from six (6) drillholes from the JA-07 series. The S.G. measurements were completed by Lab Expert laboratory using the water displacement method. A total of 1,255 S.G. measurements were conducted on drill core samples selected from six (6) drillholes from the JA-07 series located in the Hosco sector, returning an average S.G. value of 2.66 t/m^3. From these samples, a subset of 107 core samples having an analytical gold value above 0.5 g/t also returned an average S.G. value of 2.66 t/m^3. Table 11.5 summarises the statistics of the S.G. readings on the selected mineralized drill core samples.

Table 11.5: Summary of 2007 S.G. Measurements on Mineralised Core Samples

Hosco Deposit - S.G. (t/m^3)	
Count	1255
Mean	2.66
Std. Dev.	0.09
Minimum	2.26
Median	2.67
Maximum	3.07

A new S.G. measurement program was completed in 2010 for the Hosco deposit following indication that the average S.G. values for the mineralized rocks was significantly higher than the values reported in 2007 by Lab Expert. In reports dated October 27, 2009 and April 30, 2010, S.G. measurements conducted as part of SAG mill comminution (SMC) testworks under the supervision of SGS Canada Inc. – Mineral Services (Lakefield) ("SGS Lakefield") returned S.G. values between 2.72 and 2.76 t/m^3 with an average S.G. value of 2.74 t/m^3.

In order to confirm the average S.G. value of the mineralized material from Hosco, Aurizon conducted additional S.G. measurements on selected core samples from the Hosco deposit. These additional measurements were performed by Technominex of Rouyn-Noranda, Quebec. Aurizon also conducted validation measurements on selected samples at ALS Chemex located in Val d'Or, Quebec. Aurizon mandated SGS Geostat to conduct audits of the S.G. methodologies used by Technominex and ALS Chemex.

Two data verification programs were conducted to validate the work done by Technominex with the objective of validating the work and verifying some of the S.G. values reported by Lab Expert in 2007. The first program includes the validation of S.G. values by ALS Chemex of 185 drill core samples tested by Technominex. The second verification program was conducted by SGS Geostat on 35 drill core samples tested by both Lab Expert in 2007 and Technominex in 2010.

11.3.1 ALS Chemex S.G. Data Verification

Results from ALS Chemex data verification compared to Technominex S.G. data for the same drill core samples are shown in Table 11.6 and Figure 11.20 below. The results obtained by

ALS Chemex show a very strong correlation with Technominex S.G. data and no difference in the mean S.G. value.

Table 11.6: Summary of ALS Chemex S.G. Measurements Versus Technominex Results

Hosco Deposit - SG from Core Samples (t/m3)		
	Technominex 2010	ALS Chemex 2010
Count	185	185
Mean	2.76	2.76
Std Dev	0.05	0.05
Minimum	2.70	2.71
Median	2.75	2.75
Maximum	3.01	3.04

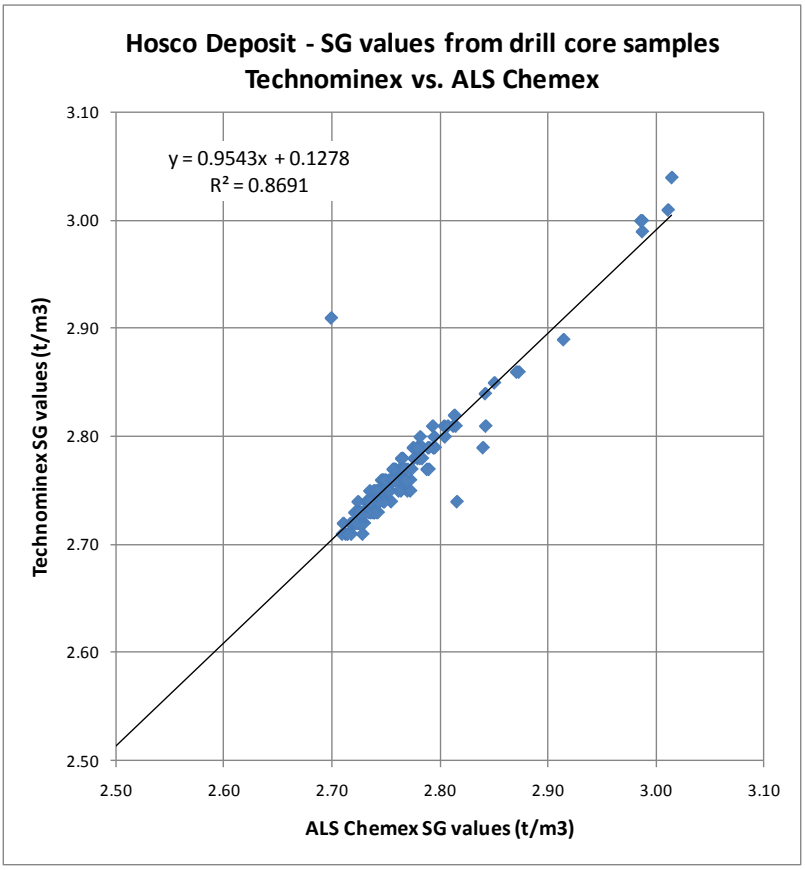


Figure 11.20: Comparative Chart Showing Technominex and ALS Chemex S.G. Results

Results from ALS Chemex data verification compared to Lab Expert 2007 S.G. data for the same drill core samples are shown in Table 11.7 and Figure 11.21 below. The results obtained by ALS Chemex show a poor correlation with Lab Expert S.G. data and a significant difference in the mean S.G. value (more than 3% difference).

Table 11.7: Summary of ALS Chemex S.G. Measurements Versus Lab Expert Results

Hosco Deposit - SG from Core Samples (t/m3)		
	Lab Expert 2007	ALS Chemex 2010
Count	121	121
Mean	2.65	2.76
Std Dev	0.07	0.03
Minimum	2.45	2.71
Median	2.66	2.75
Maximum	2.79	2.91

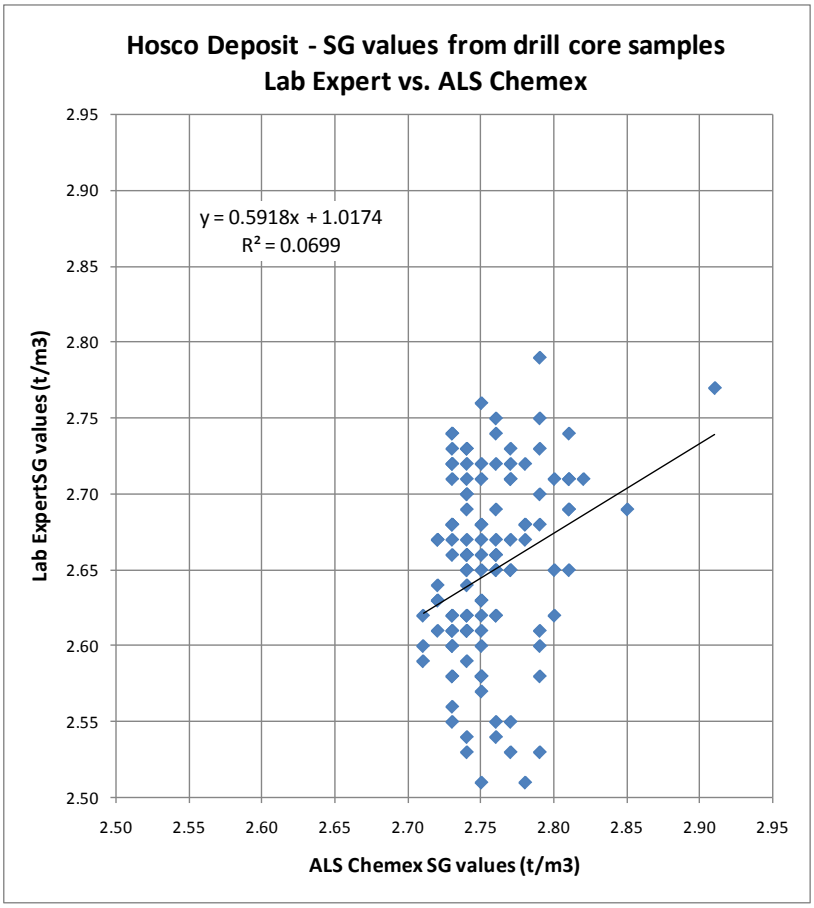


Figure 11.21: Comparative Chart Showing Lab Expert and ALS Chemex S.G. Results

11.3.2 SGS Geostat S.G. Data Verification

SGS Geostat conducted an independent data verification program during the visit at Technominex work facilities on September 2010. The program included the verification of S.G. measurements from 35 NQ-size half-core samples selected from holes JA-07-12, JA-07-17 and JA-07-19. The samples selected are part of the samples characterised by Lab Expert in 2007 and Technominex in 2010. The methodology used by SGS Geostat is the water displacement method measured by a graduated cylinder. The method consists of weighing the dry sample using a balance with a precision of ±0.5 g. Following this, the volume of water displaced by the sample is measured by submerging the sample in water in a graduated cylinder precise to ±10 ml. The S.G. can then be calculated using the following formula:

SG = weight in air (kg) / volume of water displaced (l)

Results of the SGS Geostat data verification program are summarised in Table 11.8 and Figure 11.22 below.

Table 11.8: Summary Results from SGS Geostat S.G. Data Verification Program

SG Data Verification by SGS Geostat (t/m^3)			
	Lab Expert (2007)	**Technominex (2010)**	**SGS Geostat (2010)**
Hole JA-07-12			
Count	10	10	10
Mean	2.76	2.76	2.74
Std. Dev.	0.05	0.03	0.03
Minimum	2.68	2.72	2.68
Median	2.77	2.75	2.74
Maximum	2.85	2.80	2.79
Hole JA-07-17			
Count	11	11	11
Mean	2.61	2.75	2.73
Std. Dev.	0.12	0.02	0.03
Minimum	2.35	2.73	2.69
Median	2.6	2.75	2.73
Maximum	2.79	2.78	2.80
Hole JA-07-19			
Count	14	14	14
Mean	2.62	2.75	2.71
Std. Dev.	0.05	0.01	0.03
Minimum	2.52	2.72	2.67
Median	2.63	2.75	2.71
Maximum	2.69	2.76	2.77
Overall 3 holes			
Count	35	35	35
Mean	2.66	2.75	2.73
Std. Dev.	0.10	0.02	0.03
Minimum	2.35	2.72	2.67
Median	2.65	2.75	2.72
Maximum	2.85	2.80	2.80

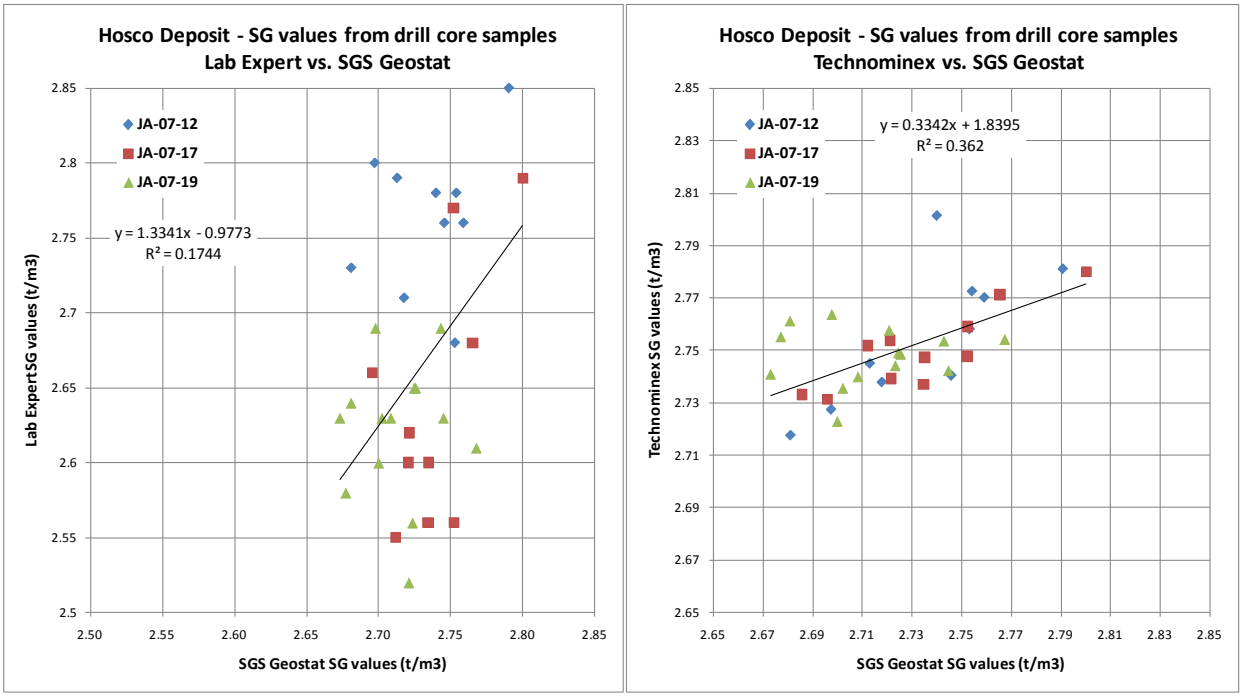


Figure 11.22: Charts Comparing Lab Expert and Technominex versus SGS Geostat SG Values

The data verification program conducted by SGS Geostat displays similar results to the verification done by ALS Chemex. The correlation between Technominex and SGS Geostat S.G. data is generally good with a difference in the mean S.G. value of less than 1%. The correlation between Lab Expert and SGS Geostat S.G. data is significantly less defined and shows a similar pattern as with the ALS Chemex data. The difference between the mean S.G. values is more pronounced with a mean S.G. value for Lab Expert 2.5% lower than SGS Geostat.

Based on the results of the data verification programs, we can conclude that the S.G. data collected by Technominex in 2010 correlated well with the check data returned by ALS Chemex, as well as the SGS Geostat verification program and can therefore be characterised as reliable for use in mineral resource estimation process. The S.G. data gathered by Lab Expert in 2007 cannot be validated with the check data gathered by ALS Chemex and SGS Geostat. The Lab Expert S.G. data must be considered unreliable and should not be used to calculate the average S.G. value for the Hosco deposit. Based on these conclusions, SGS Geostat

recommends defining a new average S.G. value for Hosco based on the Technominex 2010 S.G. data.

11.3.3 New S.G. Average Value for Hosco Deposit

During 2010, Technominex completed a series of new S.G. measurements on core sample from Hosco. A total of 1490 new S.G. values were collected from 16 different drillholes. Table 11.9 shows the summary statistics for all S.G. values collected by Technominex and S.G. values for samples above 0.5 g/t Au.

Table 11.9: Summary of S.G. Results from Technominex 2010

Hosco Deposit - 2010 SG (t/m^3)		
	All Samples	Samples > 0.5 g/t
Count	1490	357
Mean	2.76	2.76
Std. Dev.	0.002	0.002
Minimum	0.96	2.71
Median	2.76	2.75
Maximum	3.01	2.99

Based on the S.G. work conducted by Technominex in 2010 and considering the S.G. value reported by SGS Lakefield from metallurgical testing completed on composite samples, the author suggests updating the average S.G. value for the Hosco deposit to 2.75 t/m^3.

11.4 Conclusions

SGS Geostat completed a review of the sample preparation and analysis including the QA/QC analytical protocol implemented by Aurizon for the Project. The Author visited the Company core logging facilities on August 24, 2011 to review the Company sample preparation procedures. A statistical analysis of the QA/QC data for the Project did not outline any significant analytical issues.

During 2010, the Company conducted a thorough review of the average S.G. value for the Hosco deposit including 1,490 new S.G. measurements conducted on core samples and data verification completed by ALS Chemex and SGS Geostat. The new measurements outlined a significantly higher average S.G. value for the Hosco deposit than previously thought. Based on the new S.G. readings, the average S.G. value for Hosco has now been set to 2.75 t/m^3.

The average S.G. value for mineralized rocks of the Alexandria deposit has also been changed to 2.75 t/m^3 based on the new average value for the Hosco deposit. The average S.G. value for the Heva deposit is set to 2.81 t/m^3.

The author and SGS Geostat are of the opinion that the sample preparation, analysis and QA/QC protocol used by Aurizon for the Joanna project follow generally accepted industry standards and that the Project data is of sufficient quality to be used for mineral resource estimation.

12 DATA VERIFICATION

SGS Geostat completed analytical checks of drill core duplicate samples taken from selected Aurizon 2011 diamond drillholes on the Hosco sector as part of the independent data verification program. SGS Geostat also conducted verification of the laboratories analytical certificates and validation of the Hosco Sector database supplied by Aurizon for errors and discrepancies.

SGS Geostat also completed analytical checks of drill core duplicate samples taken from selected Aurizon diamond drillholes on the Heva and Alexandria sectors as part of the independent data verification program described in previous reports starting in 2007. A total of 38 independent check samples (quarter core) in 2007 and an additional 147 in 2008 as part of the 2007 and 2008 were collected during the site visits done by SGS Geostat. Despite a fairly significant scatter of original and check values, that second set of check sample data did not show any sign of bias with the average duplicate grade equal to the average original grade (both are 1.55g/t). SGS Geostat considers the data verification done on the Heva and Alexandria sectors to be current and reliable for resources estimation purposes. The 2007 and 2008 data verification full description is available in the April 7th, 2009 technical report from SGS addressed to Aurizon and available on www.sedar.com.

During the site visit conducted from August 24 to 26, 2011 by the author, Maxime Dupéré, P.Geo., a total of 38 mineralized core duplicates from the Hosco sector were collected from holes JA-11-868, JA-11-879, JA-11-917, JA-11-918, JA-11-931 and JA-11-935 by the author and submitted for Au analysis at SGS Minerals laboratory in Toronto, Ontario, Canada. The duplicate samples were processed using fire assay with ICP-OES finish (SGS code FAI323). Two blanks were inserted respectively at the beginning and ending of the sample series. One certified reference material was also inserted in the samples series (OREAS 65a). Figure 12.1 shows correlation plots for the duplicate data versus the original data. A summary of the statistical analysis conducted on the data is shown in Table 12.1.

A statistical analysis of the original and duplicate analytical values shows comparable results with 39% of the original values returning greater than the duplicate values for Au grades ranging between 0.005 g/t and 14.25 g/t.

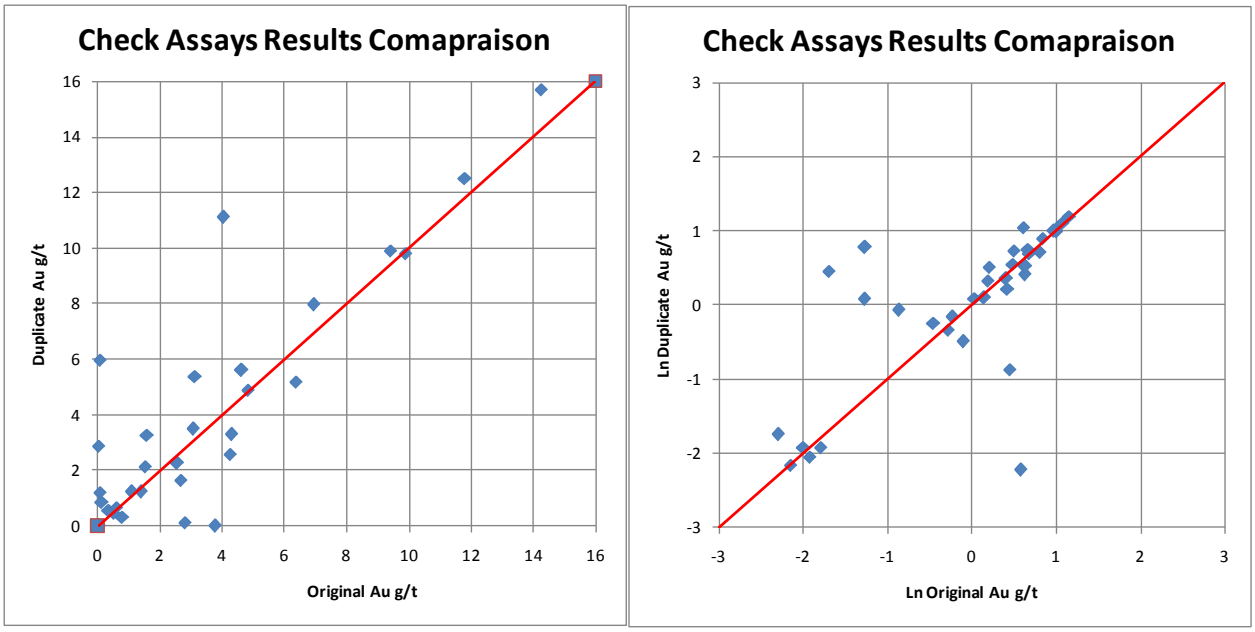


Figure 12.1: Correlation Plot for Independent Check Samples

Table 12.1: Summary of Analytical Results for Independent Check Samples

Criteria	Count	Original ≥ Duplicate	Original < Duplicate
All samples	35	14	21
		40%	60%
> 0.15 g/t	25	11	14
		44%	56%
> 0.15 g/t & < = 1.0 g/t	4	2	2
		50%	50%
> 1.0 g/t & < = 5.0 g/t	15	7	8
		47%	53%
> 5 g/t	6	2	4
		33%	67%

Criteria	Count	Sample Within % Relative Difference		
		±10%	±25%	±50%
All samples	35	6	19	22
		17%	54%	63%
> 0.15 g/t	25	5	15	18
		0.2	0.6	0.72
> 0.15 g/t & < = 1.0 g/t	4	0	3	3
		0%	75%	75%
> 1.0 g/t & < = 5.0 g/t	15	1	6	9
		7%	40%	60%
> 5 g/t	6	4	6	0
		67%	100%	0%

The digital drillhole database supplied by Aurizon has been validated for the following field: collar location, azimuth, dip, hole length, survey data and analytical values. The validation did not return any significant issues. As part of the data verification, the analytical data from the database has been validated with values reported in the laboratories analytical certificates. The total laboratory certificates verified amounts to approximately 5% of the overall laboratory certificates available for the Project. No errors or discrepancies were noted during the validation.

The final drillhole database includes historical underground channels, selected historical surface holes and all the Aurizon holes drilled in the vicinity of the Hosco deposit until hole JA-11-935. The database cut-off date is August 26, 2011. Table 12.2 summarises the data contained in the final drillhole database used for the mineral resource estimate. The author and SGS Geostat are of the opinion that the final drillhole database is adequate to support a mineral resource estimate.

Table 12.2: Summary of the Final Drillhole Hosco Database as of August 26th, 2011

Period	Type	Number of Holes	Metres Drilled	Number of Survey Records	Number of Lithology Records	Number of Assay Records	% Assayed Metres
Historical	Surface drilling	109	15326	404	324	7724	56
	Underground drilling/channels	3243	10293	1	23	8206	78
Aurizon	2007 surface drilling	62	20480	664	521	12552	81
	2008 surface drilling	356	86706	3075	2102	47481	82
	2009 surface drilling	12	6272	221	125	2740	60
	2010 surface drilling	273	57611	2085	1545	39336	88
	2011 surface drilling	54	13114	789	324	5550	84
	Total Aurizon	757	184183	6834	4617	107659	83
Total Surface		866	199509	7238	4941	115383	
Total Underground		3243	10293	1	23	8206	
Total All		4109	209802	7239	4964	123589	

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testwork has been performed at the various study stages of the Joanna Project. This Section of the Report provides an overview of the testwork as well as how the results were analyzed, interpreted and used to define the Process Flowsheet, mass balances and other process design parameters and equipment sizing. Testwork was performed in the following areas:

- Comminution
- Gravity separation of gravity recoverable gold
- Flotation
- Sulphide oxidation
- Cyanide leaching
- Thickening
- Cyanide destruction

13.1 Ore Sample Selection and Preparation for the FS

13.1.1 Samples from PFS Metallurgical Testing

Two (2) pallets of bags containing samples representing the North and South zones of the Joanna deposit were received at SGS on July 2nd, 2008 (SGS October 2009). Five (5) composite samples as well as a Master Composite were made, as shown in Figure 13.1. Most of the composite samples 1 to 5 were used for the PFS testwork however, a portion of each of these composite samples was sent to Knelson in order to have gravity recoverable gold (GRG) testwork performed for the Feasibility Study (FS). The Master Composite sample was used to perform flotation optimization testwork at SGS. The Master Composite sample head assay is shown in Table 13.1.

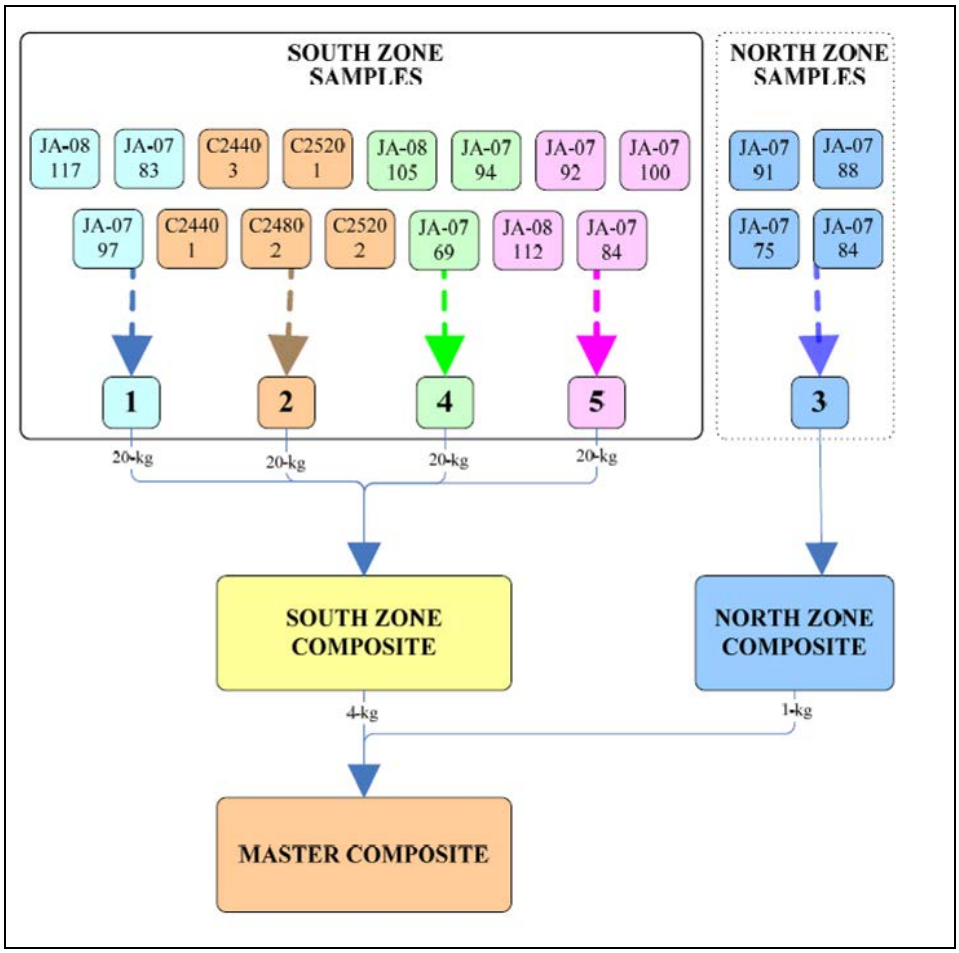


Figure 13.1: Composite Sample Preparation

Table 13.1: Master Composite Head Assay

Assay (g/t)		Assay (%)			
Au*	Ag	As	Fe	S	S=
1.52	N/A	0.082	3.96	0.87	0.43

* Calculated average of Composite 1 to Composite 5

13.1.2 Samples for FS Comminution Testing

Twenty-seven (27) bags weighing approximately 20-kg, representing samples from the Joanna deposit, were received at SGS on February 16th, 2010 (SGS April 2010). The shipment

contained seven (7) samples from which six (6) consisted of NQ half-core and one was a full HQ core. The sample identification numbers are shown in Table 13.2. A description of the geological zones referred to in this table is found in section 7.

Table 13.2: Samples Used for Comminution Testwork

Sample ID	Description
GT-02	Bulk sample prepared from core size HQ
ZN JA-08-379	North Zone
ZS2 JA-08 314 & 315	South Zone 2
ZS2 JA-08 358 & 359	South Zone 2
ZS2 JA-08 315 & 316	South Zone 3
ZS2 JA-08 358	South Zone 3
Dilution JA-08-343	Sample below cut-off grade

13.1.3 Sample for FS Flotation Pilot Plant Testing (Albion)

A shipment comprising 22 pallets of drill core samples from the Joanna deposit was received at the SGS Lakefield site on November 9th, 2009 (SGS March 2010). The total weight of the shipment was approximately 11 tonnes and contained samples from two (2) different zones of the deposit. Approximately 10 tonnes originated from the South Zone was used in the pre-pilot plant laboratory test program and the FS pilot plant for producing feed for subsequent Albion pilot plant testing. The remaining one tonne of sample originated from the North Zone was not used in this test program. The head assay for the South Zone composite is presented in Table 13.3. The primary objective of this test program was to produce a sufficient quantity of concentrate for subsequent Albion Process testwork but also to evaluate flotation performance at a pilot scale for subsequent scale-up for process development. The testwork for the Albion Process development was under the responsibility of Core Resources and Xsrata Technologies.

Table 13.3: South-Zone Composite Head Assays for Pilot Plant Flotation

Assay (g/t)		Assay (%)			
Au	Ag	As	Fe	S	S=
1.34	0.6	0.29	3.74	0.66	0.67

13.1.4 Sample for FS Flotation Pilot Plant Testing (POX)

Two (2) shipments, one (1) consisting of 19 tonnes of NQ drill half-core (referred to as DC) and the other consisting of 22 tonnes of reverse circulation (referred to as RC) material were sent to Inspectorate Exploration & Mining Services Ltd. – Metallurgical Division's Richmond Lab on November 16th and 17th, 2010 (Inspectorate February 2011). The DC and the RC material were processed separately. The primary objective was to produce a representative flotation concentrate of sufficient quantity to be used for testwork and process development in a pressure oxidation (POX) pilot plant. The testwork and process development for POX was under the responsibility of Sherrit Technologies. The head assays of the DC and RC materials are presented in Table 13.4.

Table 13.4: RC and DC Material Head Assays

Sample	Au (g/t)	As (%)	Fe (%)	S (%)	S^{-2} (%)
DC	1.11	0.258	4.20	0.73	0.71
RC	1.19	0.385	3.96	0.65	0.63

13.1.5 FS Grindability and Flotation Variability Testing Samples

Twelve (12) drill half-core samples were sent to SGS for grindability and flotation variability testing (SGS April 2012). The purpose was to evaluate how ore variability affects grindability parameters and flotation behavior of the ore over the deposit. The head assays of the variability composite samples are presented in Table 13.5.

Table 13.5: Head Assays of the Variability Testing Samples

Compostite #	Au (g/t)	As (%)	S (%)
1	0.89	0.26	0.69
2	1.10	0.19	0.57
3	1.26	0.37	0.74
4	1.38	0.20	0.62
5	0.74	0.31	0.12
6	1.48	0.42	0.52
7	0.67	0.13	0.54
8	0.92	0.25	0.84
9	1.12	0.43	0.64
10	0.98	0.52	0.72
11	1.00	0.25	0.74
12	0.77	0.21	0.78
Average	1.03	0.30	0.63
Standard Deviation	0.254	0.117	0.186
Relative Standard Deviation	24.7%	39.6%	29.7%

The Au head grades were in the range of 0.67 g/t to 1.48 g/t. The average gold head assay was considerably lower than the average LOM head grade.

13.1.6 Sample Locations and Representativity

Samples taken for SMC and Drop weight comminution tests originated from two fairly concentrated areas of the pit. The results from these tests were used to determine SAG mill and Ball mill size and power. Considering that these samples were from concentrated areas of the pit, twelve (12) additional samples were taken in a more spread out pattern in order to perform hardness variability tests (done using the SPI test method). Considering the relatively small quantity of samples, it is recommended that more samples be taken and additional full JK Drop Weight tests and SMC tests be performed to validate mill sizes and power. Samples should be taken to match the mine plan phases in order to validate ore hardness in the first months and years of operation in a more detailed manner.

The material used for making the flotation test pilot plant composites were generally well distributed within the pit shell and were deemed to be a representative blend of the ore. The purpose of the flotation pilot testwork was to produce a sufficient quantity of concentrate for performing subsequent POX testwork. In order to make enough material for the pilot scale testwork, about half of the material came from RC material, which originated from only 3 concentrated areas near the pit surface. The balance of the material came from DC material which was derived from a very large area of the pit, both in plan and in depth. During flotation, the two materials were treated separately to produce separate concentrates. Subsequently, the two concentrates were also processed separately in the POX testwork.

13.2 Comminution Testwork

13.2.1 PFS Comminution Testwork Results

As part of the PFS testwork program and flowsheet development process, five (5) metallurgical composites, as described in Section 13.1.1, were submitted for standard Bond ball mill grindability tests and SAG Mill Comminution (SMC) tests (SGS October 2009). The Bond ball mill work index (BWI) for the five (5) composites ranged between 12.2 and 14.3 kWh/t.

The SMC test generates an estimated Drop-Weight Index (DWI), which provides a measure of the strength of a rock when broken under impact conditions. The DWI's are directly related to the JK rock breakage parameters A and b, which are required for the JKSimMet comminuting modeling. The DWI for the five (5) metallurgical Composite samples ranged from 7.0 - 8.0.

The value of A x b, which is also a measure of resistance to impact breakage, was calculated and compared to compiled values in the JKTech DW database. According to this database of 2,304 recordings, the PFS grindability data for the Joanna Project suggested that the ore is in the 28th percentile of hardest ore for SAG milling.

13.2.2 FS Testwork Results

As discussed in Section 13.1.2, seven (7) samples were prepared and were subjected to the following tests (SGS April 2010):

- SMC test
- Bond rod mill grindability test
- Bond ball mill grindability test
- Bond abrasion index

One (1) of these samples (GT-02) was also analyzed using the standard JKTech drop weight impact test and Bond low-energy impact test in addition to the tests listed above, as much more material was available for this sample. The SMC test is a simplified version of the standard JKTech drop-weight test performed on rocks from a single size fraction. The SMC test was performed on all samples. The data from the JKTech drop-weight test performed on the GT-02 sample was interpreted by Contract Support Services (CSS), the North-American agent for JKTech. The test results are presented in Table 13.6 and Table 13.7.

Based on the SGS database, the GT-02 sample was categorized as very hard with respect to resistance to impact (A x b) breakage and hard in terms of resistance to abrasion breakage (t_a). The SMC and DWT results for the GT-02 sample were in agreement (about 3% difference in A x b). The other six (6) samples were characterized as hard with respect to the SGS database. The average relative density ranged from 2.73 to 2.76 g/cm^3.

Table 13.6: Grindability Test Results

Sample Name	Relative Density	DWT Results A x b	DWT Results t_a	SMC Results A x b	SMC Results t_a	SMC Results DWI	Work Indices (kWh/t) CWI	Work Indices (kWh/t) RWI	Work Indices (kWh/t) BWI	Ratio RWI/BWI	AI (g)
GT-02	2.76	26.6	0.25	27.5	0.26	9.93	16.2	17.5	14.2	1.23	0.341
ZN JA-08-379	2.76	-	-	32.1	0.30	8.64	-	16.0	14.1	1.14	0.280
ZS2 JA-08 314 & 315	2.73	-	-	30.4	0.29	9.07	-	17.1	15.7	1.09	0.436
ZS2 JA-08 358 & 359	2.74	-	-	31.2	0.29	8.86	-	16.1	14.4	1.11	0.407
ZS3 JA-08 315 & 316	2.75	-	-	31.6	0.30	8.62	-	17.6	14.5	1.22	0.405
ZS3 JA-08 358	2.75	-	-	33.7	0.32	8.08	-	15.9	13.1	1.22	0.268
Dilution JA-08-343	2.73	-	-	31.2	0.30	8.73	-	15.7	12.8	1.23	0.382

Table 13.7: Grindability Test Statistics

Statistics	Relative Density	SMC Results A x b	SMC Results t_a	SMC Results DWI	Work Indices (kWh/t) CWI	Work Indices (kWh/t) RWI	Work Indices (kWh/t) BWI	AI (g)
Average	**2.75**	**31.1**	**0.29**	**8.85**	**-**	**16.5**	**14.1**	**0.360**
Std. Dev.	0.01	1.9	0.02	0.57	-	0.8	1.0	0.066
Rel. Std. Dev.	0	6	6	6	-	5	7	18
Minimum	2.73	32.1	0.30	8.62	16.2	15.7	12.8	0.268
Median	**2.75**	**31.2**	**0.29**	**8.86**	**16.2**	**16.1**	**14.2**	**0.382**
Maximum	2.76	27.5	0.26	9.93	16.2	17.6	15.7	0.436

When comparing these test results to the results obtained in the PFS, it can be seen that the FS results indicate that the ore tested is harder for both SAG and ball milling. The Ball Mill Work Index (BWI), which ranged from 12.2 and 14.3 kWh/t in the PFS ore tested, ranged from 12.8 to 15.7 kWh/t in the FS ore tested. The DWI, which ranged from 7.0 - 8.0 in the PFS ore tested, ranged from 8.1 - 10.0 in the FS ore tested.

It should be noted that additional drilling performed during the FS period increased the Joanna deposit resource, and samples for the FS covered a larger pit shell than was the case in the PFS.

The Bond low-energy impact test determines the Bond crusher work index (CWI), which can be used with Bond's Third Theory of Comminution to calculate power requirements for crusher sizing. The CWI for sample GT-02 was found to be 16.2 kWh/t. When compared to the SGS database, the sample GT-02 was categorized as hard.

The Bond rod mill grindability test was performed at a grind of 14 mesh (1,180 µm) on the seven (7) samples. The Rod Mill Work Index (RWI) ranges from 15.7 to 17.6 kWh/t. The samples were generally categorized as moderately hard when compared to the SGS database.
The Bond ball mill grindability test was performed on the seven (7) samples. The BWI varied from 12.8 to 15.7 kWh/t. The samples were generally categorized as having a medium hardness when compared to the SGS database.

The seven (7) samples were submitted for the Bond abrasion test to determine the Abrasion Index (AI). The AI value varied from 0.268 g to 0.436 g. The samples mostly fell in the moderately abrasive range when compared to the SGS database.

13.2.3 Crusher Selection

The crusher selection for the FS was based on the CWI result obtained as part of the comminution tests performed (SGS April 2010). The CWI value used for equipment sizing was 16.2 kWh/t. The simulation software Bruno was used to size the crusher. The crusher selected is a jaw-type crusher with a feed opening measuring 1,250 mm x 950 mm and an installed power of 200 kW. It is fed ROM ore having F_{80} of 600 mm and generates product having P_{80} of 110 mm. The use of a gyratory crusher was evaluated but for the size of the Joanna operation, it was determined that even the smallest gyratory crusher would be too big and much more costly than a jaw crusher, which is much better suited for the application. In sizing the crusher, the crusher availability was assumed to be in the order of 65% with a nominal capacity of 545 t/h. This would provide the nominal daily throughput of 8,500 t/d and coupled with the crushed ore storage capacity, would assure continuous feed to the concentrator.

13.2.4 Grinding Circuit Modeling and Mill Selection

Based on the Joanna ore grindability test results, the grinding circuit for the Joanna ore was based on a conventional SAG/Ball Mill grinding circuit. There are several simulation methods available for sizing the grinding circuit. Based on BBA experience on other projects, the JKSimMet grinding circuit simulation method was adopted. The services of Contract Support Services (CSS) were retained to perform the following scope of work under supervision of BBA (Contract Support Services October 2010):

- Analyze available grindability testwork data, mainly in the form of grindability tests performed on the seven (7) Composite samples tested, to derive the basis and the design parameters for mill capacity calculations.
- Based on the design ore sample, using JKSimMet modeling software:
  - Determine optimum SAG and ball mill sizes for various grinding scenarios whereby ore hardness, throughput and final grind size are varied within a design envelope.
  - Evaluate the impact of incorporating pebble crushing in closed circuit with the SAG mill.
  - Investigate the effects of the aforementioned parameters and variables on mill size and power requirements.
  - Determine the most cost effective solution for grinding at the required grind size.

It is important to note that the function of the grinding circuit is to efficiently grind crushed ore to the required particle size to allow for cost effective gold recovery in the subsequent gravity gold recovery and sulphide flotation processing steps. The targeted final grind particle size is therefore determined based on gravity and flotation performance. Grinding finer requires higher power and bigger mills therefore there is an increase in both capital and operating costs as the grind gets finer. In general, finer grind provides a higher overall gold recovery but at some point, an incremental increase in grind fineness will not result in a cost effective incremental increase in gold recovery. It stands to reason therefore, that the targeted (or nominal) grind size is determined by the combined gravity and flotation gold recovery.

13.2.4.1 Selection of Design Basis Ore

Table 13.8 presents the A x b and BWI parameters for each sample as well as statistics for ore with and without dilution. The A x b parameter is inversely proportional to ore hardness (resistance to impact breakage in the SAG mill). The sample designated as "ZS2 JA-08 314 & 315" was selected as the “Design Basis Ore”. This sample has the highest BWI value and an average A x b value. For the purpose of performing a sensitivity analysis of the grinding circuit to hard ore and soft ore, samples were selected as follows. The GT-02 sample was chosen as the “Hard Ore” sample, based on having the lowest A x b value (indicating greatest resistance to impact breakage, which has the strongest influence on SAG mill performance). The ZS3 JA-08 358 sample was chosen as the “Soft Ore” sample, because it has the highest A x b value (indicating least resistance to impact breakage). It should be noted

however that the Design Basis Ore sample BWI value of 15.7 is significantly harder than either the Hard Ore (GT-02) at 14.2 or the Soft Ore (ZS3 JA-08 358) at 13.1. This indicates that the ball mill will likely be conservatively sized.

Table 13.8: Joanna Composite Sample Grindability Rankings

SAMPLE	A x b	BMWI
GT-02 (drop)	26.6	14.2
GT-02 (SMC)	27.5	
ZN JA-08-379	32.1	14.1
ZS2 JA-08 314 & 315	**30.4**	**15.7**
ZS2 JA-08 358 & 359	31.2	14.4
ZS3 JA-08 315 & 316	31.6	14.5
ZS3 JA-08 358	33.7	13.1
Dilution JA-08-343	31.2	12.8
With Dilution		
Average	30.5	14.1
min	26.6	12.8
max	33.7	15.7
median	31.2	14.2
st dev	2.4	1.0
75th %-ile	29.7	13.6
80th %-ile	28.7	13.3
Without Dilution		
Average	**30.4**	14.3
min	26.6	13.1
max	33.7	**15.7**
median	31.2	14.3
st dev	2.5	0.8
75th %-ile	29.0	14.1
80th %-ile	28.1	14.1

13.2.4.2 Comminution Modeling Design Criteria

The size and power of the grinding circuit is determined by two (2) major factors; ore grindability and throughput. Ore grindability has already been discussed. The throughput is determined as part of the plant design basis and is discussed further in Section 17 of this Report. In sizing the grinding circuit, it is important to define and to distinguish nominal conditions and design conditions. The nominal throughput considered in the grinding circuit simulations were based on a throughput of 381 t/h, which corresponds to the nominal daily throughput of 8,500 t/d at a 93% mill utilization rate. This is the same value used in the PFS.

In the FS, grinding circuit modeling was performed on a range of final product sizes of 75 µm to 135 µm. Testwork and modeling for the gravity concentrator and for the flotation circuit was also

performed on material in this size range. The results of this testwork are presented later in this section.

Simulations were conducted to determine the impact of incorporating a pebble crusher in closed circuit with the SAG mill. Although there was a positive impact on overall energy consumption, it was determined that the benefit was not sufficient for the investment required so this option was not pursued further in the FS.

13.2.4.3 Grinding Circuit Design

The grinding circuit design was based on a conventional SAG/Ball mill design. The SAG mill is in open circuit and the ball mill is in a closed circuit with hydrocyclones. The final product from the grinding circuit comes from the hydrocyclone overflow. A portion of the ball mill discharge is directed to the gravity concentrator. Grinding circuit modeling and simulations were conducted accordingly.

As stated earlier, grinding simulations were conducted for a range of final particle sizes. The gravity concentration and flotation testwork indicated that the most cost effective grind would be in the order of P_{80}=90 microns. Based on the simulations conducted, the SAG mill and ball mill were sized as shown in Table 13.9. This sizing was based on design ore (A x b of 30.4, BWI of 15.7), nominal throughput of 381 t/h nominal grind of 90 µm SAG mill power of 85% of maximum and Ball mill power of 82.5% of maximum. The simulations indicated a Ball mill motor power lower that what was chosen was possible, but it was decided that the same size motor should be used in the SAG mill and Ball mill to allow for common spares. Motor size can be further optimized in the next engineering phase, in collaboration with the selected Vendor, to choose a motor size that is matched to the full power that the mill is capable of drawing at full capacity.

Table 13.9: Summary of Mill Selections

Mill	Diameter	Effective Grinding Length (EGL)	Motor size	Specific Energy at Motor (kWh/t)
SAG	30 ft (9.14m)	9.5 ft (2.90m)	5,040 kW (6,760 hp)	11.3
Ball	18 ft (5.49m)	28 ft (8.53m)	5,040 kW (6,760 hp)	10.9

13.2.4.4 Sensitivity of Selected Mills to Ore Variations

As mentioned earlier, the grinding circuit design was based on design ore and nominal throughput and grind. Simulations were then carried out to determine the effect of ore hardness variability on throughput and on grind size. The maximum power utilization was capped at 95% of available power for both mills in these simulations, and the minimum throughput was maintained at 381 t/h. The results are presented in Table 13.10. The simulations show that for all ores, at nominal throughput, grinds that are finer than 90 µm are achievable. Also, throughput considerably higher than nominal can be achieved for both hard and soft ore, while maintaining grind at 90 µm or finer.

During the second phase of the FS, when it was decided that the pressure oxidation (POX) would be used, the availability of the plant was changed from 93% to 85% (the availability of the autoclave). At 85% utilization, the grinding circuit must process 417 t/h. Based on the data shown above in Table 13.10, the grinding circuit can accommodate this new throughput, for all ore samples tested in grindability testwork, at power draws lower than 95% of available power.

Table 13.10: Performance of Selected Mill Sizes

Ore Sample	Maximum Possible Throughput (t/h)	Finest Possible Product P_{80} (µm)
Design (ZS2 JA-08 314 & 315)	467 (P_{80} of 106 µm)	75 (381 t/h)
Hard (GT-02)	437 (P_{80} of 85 µm)	67 (381 t/h)
Soft (ZS3 JA-08 358)	489 (P_{80} of 92 µm)	59 (381 t/h)

13.2.5 Comminution Variability Testing

The objective of this testwork was to obtain an ore hardness profile within the pit. Twelve (12) Variability Composite samples, were submitted for this testwork (SGS April 2012). Following discussions with SGS, it was decided to perform SPI tests rather than SMC tests in order to provide indications of ore grindability using a different method. The tests included the SAG Power Index (SPI®), and CEET Crusher Index (CEET Ci) measurements. The SPI is a measure of the hardness. The CEET Ci, along with the SPI value, can be used to predict the SAG feed size distribution of the ore, and is measured during the SPI feed preparation procedure. The samples were categorized as moderately hard, to hard with respect to the SGS database. The testwork results are shown in Table 13.11.

The Modified Bond (ModBond) ball mill grindability test was performed on each variability composite sample. The work index was calibrated against the standard Bond work index. In order to calibrate the ModBond test the Bond ball mill grindability tests were performed at 150 mesh of grind (106 µm). Samples from the previous testing program were used for the calibration; five (5) samples, stored at the SGS Lakefield facility were recovered from storage and submitted for Bond ball mill and ModBond calibration testing. The results are shown in Table 13.11. The ModBond test was performed on the five (5) calibration samples and all fifteen variability samples. The Variability samples were classified as soft to medium in terms of hardness.

Table 13.11: Summary of Grindability Test Results for Variability Samples

Composite #	Sample ID	CEET Ci	SPI (min)	BWI (kwh/t)	MODBOND (kwh/t)
PFS 1				13.5	14.4
PFS 2				15.5	15.3
PFS 3				12.7	12.6
PFS 4				13.8	13.8
PFS 5				14.5	14.1
1	8500	9.6	142		13.9
2	7200	6.7	112		11.9
3	7400	7.6	114		13.1
4	7900	7.2	110		14.4
5	8100	4.1	132		14.2
6	8300	6.8	146		14.7
7	8700	5.7	121		14.6
8	8900	7.0	147		16.1
9	9100	9.6	165		14.2
10	9300	19.0	108		13.6
11	9500	11.8	131		16.5
12	9700	7.7	123		15.7
Average		8.6	129	14.0	14.3
Std. Dev.		3.8	18	1.1	1.2
Rel. Std. Dev.		45%	14%	8%	8%
Minimum		4.1	108	12.7	11.9
10th percentile		5.8	110	13.0	12.9
25th percentile		6.8	114	13.5	13.8
Median		7.4	127	13.8	14.2
75th percentile		9.6	143	14.5	14.7
90th percentile		11.6	147	15.1	15.9
Maximum		19.0	165	15.5	16.5

In reviewing the SPI and ModBond test results, there does not seem to be excessive variability in ore hardness. The CEET Ci exhibits a higher degree of variability much of which can be attributed to one (1) outlier.

13.3 Gravity Recoverable Gold

13.3.1 PFS Gravity Testwork

As part of the PFS, the potential for gold recovery by gravity concentration was evaluated by testing five (5) Composite samples at a grind size of ~140 µm (average P_{80}). Results from this testwork showed gold recovery varying between 16% and 40% (SGS October 27 2009). A gold recovery of 28.3% in gravity was used for the purposes of the PFS.

13.3.2 FS Gravity Recoverable Gold (GRG) Tests

Five (5) Composite samples, each weighing about 30 kg, were prepared by SGS from samples used in the PFS and were sent to Knelson Research & Technology Centre (Knelson January 2010). The objective of the testwork was to quantify GRG in the Joanna ore as well as its amenability to be recovered through intensive leaching.

The GRG value of an ore sample provides an indication of the amenability of the ore to gravity concentration. The GRG testing scheme is based on progressive particle size reduction, which allows recovery of gold as liberated while minimizing over grinding. The GRG test consists of sequential liberation and recovery stages. The E-GRG test includes screen analysis on the concentrates and last-stage tailings with assays on each size fraction. The total GRG results for each of the composites are shown in Table 13.12.

Table 13.12: GRG Results for Joanna Composites

Composite	Final Grind P_{80} (µm)	Total GRG (%)
1	69	**53.3**
2	77	**55.7**
3	63	**55.8**
4	50	**54.7**
5	68	**52.3**

Sub-samples of the composites were milled and then processed through a KC-MD3 gravity concentration unit to produce gravity concentrates which were then subjected to 24-hour intensive leach tests. From these intensive leach tests, a gold extraction of **53.4%** for Comp 1,2,4,5 and **69.4%** for Comp 3 were achieved. These relatively poor results are in agreement with similar testwork performed during the PFS testwork campaign and are attributed to refractory nature of the ore. Based on these results, the decision was made not to use intensive cyanidation on the gravity concentrate, but rather upgrade the concentrate using a shaking table and subject the table tails to oxidation pretreatment along with the flotation concentrate.

13.3.3 Gravity Concentrator Modeling and Equipment Selection

The results of the GRG tests for the five (5) Composite samples were used by Knelson to predict the gold recovery that can be expected by the use of a gravity concentration circuit. Knelson has developed a size-by-size mathematical model to predict gravity recovery within a grinding circuit. The model has been used numerous times for both greenfield as well as operating plants, which has allowed Knelson to compare predicted results with actual results. The model is also a very powerful tool for determining the dominant factor(s) affecting gravity performance, and for optimizing a gravity circuit once it is commissioned. The modeling results are presented in Figure 13.2.

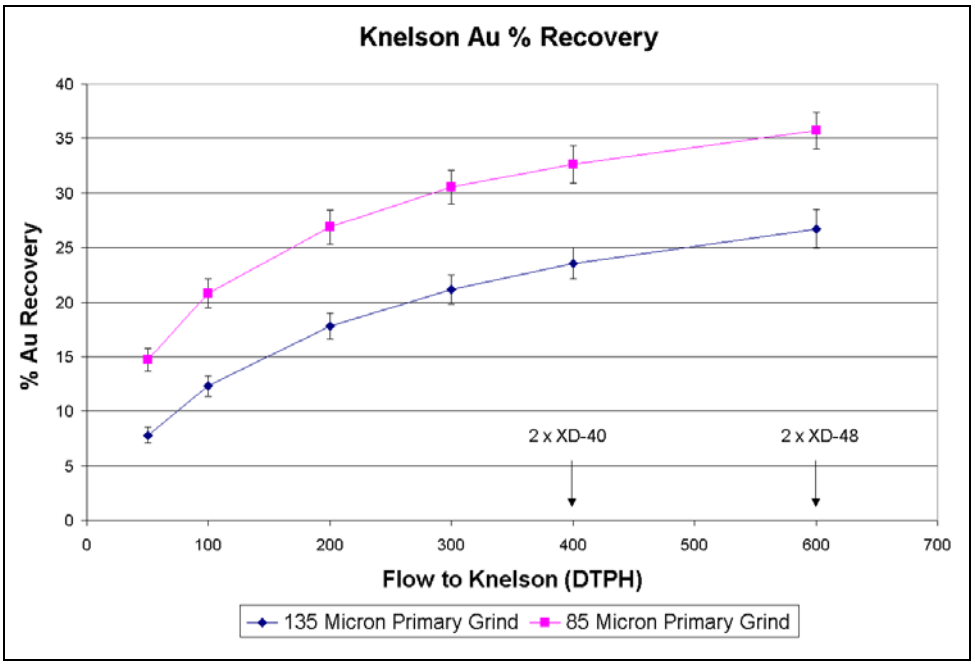


Figure 13.2: Knelson Gravity Concentration Modeling Results

From the results, it is apparent that over the range modeled, finer grinds and higher flows to gravity concentration yield higher gold recoveries. Also, the incremental benefits become lower with higher flow rates. For this reason, Knelson recommended diverting 300 t/h or 400 t/h to the gravity concentrators. These flows correspond to two (2) XD-30 units or two (2) XD-40 units (Knelson December 2009).

The tails from the Knelson concentrator are to be pumped back to the hydrocyclones and most are eventually processed in the flotation circuit. There was a concern that some of the GRG may be lost in the flotation circuit, therefore it was decided that two (2) XD-48 units, having the capacity to process 600 t/h, should be used in order to maximize GRG recovery. Furthermore, considering the marked effect of grind size on GRG recovery, a finer grind was favoured, hence a P_{80} of 90 microns was the selected grind. By interpolation, it was estimated that GRG recovery would be in the order of 34.8%. This value was used in the process flowsheet mass balance.

A mass pull in the gravity concentrators of 0.015% was assumed in the process design for development of the mass balances, based on discussions with Knelson. This figure is typical for the operation of the full-scale units.

13.4 Flotation

13.4.1 PFS Testing

The results of the locked cycle flotation tests undertaken for the PFS are presented in Table 13.13 (SGS October 2009). The tests were performed on the South Zone composite sample (as described in Section 13.1.1). The 5.5% weight recovery for the locked cycle test was higher than the previous flotation test performed with a cleaner step in open circuit (+/-2.5%). Further optimization tests to reduce the weight recovery and increase sulphur content, while maintaining gold recovery, were recommended for FS testwork. Discussions between BBA and SGS indicate that the weight recovery could be decreased by optimizing the flotation circuit.

Table 13.13: PFS Flotation Locked-Cycle Test Results

Product	Mass %	Assays		% Distribution Au		
		Au, g/t	$S^{=}$, %	Flot	Grav + Flot	$S^{=}$
Gravity Concentrate	*0.084*				*25.7*	
2nd Cleaner Concentrate	**5.50**	**21.5**	**14.4**	**91.0**	**93.3**	**93.8**
1st Cleaner Scavenger Tailing	9.45	0.19	0.11	1.41		1.19
Rougher Tailing	85.1	0.12	0.05	7.60		5.02
* Combined Tailing	94.5	0.12	0.06	9.02		6.22
Calculated Head (Flot)	100.0	1.30	0.85	100.0		100.0
Calculated Head (Flot+Grav)		1.64				

* Rougher + 1st Cleaner Scavenger Tailing

13.4.2 Laboratory-Scale Testwork for the FS

13.4.2.1 Optimization Program

In an effort to improve on the gold recovery and the flotation selectivity compared to the PFS testing, the Master Composite prepared during the PFS was subjected to a series of rougher kinetics and batch cleaner tests (SGS March 2010). Flotation conditions that were varied included flowsheet configuration, primary grind size as well as reagent suite and dosages.

A total of four (4) rougher kinetics tests were completed on the Master Composite. Based on the results of these tests, it was recommended that the strong frother DF250 be substituted with MIBC because it was found that DF250 often ends up in the circulating loads during closed circuit operation and creates a froth that is very wet and difficult to control. Furthermore, MIBC is less expensive and more readily available. The rougher kinetics tests also revealed that between 65% and 85% of the Au units were recovered during the first four (4) minutes of flotation at S grades between 14% S and 23% S.

A series of eight (8) open circuit batch cleaner tests were also completed on the Master Composite to evaluate different circuit and reagent configurations. Two (2) main conclusions were drawn from the results of these tests:

- The best batch-cleaner test results were obtained with a finer primary grind and no regrind stage.
- As observed in the rougher kinetics tests, the concentrate of the first rougher stage contains a very high proportion of the gold and adequately high sulphur content. In one (1) test, the first incremental rougher concentrate recovered almost 76% of the Au units and had a concentrate grade of over 20% S, which is well above the minimum grade target of 15% S for the Albion Process. This suggested that the concentrate from the first rougher stages can bypass the cleaner stages in operation to increase cleaning capacity for the remaining concentrate and minimize the risk of gold loss in cleaner stages.

13.4.2.2 Locked Cycle Testing

A total of ten (10) locked cycle tests (LCT) were completed on the five (5) composites from the PFS (SGS March 2010). Each composite was subjected to a test at a fine grind size target of P_{80} = 85µm and a coarse grind size target of P_{80} = 135 µm. The two (2) objectives of the LCTs were to evaluate the flotation response of the five (5) composites (i.e. evaluate effect of grind size) in a closed circuit operation and to produce concentrates for downstream hydrometallurgical testing. The flotation conditions that were developed in the optimization program were used for the LCTs.

Figure 13.3 compares S grade versus Au recovery for the 2nd cleaner concentrates. With the exception of Composite #5, the finer grind size produced a superior flotation performance. This difference in performance was most pronounced for composites #1, #2, and #4. This is in agreement with previous findings. It should be noted that this data was used in subsequent analysis to estimate flotation performance at intermediate grind sizes.

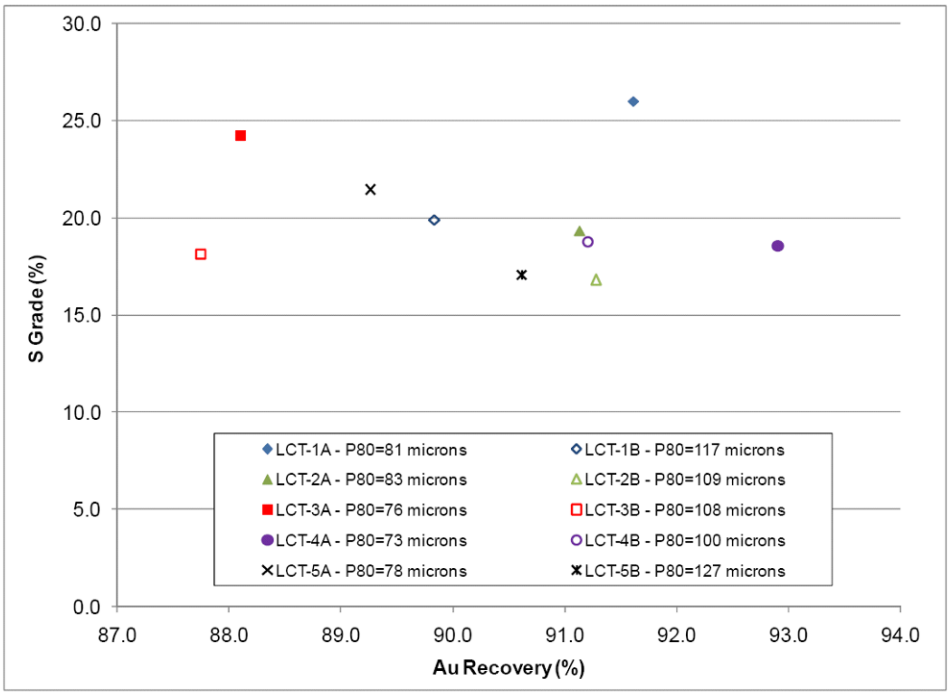


Figure 13.3: S Grade Versus Au Recovery Points for Locked Cycle Tests

13.4.2.3 Pilot Plant Benchmarking Tests

As a prelude to pilot-scale testwork, three (3) batch cleaner tests were carried out on the pilot plant composite for benchmarking purposes (SGS March 2010). The first two (2) tests used the flotation conditions that were employed in the locked cycle tests at primary grind size targets of P_{80} = 85 µm and P_{80} = 135 µm, respectively. In the third test, the frother MIBC was substituted by Dowfroth 250 at the finer primary grind size. The highest Au recovery was obtained for the first test (finer primary grind and MIBC as the frother). The Au recovery of the second test decreased by 3%, which was likely the result of an inferior liberation as a result of the coarser primary grind size. The third test, (which used Dowfroth 250 as the frother) produced an inferior

flotation selectivity and the overall Au recovery into the final Au concentrate was lower than the other two (2) tests. The test results of the three (3) open circuit batch cleaner tests on the pilot plant composites confirmed the choice of a finer primary grind size and the use of MIBC as preferable flotation conditions for the pilot plant campaign.

13.4.3 Pilot Plant Tests for the Production of Albion Feed

The objectives of this testwork were twofold; first, the pilot plant was operated at a larger scale than previously done in order to see if results previously obtained could be replicated on a larger scale to reduce risks for full plant scale-up and secondly, the pilot plant was to produce a relatively large quantity of concentrate for feed to pilot scale Albion testwork (SGS March 2010). The final concentrate S level was targeted at a minimum 15%, as required for the Albion testwork. The tailings produced in flotation were kept for subsequent environmental testwork.

The pilot plant composite sample was processed in a pilot-scale gravity separation circuit prior to feeding to the flotation pilot plant. The gravity concentrate was further processed on a Wilfley table in an effort to upgrade the material. This step removed some of the GRG from the flotation feed in order to replicate the process flowsheet envisaged.

The flotation pilot plant testwork performed at SGS processed approximately 8.6 tonnes of ore. The flowsheet that was used in the flotation pilot plant campaign is shown in Figure 13.4.

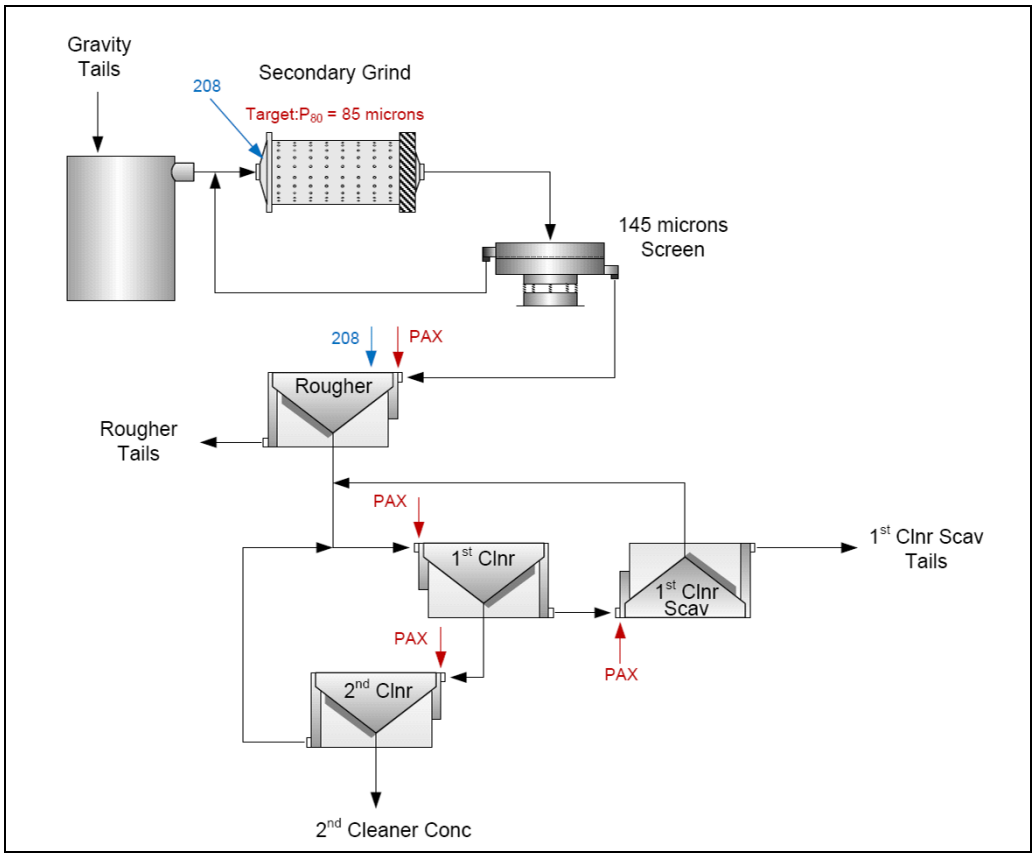


Figure 13.4: Flotation Pilot Plant Flowsheet (for Albion Pilot Test Feed)

The average grind feeding the flotation circuit over the course of the test period had a P_{80} of approximately 77 µm. The pilot plant campaign used the reagent suite that was developed in the flotation optimization program as follows:

- PAX – Potassium Amyl Xanthate – Sulphide collector – average 78 g/t;
- 208 – AERO 208 – Au co-collector (Dithiophosphate) – average 26 g/t;
- MIBC - Methylisobutyl Carbinol – frother – average 127 g/t.

These were added to the rougher and cleaner stages.

The head assays of the last six (6) surveys are shown in Table 13.14. The average adjusted flotation circuit head grade for the South Zone composite was 0.96 g/t Au. The relative standard deviation of the direct and adjusted head grades for Au was 14.0% and 12.2%.

Table 13.14: Flotation Pilot Plant Circuit Feed Head Grades

PP Run	Assays (%)					
	Au		As		S	
	Dir.	Adj.	Dir.	Adj.	Dir.	Adj.
PP-03A	0.82	0.83	0.20	0.19	0.66	0.66
PP-03B	1.06	1.06	0.24	0.24	0.71	0.71
PP-04	1.03	0.95	0.21	0.22	0.70	0.74
PP-05A	0.88	0.94	0.24	0.23	0.72	0.67
PP-05B	1.15	1.17	0.29	0.28	0.78	0.79
PP-06	0.84	0.92	0.25	0.21	0.68	0.65
Average	***0.96***	***0.98***	***0.24***	***0.23***	***0.71***	***0.70***
StdDev	***0.14***	***0.12***	***0.03***	***0.03***	***0.04***	***0.05***
Rel StdDev	***14.0***	***12.2***	***13.4***	***13.4***	***5.82***	***7.72***

A summary of the mass balances and metallurgical results of the six (6) surveys of the flotation products is presented in Table 13.15. The average Au recovery achieved in the flotation pilot plant was 90.7% with a weight recovery of 4.33% at a grind of 77 µm. The results show that the final concentrate %S was well controlled at the target of 15%.

Table 13.15: Flotation Pilot Plant Metallurgical Results

Survey	Product	wt (%)	Assays (%, g/t)			Distribution (%)		
			Au	As	S	Au	As	S
PP-03A	2nd Cleaner Conc	4.50	16.2	3.98	13.8	88.9	93.5	95.2
	1st Clnr Scav Tails	6.00	0.35	0.06	0.37	2.5	2.0	3.4
	Rougher Tails	89.5	0.08	0.01	0.01	8.6	4.5	1.4
	Feed	100	0.83	0.19	0.66	100	100	100
PP-03B	2nd Cleaner Conc	4.34	21.6	5.25	15.3	88.7	94.1	93.3
	1st Clnr Scav Tails	11.5	0.38	0.07	0.38	4.1	3.1	6.1
	Rougher Tails	84.1	0.09	0.01	0.005	7.2	2.7	0.6
	Feed	100	1.06	0.24	0.71	100	100	100
PP-04	2nd Cleaner Conc	4.49	19.0	4.61	15.0	90.1	94.4	91.1
	1st Clnr Scav Tails	4.86	0.43	0.11	0.45	2.2	2.3	3.0
	Rougher Tails	90.6	0.08	0.01	0.05	7.7	3.3	6.0
	Feed	100	0.95	0.22	0.74	100	100	100
PP-05A	2nd Cleaner Conc	4.25	20.3	5.11	14.7	91.2	95.4	92.9
	1st Clnr Scav Tails	2.77	0.33	0.07	0.32	1.0	0.9	1.3
	Rougher Tails	93.0	0.08	0.01	0.04	7.8	3.7	5.7
	Feed	100	0.94	0.23	0.67	100	100	100
PP-05B	2nd Cleaner Conc	4.59	23.6	5.81	15.7	92.5	95.9	92.0
	1st Clnr Scav Tails	4.57	0.34	0.07	0.38	1.3	1.2	2.2
	Rougher Tails	90.8	0.08	0.01	0.05	6.2	2.9	5.8
	Feed	100	1.17	0.28	0.79	100	100	100
PP-06	2nd Cleaner Conc	3.81	22.3	5.18	15.5	92.7	93.7	91.0
	1st Clnr Scav Tails	2.87	0.40	0.09	0.37	1.2	1.3	1.7
	Rougher Tails	93.3	0.06	0.01	0.05	6.1	5.1	7.4
	Feed	100	0.92	0.21	0.65	100	100	100
Average PP-03A to PP-06	2nd Cleaner Conc	4.33	20.5	4.99	15.0	90.7	94.5	92.6
	1st Clnr Scav Tails	5.43	0.37	0.08	0.38	2.1	1.8	2.9
	Rougher Tails	90.2	0.08	0.01	0.03	7.3	3.7	4.5
	Feed	100	0.98	0.23	0.70	100	100	100

The kinetics curves, as shown in Figure 13.5, clearly illustrate that the recovery of Au is complete in the Rougher #5 cell and the remaining three (3) flotation cells did not significantly improve the gold recovery. Based on the residence time calculations provided earlier, a rougher flotation time of 20 minutes in the plant appears sufficient to achieve the maximum gold recovery.

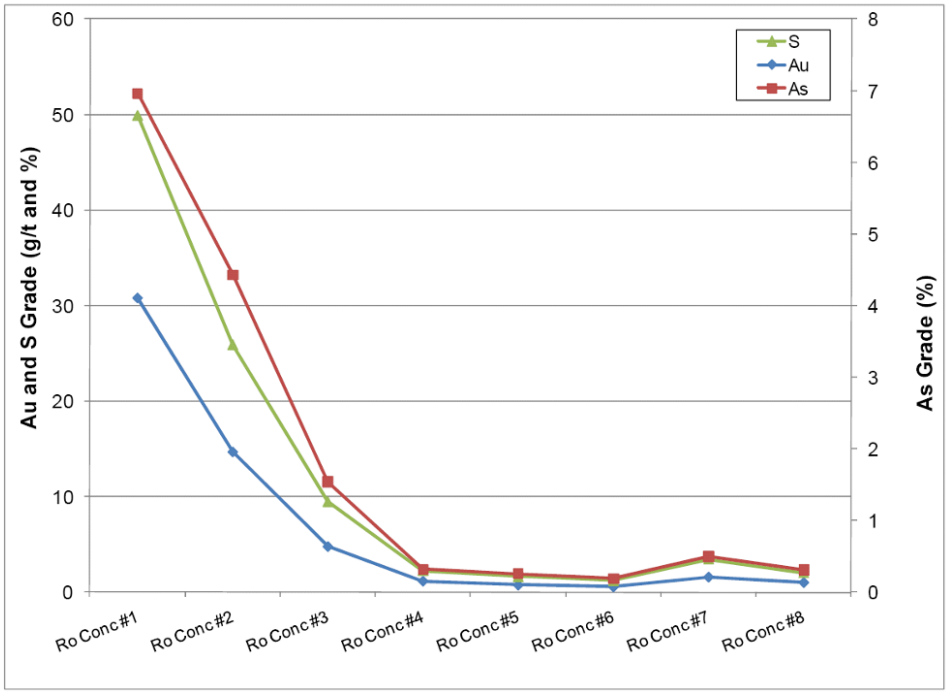


Figure 13.5: Flotation Pilot Plant Rougher Concentrate Kinetics

13.4.4 Pilot Plant Flotation for the Production of POX Feed

Following the decision by Aurizon Mines to pursue a more detailed evaluation of the POX process for flotation concentrate oxidation prior to cyanide leaching (as an alternative to the Albion Process), flotation concentrate needed to be produced in order to undertake a pilot scale test program to assess the performance of the POX process on the Joanna ore concentrate. To produce the quantity of concentrate required for the POX testwork, a pilot scale grinding/gravity/flotation circuit was required. Aurizon Mines retained the services of Inspectorate to perform this work. A total of 41 t of ore was shipped to the laboratory with the objective of producing 1.6 t of concentrate (Inspectorate February 2011).

The flowsheet for this testwork was similar to the one developed by SGS to produce the concentrate for the Albion pilot test program. The main difference between the SGS and Inspectorate flowsheet was that the latter, the second cleaner, was removed. The purpose of this cleaner was to assure a minimum concentrate sulphur level of 15%, which was necessary for the Albion Process but not for POX. The pilot plant circuit set up by Inspectorate consisted of

the integration of continuous rod mill grinding followed by gravity concentration of the rod mill product in a pilot scale Knelson gravity concentrator. The Knelson tails constituted the feed to the flotation circuit.

The average residence times achieved in the flotation circuit are presented in Table 13.16. It should be noted that flotation kinetics were not evaluated in this work.

Table 13.16: Pilot Plant Flotation Residence Times

Stage	Dry feed rate kg/hr	Equipment Vol. L	PD % Solids	Residence time min.
Rougher	148.1	216	35	40
1st Cleaner	24.4	22	8	5
1st Cleaner Scav	11.0	44	7	18

The Inspectorate pilot plant campaign used the same reagent suite that was used in the SGS pilot plant campaign. They are as follows and were added as required:

- PAX – Potassium Amyl Xanthate – Sulphide collector – average 50 g/t;
- 208 – AERO 208 – Au co-collector (Dithiophosphate) – average 30 g/t;
- MIBC - Methylisobutyl Carbinol – frother – average 208 g/t.

As described previously, the ore for the production of the concentrate for POX development was derived from drill core material (DC) and from reverse circulation material (RC). These were processed separately. The grind of the two (2) ores were similar with a P_{80} of 83 µm for the DC ore and 81 µm for the RC ore. The flotation performance results for the DC and RC ores are presented in Table 13.17 and Table 13.18. The average combined gold recovery for gravity and flotation for the DC material was 88.8% while that of the RC was 92.3%. It is important to note that Inspectorate operated the gravity/flotation circuit differently than SGS in that Inspectorate had a much higher mass pull in the gravity concentrator and a lower mass pull in flotation. Also, the behavior of the DC material was found to be quite different than that of the RC material.

Table 13.17: Gravity and Flotation Pilot Plant Metallurgical Results for DC Material

Product	Weight kg	Weight %	Assay Au, g/t	Recovery Au, %
Gravity Pre-concentration				
Table Concentrate	85.0	0.4	102.10	40.9
Table Tails	184.0	1.0	5.40	4.7
Knelson Concentrate	**269.0**	**1.4**	**35.96**	**45.6**
Flotation				
Flotation Cleaner Concentrate	520.5	2.7	17.60	43.2
Gravity + Flotation	**789.5**	**4.2**	**23.85**	**88.8**
Final Tails	18,203.5	95.8	0.13	11.2
Calculated Head	**18,993.0**	**100.0**	**1.12**	**100.0**
Measured Head			1.12	

Table 13.18: Gravity and Flotation Pilot Plant Metallurgical Results for RC Material

Product	Weight kg	Weight %	Assay Au, g/t	Recovery Au, %
Gravity Pre-concentration				
Table Concentrate	23.0	0.1	102.20	10.6
Table Tails	253.0	1.1	13.30	15.2
Knelson Concentrate	**276.0**	**1.2**	**20.71**	**25.9**
Flotation				
Flotation Cleaner Concentrate	632.5	2.8	23.20	66.4
Gravity + Flotation	**908.5**	**4.1**	**22.44**	**92.3**
Final Tails	21,301.5	95.9	0.08	7.7
Calculated Head	**22,210.0**	**100.0**	**0.99**	**100.0**
Measured Head			1.19	

13.4.5 Flotation Pilot Plant Results Comparison and Process Design Parameters

Table 13.19 shows a comparison of the gravity/flotation results of the SGS and Inspectorate pilot plant campaigns. Also presented are the process design parameters established in this FS based on all testwork results and adjusted for average LOM head grade as well as for targeted grind. In looking strictly at flotation gold recovery, the process design value is in line with the SGS value and the Inspectorate RC value and all three (3) have similar gold grade at flotation feed. The DC material exhibited a lower flotation gold recovery explained by the considerably lower flotation feed gold grade and higher gold concentration in the flotation tails.

Table 13.19: Comparison of Gravity/Flotation Pilot Plant Results and FS Design Parameters

	Process Design Criteria	SGS Pilot Plant	Inspectorate DC Material	Inspectorate RC Material
Feed to Gravity, Average P_{80} (µm)	90	77*	83	81
Au Assays (g/t)				
Avg. Head	1.25	1.34	1.12	1.19
Avg. Flotation Feed	0.85	0.98	0.61 (calc.)	0.89 (calc.)
Avg. Au in Flotation Con.	17.4	20.5	17.5	22.7
Avg. Au in Flotation Tails	0.09	0.097	0.13	0.08
S Assays (%)				
Avg. in Flotation Con.	14.5	15	14.6	17.1
Avg. in Flotation Con.(max)	18.5			
Recovery				
Gravity Au (%)	34.8	27	45.6	25.6
Shaking Table Au (%)	60	96.1	90.0	40.9
Flotation Au (%)	89.3	90.7	79.4	89.6
Gravity Weight (%)	0.015	0.29	1.4	1.2
Flotation Weight (%)	5	4.33	2.7	2.8
Residence Time				
Rougher	37	34	40	40
1st Cleaner	13	4	5	5
1st Cleaner Scavenger	8.5	7	18	18
2nd Cleaner	removed	3	-	-

*SGS pilot gravity circuit feed P_{100}=250 µm, reground before flotation

Originally, the flotation circuit was designed for sulphide oxidation using the Albion Process. This required that design take into consideration a plant utilization rate of 92.5% and flotation concentrate with a minimum of 15% sulphur. When it was decided that POX would replace the Albion Process, plant utilization was changed from 92.5% to 85% (based on the availability of the autoclave) and the second cleaner column was removed. The flotation cells and columns were designed to provide the required nominal residence times as indicated by the testwork results. Based on the flotation kinetics results, the rougher cell configuration selected is deemed

to be conservatively sized and provides flexibility for increased tonnage and/or higher residence time for more difficult ore.

13.4.6 Flotation Variability Testing

The objective of this testwork was to subject ore samples from various areas in the Joanna deposit to similar flotation conditions and to evaluate how flotation performance varies with ore variation. Each of the twelve (12) Variability Composite samples was subjected to a locked cycle test (LCT) using the flowsheet that is shown in Figure 13.6 (SGS April 2012). This flowsheet and reagent regime was based on previous testwork done at SGS. Although the commercial plant design is based on using a flotation column as the first cleaner, a single stage of cleaning in a conventional cell was used in the LCTs.

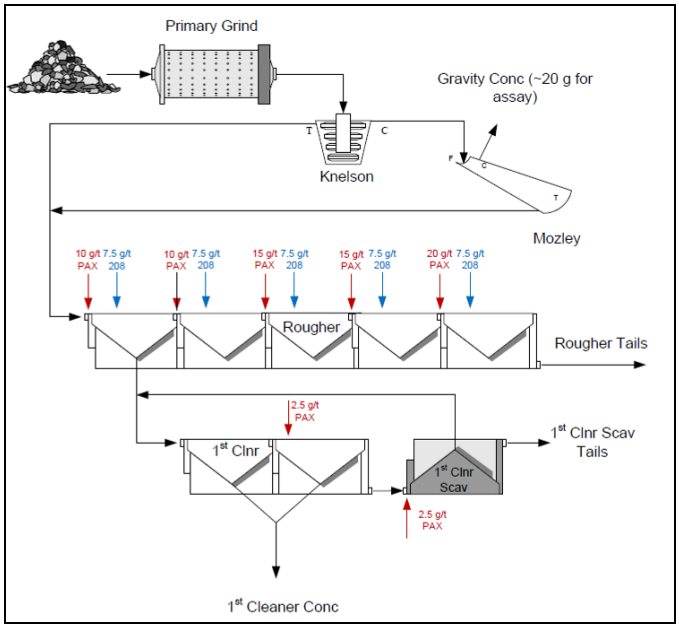


Figure 13.6: Flowsheet for Locked Cycle Testing of Variability Samples

In order to generate a consistent feed for the flotation component of the locked cycle test, the entire sample that is required for the LCT was subjected to a gravity separation stage before splitting it into the flotation test charges. After grinding the sample to a P_{80} of approximately 85 µm, the mill discharge was treated in a Knelson concentrator followed by upgrading of the Knelson concentrate on a Mozley table. Mozley tails combined with the Knelson tails and

directed to flotation. The results of the LCT on the Variability samples are summarized in Table 13.20.

Table 13.20: Flotation LCT Results on Variability Samples

Composite #	Head Grade	Weight Rec.	Combined Mozley and Flotation Concentrates					
			Assays, %, g/t			% Distribution		
	g/t	%	Au	As	S	Au	As	S
1	1.32	3.83	31.0	4.0	16.8	90.2	91.1	90.4
2	1.14	4.02	25.2	3.0	11.5	88.9	97.5	82.8
3	1.38	3.79	32.5	7.2	17.1	89.4	97.3	85.2
4	1.26	2.90	40.4	3.7	17.7	93.1	97.2	91.5
5	0.94	9.54	8.8	2.4	11.8	89.5	92.9	92.7
6	1.77	3.21	54.0	12.5	16.0	97.7	98.4	90.4
7	0.97	3.49	24.9	1.9	12.8	89.5	97.1	90.3
8	1.11	4.60	21.7	3.7	15.0	89.6	97.5	87.1
9	1.02	4.17	21.8	8.4	15.6	89.3	98.2	91.6
10	0.92	3.95	22.3	11.6	18.1	95.2	99.0	94.7
11	1.59	4.02	36.8	5.0	17.7	93.0	97.9	93.0
12	0.88	4.06	19.4	4.4	18.0	89.8	97.9	92.6
Average	**1.19**	**4.30**	**28.2**	**5.6**	**15.6**	**91.3**	**96.8**	**90.2**
Std. Dev.	0.28	1.71	11.7	3.5	2.4	2.8	2.4	3.5
Minimum	0.88	2.90	8.8	1.9	11.5	88.9	91.1	82.8
10th percentile	0.92	3.24	19.6	2.4	11.9	89.3	93.3	85.4
25th percentile	0.96	3.71	21.8	3.5	14.4	89.5	97.1	89.5
Median	**1.13**	**3.98**	**25.0**	**4.2**	**16.4**	**89.7**	**97.5**	**91.0**
75th percentile	1.34	4.09	33.6	7.5	17.7	93.0	98.0	92.6
90th percentile	1.57	4.56	40.0	11.3	17.9	95.0	98.4	93.0
Maximum	**1.77**	**9.54**	**54.0**	**12.5**	**18.1**	**97.7**	**99.0**	**94.7**

The combined gravity/flotation Au recovery varied between 88.9% and 97.7% indicating that even with varying ore conditions, response to gravity/flotation is fairly consistent. Mass pull, as

expressed in Wt %, is also generally well controlled however; statistics are influenced by one (1) outlier, LCT5. Observations made during the test indicated that concentrate appeared to be unusually slimy, which resulted in high mass pull of 9.5% and low concentrate grade at 8.8 g/t. Sulphur grade average was 14.8% and ranged between 11.5% and 18.1%, which is well within POX design.

13.5 Cyanide Leaching Process Options Evaluated

13.5.1 Direct Ore Cyanidation

Whole ore cyanidation was briefly evaluated during the PFS testwork campaign on the South Zone Composite and the North Zone Composite. Results as a whole were deemed to be poor for both the South Zone and North Zone Composites, indicating a refractory association of a significant proportion of the contained gold values. North Zone Composite gold extraction (at ~61%) was approximately 12% higher than South Zone Composite extraction (at 49%). The refractory nature of this material was verified in the mineralogical evaluation. Based on these results, whole ore cyanidation was not considered further (SGS October 2009).

13.5.2 Fine Grinding of Flotation Concentrate Followed by Cyanidation

During the PFS testwork campaign, a program to evaluate the potential to recover gold from reground flotation concentrate was initiated. The best gold extraction result achieved was 72% under intensive cyanide leaching conditions and at a P_{80} of 6 µm. This result further confirmed the refractory nature of the ore and the need for an oxidative pretreatment prior to cyanidation (SGS October 2009).

13.5.3 Oxidation by Albion Process Followed by Cyanidation

The PFS for the Joanna Property development was based on using the Albion Process, developed by Xstrata Plc, to oxidize the refractory concentrate prior to cyanidation. Preliminary testwork results were encouraging and it was recommended in the PFS to pursue the FS based on the Albion Process (SGS October 2009). A pilot plant Albion test program was defined and undertaken early during the FS using concentrate produced at SGS as part of the flotation pilot plant testwork.

The Albion Process combines ultrafine grinding using an IsaMill with oxidative leaching at atmospheric pressure. This process was deemed to be much more cost effective than the more conventional pressure oxidation process (POX).

During the PFS testwork campaign, a series of tests were carried out by SGS to determine the performance of this process on samples for the Joanna deposit. The first series of Albion tests was run on rougher concentrate material giving a gold recovery of about 83% by carbon in leach (CIL) cyanidation. Two (2) additional tests on cleaner concentrate (higher grade gold and sulphide) gave an average recovery of 92.8% by CIL. This bench scale testwork indicated a high consumption of lime and cyanide however, Xstrata Technology, based on their experience recommended that a lime consumption of 12 kg per tonne of concentrate and a cyanide consumption of 6 kg per tonne of oxidized residue be used to estimate actual consumptions for a scaled-up operation.

Prior to undertaking the full pilot scale testwork, a series of bench scale tests were conducted to identify optimal operating parameters by which to conduct the pilot testwork (Hrltesting May 2010). Results from these tests were not encouraging and showed relatively low and inconsistent gold recoveries and high acid, lime and cyanide consumptions. The best gold Leachwell recovery obtained was 90% with an associated 99% sulphide oxidation.

The pilot plant treated approximately 195 kg of concentrate in continuous mode (Hrltesting August 2010). Key parameters were monitored regularly to ascertain stability. Sulphur oxidation ranged from 68 to 94% and the (weight adjusted) mean sulphur oxidation was 82.5%. The LeachWell gold recoveries ranged from 70 to 96%. Disregarding values that were deemed outliers, for sulphur oxidation levels in the range of 80.3% – 83.8%, Leach Well gold recovery ranged between 89.4% – 93.9%. The oxidized product was forwarded to a neutralization reactor train for neutralization with limestone slurry. Gold (LeachWell) recovery from the neutralized product ranged from 84.6 to 95.7%.

Following the pilot plant oxidation of the concentrate, an extensive test program was undertaken to evaluate and optimize gold recovery as well as cyanide consumption (Core Resources 2010). The initial cyanidation results were quite poor, with gold recovery varying between 70 and 80%.

CIL (carbon in leach) was found to perform better than CIP, (carbon in pulp) indicating the presence of “preg-robbing” constituents. The best results, which approached 90% gold recovery, were achieved by leaching at higher temperatures, but this came with an extremely high cyanide consumption as well as very high lime consumption.

Considering the results obtained with the Albion Process, Aurizon Mines decided to delay the FS to allow for a reorientation towards the POX process.

13.6 Pressure Oxidation (POX) Followed by Cyanidation

Pressure oxidation (POX) is a process used to oxidize the sulphide matrix in refractory minerals at elevated pressure and temperature, causing the contained gold to be more amenable to cyanide leaching. Unlike the Albion Process which relies on ultrafine grinding followed by atmospheric oxidation, oxidation in the POX process is achieved in a high pressure autoclave, and does not require ultrafine grinding.

13.6.1 PFS Testing

A series of baseline POX tests were initially carried out during the PFS testwork campaign on flotation concentrates to verify the performance of the POX process on samples from the Joanna ore body (SGS October 2009). In general, the POX tests were successful and achieved gold extraction values above 95% by CIL leaching. Cyanide consumption was also considerably lower in the POX residue leach tests compared to the direct leach of flotation concentrate. The POX option was however not retained for the PFS because of its purported higher capital and operating costs relative to the Albion Process and also because of the encouraging results obtained with the Albion testwork. One driving factor for this decision was the comparison of oxygen consumption which was estimated to be 190 t/d for the POX process and only 26 t/d for the Albion Process, as had been estimated by Xstrata Technology at the time.

13.6.2 High Level Assessment of the POX Process for the Joanna Ore

During the course of the FS Albion Process development, Aurizon Mines contracted a third party consultant with expertise in the field to undertake an order-of-magnitude study on the applicability of the POX process to the Joanna ore concentrate based on available testwork data

(Hatch July 2010). No testwork specific to this study was undertaken therefore the study was based on data from previous testwork.

The design basis for the aforementioned study considered a POX plant having a throughput of 350 t/d of flotation concentrate at 17% sulphur. The autoclave design was based on an operating temperature of 225 °C, a retention time of 60 min, an oxygen purity of 99% and an oxygen partial pressure of 690 kPa(a). It was assumed that 85% sulphur oxidation would be required to achieve acceptable gold recovery. Plant utilization of 85% was also assumed. Based on these design assumptions, an order of magnitude estimate of CAPEX and OPEX was developed.

Based on this assessment, Aurizon Mines and BBA came to the decision that the Albion Process should not be pursued further and that the FS should be based on POX.

13.6.3 Bench-Scale Testing Prior to Pilot Plant Testwork

For the FS, Aurizon Mines mandated Sherritt Technologies to undertake testwork and develop the process flowsheet and process design for the POX area of the processing plant and concentrator (Sherritt May 2011). An advance sample of the Drill Core (DC) flotation concentrate from the Inspectorate pilot plant testwork was tested initially in the batch test program by Sherritt to determine the preferred operating conditions for the POX pilot plant campaign.

Chemical assays for the advance sample of DC flotation concentrate are provided in Table 13.21.

Table 13.21: Analysis of Advance DC Sample from Inspectorate Flotation Pilot Plant

Advance Drill Core Sample Analysis, wt% or g/t for Au and Ag								
Au	Ag	Al	As	C	C_{NAL}[1]	Fe	Si	S
20.6	1.2	4.64	5.13	0.24	0.09	21.7	17.5	13.1

[1]Non-acid leachable carbon, i.e., organic carbon

The preliminary testwork results indicated that the concentrate required relatively low acid additions (20 to 50 kg/t) and the concentrate tested in acidulation was highly reactive, resulting in the expected formation of elemental sulphur from the contained pyrrhotite. Based on these results, Sherritt determined that acidulation was not required and therefore was not included as part of the POX pilot plant flowsheet.

Thirteen batch POX rate tests were conducted, prior to the pilot plant campaign on the advance DC concentrate sample, in order to investigate the effects of feed particle size, pressure oxidation temperature, oxygen partial pressure, retention time, chloride concentration in the leach solution, and solids recycle and to determine the preferred starting conditions in pressure oxidation for the pilot plant campaign.

Batch pressure oxidation of the advance DC sample gave sulphide oxidations of over 98% within 40 minutes over a wide range of operating conditions. Corresponding cyanide amenability (CAN) gold extractions from the POX product solids were typically over 96%. The following recommendations for the pilot plant campaign were made based on the results of the batch pressure oxidation tests:

- baseline operating conditions of 60 minutes retention time, 220ºC and 500 kPa O_2 pressure were selected;
- additional grinding of the concentrate was not required to achieve higher gold extraction;
- no indication of “preg-robbing” behaviour was noted during pressure oxidation with either high chloride concentrations in solution or with a solids recycle;
- no solids recycle was suggested in the pilot plant testing due to the limited duration of the pilot plant campaign and the decision to test two feed materials;
- no appreciable difference was noted between the CNA gold extraction or the sulphate content of the solids for solids before and after conditioning for the DC materials and a relatively short conditioning time of 20 to 30 minutes was selected for continuous testing.

Additional batch POX tests were performed prior to continuous testing to determine the behaviour of the pilot plant feed samples from Inspectorate and table tails materials under the selected pressure oxidation conditions for the pilot plant campaign.

Batch pressure oxidation of the pilot plant feed samples under the defined test conditions resulted in sulphide oxidation of over 97.5% within 40 minutes. Corresponding CNA gold extractions were typically over 96% for the DC concentrate and up to 92% for the RC concentrate. The low gold extractions for the RC material led to the consideration of longer retention times in pressure oxidation (i.e., greater than 60 minutes) and higher temperatures (230ºC) for the test conditions of the pilot plant campaign. The CNA leaching conditions used for the batch test samples, including 24 h of leaching, were retained for the pilot plant campaign.

Two (2) solution neutralization batch rate tests were conducted on selected solutions that were produced in the POX batch tests. Good removal of acid and metals from the POX solutions was achieved in the two solution neutralization tests using the staged addition of limestone and lime. Based on the batch test results, a target final pH 9 and retention time of five to six hours were selected for the solution neutralization circuit in the pilot plant.

The settling behavior of the feed concentrate and POX solids were evaluated in batch tests prior to continuous testing and were both very good. The settled solids density from the POX batch settling test was used to determine initial target wash water flows to the countercurrent decantation (CCD) wash circuit for the pilot plant campaign.

13.6.4 Pilot Plant

The pilot plant operations carried out by Sherrit commenced at 6:00 pm on February 22nd, 2011 and ended at 12:00 am on March 1st (Sherritt 2011). The analyses of the feed concentrates (DC and RC) are presented in Table 13.22.

The pressure oxidation pilot plant operation was divided into five (5) periods, based on concentrate type (DC or RC), autoclave temperature (220 or 230°C), and retention time (60 or 75 min). Cyanidation tests were performed on selected POX solids samples throughout the campaign. The block flow diagram for the pilot plant is shown in Figure 13.7.

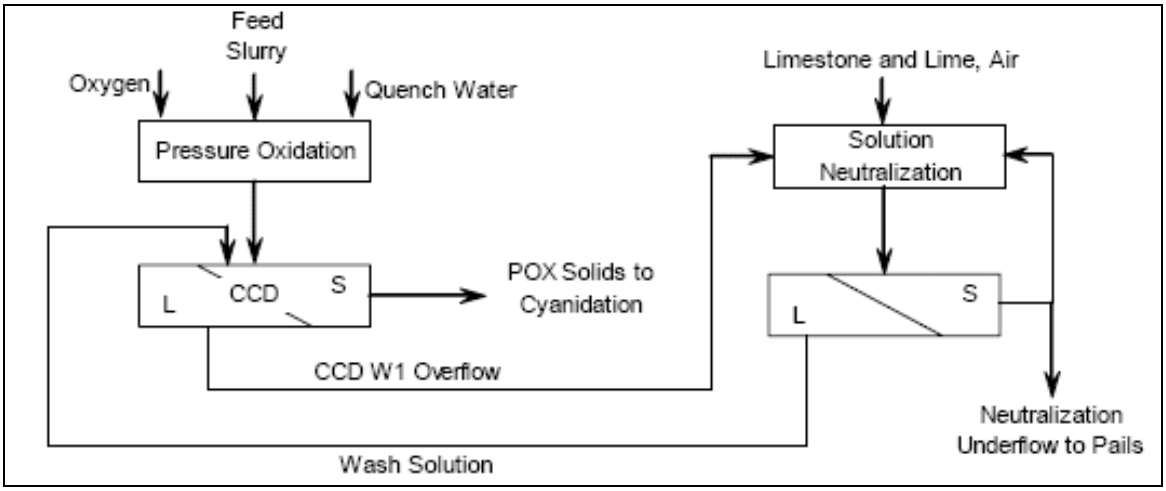


Figure 13.7: POX Pilot Plant Block Diagram

13.6.4.1 Feed Material

Two (2) types of feed material were treated in the pilot plant campaign, which were generated from material produced by reverse circulation (RC) drilling near the top of the Joanna deposit and an overall composite of drill core (DC). These materials were processed at Inspectorate to produce flotation concentrate and table tails samples for both the RC and DC materials. Each of the flotation concentrate and corresponding table tails were blended to produce a homogeneous feed for each concentrate type. The compositions of each of the concentrate types, constituting the feed blends for the POX pilot plant are shown in Table 13.22. The particle size of the feed blend was in the order of 74% to 83% passing 88 µm. The DC and RC concentrates were re-pulped to 66% and 68% solids respectively.

Table 13.22: Analysis of DC and RC Samples from Inspectorate Flotation Pilot Plant

Feed Slurry	Feed Blend Analysis, wt% or g/t for Au and Ag								
	Au	Ag	Al	As	C	C_{NAL}[1]	Fe	Si	S
RC Blend	16.4	1.8	5.10	6.36	0.35	0.22	20.6	18.2	11.7
DC Blend	12.1	2.2	5.33	3.10	0.24	0.08	17.5	21.1	9.88

[1]Non-acid leachable carbon, i.e., organic carbon

13.6.4.2 Pressure Oxidation (POX)

The pressure oxidation circuit consisted of feed systems for feed slurry and quench water, the pressure oxidation autoclave, pressurized and atmospheric discharge tanks, and a conditioning tank/evaporator to simulate flashing from a commercial pressure oxidation autoclave and to dissolve basic iron sulphates.

The pressure oxidation circuit was operated for 150 hours. The circuit processed, 410 kg of solids for each of the RC and DC Blends,.

The results of the continuous POX pilot plant campaign are shown in Table 13.23.

Table 13.23: Continuous POX Pilot Plant Results

Period	1	2	3	4	5
Run Time, h	0 to 24	24 to 57	57 to 81	81 to 123	123 to 150
Feed Blend	RC	RC	RC	DC	DC
Temperature, °C	220	220	230	220	220
Pressure, kPa(gauge)	2,720	2,272	3,200	2,720	2,720
Calculated Oxygen Pressure, kPa	500	500	500	500	500
Retention Time, min	75	60	60	60	75
Sulphide Oxidation, %	98.2	97.7	98.1	97.6	98.4
Gold Extraction, %	88.0	86.1	88.9	93.9	92.8

Results indicate that for both the DC and RC concentrates, sulphide oxidation is superior at the higher retention time (75 min). For RC, the combination of higher temperature and lower retention time (60 min) is equivalent to lower temperature combined with higher retention time. Overall gold extraction was improved for the RC concentrate by operating at a longer retention time or higher temperature, but average gold extraction from the conditioning tank solids was typically 88 to 90% when treating the RC concentrate under these conditions. Gold extraction was higher for the DC concentrate at 93 to 94% on average for the conditioning tank solids. In both cases, lower gold extractions were obtained in cyanide leaching of the solids from continuous test solids, than were obtained in the batch test program.

A strong relationship between the extent of sulphur oxidation and gold extraction was observed for the DC concentrate. This relationship was not as clear for the RC concentrate and it appeared that a small quantity of organic carbon in the RC concentrate was also affecting gold extraction. High gold extraction for the RC concentrate was not typically achieved until most of the sulphur and organic carbon were oxidized in the autoclave. Organic carbon decomposition was enhanced at 230ºC.

Based on the results from this testwork, it was concluded that plant design should be based on a 70-minute retention time and 220ºC autoclave operating temperature.

13.6.4.3 Conditioning

The pilot plant autoclave discharge slurry passed through a conditioning tank with a retention time of 20 to 30 minutes. Samples of this conditioned slurry were held at temperature in a small batch reactor during the pilot test campaign to investigate the effect of extended conditioning time. Conditioning times of up to 4 h did not significantly affect the gold extraction from the conditioned solids, but a longer retention time would be beneficial for decomposing more of the basic sulphates in the POX solids prior to the CCD wash circuit. Sulphate in solids was overall quite low, at between 2.5 and 3.5%, resulting in relatively low lime consumption in cyanidation of about 10 to 12 kg/t.

Based on the results, and the possibility of more sulphate in the pressure oxidation solids in commercial practice, for which the feed is expected to have higher sulphur content, a conditioning tank retention time of 120 minutes was recommended for the commercial plant.

13.6.4.4 Counter Current Decantation (CCD)

All of the conditioning tank discharge slurry from the pressure oxidation circuit was washed with water to start and neutralization thickener overflow solution once steady state was achieved in a countercurrent decantation (CCD) wash circuit in order to remove the bulk of the acid from the POX solids prior to cyanidation for gold recovery.

Overflow solution from the first wash thickener was forwarded to the solution neutralization circuit for neutralization of acid and precipitation of contained metals. Subsamples of the third wash thickener underflow slurry were taken, further washed and subjected to cyanidation tests to determine gold extractions.

The acid concentration in the wash circuit solutions depended on the feed acid and the wash ratio. The desired extent of washing in the CCD circuit (i.e., to the target of 2.5 g/L H_2SO_4 in the final thickener underflow solution) was generally met or exceeded using a wash ratio of about 2.8. If two (2) thickeners were to be used commercially, a higher wash ratio would be required.

The POX solids settled well in the CCD wash circuit to about 50% solids, with little difference observed between the final solids content of the underflow slurry when processing POX slurry was produced from the RC and DC blends. Between 97 and 99% of the sulphuric acid and metal salts were washed from the conditioning tanks discharge slurry during the pilot plant campaign. The results are summarized in Table 13.24.

Table 13.24: Summary of Operating Parameters in CCD Circuit for POX Pilot Plant

Operating Period	1	2	3	4	5
Wash Ratio to W3, L/L of U/F solution	2.0	2.4	2.8	2.9	2.7
Flocculant Dosage, g/t solids					
W1	39	47	63	54	64
W2	22	26	34	29	34
W3	17	22	30	25	28
Total	77	95	128	108	127
W3 Overflow H_2SO_4, g/L	2.9	3.3	2.6	1.3	1.2
Acid Removal, %	98	97	98	99	99

13.6.4.5 Neutralization

In the neutralization step, acid wash solution from the CCD was reacted with limestone and slaked lime in two (2) stages at 80°C to neutralize the free acid and precipitate metals. The solution neutralization circuit consisted of a solution feed tank, slurry feed tanks for limestone and lime, and a series of five (5) stirred tank reactors in series followed by a thickener.

Overflow solution from the neutralization thickener was forwarded to the CCD wash circuit to serve as wash solution. The underflow from the solution neutralization thickener was collected in pails.

The solution neutralization (NE) wash circuit was operated for a total of 113 h. Circuit operation was divided into five (5) periods, based on the changes in the POX operating conditions. Average values of the operating parameters and important results are summarized by period in Table 13.25.

Table 13.25: Summary of Operating Parameters in Neutralization Circuit for POX Pilot Plant

Operating Period	1	2	3	4	5
Retention Time, h	4.7	4.9	5.2	5.0	6.1
Average Temperature, °C	81	79	79	80	81
Stage 1 (Tanks 1 and 2)					
Tank 2 Median pH @ 25°C	4.6	4.7	4.9	4.4	4.9
Limestone Addition, kg/t feed[1]	372	197	123	92	96
Limestone Utilization, %	98.5	97.1	96.8	95.4	94.3
Stage 2 (Tanks 3 to 5)					
Tank 5 Median pH @ 25°C	8.6	8.5	8.5	8.5	8.5
Lime Addition, kg/t feed[1]	47	36	41	41	47
Thickener					
Flocculant (MF338) Dosage, g/t	71	38	25	28	24
Underflow Slurry Solids Content, wt%	24.8	40.7	40.9	43.4	48.7
Thickener Overflow As, mg/L	0.24	0.78	1.26	0.74	1.33

[1] Calculated from solution and feed solids analyses using an iron tie

Once steady operation was achieved in Period 2, limestone consumption was between 90 and 200 kg of limestone per tonne of feed to the POX circuit to give a pH in Tank 2 of 4.4 to 4.9. Limestone utilization was high, between 94 and 98%. Lime consumption was between 36 and 47 kg per tonne of feed to the POX circuit, resulting in a pH in Tank 5 of 8.5 to 8.6.

Optimization in the control of the pH in the second stage of neutralization during commercial operation could allow for a better balance between limestone and lime addition.

During the pilot plant campaign, all elements analyzed, except for calcium and magnesium, were precipitated to near or below the detection limit. Arsenic in the thickener overflow solution was about 1 mg/L.

13.6.4.6 Cyanidation

Selected pressure oxidation and countercurrent decantation (CCD) wash circuit slurry samples were collected at regular intervals throughout the pilot plant campaign for cyanidation testing. The objectives were to determine gold extractions and reagent consumptions in the cyanidation of the oxidized solids.

CNA tests at low pulp density and without intermediate sampling provide information on the ultimate gold extractions achievable. A total of 213 CNA tests were conducted, with excess cyanide relative to the small quantity of solids, and with activated carbon present, in small roll bottles for 24 h.

Larger scale bottle-roll cyanidation rate tests in CIL or CIP modes were conducted at higher pulp density and with cyanide additions more representative of commercial operation. This type of test provides more accurate information on reagent requirements and consumption than do the smaller-scale CNA tests. A total of 22 CIL and 10 CIP tests were conducted during the pilot plant campaign. Results are shown in Figure 13.8.

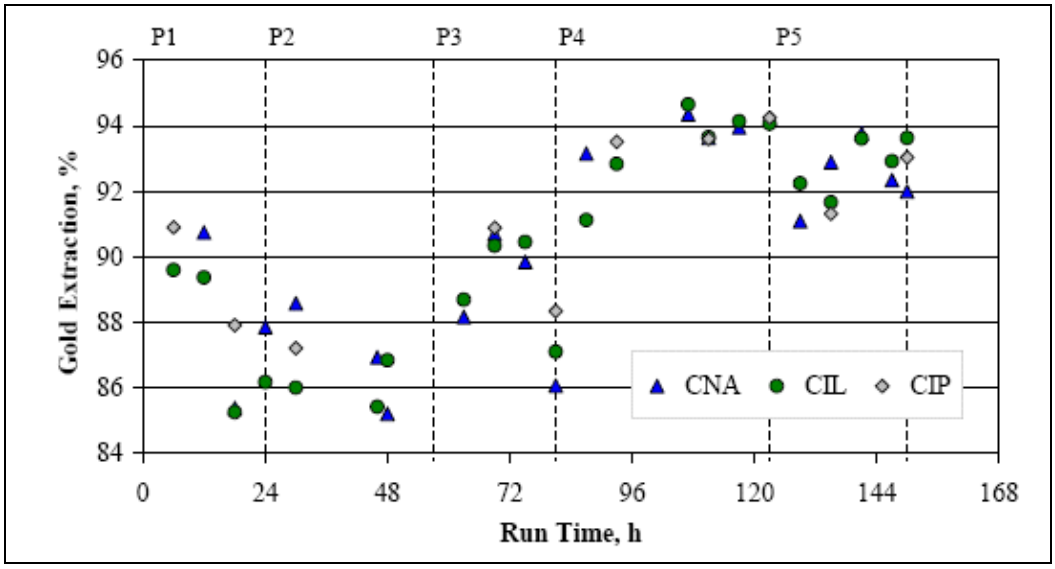


Figure 13.8: Comparison of CNA, CIL and CIP Gold Extraction Values

There was generally good agreement between the CNA, CIL and CIP extraction results. No significant difference in gold extraction was observed between the CIL and CIP tests, indicating little, if any evidence of preg-robbing behavior of the POX solids.

Gold extraction typically reached between 88 and 90% for the RC concentrate and up to 94% for DC concentrate. Gold extraction was nearly complete after 24 h during high pulp density CIL tests. Operating for extended cyanide leach times (i.e., up to 72 h) increased gold extractions between 0.5 and 2%, but required significantly higher lime and NaCN additions. Higher cyanide

concentrations in cyanide leaching did not significantly increase gold extraction, but increased cyanide consumption.

13.6.5 Post-Pilot Plant Batch Tests

Following the pilot plant campaign, two (2) additional batch POX tests were also conducted in order to investigate the low gold extractions observed for the RC concentrate. Additional grinding of the RC concentrate improved the CNA gold extraction by 1 to 2%. Blending of the RC and DC concentrates resulted in lower overall CNA gold extraction than was obtained by treating the materials separately, indicating that blending may not be a good option to try to improve the gold recovery for the RC concentrate.

13.6.6 POX Design Parameters

13.6.6.1 Sulphur Head Grade

The function of the POX process is to oxidize sulphur. Therefore, it is important to determine the sulphur content of the material to be processed there. To do so the sulphur content of the DC and RC concentrates were analyzed. Table 13.26 shows the distribution of sulphur in the two (2) concentrate types.

Table 13.26: Sulphur Distribution of DC and RC Ore Samples

Drill Core (DC) 285 Data Points		RC Drilling 112 Data Points	
	% Sulphur		% Sulphur
Average	0.78%	Average	0.65%
50th percentile	0.74%	50th percentile	0.53%
75th percentile	0.98%	75th percentile	0.80%
80th percentile	1.02%	80th percentile	0.84%
95th percentile	1.30%	95th percentile	1.19%

The design conditions for the POX process were determined based on the following assumptions:

- Plant utilization rate of 85%;
- Annualized plant feed rate;
  - "Nominal": 8500 t/d;
  - "Design": 9500 t/d;
- Sulphur content based on DC sample since it contained the higher sulphur load;
  - "Nominal" sulphur content based on the average (0.78%);
  - "Design;" sulphur content based on 80th percentile (1.0%);
- Flotation recovery;
  - 5% overall weight recovery;
  - 92.5% sulphur recovery.

These aforementioned parameters provided the design basis for sizing process equipment. Table 13.27 provides a summary of the process design basis.

Table 13.27: POX Process Design Basis

Case	Ore Feed Rate (Annualized t/day)	Sulphur Head Grade (%)	POX Feed Rate (t/op. h)	Sulphur Concentration of POX feed (%)	Sulphur Fed To POX (t/op. h)
Nominal	8500	0.78	20.8	14.5%	3.02
Design Case 1	9500	0.78	23.3	14.5%	3.38
Design Case 2	8500	1.0	20.8	18.5%	3.85

It was decided against designing for high feed rate combined with high sulphur since this would be exceptional and would result in needlessly overdesigning the oxygen plant. Design Case 1 has the highest throughput and is therefore used for vessel sizing (for residence time) whereas Design Case 2 requires the highest rate of sulphur oxidation and is therefore used to size the oxygen plant and the cooling systems.

13.6.6.2 POX Plant Design

Based on the positive pilot plant results, the POX process was retained as the oxidation technology for the purposes of this FS. Table 13.28 summarizes the design parameters developed by Sherrit for the POX process design. It should be noted that the pressure in the autoclave during the pilot testing was 2,720 kPa(gauge), while in the commercial plant design it is based on 3,000 kPa(gauge) (Sherritt March 2012). This is because lower purity oxygen produced by a VPSA oxygen plant will be used in the commercial plant and the higher autoclave pressure is required to maintain the same partial pressure of oxygen in the autoclave. A high purity cryogenic oxygen plant was considered but the VPSA plant proved to be a more cost effective solution.

Table 13.28: POX Circuit Design Parameters

POX Autoclave Residence Time (min)	70
Autoclave Operating Temperature (°C)	220
Autoclave Pressure (kPa)	3,000
Oxygen Partial Pressure in Autoclave (kPa)	500
Oxygen Purity (%)	92.5
Post-Autoclave Conditioning Tank Residence Time (min)	120
Nominal Oxygen Consumption (kg/h)	9,293
Oxygen Plant Design Capacity (t/d)	286
Total Neutralization Residence Time (hrs)	6
Nominal Limestone Consumption (kg/h)	6,045
Nominal Lime Consumption (kg/h)	1,045

13.6.6.3 Cyanidation

The design of the cyanide leaching circuit was based on a CIP-type circuit as no “preg-robbing” behavior was exhibited in the leaching tests on the pilot plant samples. The residence time in the cyanide leaching portion of the circuit was selected to be 48 hours. The Pumpcell technology by Kemix was selected for use as the activated carbon adsorption portion of the circuit. This technology allows the circuit to be operated in a carousel mode of operation as described in detail in Section 17 of this Report. The sizing of this portion of the circuit was performed by the Vendor based on operating parameters provided. The design parameters for the leaching and CIP circuit are presented in the Table 13.29.

Table 13.29: Leach and CIP Circuit Design Parameters

Number of Leach Tanks	4
Total Cyanide Leach Residence Time (hrs)	48
Number of CIP Tanks	10
CIP Tank Volume (m^3)	20
CIP Carbon Concentration (g/L)	50
Gold on Carbon Loading (g/t)	10,235
Required Carbon Regeneration Capacity (t/d)	1.0

13.7 Thickener Testwork

13.7.1 Flotation Concentrate and Tailings Thickeners

Testwork was undertaken by Outotec in May 2010 using samples from SGS to determine design parameters for the flotation concentrate and flotation tailings thickeners.

The Flocculant Flomin 905 MC showed the most efficient flocculation and fastest settling at dosages of 50 and 15 g/t for both tailings and concentrate samples, respectively.

The tailings sample exceeded the targeted underflow density, achieving 60.8% w/w solids at a solids loading rate of 0.81 t/m^2h. The overflow clarity of 200 ppm TSS was achieved.

The concentrate sample achieved the highest underflow density of 64.6% w/w solids at a solids loading rate of 0.25 t/m^2h.

13.7.2 POX Area Thickeners

Liquid solid separation tests were carried out by Pocock Industrial, on behalf of Sherritt, on the POX discharge slurry and the solution neutralization discharge slurry during the pilot plant campaign (Sherrit May 2011).

Thickening tests from four (4) operating periods were performed on POX discharge slurries in order to determine the recommended maximum hydraulic design basis for high rate thickener

design. There appeared to be relatively minor effects of POX parameter variations on the thickening characteristics. Results are summarized in Table 13.30.

Table 13.30: Results From Thickening Tests Performed on POX Discharge Slurry

Material Tested	Thickener Feed Solids[1] %	Flocculant Dose[2] g/t	Design Basis Net Feed Loading[3] m^3/m^2h	High Rate Thickener Diameter and Predicted Underflow Density
Period 2	13.7	100 to 120 100 to 120	1.89 2.51	9.92 m; 54% underflow 8.61 m; 50% underflow
Period 3	17.6	75 to 90 75 to 90	1.91 2.76	8.44 m; 58% underflow 7.02 m; 50% underflow
Period 4	18.1	100 to 115 100 to 115	1.83 2.58	8.55 m; 55% underflow 7.20 m; 50% underflow
Period 5	16.4	95 to 105 95 to 105	1.93 2.90	8.84m; 57% underflow 7.21 m; 50% underflow

1 Feed solids concentration range required for thickener operation at maximum design net feed loading rate

2 Hychem CP906HH

3 Recommended design net feed loading rates are provided without scale-up or safety factors

Table 13.31 shows the high rate thickener results for the neutralization discharge slurry.

Table 13.31: Results From Thickening Tests Performed on Neutralization Discharge Slurry

Material Tested	Tested Feed Solids[1] %	Flocculant Dose[2] g/t	Design Basis Net Feed Loading[3] m^3/m^2h	High Rate Thickener Diameter and Predicted Underflow Density
2-28-11 18:00 h	12.5	15 to 20	3.48 4.21	10.0 m; 56% underflow 9.12 m; 50% underflow

1 Feed solids concentration range required for thickener operation at maximum design net feed loading rate
2 Hychem AF304
3 Recommended design net feed loading rates are provided without scale-up or safety factors

13.8 Cyanide Destruction

Various cyanide destruction options, including Caro's acid, SO_2/Air and CombinOx processes, were investigated on a laboratory scale for treatment of slurry effluent samples from the Joanna Project. The ore sample was shipped to Cyanco Corporation's laboratory in Piscataway, New Jersey in June 2011 (Cyanco August 2011).

Performance and reagent consumption data for each process were compared using a weak acid dissociable cyanide (CN_{WAD}) treatment target of 20 ppm or less, while also recognizing the 20 ppm total cyanide (CN_T) limit. All three (3) of the processes tested (Caro's Acid, SO_2/AIR and CombinOx) were successful in reducing the CN_{WAD} to levels at or below the target.

The 20 ppm CN_T limit could be met using all the processes tried when treating more typical effluents with soluble iron levels around 1 ppm. However, due to the elevated iron cyanide levels in one (1) of the feed samples, over 10 ppm Fe, the CN_T limit could not be met using the same processes. By adding more copper and reducing the pH, it was possible to meet the CN_T limit for the high iron sample for both Caro's Acid and SO_2/AIR.

OPEX and CAPEX estimates were prepared by Cyanco for the three (3) processes based on the results of the testwork. The SO_2/Air process was recommended by Cyanco because although it was the least economical in terms of OPEX, it had a much lower CAPEX than the other processes. For plant design, the SO_2/Air system was designed with a residence time of 4 hrs.

13.9 Gold Recovery

13.9.1 Effect of Particle Grind Size on Overall Gold Recovery

An analysis was performed to quantify the effect of particle grind size on gold recovery. The results of the gravity circuit modeling performed by Knelson and flotation tests, show that the grind size has an important effect on recovery in these process areas. Modeling of the gravity circuit predicted that at a 600 tph feed rate, the gravity gold recovery at a P_{80} of 135 µm would be in the order of 26.4% while a P_{80} of 85 µm gravity gold recovery would be 35.7%. When comparing flotation locked cycle test results at a feed P_{80} in the range of 73-83 µm to test results in the range of 100-127 µm , it was observed that for the finer grind range, approximately 1% higher gold recovery is achieved.

As the gold recovery in the gravity circuit varies, the head grade to the flotation circuit varies accordingly. Therefore, the gold recovery model needs to take this into consideration. Because of the complexity of the interactions between these two (2) process areas and the rest of the process, simulations were carried out to estimate the impact of grind size on overall recovery. Simulations were carried out at various P_{80} product sizes between 75 and 135 µm. The following assumptions were made to estimate the recovery in the individual process areas:

- A head grade of 1.3 g/t
- The modeled gold recovery in the gravity circuit was interpolated linearly between 85 µm and 135 µm, and extrapolated to 75 µm using the same slope.
- The gold recovery in the shaking tables treating the Knelson concentrate to achieve further concentration was assumed to be 60%. The tails from the shaking tables are sent directly to the oxidation stage by-passing flotation.
- The concentration of gold obtained in the pilot plant tailings was used as a reference. This result was then corrected for coarser grinds by taking an equivalent feed grade to flotation. The recovery for 135 µm primary grind was assumed to be 1% lower than for 75 µm, (based on locked cycle testing results). The resulting flotation tails concentration was then taken and used to calculate the recovery for that grind size based on the feed grade obtained from the gravity concentrator. The same method was applied to the intermediate grinds, however, the 1% correction was interpolated towards zero at 75 µm.

- Recovery in oxidation and cyanidation was assumed to be 92.5%.

Table 13.32 below summarizes the simulation results.

Table 13.32: Effect of Comminution Circuit Product Size P_{80} on Overall Gold Recovery

Head Grade (g/t)	1.3				
Grind Size P_{80} (µm)	75	90	100	120	135
Gravity Recovery (%)	37.6	34.8	32.9	29.2	26.4
Shaking Table Recovery (%)	60				
Flotation Recovery (%)	88.6	88.9	89.1	89.4	89.6
Post-Oxidation CN Recovery(%)	92.5				
Overall Recovery (%)	87.6	87.4	87.2	86.9	86.6

The results of the simulation show that the overall recovery may vary by 1% over the range of grind sizes evaluated.

Following a high level analysis of incremental capital and operating costs associated with finer grinding, a decision was taken to choose a P_{80} of 90 µm as the grinding circuit product size.

13.9.2 Gravity Circuit Gold Recovery

In order to estimate the gold recovery in the gravity circuit for the selected throughput of 600 t/h and grind size P_{80} of 90 µm, the average recoveries generated from the Knelson modeling results previously presented in this section were interpolated linearly between the points at 85 and 135 µm. A gold recovery of 34.8 % in the gravity concentration circuit was determined and used for the FS.

It was further assumed that the Knelson product will be further concentrated using shaking tables and gold recovery in the shaking table concentrate is in the order of 60% of the gold in the shaking table feed. Shaking table concentrate is directed to the melting furnace where 100% of the gold is assumed to be recovered. The shaking table tails contain the gold not recovered in the shaking table concentrate, this stream is directed to the POX feed.

13.9.3 Flotation Gold Recovery

The gold which is not extracted by the aforementioned gravity circuit is directed with the ground ore to the flotation circuit. As was previously indicated, the average gold recovery in the SGS flotation pilot plant was 90.7% at 4.33% weight recovery. In order to establish the gold recovery at nominal conditions, it was necessary to adjust gold recovery for the following factors:

- The average gold concentration of the pilot plant feed ore was 1.34 g/t versus the average head grade in the mine plan for the LOM, which is 1.26 g/t.
- In the SGS pilot plant, gold recovery at the gravity circuit was 27% while the value adopted for the FS was 34.8%.
- Average P_{80} of the feed in the pilot plant was 77 µm versus the adopted plant design P_{80} of 90 µm.
- To achieve the required sulphur concentration in the Albion concentrate , the pilot plant was operated using two-stage cleaning resulting in a 4.33% weight recovery and 88.9% gold recovery. Considering that sulphur target for POX is lower, plant design requires only one-stage cleaning. It was estimated that the weight recovery would be in the order of 5% and gold recovery would increase slightly to 89.15%.

13.9.4 POX Gold Recovery

As previously discussed, testwork results indicated that, at the sulphide oxidations achieved, gold extraction in cyanidation following POX typically reached up to 94% for the DC concentrate and between 88 and 90% for the RC concentrate. Considering that the majority of the Joanna ore deposit is better represented by the DC ore, it was determined that in the commercial operation, under stable conditions, recoveries will approach the higher end of what was observed in the pilot plant operation. For the purposes of the FS, it was estimated that the gold recovery in the CIP stage following POX will be in the order of 92.5%.

13.9.5 Overall Gold Recovery

By consolidating the gold recoveries for the individual process areas previously mentioned, the overall gold recovery at nominal conditions was estimated to be in the order of 87.5%. Figure 13.9 presents gold recoveries in the various process areas for nominal operating conditions.

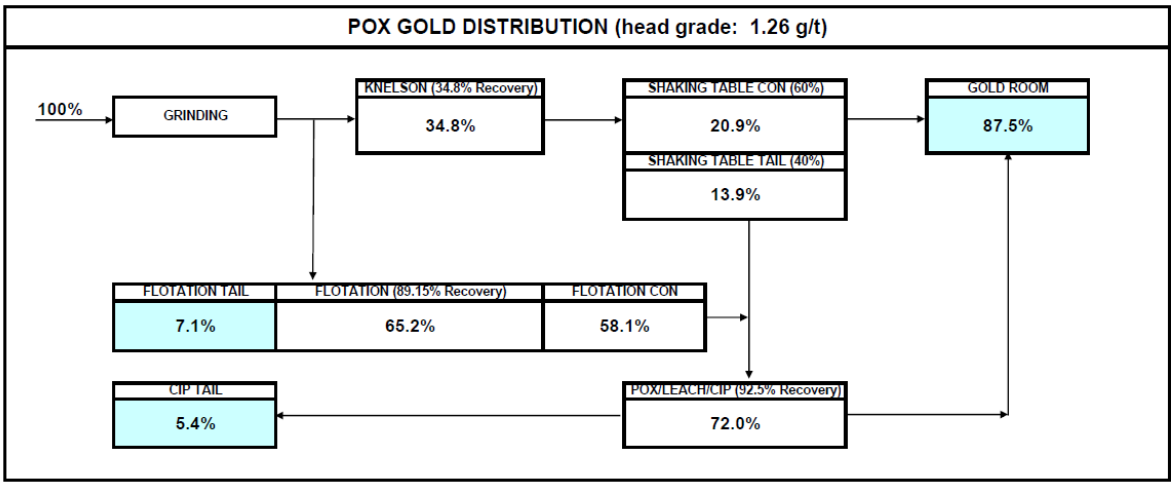


Figure 13.9: Overall Gold Recovery

The previous analysis was performed to estimate gold recovery under nominal conditions, including and average LOM gold head grade of 1.26 g/t. In reality however, the head grade is not constant and varies with the mine plan. In order to adjust overall gold recovery to varying mine head grade, a gold recovery versus head grade curve was developed and is presented in Figure 13.10. To generate this curve, it was assumed that the recovery in gravity and in post-POX CN leaching is constant. Flotation recovery was not assumed to be constant but rather, for flotation it was assumed that the flotation tailings had a fixed gold concentration of 0.10 g/t.

It should be noted that this curve is only considered to be valid for gold head grades above approximately 1.0 g/t; below this value, gravity gold recovery may not be constant.

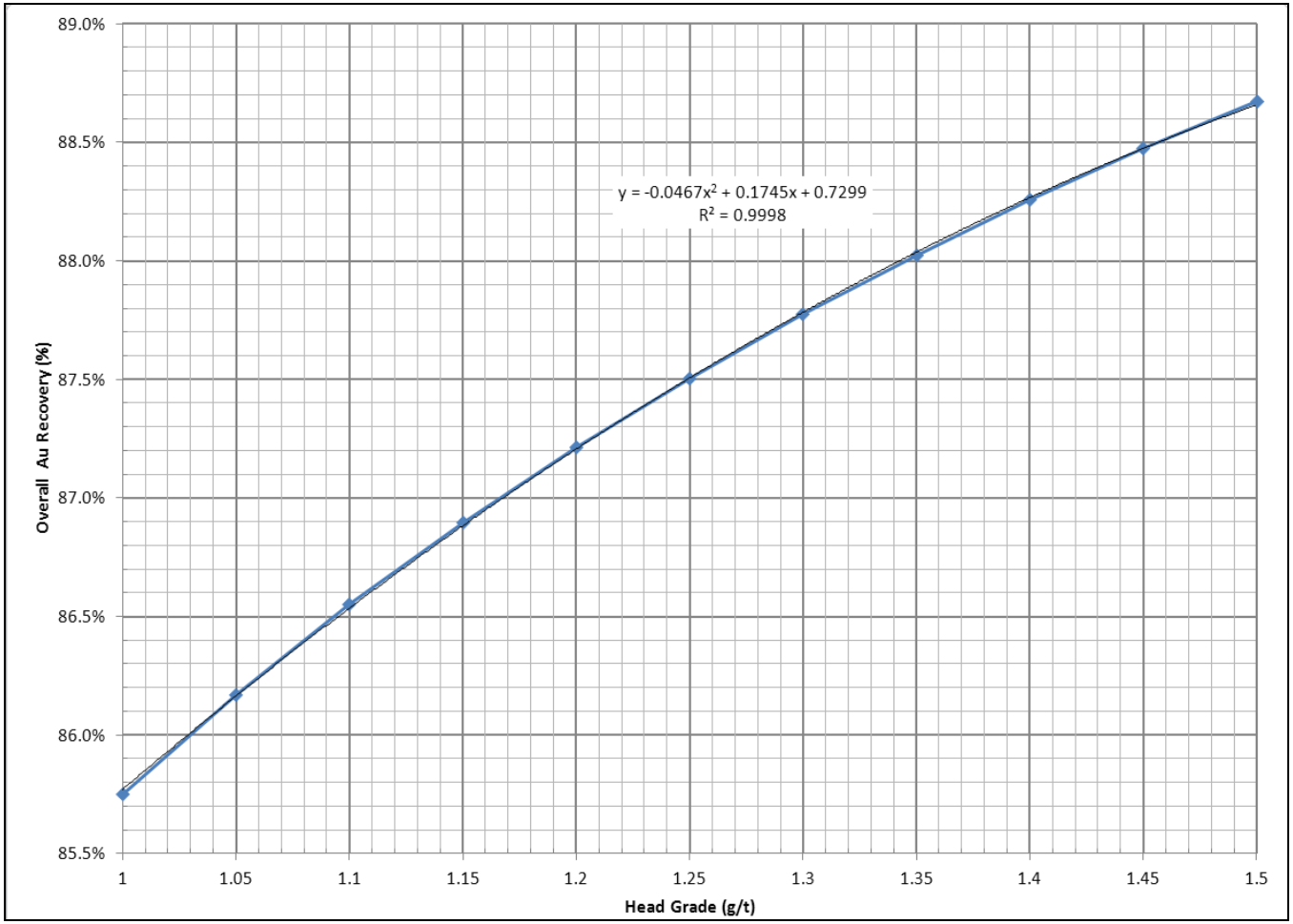


Figure 13.10: Variation of Overall Gold Recovery with Head Grade

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

This section reports the results of a mineral resource estimate for the Hosco deposit updated to September 22, 2011, which is based on new analytical data sampled from the drilling completed since the previous mineral resource estimate. The previous mineral resource estimate was completed by SGS Geostat and was disclosed in the Company news release dated June 13, 2011.. The new mineral resource estimate is now defined based on two distinct mining scenarios, open-pit and underground, using distinct base case gold cut-off grade.

As described in the Company's June 13, 2011 news release, as the updated resource outline of the Hosco deposit disclosed in the news release crossed over the Alexandria claim boundary, all the resources of the Alexandria sector were restated. However, the Alexandria resource estimate was not updated in June 2011. The restated mineral resource estimate for the Alexandria sector disclosed by the Company on June 13, 2011 is used in this Report.

Also as described in the Company's June 13, 2011 news release, the mineral resources of the Heva deposit were not updated at that time but were restated using a modified cut-off grade for the mineral resources located below 300 m from surface. The Heva deposit mineral resources were restated as disclosed by the Company on June 13, 2011 (date of restatement: May 31, 2011) to better reflect the reasonable prospect of economic extraction in relation to the advanced stage of exploration of the Hosco deposit located to the east. The restated mineral resource estimate for the Heva deposit disclosed by the Company on June 13, 2011 is used in this Report.

The mineral resource has been estimated by Maxime Dupéré P.Geo., Geologist for SGS Geostat. Mr. Dupéré is a professional geologist registered with the Ordre des Géologues du Québec and has worked in exploration for gold and diamonds, silver, base metals and iron ore. The author has been involved in mineral resource estimation work over different gold deposits on a continuous basis since he joined SGS Canada Inc. in 2006, which includes the participation in mineral resource estimate for the Joanna gold project in 2011. Mr. Dupéré is an independent Qualified Person as per Section 1.4 of the NI 43-101 Standards of Disclosure for Mineral Projects with respect to the owner of the mineral titles included in the Property.

14.2 Hosco Deposit Resource Estimate

SGS Geostat conducted the current mineral resource estimate for the Hosco sector using historical underground channels and drillholes, selected historical surface drillholes and updated surface drillhole data compiled from the 2007 to 2011 exploration programs conducted by Aurizon. The database used to produce the mineral resource estimate is derived from a total of 755 recent surface drillholes, 109 historical surface drillholes and 3,243 historical underground channels or drillholes and contains the collar, survey, lithology, and analytical results information. The database cut-off date is August 26, 2011. The database includes 21 additional surface drillholes from the previous resources estimation disclosure dated June 13th, 2011 located in the Hosco West sector between 7000 mE and 7650 mE. Please refer Section 12 for a summary of the records contained in the final drillhole database.

The lateral extension (east-west direction) of the Hosco block models has been modified since the last mineral resource estimation and is now defined within sections 7,000 mE and 10,000 mE of the Project. The mineral resource estimate for the Hosco deposit is derived from two distinct computerised resource block models: a constrained high grade block model and a residual low grade block model. The construction of the high grade block model begins with the modelling of 3D wireframe envelopes or solids of the mineralization based on defined mineralised intercepts from channels and drillholes. Once the modelling is complete, the analytical data contained within the wireframe solids is normalised to generate fixed length composites. Composite data capped to a defined Au grade is used to interpolate the grade of blocks regularly spaced on a defined grid that fills the 3D wireframe solids. The interpolated blocks located below the bedrock/overburden interface and outside known barren geological features comprise the high grade mineral resources. For the low grade model, the blocks are interpolated using composite data located outside the 3D wireframe solid defined for the high grade block model. Mineral resources of the low grade block model are defined using the same process as with the high grade model. The final mineral resources comprise both high and low grade block models. The blocks for both models are then classified based on confidence level using proximity to composites, composite grade variance and mineralised solids geometry. The 3D wireframe modelling, block models and mineral resource estimation were conducted by SGS Geostat based on information provided by Aurizon.

The mineral resources of the Hosco deposit have been reported with two distinct mining scenarios: open-pit and underground mining perspectives. The open-pit mineral resources are reported within a Whittle pit shell optimised using the measured, indicated and inferred categories of the estimated global block model. The pit optimisation parameters have been supplied by Aurizon Mines and are based on the parameters used in the project's Prefeasibility Study. The base case cut-off grade for the in-pit mineral resources using a gold price of US$1,000 per oz Au is 0.33 g/t Au. The base case cut-off grade for the underground ("U/G") mineral resources at Hosco has been set at 2.0 g/t Au and is based on a conceptual U/G mining model completed by SGS Geostat. The reported U/G mineral resources are exclusive of the in-pit mineral resources.

Important Note: During the mineral resource estimation process, different assumptions were made. These assumptions were used in order to calculate modelling cut-off grades and resources cut-off grades following the "reasonable prospect for economic extraction" stated by the NI 43-101 regulation. A Whittle optimised shell was done with the same parameters as described in Table 14.5. The term in-pit refers to the resources within the optimised shell according to the different cut-off grades. The term underground (U/G) refers to a conceptual U/G mining model completed in-house by SGS Geostat. The terms in-pit and Underground (U/G) do not imply that any pit design and underground mining scenarios were done by SGS Geostat. Furthermore, it should not imply that the resources stated herein have demonstrated economic viability.

14.2.1 Exploratory Data Analysis

14.2.1.1 Analytical Data

Most of the drillhole intervals defining the modelled mineralised solids have been sampled continuously. The sampling gaps where no analytical data is reported relates to early exploration holes from the historical drilling programs. The sampling gaps in the mineralised intervals were considered having zero grade for the purpose of the block model interpolation process. There are a total of 127,773 assay intervals with an assigned Au value reported in the database used for the current mineral resource estimate. The grade distribution of the Au assay data is lognormal with more than one Au grade population observed in the data. Table 14.1

summarises the statistics of the Au assays reported in the drillhole database. Figure 14.1 shows the histogram of the Au assay data. Assays containing -1 and 0 grade values were not considered in this histogram.

Table 14.1: Summary Statistics for Au Assay Data from the Hosco Database

	Au Assays (g/t)				
Records	Mean	Std. Dev.	Min	Median	Max
127,772	0.491	3.77	0	0.04	1006.97

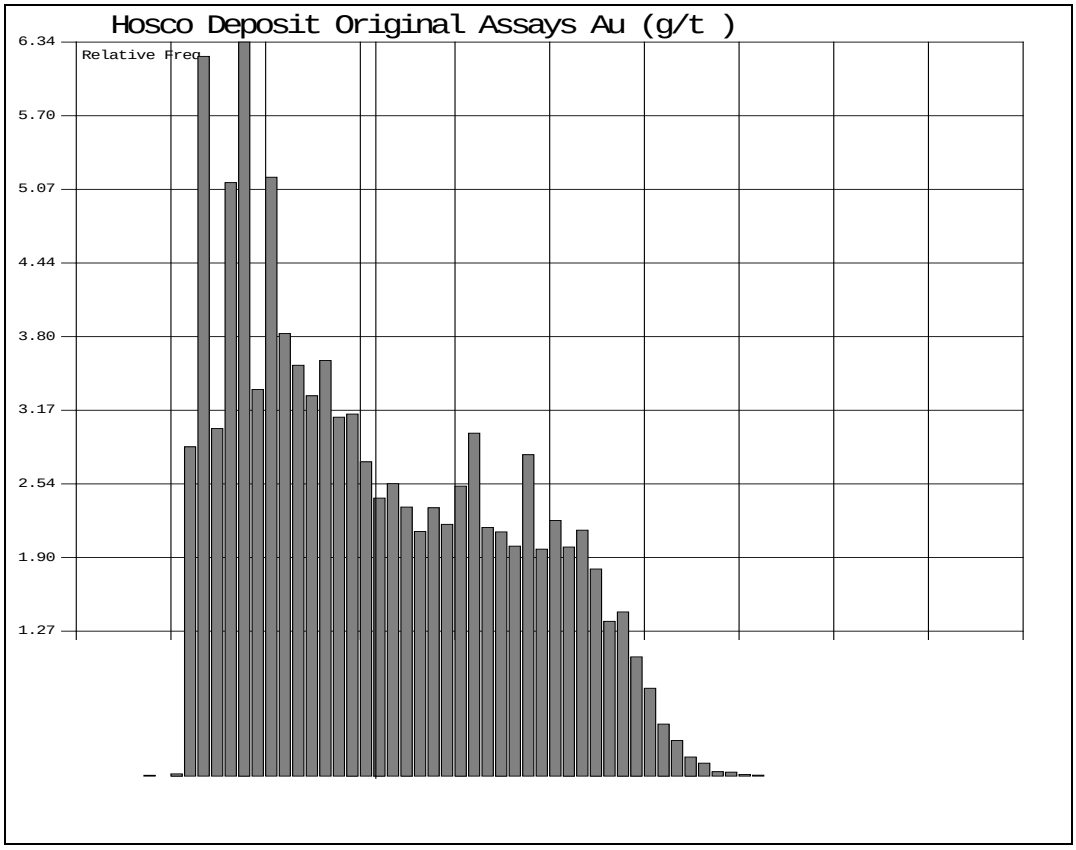


Figure 14.1: Histogram of Au Analytical Data from the Hosco Database

14.2.1.2 Composite Data

Block model grade interpolation is conducted on composited assay data. A composite length of 2 m has been selected to reflect the 5 m N-S thickness of the block size defined for the resource block model. Compositing is conducted from the start of each mineralized intercept

(HG mineralization) or in bedrock (LG mineralization) of drillholes or channels. The last composite kept at the end of the mineralised Intercept (HG mineralization) or the hole (LG mineralization) has a minimum length of 1 m. Composite grades were capped at 15 g/t which represent a gold loss of 5.2% from 73 composites capped. The first 1% of the highest Au grade final capped composites (190 composites) represents 8.0% of the contained metal versus 13.3% before capping.

At total of 19,214 composites were generated for the high grade block model and 84,980 composites for the low grade block model for a total of 104,194 composites. The modeled 3D wireframe envelopes of the high grade mineralization were used to constrain the high grade composites. The low grade composites are generated outside the 3D wireframe envelopes. Table 14.2 summarises the statistics of the capped high grade and low grade composite data. Figure 14.2 shows the histogram of the composites for the low grade and high grade composite. Figure 14.3 shows cumulative frequency plots of the high grade composite data with the capping level indicated. Figure 14.4 and Figure 14.5 display the spatial distribution of the composites in plan and longitudinal views respectively.

Table 14.2: Summary Statistics for Capped Au Composites

2m Composites (g/t)		
	High Grade	**Low Grade**
Count	19214	84980
Mean	1.42	0.07
Std.Dev.	1.65	0.24
Min	0.00	0.00
Median	0.97	0.01
Max	15.00	15.00

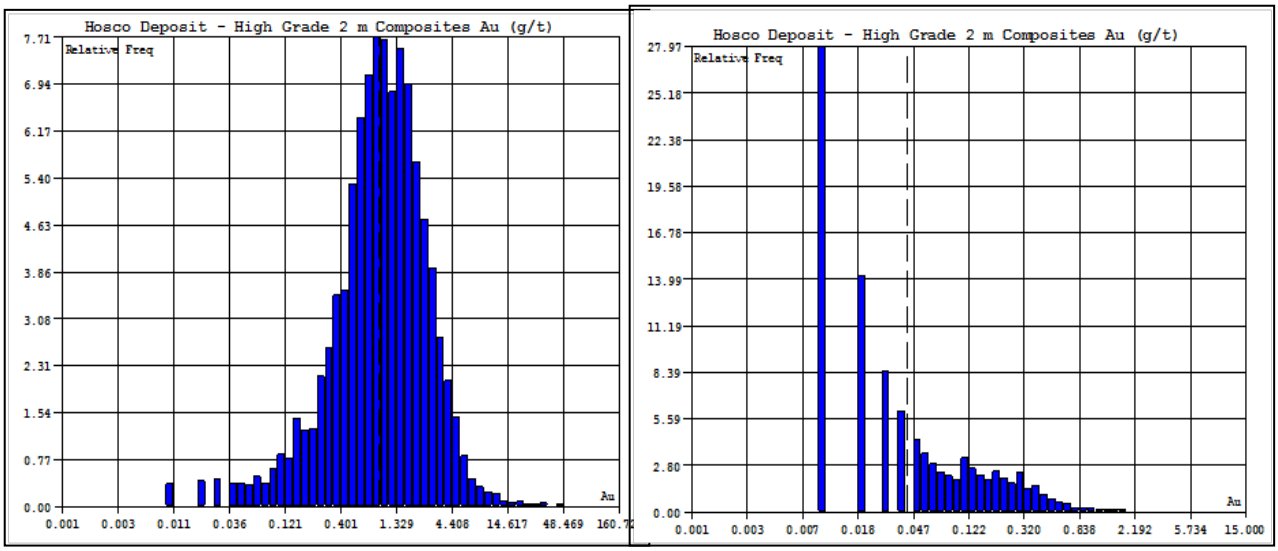


Figure 14.2: Histograms of High Grade and Low Grade Au Composites

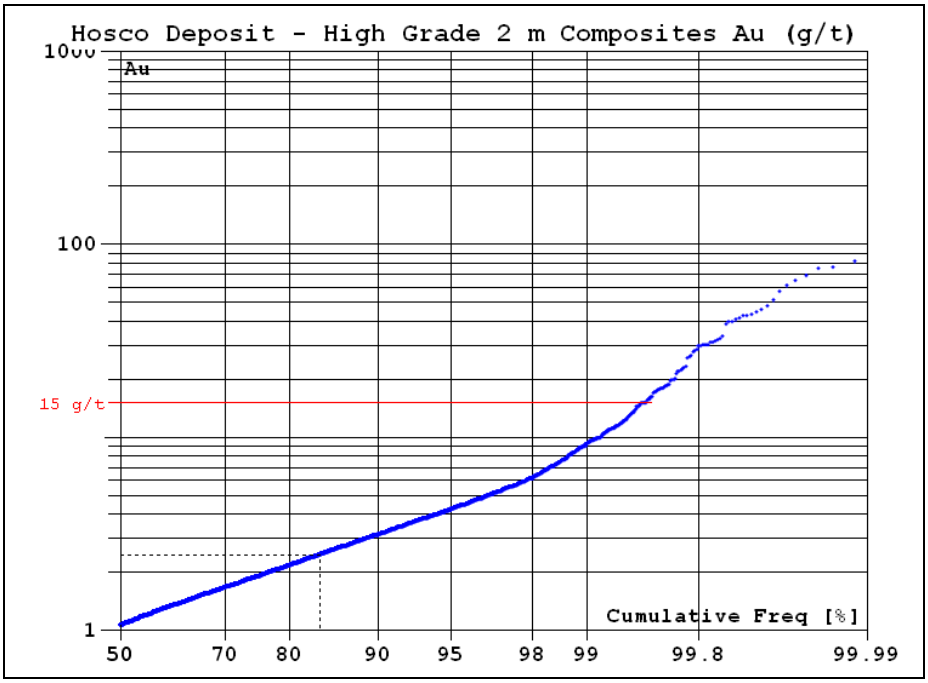


Figure 14.3: Cumulative Frequency Plot of High Grade Au Composites

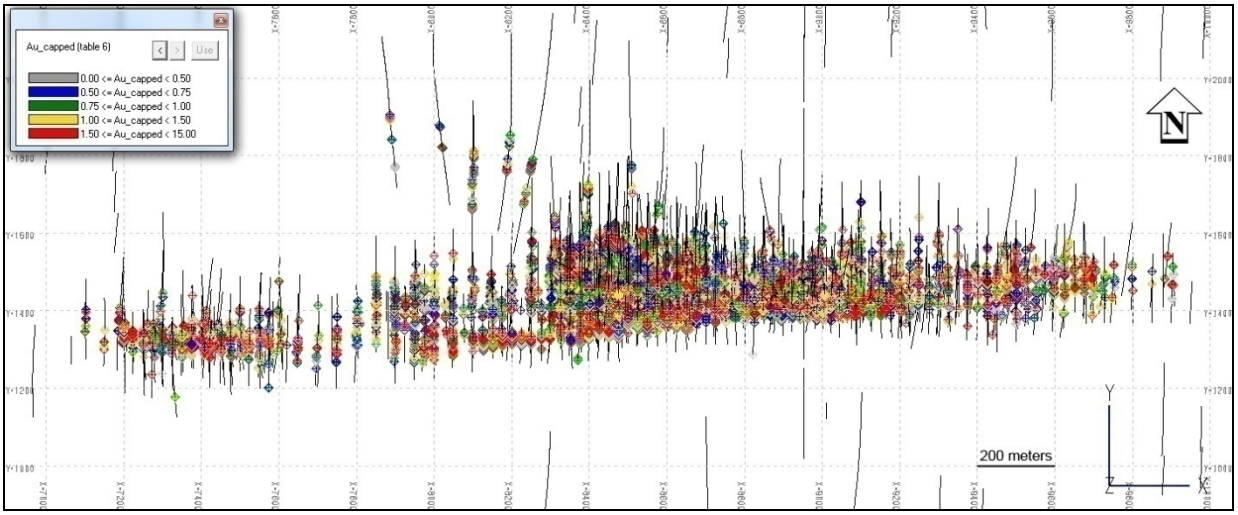


Figure 14.4: Plan View Showing the Spatial Distribution of High Grade Au Composites

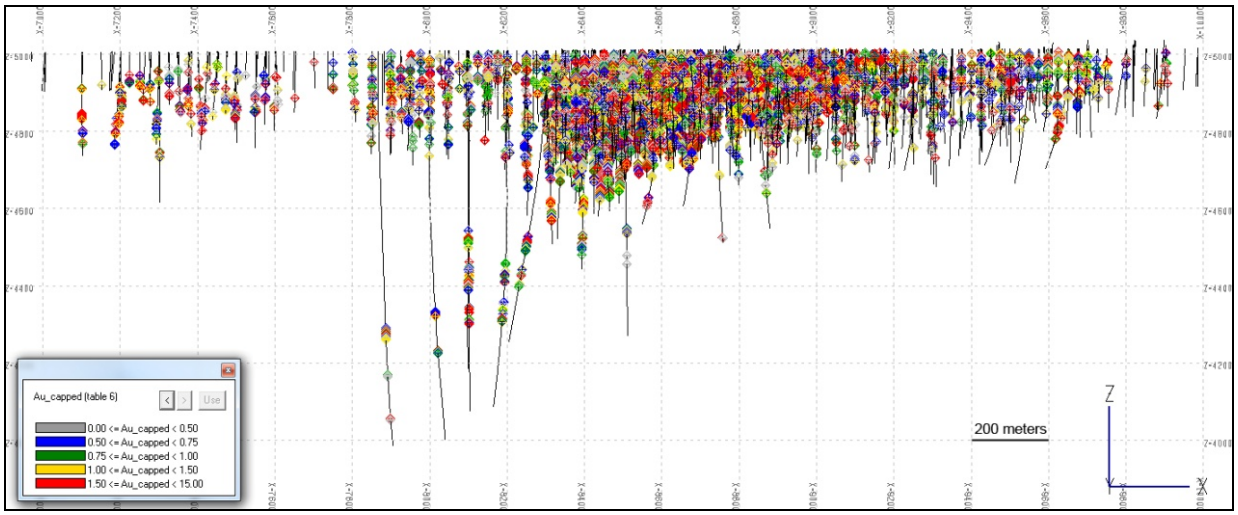


Figure 14.5: Plan View Showing the Spatial Distribution of High Grade Au Composites (Looking North)

14.2.1.3 Specific Gravity

The section Specific Gravity summarises the S.G. determination for the Hosco deposit conducted by the Company. The results of the S.G. measurements conducted on representative mineralised intervals returned an average S.G. value of 2.75 t/m^3. This value has been used for the calculation of the tonnages from the volumetric estimates of the resource block models.

14.2.2 Geological Interpretation

SGS Geostat conducted the interpretation and modelling of the 3D wireframe envelopes of the high grade Au mineralization in collaboration with Aurizon personnel. The interpretation was first completed on sections to define mineralised prisms using Au assays and based on observed mineralised zones. A minimum Au grade of 0.5 g/t over a minimum drillhole interval length of 4 to 5 m was generally used as guideline to define the width of the mineralised prisms. The final 3D wireframe model was constructed on a bench by bench basis by connecting the defined mineralised prisms based on geological interpretation. A bench height of 8 m was used for the wireframe model which corresponds to the thickness of the blocks in the resource block models.

A bedrock-overburden interface 3D surface has been generated by triangulating the lower intercepts of the overburden-coded lithology field of the drillhole database. Finally, 3D wireframe solids of known barren late intrusive units observed crossing the deposit were modelled. Resource blocks located above the bedrock-overburden interface and falling within the wireframe solids defining the late intrusive units were removed from the final mineral resource estimate. The barren late intrusives correspond to two, 10 to 60 m wide, NNE-SSW to NE-SW diabase dykes, steeply dipping to the WNW. The easternmost dyke is encountered around 9400 mE with varying direction from NNE-SSW to NE-SW and another is encountered from around 7475 mE to 7600 mE. The westernmost dyke is cut by the regional Cadillac fault with a sinistral displacement of approximately 200 m. The trace of these dykes is shown on Figure 14.7, Figure 14.11, and Figure 14.13

Figure 14.6 and Figure 14.7 shows the contour of the final mineralized solids and the mineralized intervals used for the modelling on representative sections and plan levels (late intrusive units shown in orange).

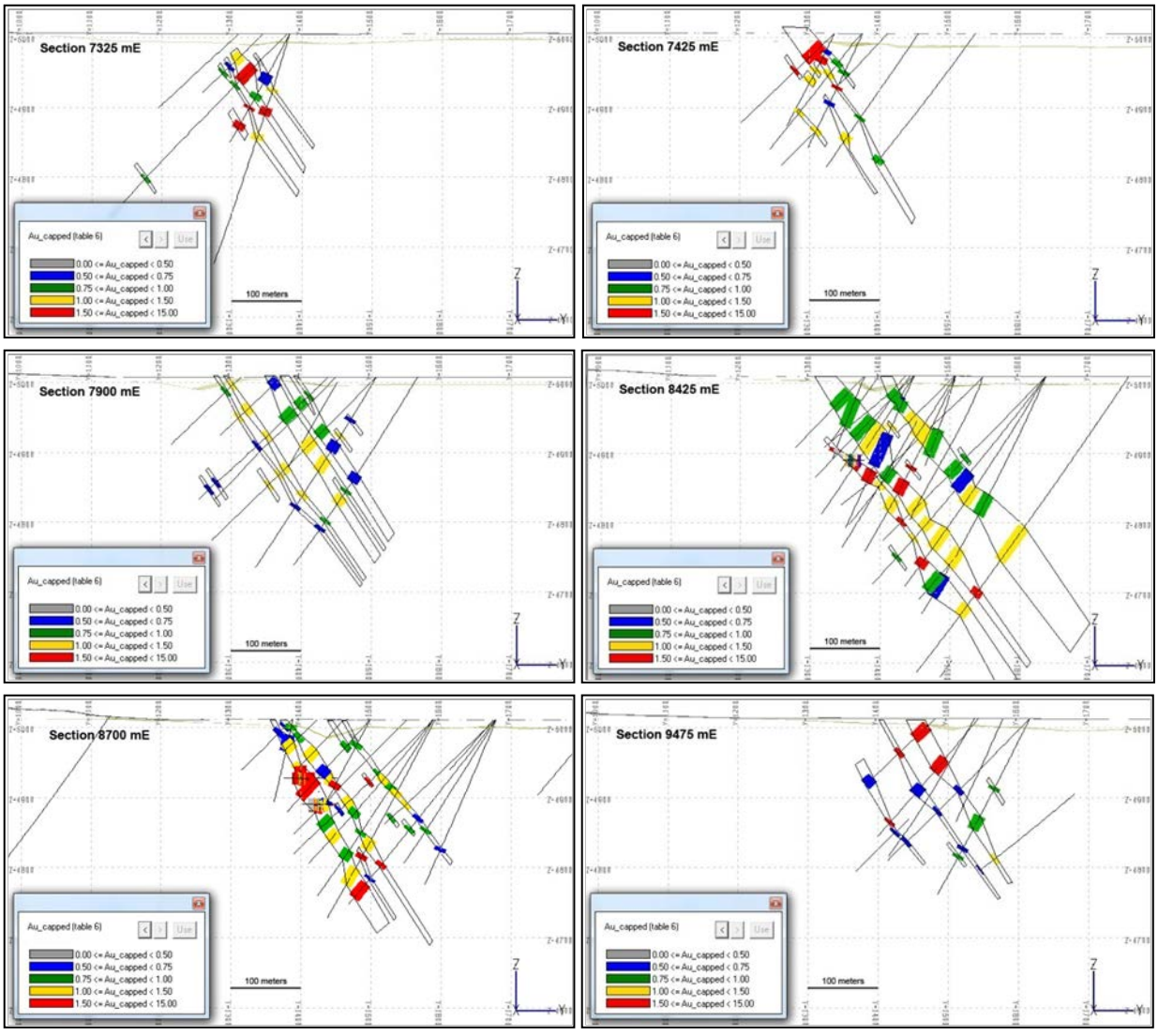


Figure 14.6: Sections with Mineralized Intervals and High Grade Au Solids (Looking West)

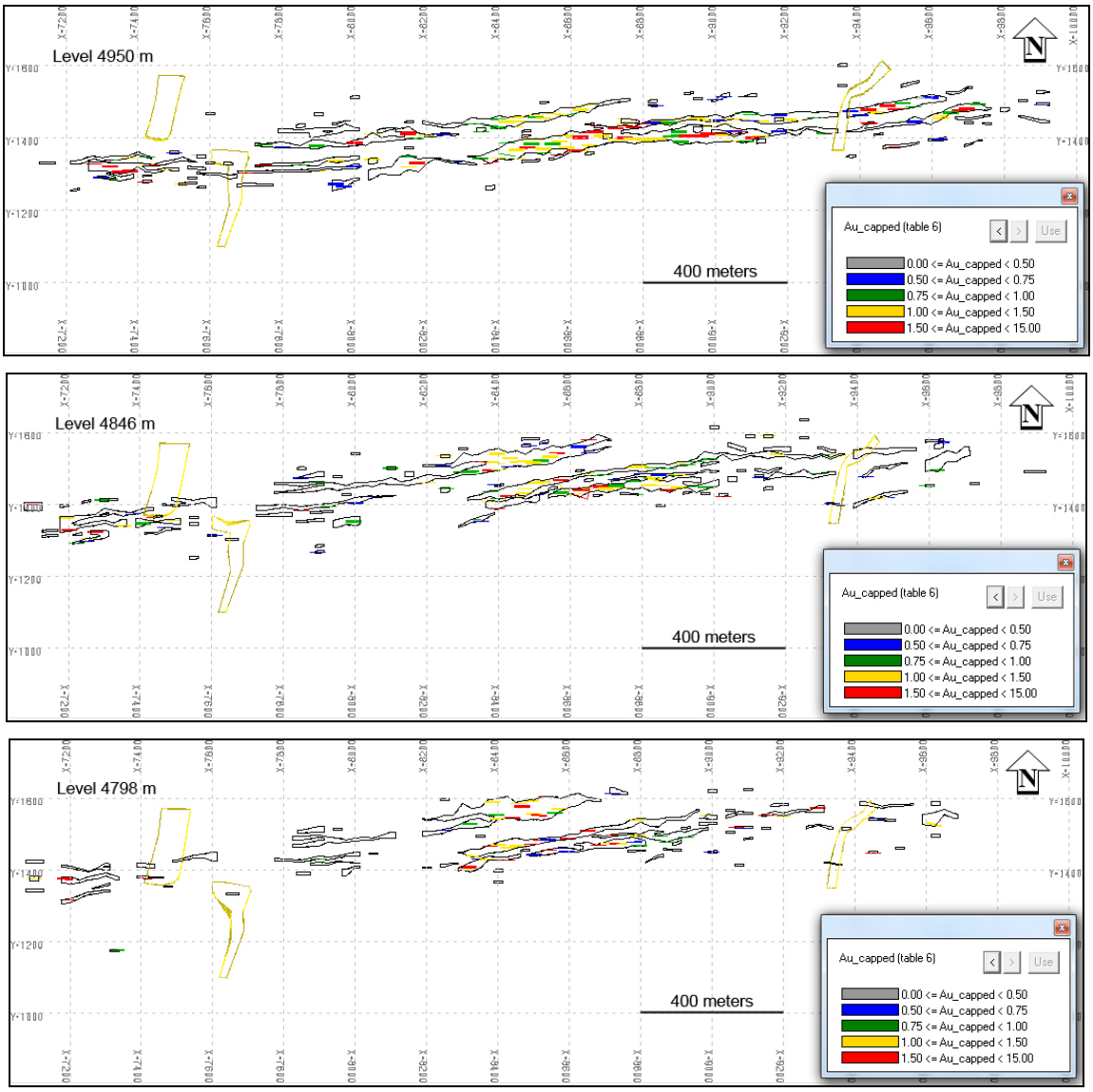


Figure 14.7: Level Views Showing Mineralized Intervals and High Grade Au Solids

14.2.3 Spatial Analysis

In June 2011, the spatial continuity of the high grade Au composites was assessed by variography. Experimental correlograms, which are the calculated correlation coefficient of grade from composite pairs separated by a given distance for a given direction, have been generated for 2 m composite data. Table 14.3 presents the June 2011 experimental variogram model. The spatial continuity outlined by the variography is characterised by: 1) a relative nugget effect of 40%, 2) a relatively strong anisotropy with the best continuity along the dip

direction (up to 25-30 m range) corresponding to the general dipping orientation of the high grade 3D envelopes, and worst continuity in the north-south direction across deposit.

Table 14.3: June 2011 Experimental Variogram Model for High Grade 2m Composites

Nugget effect	First Exponential Variogram Component							Second Exponential Variogram Component						
	Sill (C)	Ranges (in metre)			Orientation (in degrees)			Sill (C)	Ranges (in metre)			Orientation (in degrees)		
		Max	Interm.	Min	Azimuth	Dip	Spin		Max	Interm.	Min	Azimuth	Dip	Spin
0.04 (40%)	0.04 (40%)	3	3	1.5	350	-50	0	0.02 (20%)	30	5	3	350	-50	0

In September 2011, as before, correlograms were computed for the cut grade of the 19,214 of 2 m composites in the high grade envelope. They are computed in all directions together (average variogram with lag of 2 m) as well as along the principal directions of the mineralized zones i.e. (1) the average direction of surface drillholes (dip of 45° to south with lag of 2m) (2) the average horizontal strike (azimuth N80° with 5m lags) (3) the average dip (dip of 50° to N350° with 5m lags) (4) the horizontal NS (azimuth N0° with lags of 5m).

Experimental curves and fitted models are shown on Figure 14.8. As expected, they closely resemble those of the previous study i.e., relative nugget effect is 40% and continuity is better along dip and strike than across. Like before, the fitted model is the sum of that nugget effect of 40% plus a short range (local) component of 40% magnitude and a long range (regional) component of 20% magnitude. Ranges of the local component are about 15 m along dip, 10 m along strike and 5 m across dip and strike (vs. 10 m, 10 m and 5 m in the previous model) while the ranges of the regional component are about 90 m along dip, 60 m along strike and 15 m across dip+strike (vs. 90 m, 15 m and 10 m in the previous model). In other words, the long distance continuity along strike is more extensive in comparison to the previous study in which correlograms were computed on the uncapped grade of composites.

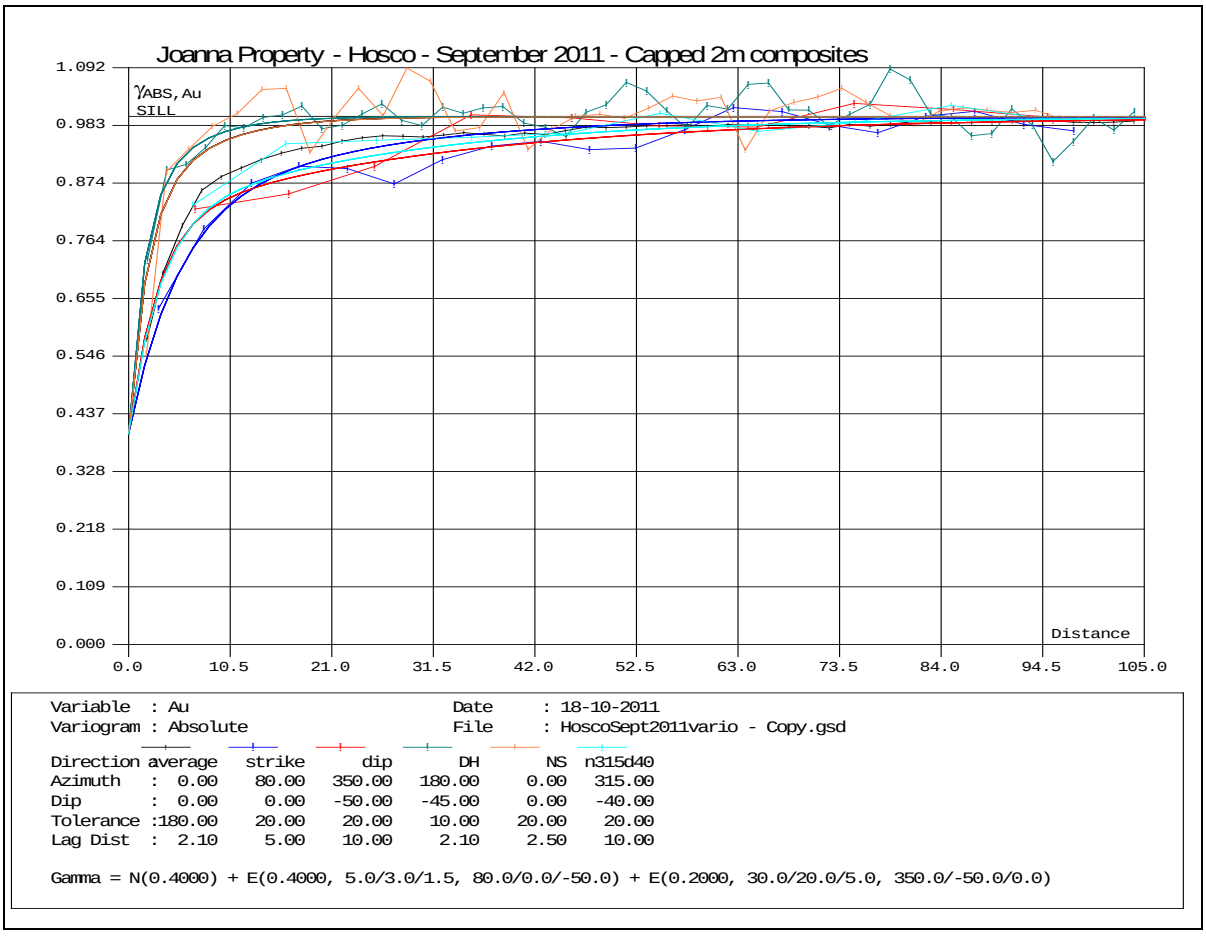


Figure 14.8: September 2011 Correlogram of 2 m Capped High Grade Au Composite Data

*Graphs shown are actually 1 correlogram so as to resemble variograms.

Table 14.4: September 2011 Experimental Variogram Model for High Grade Au 2 m Composites

Nugget Effect	First Exponential Variogram Component							Second Exponential Variogram Component						
	Sill (C)	Ranges (in metres)			Orientation (in degrees)			Sill (C)	Ranges (in metres)			Orientation (in degrees)		
		Max	Interm.	Min.	Azimuth	Dip	Spin		Max	Interm.	Min.	Azimuth	Dip	Spin
0.04 (40%)	0.04 (40%)	5	3	1.5	80	0	-50	0.02 (20%)	30	20	5	350	-50	0

14.2.4 Resource Block Modelling

A block size of 8 m (E-W) by 5 m (N-S) by 8 m (vertical) has been selected for the resource block models based on drillhole spacing, width and geometry of mineralization, and mining bench height defined in the Prefeasibility Study. The high grade resource block model which is constrained by the 3D wireframe solids contains a total of 93,509 blocks located outside the

defined intrusive units and below the bedrock-overburden surface. The maximum volume of the high grade block model totals more than 29,923,000 m^3. The low grade block model defines the area located outside the high grade 3D wireframe solids, below the bedrock-overburden surface and outside the defined intrusive unit limits. The low grade block model has not been constrained by other 3D wireframe envelopes but is limited to the defined block model maximum extent limits shown in Table 14.5. The coordinates used relate to the old mine grid parameters.

Table 14.5: Resource Block Model Parameters

Direction	Block Size	Number of Blocks	Coordinates (m)	
			Minimum	Maximum
East-West	8 m	376	7000 mE	10,000 mE
North-South	5 m	201	1000 mN	2000 mN
Vertical (Elevation)	8 m	126	4006 mZ	5006 mZ

14.2.5 Grade Interpolation Methodology

The grade interpolation for the Project resource block models was completed using Ordinary Kriging methodology ("OK"). Anisotropic search ellipsoids were selected for the grade interpolation process based on the analysis of the spatial continuity of capped high grade Au composites using variography. Limits are set for the minimum and maximum number of composites to be used for the interpolation and restrictions are applied on the number of composites used from each drillhole.

The interpolation process for the high grade block model was conducted using three (3) successive passes with relaxed search conditions from one pass to the next until all blocks were interpolated. The orientation of the search ellipsoid, which is identical for each interpolation pass is N350° azimuth, negative -55° dip and 0°spin. Figure 14.9 shows the different ellipsoids used during the interpolation process.

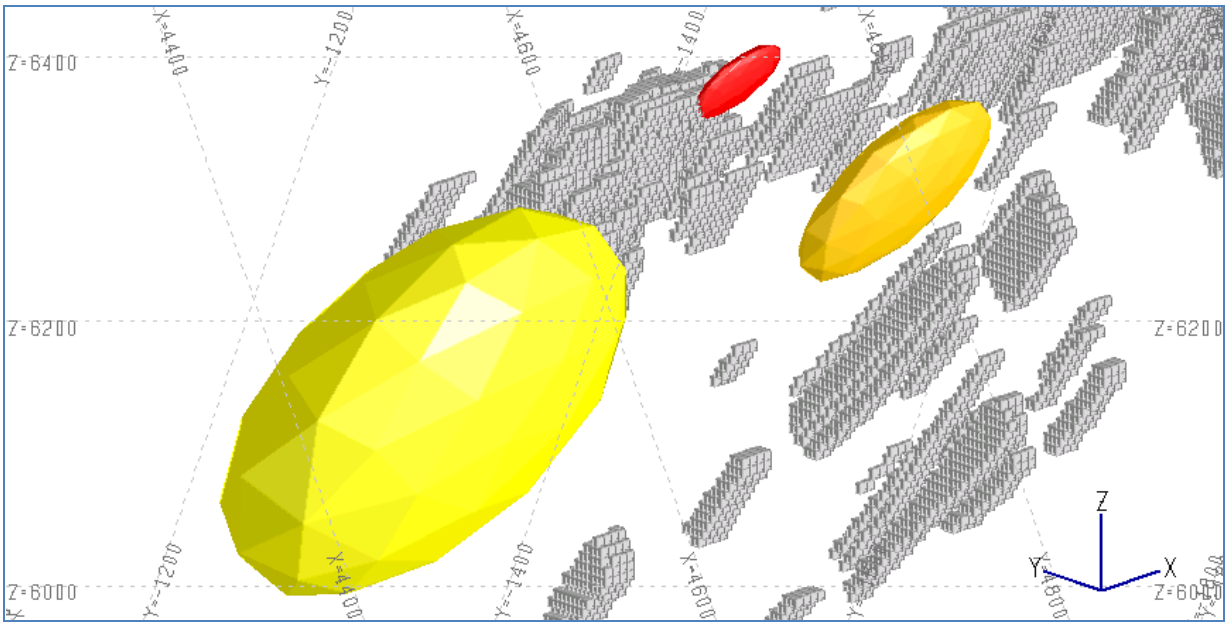


Figure 14.9: Different Search Ellipsoids Used for the Interpolation Process

In the first pass, the search ellipsoid dimension was 50 m (long axis) along strike, 35 m (intermediate axis) along dip and 10 m (short axis) along the direction perpendicular to strike and dip. The shape of the ellipsoid corresponds to the anisotropy of the correlogram. Search conditions required to estimate each block were defined by a minimum of seven (7) composites and a maximum of 30 composites with a maximum of three (3) composites selected from each hole or channel. Sixty-three percent (63%) of the blocks were estimated in the first pass.

In the second pass, the search ellipsoid dimension was increased to 100 m by 100 m by 30 m with a reduction of the anisotropy. The search conditions were the same as the first pass interpolation. The second pass resulted in the interpolation of 30% of the blocks.

The third interpolation pass estimated the remaining blocks (7% of total). The ellipsoid dimension for the last pass was increased to 200 m by 200 m by 100 m with search conditions defined as a 30 composite maximum and a three (3) composite minimum with a maximum number of composites per hole set to three (3).

The interpolation process for the low grade block model was limited to two (2) successive passes using the same ellipsoid dimensions and search conditions as defined for the high grade block model. A total of 310,824 blocks were interpolated during the first pass and a total of 838,481 blocks were populated after the second pass. The low grade block model resulted in only 154,927 blocks having Au grades above 0.1 g/t. Figure 14.10 and Figure 14.11 show the interpolation results on representative sections and plan levels (late intrusive units shown in orange outline).

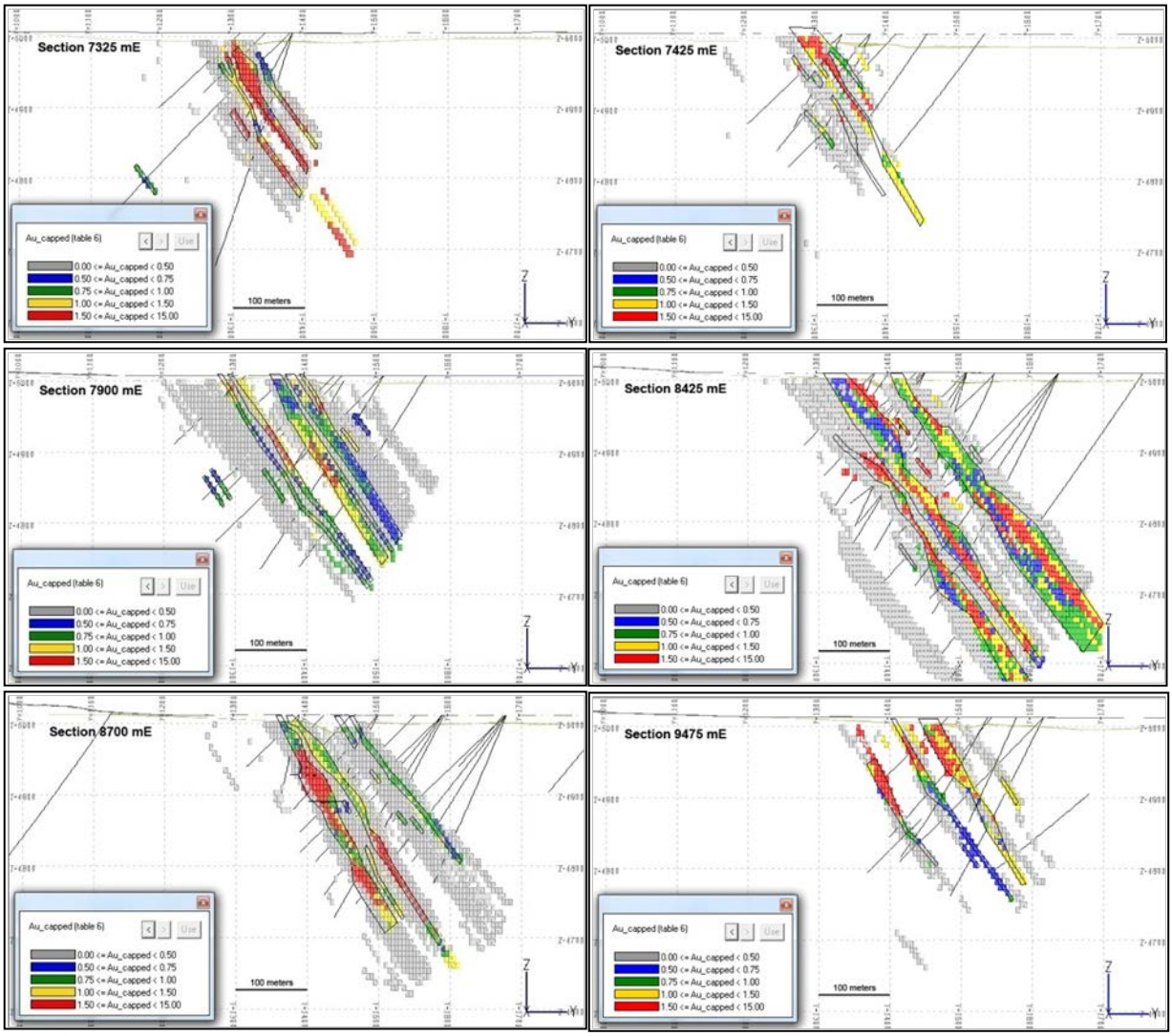


Figure 14.10: Sections Showing Au Block Model Interpolation Results (Looking West)

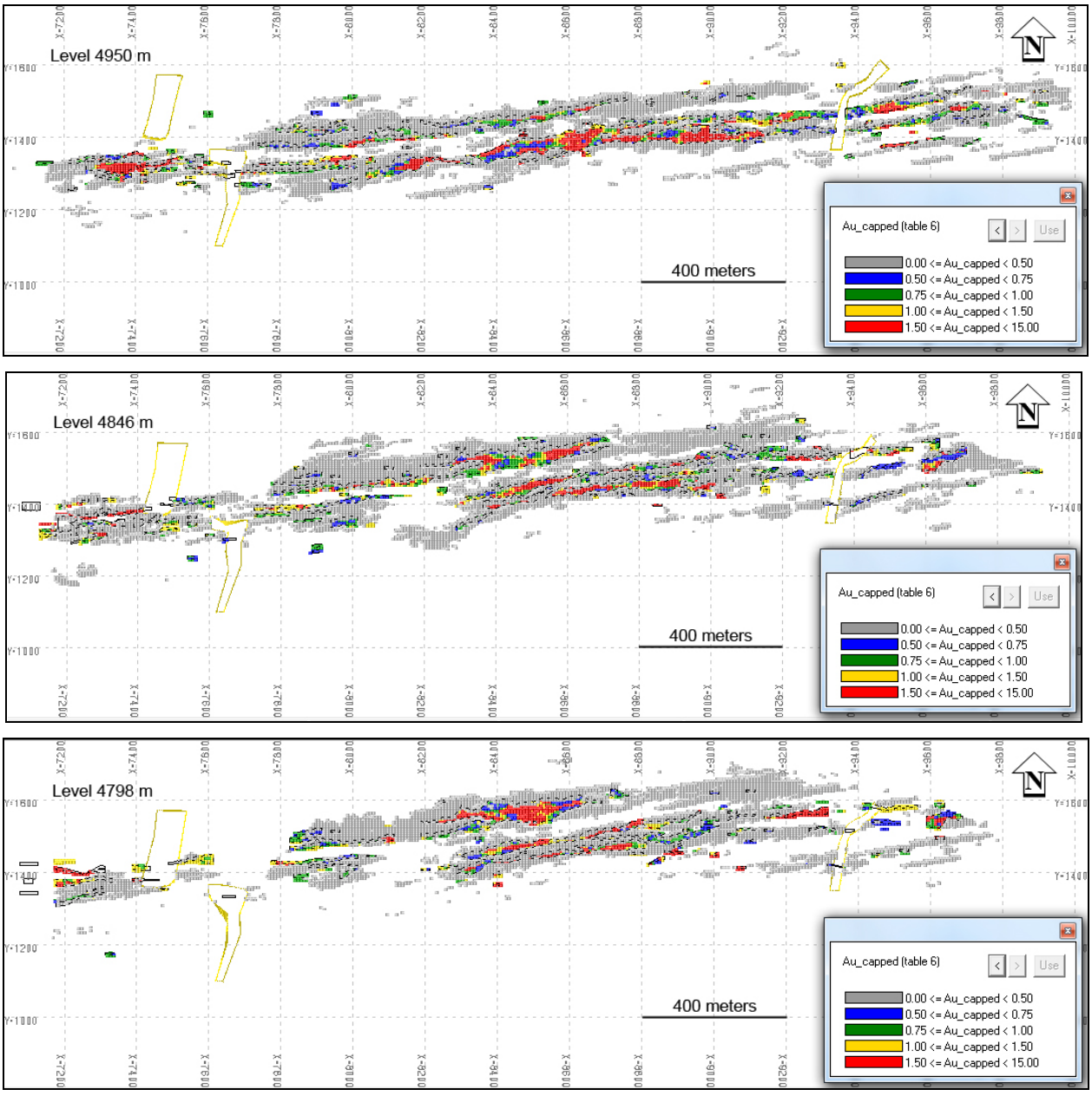


Figure 14.11: Level Views Showing Au Block Model Interpolation Results

14.2.6 Mineral Resource Classification

The mineral resources at Hosco are classified into measured, indicated and inferred categories. The parameters used to determine the mineral resources classification follow the CIM requirements and guidelines. The mineral resources were classified in two successive stages: automated classification followed by manual editing of the final resource categories.

The first classification stage is conducted by applying an automated classification process around each block that selects a minimum number of composites from a minimum number of holes located within a search ellipsoid of a given size and orientation. For the measured resource category, the search ellipsoid dimension is 30 m (along strike) by 30 m (dip direction) by 5 m. The search conditions for the measured category are a minimum of seven (7) composites located in at least four (4) different drillholes or channels. For the indicated category, the ellipsoid dimension is increased to 60 m by 60 m by 10 m with ellipsoid orientation and search criteria similar to the measured category. The second classification stage involves the delineation of coherent zones for the measured and indicated resource categories based on the results of the automated classification. The objective is to homogenise or “smooth” the results of the automated process by removing the “Swiss cheese” or “spotted dog” patterns typical of the automated classification results. This stage is conducted by defining 3D envelopes on a bench by bench basis for the measured and indicated categories. Figure 14.12 and Figure 14.13 show the block model final classifications in section and plan level views respectively (Categories: measured – red, indicated – blue, and inferred – grey).

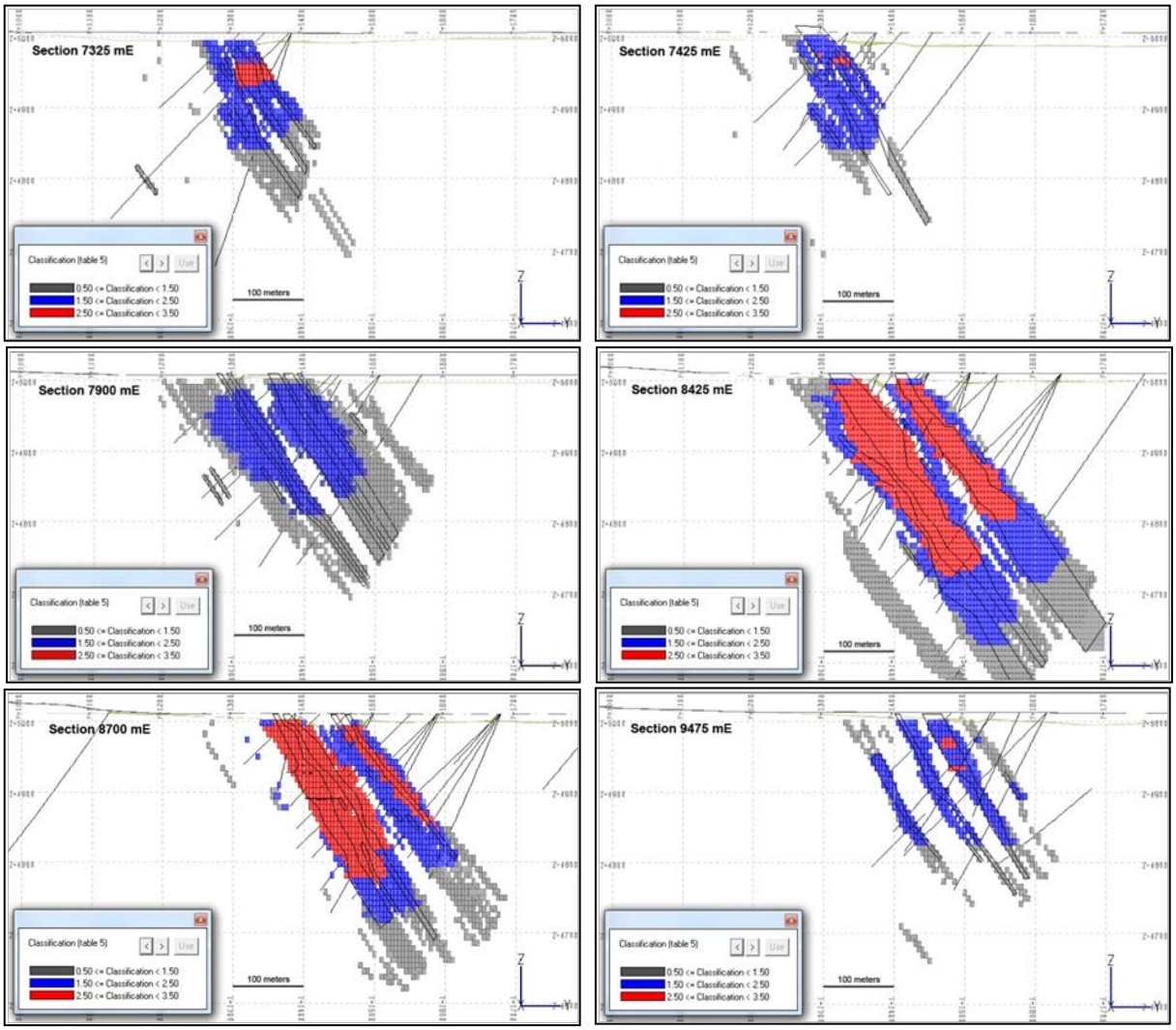


Figure 14.12: Sections Showing Final Resource Classification (Looking West)

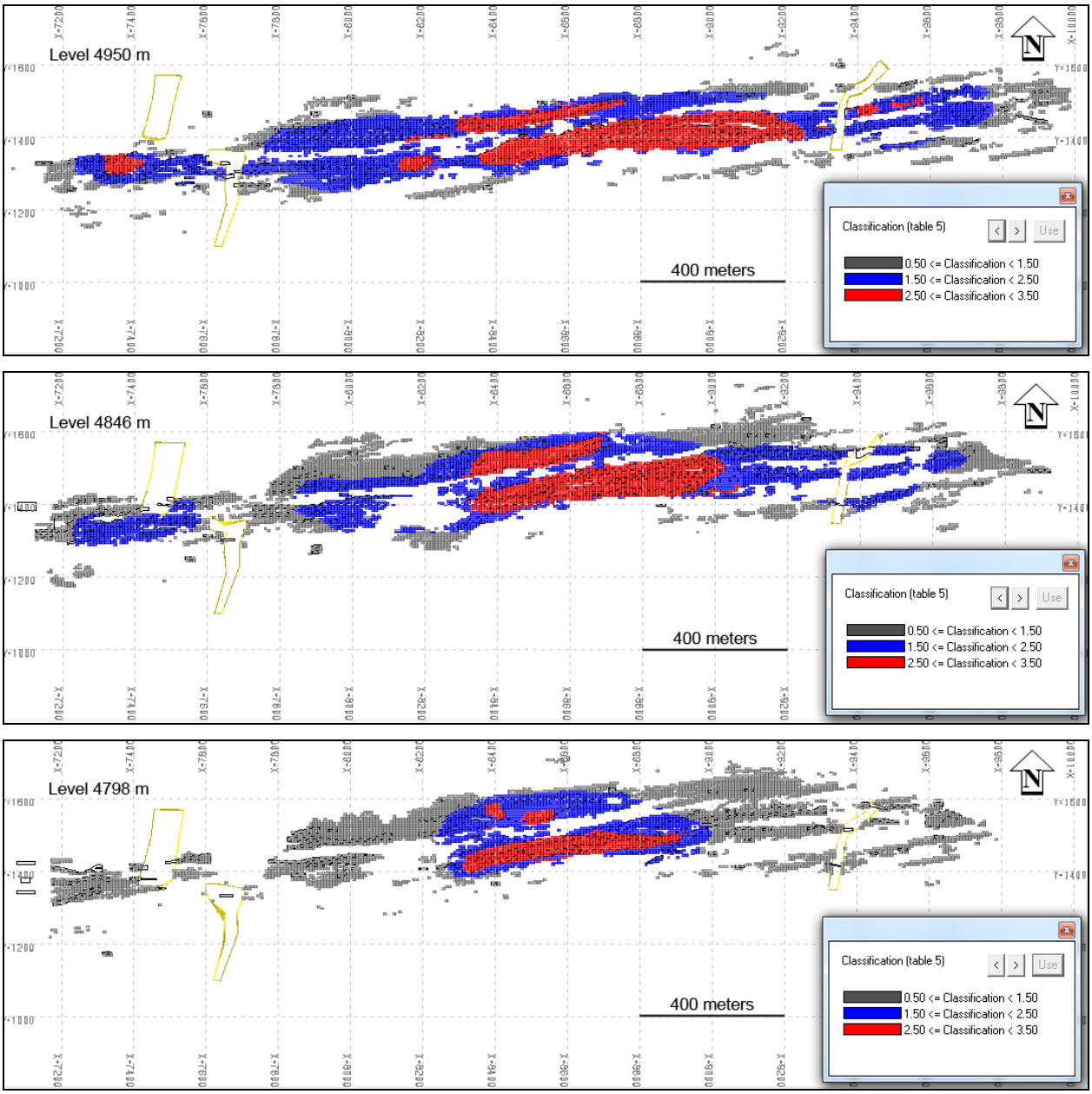


Figure 14.13: Plan Level Views Showing Final Resource Classification

14.2.7 Mineral Resource Estimation

The mineral resources of the Hosco deposit are defined using two distinct mining scenarios: open-pit and underground mining perspectives. The open pit mineral resources are reported within a Whittle pit shell optimised using the measured, indicated and inferred categories of the estimated global block model. The pit optimisation parameters supplied by Aurizon and

validated by SGS Geostat are based on the parameters used in the Project's Prefeasibility Study. Table 14.6 summarises the optimisation parameters. The base case cut-off grade for the in-pit mineral resources using a gold price of US$1,000 per oz Au is 0.33 g/t Au. The base case cut-off grade for the underground ("U/G") mineral resources at Hosco has been set at 2.0 g/t Au and is based on a conceptual U/G mining model completed by SGS Geostat. The reported U/G mineral resources are exclusive of the in-pit mineral resources. SGS Geostat considers that mineral resources defined at Hosco meet the requirement of a reasonable prospect for economic extraction.

Table 14.6: Parameters Used for the Whittle Open-Pit Optimisation

Parameters	Value	Unit
Sales Revenue		
Gold price	1,000	US$/oz
Exchange rate (CDN$/US$)	1	
Operating Costs		
Mining ore	1.75	CDN$/t
Mining waste	1.75	CDN$/t
Mining overburden	2	CDN$/t
Incremental mining cost per bench	0.01	CDN$/t
Crushing and milling	7.57	CDN$/t
Transportation	0.95	CDN$/t
General and administration	0.73	CDN$/t
Metallurgy and Refining		
Metallurgical recovery	86.8	%
Refining and transport charges	5	CDN$/oz
Gold payment	99.935	%
Geotechnical Parameters		
Pit slope (north wall)	49	degrees
Pit slope (south, east and west walls)	53	degrees
Bulk density (ore and waste)	2.75	t/m3
Bulk density (overburden)	2	t/m3

The mineral resource estimate includes the results of both high grade and low grade Au block models. The mineral resource tonnage has been calculated from the volumetric estimates of the resource block models using an average bulk density of 2.75 t/m^3.

The updated mineral resources for the Hosco deposit are as follows: The in-pit mineral resources using a base case cut-off grade of 0.33 g/t Au totals 30,690,000 tonnes grading

1.33 g/t Au for 1,311,000 oz Au in the measured category, 27,150,000 tonnes grading 1.18 g/t Au for 1,033,000 oz Au in the indicated category with an additional 7,050,000 tonnes grading 1.18 g/t Au for 267,000 oz Au in the inferred resources category. The U/G mineral resources using a base case cut-off grade of 2.0 g/t Au totals 50,000 tonnes grading 2.65 g/t Au for 5,000 oz Au in the indicated category with an additional 590,000 tonnes grading 2.54 g/t Au for 48,000 oz Au in the inferred resources category. Table 14.7 summarises the in-pit mineral resources for Hosco for cut-off grades of 0.33 g/t Au (base case), 0.5 g/t Au, and 1.0 g/t Au. Table 14.8 reports the U/G mineral resources for Hosco for cut-off grades of 2.0 g/t Au (base case), 2.5 g/t Au, and 3.0 g/t Au.

Table 14.7: Updated Mineral Resources for the Hosco Deposit (In-Pit)

Grade Cut-off (Au g/t)	Category	Tonnage* (t)	Grade (g/t)	Au metal** (oz)
0.33 (Base Case)	Measured (M)	30,690,000	1.33	1,311,000
	Indicated (I)	27,150,000	1.18	1,033,000
	Total (M+I)	57,840,000	1.26	2,344,000
	Inferred	7,050,000	1.18	267,000
0.5	Measured (M)	29,460,000	1.37	1,295,000
	Indicated (I)	26,410,000	1.21	1,024,000
	Total (M+I)	55,870,000	1.29	2,319,000
	Inferred	6,950,000	1.19	266,000
1.0	Measured (M)	20,370,000	1.62	1,058,000
	Indicated (I)	15,130,000	1.51	736,000
	Total (M+I)	35,500,000	1.57	1,794,000
	Inferred	4,520,000	1.39	201,000

Updated: September 22, 2011- *Rounded to nearest 10k- **Rounded to 1k
Relative density used: 2.75t/m3
CIM Definitions were followed for mineral resources
Mineral resources which are not mineral reserves do not have demonstrated economic viablilty
Historical production of 9,704 oz has not been removed from mineral resources

Table 14.8: Updated Mineral Resources for the Hosco Deposit (Underground)

Grade Cut-off	Category	Tonnage (t) (t)	Grade (g/t)	AuMetal** (oz)
2.0 (Base Case)	Indicated (I)	50,000	2.65	5,000
	Inferred	590,000	2.54	48,000
2.5	Indicated (I)	20,000	3.33	2,000
	Inferred	180,000	3.39	19,000
3.0	Indicated (I)	10,000	3.90	1,000
	Inferred	100,000	4.00	12,000

Updated: September 22, 2011- *Rounded to nearest 10k- **Rounded to 1k
Relative density used: 2.75t/m3
CIM Definitions were followed for mineral resources
Mineral resources which are not mineral reserves do not have demonstrated economic viablilty

14.2.8 Sensitivity Analysis

During March 2011, some sensitivity analysis of the high grade block model was conducted using two other interpolation methodologies from the previous resources and was updated as stated in the Company's June 13, 2011 press release. The interpolation methods used for the analysis are inverse distance to the power square ("ID2") and nearest neighbour ("NN"). The block model, ellipsoid and search conditions parameters are the same as with the OK interpolation methodology. Table 14.9 compares the results of the sensitivity analysis to the OK mineral resource estimate for an Au cut-off grade of 0.33 g/t. Sensitivities analysis using ID2 and NN returned relative differences for Au metal of less than 2% with the exception of the NN inferred resources where the relative difference for Au metal returned 9%. The results of the sensitivity analysis are typical for the data distribution of the Hosco deposit where relatively high Au grade analytical data occurs in clusters where underground channel sampling was historically conducted in the old mining levels. OK is known to have de-clustering properties which can lower the effect of the high density channel data observed on the Project.

Table 14.9: June 2011 Sensitivity Analysis of the High Grade Block Model Using a Cut-Off Grade of 0.33 g/t Au

Interpolation Method	Resource Category	Tonnage* (t)	Grade (g/t)	Au metal** (oz)
OK	Measured	29,490,000	1.35	1,283,000
	Indicated	25,840,000	1.18	977,000
	M+I	55,330,000	1.27	2,260,000
ID2	Measured	29,770,000	1.34	1,279,000
	Indicated	26,200,000	1.16	975,000
	M+I	55,960,000	1.25	2,254,000
	Relative difference	*1%*	*-1%*	*0%*
NN	Measured	28,190,000	1.39	1,264,000
	Indicated	26,130,000	1.22	1,024,000
	M+I	54,320,000	1.31	2,288,000
	Relative difference	*-2%*	*3%*	*1%*

Interpolation Method	Resource Category	Tonnage* (t)	Grade (g/t)	Au metal** (oz)
OK	Inferred	7,730,000	1.15	285,000
ID2	Inferred	7,890,000	1.11	282,000
	Relative difference	*2%*	*-3%*	*-1%*
NN	Inferred	9,000,000	1.07	310,000
	Relative difference	*16%*	*-6%*	*9%*

* Rounded to nearest 10K - ** Rounded to nearest 1K

14.2.9 Mineral Resource Validation

A validation of the mineral resource Au grade was conducted as part of the verification process. The validation includes: 1) a visual comparison of the color-coded block values versus the composite data in the vicinity of the interpolated blocks on random sections and plan views, and 2) a validation of the blocks of the different mineralised envelopes that were used for the final block model. The author also verified that blocks did not belong to two distinct mineralised envelopes within the final block model and that no blocks belonging to the late intrusive [dykes] were included in the final block model.

14.3 Alexandria Deposit Resource Estimate

The mineral resources of the Alexandria deposit were restated as disclosed in the Company's June 13, 2011 news release to reflect the modification of the limit separating the eastern end of the Hosco deposit and the western end of the Alexandria deposit. The mineral resource

estimate was also restated at that time by adjusting the average bulk density of the mineralised lithologies in the area.

As described in the previous sections, the easternmost limit of the Hosco deposit has been moved from 9,700 mE to 10,000 mE, which is now selected as the limit defining Hosco versus Alexandria. Since the current mineral resources of Alexandria (November 2009) range from 9,700 mE to approximately 11,500 mE, an adjustment of the mineral resource estimates for Alexandria was necessary. The restated resources estimates for Alexandria exclude the previous mineral resources located between 9,700 mE and 10,000 mE. No additional drilling data was included in the mineral resource estimates for Alexandria and no new grade interpolation was completed. Since the mineral resources at Alexandria are all located within 175 m below surface and thus potentially mineable by open pit, the base case cut-off grade has been set at 0.5 g/t Au. This corresponds to approximately twice the cut-off grade defined for Hosco. No pit optimisation was conducted on the Alexandria mineral resources.

The mineral resources for the Alexandria deposit were estimated by QP Michel Dagbert, P.Eng. in November 2009 using analytical data from 12 recent surface drillholes and six (6) historical surface drillholes. The mineral resource estimate was completed using a 2D polygonal modelling in which mineralised intervals identified along drillholes were projected onto the east-west vertical long section. The mineral resources volume is estimated using the length of each mineralised interval and limited to an area of influence of 25 m by 25 m squares. In the restatement disclosed in the Company's June 13, 2011 news release, the average bulk density used to calculate tonnage from the volumetric estimates of the 2D polygonal model for the Alexandria deposit was changed to 2.75 t/m^3. The new average bulk density is based on the average bulk density for the Hosco deposit, which is located to the west, adjacent to the Alexandria deposit.

The restated mineral resources for the Alexandria deposit use a base case cut-off grade of 0.5 g/t Au totals 980,000 tonnes grading 1.2 g/t Au for 37,000 oz Au in the inferred category. No mineral resources were defined in the measured and indicated categories. Table 14.10 summarises the mineral resources for Alexandria for cut-off grades of 0.5 g/t Au (base case), 1.0 g/t Au, and 1.5 g/t Au.

Table 14.10: Mineral Resources for the Alexandria Deposit

Grade Cut-off (Au g/t)	Category	Tonnage* (t)	Grade (g/t)	Au metal** (oz)
0.5 (base case)	Inferred	980,000	1.2	37,000
1.0	Inferred	550,000	1.4	26,000
1.5	Inferred	140,000	2.3	10,000

Dated: May 31, 2011- *Rounded to nearest 10k- **Rounded to 1k
Relative density used: 2.75t/m3
CIM Definitions were followed for mineral resources
Mineral resources which are not mineral reserves do not have demonstrated economic viablilty

14.4 Heva Deposit Resource Estimate

The only modification to the reporting of the mineral resources of the Heva deposit made in the restatement disclosed by the Company on June 13, 2011 was the adjustment of the cut-off grade for the mineral resources located at depth. Based on results from the open-pit optimisation for the Hosco mineral resources, it was decided that a different cut-off grade should be applied to the mineral resources located at depth versus the resources closer to surface. The elevation of 4,700 m (which corresponds to approximately 300 m below surface) was selected as the threshold between the resources having open-pit potential compared to the resources accessible only by an underground mining operation. The mineral resources for Heva were not updated using additional drilling information and the average bulk density remains the same as the previous estimate of 2.81 t/m^3. No pit optimisation was conducted on the Heva mineral resources.

The mineral resources for the Heva deposit were estimated by QP Michel Dagbert, P.Eng in November 2009 using analytical data from 47 recent surface drillholes and 466 historical drillholes or channels. The analytical data was normalised to 2 m long composites and high grade values were capped at 15 g/t Au. The mineral resource estimate was completed using 3D modelling and block model interpolation methods. The interpolation was done by Ordinary Kriging. The block model covers a strike length of 2,800 m between 4,200 mE and 7,000 mE of the local grid and reaches a maximum depth of 650 m below surface. The block model were defined by block size of 8 m east-west, 5 m north-south and 5 m elevation. An average bulk density of 2.81 t/m^3 was used to calculate tonnage from the volumetric estimates of the block model.

The restated mineral resources for the Heva deposit are as follow: The mineral resources located above 4,700 m elevation and using a base case cut-off grade of 0.5 g/t Au totals 4,410,000 tonnes grading 1.91 g/t Au for 270,000 oz Au in the indicated category with an additional 7,680,000 tonnes grading 1.71 g/t Au for 421,000 oz Au in the inferred resources category. The mineral resources located below 4,700 m elevation and using a base case cut-off grade of 2.0 g/t Au totals 650,000 tonnes grading 2.83 g/t Au for 59,000 oz Au in the inferred resources category. Table 14.11 summarizes the in-pit mineral resources for Heva for cut-off grades of 0.5 g/t Au (base case), 1.0 g/t Au and 1.5 g/t Au for the resources located above 4,700 m elevation, and 2.0 g/t Au (base case), 2.5 g/t Au and 3.0 g/t Au for the resources located below 4,700 m elevation.

Table 14.11: Mineral Resources for the Heva Deposit

Resources Depth	Grade Cut-off (Au g/t)	Category	Tonnes (t)	Au Grade (g/t)	Au Metal (oz)
Above Elevation 4700 m (open pit)	0.5 (base case)	Indicated	4,410,000	1.91	270,000
		Inferred	7,680,000	1.71	421,000
	1.0	Indicated	3,860,000	2.06	256,000
		Inferred	6,970,000	1.80	403,000
	1.5	Indicated	2,600,000	2.45	205,000
		Inferred	4,700,000	2.04	308,000
Below Elevation 4700 m (underground)	2.0 (base case)	Inferred	650,000	2.83	59,000
	2.5	Inferred	410,000	3.19	42,000
	3.0	Inferred	250,000	3.48	28,000

Updated May 3, 2011
Bulk density used: 2.81 t/m3
*Rounded to nearest 10k - **Rounded to nearest 1k
CIM definitions were followed for mineral resources
Mineral resources which are not mineral reserves do not have demonstrated economic viability

14.5 Total Mineral Resource Estimate for the Joanna Property

The final mineral resources for the Joanna property, which include Hosco, Heva and Alexandria deposits, are reported in Table 14.2.

Table 14.12: Updated Mineral Resources for the Joanna Property

Sector	Resource Depth	Resource Category	Cut-off Grade (g/t)	Tonnage* (t)	Au Grade (g/t)	Au Metal** (oz)
Hosco	In-pit	Measured	0.33	30,690,000	1.3	1,311,000
Hosco	In-pit	Indicated	0.33	27,150,000	1.2	1,033,000
Heva	Above 4,700m Elev.	Indicated	0.5	4,410,000	1.9	270,000
Joanna	**In-pit/Above 4,700m Elev.**	**Total M + I**		**62,250,000**	**1.3**	**2,614,000**
Hosco	In-pit	Inferred	0.33	7,050,000	1.2	267,000
Heva	Above 4,700m Elev.	Inferred	0.5	7,680,000	1.7	421,000
Alexandria	Above 4,700m Elev.	Inferred	0.5	980,000	1.2	37,000
Joanna	**In-pit/Above 4,700m Elev.**	**Total Inferred**		**15,710,000**	**1.4**	**725,000**
Hosco	U/G	Indicated	2.0	50,000	2.6	5,000
Joanna	**U/G - Below 4,700 m Elev.**	**Total Indicated**		**50,000**	**2.6**	**5,000**
Hosco	U/G	Inferred	2.0	590,000	2.5	48,000
Heva	Below 4,700 m Elev.	Inferred	2.0	650,000	2.8	59,000
Joanna	**U/G - Below 4,700 m Elev.**	**Total Inferred**		**1,240,000**	**2.7**	**107,000**

*Rounded to nearest 10k - **Rounded to nearest 1k

Updated: September 22, 2011

CIM definitions were followed for mineral resources

Mineral resources which are not mineral reserves do not have demonstrated economic viability

Historical production of 9,704 oz (Hosco) and 10,700 oz (Heva) has not been removed from mineral resources

14.6 Comments about the Mineral Resource Estimates

There are no known factors or issues related to environment, permitting, legal, mineral title, taxation, marketing, socio-economic or political settings that could materially affect the mineral resource estimate.

Important note: Different assumptions were made during the mineral resource estimation process. The assumptions were used in order to calculate modelling cut-off grades and resources cut-off grades following the “reasonable prospect for economic extraction” stated by the NI 43-101 regulation. A Whittle optimised shell was done with the same parameters as described in Table 14.6. The term in-pit refers to the resources within the optimised shell according to the different cut-off grades. The term underground (U/G) refers to a conceptual U/G mining model completed in-house by SGS Geostat. The terms in-pit and Underground (U/G) do not imply that any pit design and underground mining scenarios were done by SGS Geostat. Furthermore, it should not imply that the resources stated herein have demonstrated economic viability.

15 MINERAL RESERVE ESTIMATES

The reserves for the detailed pit design have been calculated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM Standards on Mineral Resources and Reserves) in August, 2000. The mineral reserves (with dilution and ore loss) contained within the Main Pit and West Pit amount to 41.1 Mt of ore at an average grade of 1.26 g/t Au using a cut-off grade of 0.5 g/t Au, for a total of 1.66M oz of in-situ gold and 1.45M oz of recovered gold. Total waste, including rock, inferred material and overburden, is 184.6 Mt for a stripping ratio of 4.49. The detailed mineral reserve estimate is shown in Table 15.1.

Table 15.1: Mineral Reserve Estimate (Cut-Off 0.5 g/t Au)

Material	Main Pit		West Pit		TOTAL			
	Material (k tonnes)	Grade (g/t Au)	Material (k tonnes)	Grade (g/t Au)	Material (k tonnes)	Grade (g/t Au)	Au koz (In-Situ)	Au koz (Recovered)
Proven Reserves	27 419	1.29	817	1.71	28 236	1.30	1 185	1 040
Probable Reserves	11 356	1.11	1 496	1.47	12 851	1.15	476	414
TOTAL	**38 775**	**1.24**	**2 313**	**1.56**	**41 087**	**1.26**	**1 660**	**1 454**
Waste	150 587		11 733		162 320			
Overburden	18 338		3 973		22 311			
TOTAL	**168 925**		**15 706**		**184 631**			
Strip Ratio	**4.36**		**6.79**		**4.49**			

Note: Mill Recovery Equation = $-0.0467\ Au^2 + 0.1745\ Au + 0.7299$

16 MINING METHODS

16.1 General Description

Mining of the Joanna Hosco deposit will follow the standard practice of an open-pit operation with conventional drill and blast, load and haul cycle using a drill/truck/shovel mining fleet. The overburden and waste rock material will be hauled to the designated overburden and waste disposal areas. The run-of-mine ore will be drilled, blasted and loaded by hydraulic shovels and delivered by trucks to the primary crusher or stockpile near the crusher. Mining operations will support the nominal milling throughput rate of 8,500 tpd.

Aurizon Mines will use its own employees to operate mining equipment. It is planned however that overburden removal will be carried out by a local mining contractor.

16.2 Resource Block Model

The mining engineering work required for the Study, such as the pit optimization, engineered pit design, mine planning and economic analysis, is based on the resource block model prepared by SGS Geostat. The block model was provided in a CSV file named *Hosco_Oct_2011HG-LGclassFINAL.csv* and was transferred by BBA into the mining software MineSight. Aurizon provided BBA with topographical and bedrock mapping data. A local coordinate system named *GML* was used. The unit block size in the model is 8 m (East by West) x 5 m (North by South) x 8 m (Height).

The following data was provided by SGS Geostat in the model:

- GML Coordinates;
- Density;
- Classification (Measured, Indicated or Inferred);
- Au (gold in g/t, i.e. grade item).

Additional variables were added to BBA's model in MineSight in order to perform calculations and to determine block value.

Following the import of the block model into MineSight, a verification of the total mineral resources by category was performed in order to ensure conformity with the results provided by SGS.

16.3 Pit Optimization

The objective of pit optimization and mine design is to convert the resources in the block model into mine reserves for this Feasibility Study in order to serve as a basis for mine scheduling and economic analysis.

Given the operating costs, gold recovery, gold price, and other design parameters, the pit optimization is used to generate an optimal pit shell to maximize the undiscounted cash flow of the project. BBA has used the MineSight Lerchs-Grossman 3D ("LG 3D") routine named MS-EP – a widely accepted standard in the mining industry - to generate the pit shell for the Hosco deposit. The LG 3D is a true pit optimizer based on dynamic programming of graph theory to generate an optimized pit shell from a 3D block model. The basic optimization principle of the algorithm operates on a net value calculation for each ore block in the model, i.e., revenue from sales less total operating costs, i.e., mining, crushing, processing, G&A, and other costs.

Based on the requirement of the Canadian NI 43-101 Standards of Disclosure for Mineral Projects, only ore blocks classified in the Measured and Indicated categories are used to drive the pit optimizer for a Feasibility Study, and as such, the Inferred resource blocks bear no economic value and are not used in the pit optimization process.

16.3.1 Pit Optimization Criteria and Parameters

Operating costs and design parameters used for the LG 3D shell were developed by BBA based on information from the ''Pre-Feasibility Study of the Hosco Deposit'' on the Joanna Gold Project (December 2009), and on available results and assumptions on crushing and processing from BBA. The technical and economic parameters used to conduct the pit optimization are outlined in Table 16.1.

Table 16.1: Pit Optimization Parameters for LG 3D

Parameter	Value
Sales Revenue	
Gold Price	900 US$/oz
Exchange Rate	1.00 C$/US$
Operating Cost	
Mining Ore	2.00 C$/t
Mining Overburden	2.00 C$/t
Mining Waste	2.00 C$/t
Crushing and Processing	13.00 $/t milled
Additional Transportation Cost	0.00 $/t milled
Other Costs (environment, on-going…)	0.40 $/t milled
G/A	0.73 $/t milled
Incremental Cost per Bench	0.01 C$/t/bench
Metallurgy	
Gold Recovery	87.5 %
Refining and Transport	3 C$/oz
Gold Payment	99.935 %
Pit Parameters	
Overall Pit Slope	48°

16.3.2 Theoretical Pit Shells

Using the technical and economical parameters presented above, the LG 3D pit optimizer was carried out to generate an optimum pit shell having the highest undiscounted cash flow. A plan view of the LG 3D pit shell is shown in Figure 16.1.

The theoretical pit shell resulting from the LG 3D optimization is only preliminary and does not represent a practical design for mining. This optimized pit shell will be used to serve as a guide for the detailed mine design with the required operational haulage ramp, proper pit slope, and benching arrangement, as presented in this section.

In order to optimize the operational stripping ratio in the early years of the project and to increase the net present value of the project, a series of optimized pit shells were generated using the LG 3D MineSight routine by varying the price of gold. These pit shells served as a guide for the pit phases design, as presented later in this section.

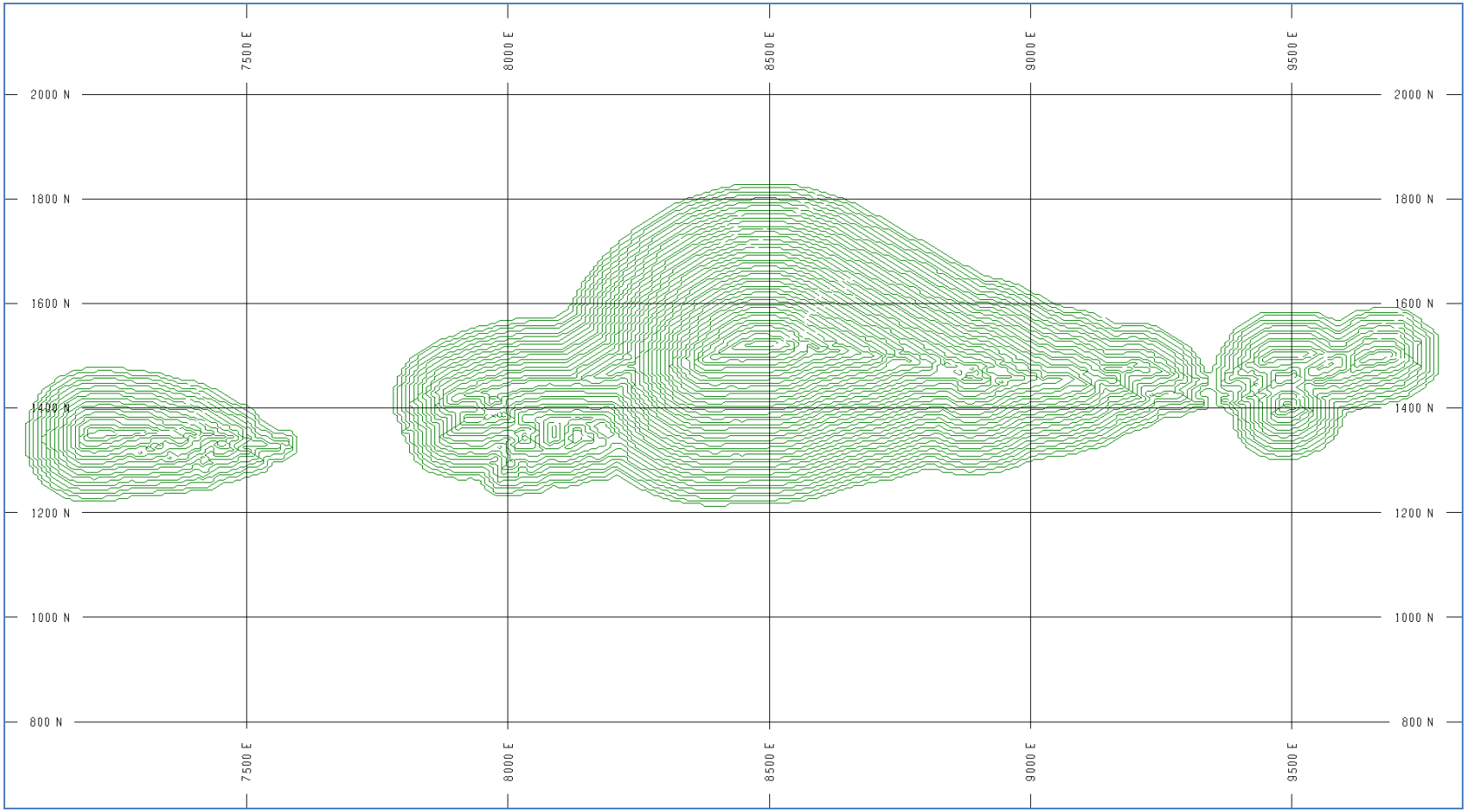


Figure 16.1: LG 3D Optimized Pit Shell

16.4 Mine Design and Reserves

16.4.1 Specific Gravity

The specific gravity of waste rock and ore is 2.75t/m^3, and specific gravity of overburden is 2.00t/m^3.

16.4.2 Mill Cut-Off Grade

The break-even cut-off grade or the milling cut-off grade (CoG) is used to classify the material inside the pit limits as ore or waste. Since the material is located inside the pit, the break-even cut-off is the grade required to cover the costs for processing, G&A, and those related to gold refining and transport only.

Using the economic and technical parameters shown in Table 16.1, the mill cut-off grade was calculated at 0.50 g/t Au, including an average dilution rate of 5%.

16.4.3 Detailed Pit Design

The slope configuration in overburden is based on LEQ's recommendations as discussed in Section 20 of their report (LEQ, 2012). The pit slopes and benching arrangement in fresh rock are based on Golder's recommendations (Golder, 2009). For the rock slopes, Golder recommends the following:

- Bench face angle (BFA) of 65° and inter-ramp angle (IRA) of 49° for Northeast to Northwest walls (including the South wall) and BFA of 70° and IRA of 53° for the North wall.
- At the current study level, it would not be appropriate to apply the recommended slope angles over extended wall heights due to the uncertainty regarding fabric continuity inherent in slope designs, based upon drillhole information. It is therefore recommended that flexibility be added to the designs by the sub-division of the final and phase walls into a series of bench stacks, no greater than 120 m high, separated by either haul road traverses or the inclusion of "geotechnical" berms at 100 m – 120 m vertical intervals. The geotechnical berms should be at least 15 m wide and would also act to protect personnel from potential major rock falls, allow for horizontal drain hole water controls, provide some flexibility in wall development and allow for periodic clean-up.

Triple benching arrangement is used to form 24 m high benches.

All in-pit ramps will be 24 m wide to accommodate the operation of 100-ton capacity off-highway class rear-dump trucks. This ramp width is sufficient to support uninterrupted 2-way haulage cycle traffic. The final ramp width also includes an external safety berm as well as an internal drainage ditch. In order to reduce the volume of waste stripping during the final phases, the ramp width is reduced to 18 m for the last benches at the bottom of the pit. The ramp gradient is 10% for all mine ramps. Temporary ramps will be used in the early years of mine operations to shorten haulage distances to the primary crusher or to the overburden and waste dumps.

The final ramp and temporary ramp exits are located on the north side of the pit to minimize the haulage distances towards the primary crushing, as well as the overburden and waste dumps.

The detailed pit design work was carried out using the LG 3D outlines described in this section. The ultimate pit design includes the entire practical geometry required in a mine including pit access and haulage ramp to all pit benches, pit slope design, benching configurations, smoothed pit walls, catch berms and geotechnical berms as described previously.

Based on the optimization pit shells, the Hosco deposit will be mined using two distinct pits: the Main pit and the West pit. The design pits are approximately 2,190 m in length by 660 m wide and 280 m deep for the Main pit, and 660 m in length by 450 m wide and 120 m deep for the West pit.

The detailed open-pit design is shown in Figure 16.2 and Figure 16.3. Also, Figure 16.45 and present cross-sections of the detailed pit design with the mineralized blocks by coloring cut-off grades.

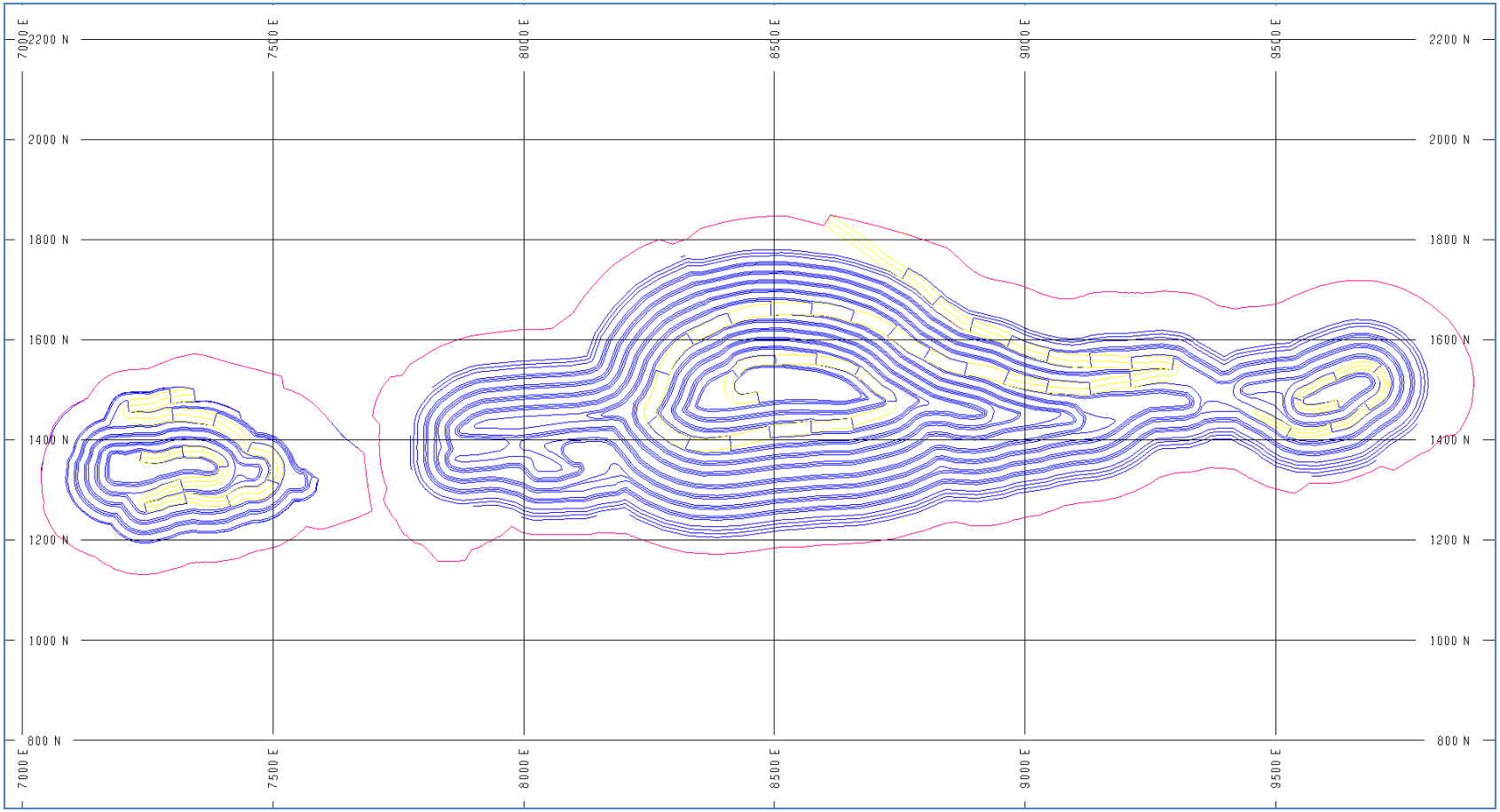


Figure 16.2: Plan View of the Detailed Open-Pit Design

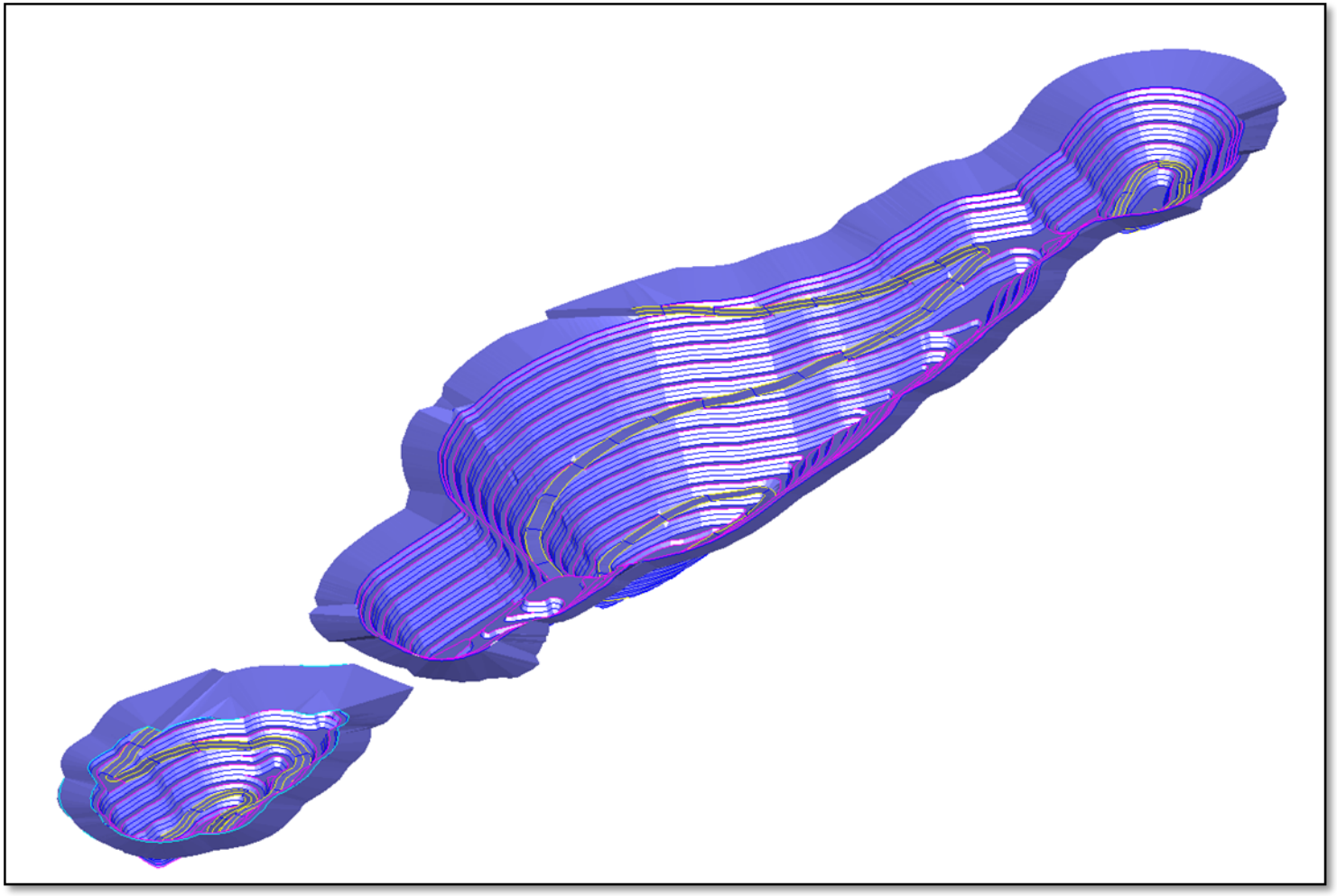


Figure 16.3: Isometric View of the Detailed Open-Pit Design

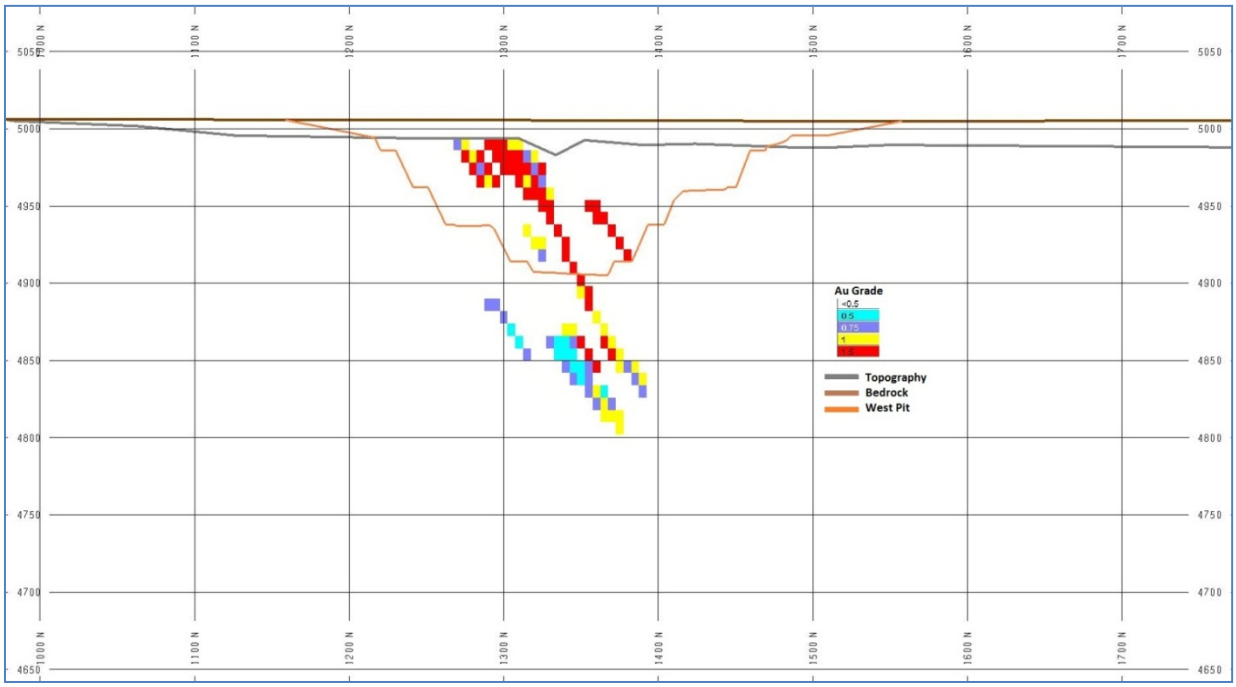


Figure 16.4: West Pit Cross Section East 7400 (Looking West)

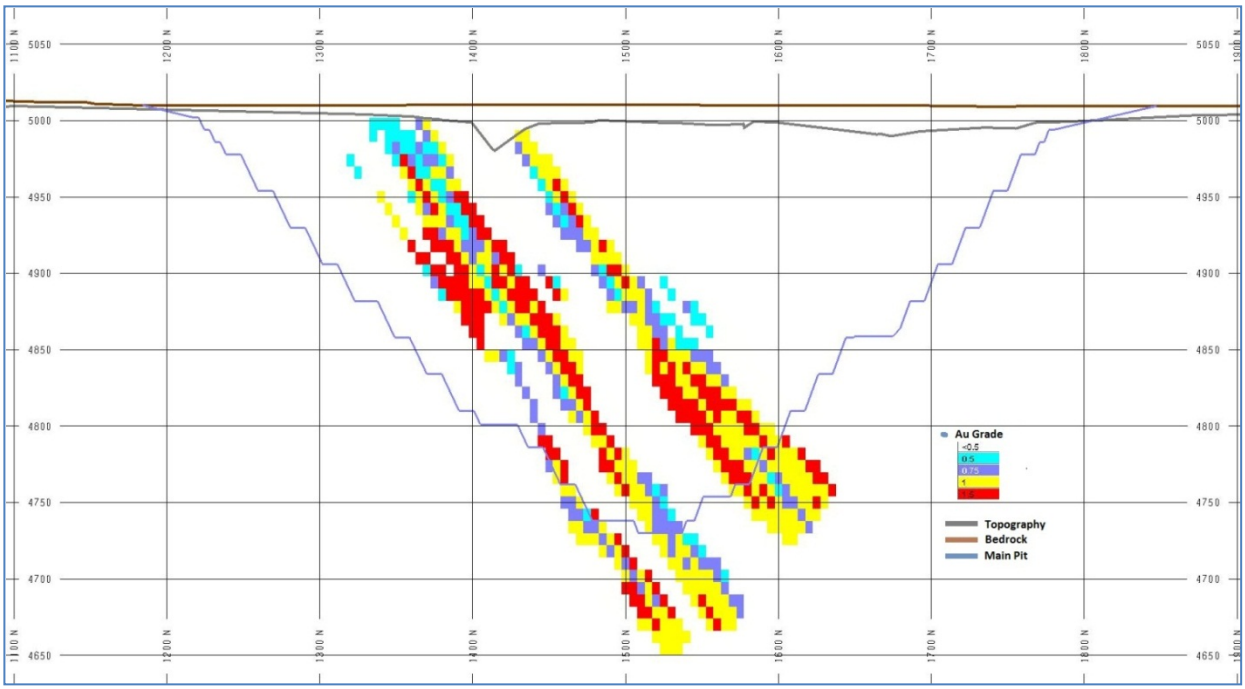


Figure 16.5: Main Pit Cross Section East 8500 (Looking West)

16.4.4 Mine Reserves

Using the resource block model, the dilution rate and the mining recovery were estimated for the mine. In the estimation, it was assumed that the selected mining method will be optimum, i.e., good blasting practice as well as a good practice of dilution control. Under this best case scenario, it was assumed that the main source of dilution and ore loss will be at the contact between the ore and waste using the following parameters:

- The minimum mining width is 5 m (1 block);
- Contact dilution of 0.65 m at the ore/waste contact;
- The "orphan" blocks are not mined.

The estimation of the contact dilution was carried out on 5 selected benches equally spaced, and the results are presented in Table 16.2.

Table 16.2: Estimation of In-pit Dilution and Mine Recovery

Pit Zone	Dilution	Grade	Ore Loss
Main Pit	6.5%	0.15 g/t Au	4%
West Pit	14%	0.07 g/t Au	5%

As described in Section 13 of this Report, the mill gold recovery is based on the following formula:

$$\%Recovery = -0.0467\ Au^2 + 0.1745\ Au + 0.7299$$

The mine reserves for the detailed pit design have been calculated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM Standards on Mineral Resources and Reserves) in August 2000, and are presented in Section 15 of this Report.

16.5 Mine Production Schedule

A total production rate of 8,500 tpd is planned for mining the Hosco deposit. The West Pit will be mined using a production rate of 1,500 tpd, and the Main Pit will be mined using a production rate of 8,500 tpd or 7,000 tpd, depending of the West Pit production.

In order to maximize cash flow and to minimize the stripping ratio in the initial years, the Hosco deposit will be mined in phases using a push-back technique. The push-backs are designed and based on a series of nested LG 3D pit shell simulations. The Hosco final pit was divided into four (4) mining phases presented in 3D view in Figure 16.6. Phase 1 (starter pit), Phase 2, Phase 3 (final Main Pit) and West Pit are indicated. All phases are designed in accordance with the design criteria, as previously presented. Pit phases served as a guide to develop the life-of-mine schedule.

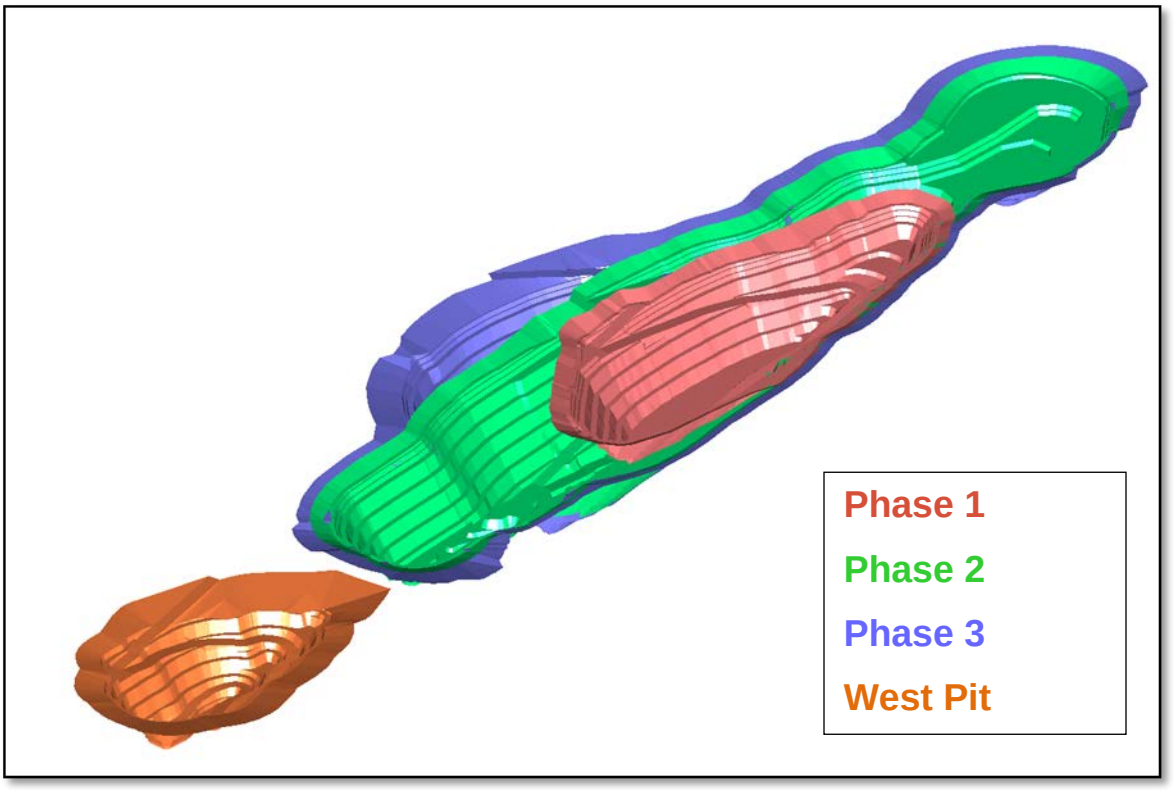


Figure 16.6: Pit Phases

A pre-production period of at least six (6) months is included in the mining schedule to provide access to ore and to provide material for the site preparation. As the ore body is near the surface, little pre-production stripping will be required in preparation for mining.

In order to increase the average grade in the early years, a cut-off grade strategy approach is used as follows:

- In Year 1: ore between 0.5 and 0.8 g/t Au will be stockpiled;
- In Year 2: ore between 0.5 and 0.7 g/t Au will be stockpiled.

Mining of ore will start in Phase 1 (starter pit) during Year 1 through the beginning of Year 5. West Pit stripping will begin in Year 4 and production will occur from Year 5 to Year 9. Overburden removal and pre-stripping in Phases 2 and 3 will begin during Year 1 and Year 3 respectively. Mining of ore in Phase 2 will start during Year 3 through the end of Year 11. Mining of ore in Phase 3 will occur from Year 8 until Year 14.

The total material stockpiled during Year 1 and Year 2 is in the order of 975 kt at an average grade of 0.77 g/t Au. Of this, 404k tonnes will be reclaimed in the first year of mill production, and the remaining 571kt is assumed to be reclaimed at the end of the open-pit life, but can be reclaimed on an as-need basis.

A total amount of 2.3 Mt of low grade ore (between 0.3 and 0.5 g/t Au) at an average grade of 0.37 g/t Au will be stockpiled and can be reclaimed at the end of the mine life if market conditions make it economically profitable at that time.

Based on the mine phases described above, a total mine life of 13.4 years at a nominal production rate of 8,500 tpd is foreseen for mining the Joanna Hosco pits. A production schedule, including dilution and mine recovery, is presented in Table 16.3.

Table 16.3: 8,500 tpd Production Schedule (CoG at 0.5g/t Au) With Dilution and Mine Recovery

Year	Mine to Mill		Mine to Stockpile [1]		Stockpile to Mill		Total Ore Milled				Total Waste	Total OVB	Total	Strip
	(kt)	Au (g/t)	(kt)	Au (g/t)	(kt)	Au (g/t)	(kt)	Au (g/t)	%Rec.[4]	Recov. kOz	(kt)	(kt)	Moved (kt) [2]	Ratio [3]
Y0			404	1.00							3 457	4 962	8 823	
Y1-1	622	1.23	127	0.62	404	1.00	1 026	1.14	86.8%	33	3 291	2 192	6 231	5.35
Y1-2	1 319	1.27	230	0.64			1 319	1.27	87.6%	47	4 622	2 207	8 379	5.18
TOTAL Y1	1 941	1.26	357	0.63	404	1.00	2 345	1.21	87.3%	80	7 913	4 399	14 610	5.25
Y2-1	1 567	1.26	111	0.58			1 567	1.26	87.6%	55	4 736	2 200	8 613	4.43
Y2-2	1 531	1.32	104	0.58			1 531	1.32	87.9%	57	4 271	2 186	8 092	4.22
TOTAL Y2	3 098	1.29	214	0.58			3 098	1.29	87.7%	113	9 007	4 386	16 705	4.32
Y3	3 099	1.38					3 099	1.38	88.2%	122	11 260	4 592	18 951	5.12
Y4	3 090	1.49					3 090	1.49	88.6%	131	12 689	3 973	19 752	5.39
Y5	3 099	1.45					3 099	1.45	88.5%	127	16 465		19 564	5.31
Y6	3 044	1.17					3 044	1.17	87.0%	100	18 024		21 067	5.92
Y7	3 048	1.09					3 048	1.09	86.5%	93	18 155		21 204	5.96
Y8	3 086	1.18					3 086	1.18	87.1%	102	15 566		18 652	5.04
Y9	3 089	1.23					3 089	1.23	87.4%	107	14 621		17 711	4.73
Y10	3 097	1.31					3 097	1.31	87.8%	115	12 460		15 557	4.02
Y11	3 070	1.21					3 070	1.21	87.3%	104	11 067		14 137	3.60
Y12	3 060	1.20					3 060	1.20	87.2%	103	6 994		10 054	2.29
Y13	3 040	1.26					3 040	1.26	87.6%	108	4 211		7 251	1.39
Y14	1 251	1.14			571	0.61	1 823	0.97	85.6%	49	430		1 681	0.24
TOTAL	**40 112**	**1.27**	**975**	**0.77**	**975**	**0.77**	**41 087**	**1.26**	**87.5%**	**1 454**	**162 320**	**22 311**	**225 719**	**4.49**

Notes:

(1) Stockpile: PP: Au > 0.5g/t, Year 1: 0.50g/t - 0.80g/t, Year 2: 0.50g/t - 0.70g/t.
(2) Total material moved excludes stockpile reclaim
(3) Strip Ratio: excluding Stockpile
(4) %Recovery = -0.0467 Au2 + 0.1745 Au + 0.7299

16.6 Pile Design

The location of the different piles is shown on the site plan in Section 18 of this Report.

16.6.1 Waste Rock Pile

Waste rock material represents the material containing 0.3 g/t Au or less. According to SGS Geostat's block modeling analysis of the As content, a total of 10% of the waste material could leach arsenic and requires environmental protection measures. The two (2) portions of waste material will be managed separately to reduce initial and restoration costs. The larger portion of the waste material (90%) will be stockpiled on the main waste dump and does not require any special environmental considerations. The remaining waste material (10%), which is potentially problematic, will be stockpiled on the secondary waste dump and requires an appropriate design, as developed by Roche and described in Section 20 of this Report. The material from grade 0.3 g/t Au to 0.5 g/t Au will also be stockpiled on the secondary dump. The total amount of low grade ore is 1.5 million m^3 (2.3 Mt). If the price of gold, at that time, makes it profitable, the low grade will be reclaimed at the end of the mine life.

The waste dumps have been designed for the LOM waste generated, and are located around the periphery of the mine to optimize the haulage distance and to minimize costs. The main waste dump and secondary waste dump, located north of the pit on the north side of the Stitchman Creek, will have a total capacity of 69.1 million m^3 (146.1 million tonnes) and 8.8 million m^3 (16.2 million tonnes of waste and 2.3Mt of low grade), respectively.

The capacities of the dumps have been estimated using a swell factor of 30%. The design parameters are as follow:

- Face angle: 33°;
- Bench height: 10 m;
- Berm width: 10 m;
- Number of bench: 7 (main waste dump), 5 (secondary waste dump);
- Minimum distance from Stitchman Creek = 100 m.

16.6.2 Overburden and Topsoil Pile

Digging the pit will require the removal of a quantity of overburden estimated at about 22.3 Mt, mainly made up of clay. Because of its water content, the overburden cannot be used directly as construction backfill material for the various facilities. It will therefore be stored on two (2) piles located near the pit and will be used later for the restoration.

Overburden material will be removed from Year 0 to Year 4. The overburden clay dump, located north-east of the Main Pit, will have a capacity of 11.3 million m^3 (18.9 million tonnes and the topsoil dump located north of the West Pit will have a capacity of 2.1 million m^3 (3.4 million tonnes) using a swell factor of 20%. The overburden dump design parameters are as follows:

- Face angle: 22°;
- Height: Overburden (clay) = 7.5 m, Topsoil = 5 m;
- Minimum distance from Stitchman Creek = 100 m.

16.6.3 Ore Stockpile

In addition to the waste dumps and overburden dumps, a 1Mt ore stockpile is required on the crusher pad to store the ore material discarded during Year 1 and Year 2.

16.7 Mining Operations

The ore of the Joanna Hosco deposit will be mined using conventional open-pit mining methods based on a truck/shovel operation. All the equipment will be diesel-powered.

The production plan, presented in Table 16.3, was used as the basis in determining the mobile equipment fleet requirements. All the equipment is assumed to be owned by Aurizon Mines Ltd. and operated by Aurizon Mines Ltd. personnel. The mine will operate on 2 x 12-hour shifts per day, 7 days per week and 365 days per year with crews rotating on a 1-week in, 1-week out schedule. The selection of the primary mining fleet is based on cycle time estimations, mechanical availability and utility factors, as well as average yearly haulage profiles.

It is assumed that the overburden removal will be conducted by a contractor.

16.7.1 Drilling

Blast holes will be carried out by diesel-powered down-the-hole drills with the following parameters:

- Diameter blast hole: 6.25 inches;
- Drilling pattern: 5.2 m by 5.2 m pattern in ore and on a 5.8 m by 5.8 m pattern in waste;
- Sub drill: 1 m;
- Re-drill: 10%;
- Penetration rate: 25 m/hr.

The net productive time for drilling is based on a 12-hour shift with a total of 75 minutes of scheduled delays. The operator efficiency for the drilling operation has been established at 45 minutes per hour (75%) to account for the time required for the drill to move between holes. Thus the remaining available productive time is estimated at 484 minutes (8.1 hours) per 12-hour shift. The operating parameters for drilling are summarized in the following Table 16.4:

Table 16.4: Net Productive Time: Drilling

Category	Time/Shift (minutes)
Scheduled time per shift (12 hrs)	720
Scheduled non-productive time	
Startup	15
Shut down	15
Coffee break	15
Lunch	30
Net scheduled time	645
Job efficiency (75% or 45 min-hr)	161
Operating minutes per shift	484
Operating Hours per Shift	**8.1**

16.7.2 Blasting

Blasting will be executed under contract with an explosives company that will supply the blasting products and technology, portable storage facility with silo and equipment to deliver explosives to the hole. The explosives supplier will also be responsible for providing a down-the-hole service. Blasting will be conducted using an emulsion-type explosive with an average density (in the hole) of 1.25 kg/m^3. Based on a drilling pattern of 5.2 m x 5.2 m in ore and 5.8 m x 5.8 m in waste, the design powder factors are 0.24 kg and 0.2 kg of explosives per tonne of rock for ore and waste material respectively. After mining commences, fragmentations will be evaluated and the drill and blast parameters may be further refined to optimize results.

16.7.3 Loading and Hauling

For the Main Pit area, loading and hauling will be conducted using a fleet of 100-ton capacity trucks (for both ore and waste) in combination with an 11 m^3 capacity hydraulic front shovel in ore and waste. This combination will allow 4-pass loading of trucks. The truck loading time has been estimated at 2 minutes for ore and waste. The maximum shovel productivity has been estimated at 18,926 tonnes of ore or waste per shift.

Due to the lower production rate, the West Pit will be excavated using a 50-ton class truck in combination with a 5.7 m^3 capacity hydraulic shovel. This combination will allow 3-pass loading of trucks. The truck loading time has been estimated at 1.5 minutes for ore and waste. The maximum shovel productivity has been estimated at 8,399 tonnes of ore or waste per shift.

A 10 m^3 capacity wheel loader will also be used to complement the hydraulic shovels in an effort to maximize the flexibility and blending capacity of the operation. The wheel loader will be used in Year 14 to load haul trucks with ore reclaimed from the ore stockpile.

Net Productive Time

Productivity parameters have been established based on a net productive time of 9.3 hours per 12-hour shift. This time incorporates a total scheduled non-productive time of 75 minutes per shift to account for shift change, routine inspection and lubrication, coffee and lunch breaks, as summarized in Table 16.5.

Table 16.5: Net Productive Time: Loading and Hauling

Category	Time/Shift (Minutes)
Scheduled time per shift (12 hrs)	720
Scheduled delays	
Shift change	15
Inspection	15
Coffee break	15
Lunch	30
Net scheduled time	645
Job efficiency (87% or 52 min-hr)	86
Operating minutes per shift	559
Operating Hours per Shift	**9.3**

Loading Parameters

The in-situ material densities average 2.75 tonnes/m^3 of ore and waste over the mine life. Blasting will result in a swell factor of approximately 30%, which will yield a loose density of 2.12 tonnes/m^3 in both ore and waste. It is anticipated that the use of high precision detonators for blasting initiation will yield good fragmentation. Based on these parameters and the annual mine plan, the yearly shovel/loader requirements were calculated. The loading parameters are summarized in Table 16.6 and 16.7.

Table 16.6: Loading Parameters (Main Pit)

Parameter	Ore and Waste
Bucket size (m^3)	11
Fill factor (%)	97%
In-situ bulk density (t/m^3)	2.75
Swell factor (%)	30%
Loose density (t/m^3)	2.12
Tonnes per bucket (t)	22.57
Truck capacity (t)	91
Time/pass (min)	0.50
Pass/truck (pass)	4.03
Rounded (pass)	4.00
Loading time (min)	2.00
Truck spot time (min)	0.67
Total time/truck (min)	2.67
Truck loads/shift	210
Tonnes per trip	90.3
Shift Production Capacity (t/shift)	**18 926**

Table 16.7: Loading Parameters (West Pit)

Parameter	Ore and Waste
Bucket size (m^3)	5.7
Fill factor (%)	90%
In-situ bulk density (t/m^3)	2.75
Swell factor (%)	30%
Loose density (t/m^3)	2.12
Tonnes per bucket (t)	10.85
Truck capacity (t)	32
Time/pass (min)	0.50
Pass/truck (pass)	2.95
Rounded (pass)	3.00
Loading time (min)	1.50
Truck spot time (min)	0.67
Total time/truck (min)	2.17
Truck loads/shift	258
Tonnes per trip	32.6
Shift Production Capacity (t/shift)	**8 399**

Hauling Parameters

Average annual haul profiles were created based on the annual mine plan. Haul routes were traced in MineSight software according to the mining centroid for each year, and for each respective haul route (i.e., ore, waste, low grade and overburden). Haul truck travel speeds were based on an internal database and were compared to manufacturer rimpull charts used to determine the average annual cycle times, including an allowance factor of approximately 10% to account for additional cycle delays, e.g., queuing and waiting. These cycle times were then used to determine the annual haulage fleet size. A summary of the annual average cycle time for ore, waste, low grade material, and overburden can be found in Table 16.8 and 16.9.

Table 16.8: Average Cycle Time (Main Pit)

Year	Cycle Time (min.)*		
	Crusher and Stockpile	Main Waste Dump	Secondary Waste Dump
Pre-prod	6.73	7.57	10.20
Year 1	7.36	9.25	11.88
Year 2	8.11	11.22	14.44
Year 3	9.90	8.03	10.87
Year 4	14.02	10.87	13.60
Year 5	10.05	10.49	13.43
Year 6	11.15	12.13	15.66
Year 7	11.79	12.76	15.88
Year 8	12.97	11.21	14.91
Year 9	13.96	12.50	15.68
Year 10	15.85	16.05	18.82
Year 11	16.42	18.82	22.18
Year 12	19.38	22.45	25.13
Year 13	21.50	24.57	27.04
Year 14	24.86	27.93	30.40

* excluding loading and dumping times

Table 16.9: Average Cycle Time (West Pit)

Year	Cycle time (min.)*		
	Crusher and Stockpile	Main Waste Dump	Secondary Waste Dump
Year 4	6.42	7.31	10.41
Year 5	7.55	8.44	11.54
Year 6	8.99	9.87	12.98
Year 7	10.77	12.19	15.08
Year 8	12.80	14.23	17.12
Year 9	14.26	16.22	18.57

* excluding loading and dumping times

Equipment Availability

Equipment availability was estimated based on the following assumptions:

- Constant utilization availability of 95% for trucks and shovels over the mine life;
- Availability profiles for shovels and trucks based on general manufacturer's benchmarks and an age-based maintenance plan. Newer units can expect highest availabilities due to the age of the fleet and potential manufacturer's guaranteed rates. A minimum availability of 85% was assumed as the fleet ages.

Mine Support Equipment

The following service equipment was selected to carry out the routine road maintenance and other miscellaneous work within and around the mining areas:

- Track-dozers: two (2) track-dozer 410 hp are used to maintain waste dumps as well as to perform general work within the mine;
- Grader: One (1) motor grader 265 hp is used for road maintenance;
- Water truck: One (1) water truck of 20,000 liter space capacity with interchangeable boxes depending on the season: a box to hold crushed stone for icy road conditions in winter, and one box to carry water for dust abatement in the summer;

- Wheel-dozer: One (1) wheel-dozer 354 hp is used for road maintenance, blast hole stemming and snow removal;
- Small excavator: One (1) backhoe excavator 188 hp is used to excavate and maintain ditches around the pits;
- Other service equipment: One (1) fuel truck, one (1) service truck, one (1) tire handler and a few pick-up trucks. Due to their accessibility in the vicinity of the mine, other service equipment will be rented.

16.7.4 Mine Equipment Annual Fleet Requirements

Table 16.10 provides a list of the mine equipment requirements during the life of the mine.

Table 16.10: Mine Equipment List for 8,500 tpd

Type	Pre-prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14
PRIMARY															
Hydraulic Front Shovel (10 m^3)	1	1	2	2	2	2	2	2	2	2	2	2	1	1	1
Front End Loader (800 hp)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Haul Truck (100-tons class)	3	4	6	6	8	8	9	10	10	10	10	10	9	7	5
SECONDARY															
Drill DTH (6”1/4)	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Track Dozer (410 hp)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Motor Grader (265 hp)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
AUXILIARY															
Wheel Dozer (354 hp)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe Excavator (188 hp)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic Crane - 75 t (rented)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Fuel/Lube Truck (20 000 litres)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Prime Mover for Low Bed (rented)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Service Truck 22,000 GWV, 250 hp	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire Changer Truck Mounted	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Loader/ToolCarrier	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Boom Truck (rented)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4 x 4 Crew cab, Pick up 3/4 t	4	5	5	5	5	5	5	5	5	5	5	5	5	5	5
4 x 4 Single cab Pick-up 3/4 t	4	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Lighting Tower 4 Post of 1000 W	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
WEST PIT															
Hydraulic Excavator (5.7 m^3)	0	0	0	0	0	1	1	1	1	1	0	0	0	0	0
Haul Truck (50-ton)	0	0	0	0	0	4	4	4	4	4	0	0	0	0	0
Drill DTH (6”1/4)	0	0	0	0	0	1	1	1	1	1	0	0	0	0	0
Total Fleet	**25**	**30**	**33**	**33**	**35**	**41**	**42**	**43**	**43**	**43**	**37**	**37**	**35**	**32**	**30**

16.7.5 Mine Labour

Labour requirements were determined according to equipment requirements. For more details, see Section 21.

16.7.6 Mining Services

Dewatering

It is recommended to construct a drainage ditch around the perimeter of the pit to collect run-off water. In addition, a suitable pumping system will be included in the design for the dewatering of the pit.

Aggregate Requirement

Considering the size of the mine and the availability of aggregate in the vicinity of the mine, there will be no aggregate plant at the Joanna mine site.

Contract Mining

The Feasibility Study assumes that mining would be carried out using the owner's equipment and personnel. However, contract mining submissions should be requested from mining contractors to provide a comparison for the owner's mining scenario.

The Feasibility Study assumes that the overburden removal will be executed by a local contractor using the contractor's equipment and personnel.

17 RECOVERY METHODS

Section 13 of this Report describes the metallurgical testwork performed and how the results were used to determine and develop the optimal method of recovering gold from the Joanna ore. In this section of the Report, a process design basis is established and the flowsheets, the mass and water balance and the process design criteria considered for the Feasibility Study (FS) are presented. The selection and sizing of the process equipment was performed based on this information and is presented in this section as well. Equipment sizing was based on a combination of the following types of information:

- Testwork results;
- Handbook references;
- BBA's experience on other projects (reference projects);
- Vendor information.

A description of the processing plant is also presented.

17.1 Process Design Basis

Design is based on an average daily ore throughput of 8,500 t/d or 3.1 Mt/y. Based on this target and the results of the metallurgical testwork performed, a mass balance was developed, which forms the basis of design of the mineral processing plant. The major design criteria for the plant are presented in Table 17.1 and a summary of the major flows entering and exiting the plant is presented in Table 17.2.

Table 17.1: Major Design Parameters

Item	Value
Nominal Daily Plant Feed (Annualized t/d)	8,500
Design Daily Plant Feed (Annualized t/d)	9,500
Annual Plant Utilization	85%
Annual Operating Hours	7,446
Nominal Gold Head Grade (g/t)	1.26
Gold Recovery	87.5%
Annual Gold Production (troy oz/y)	109,237

Table 17.2: Major Process Solid Streams

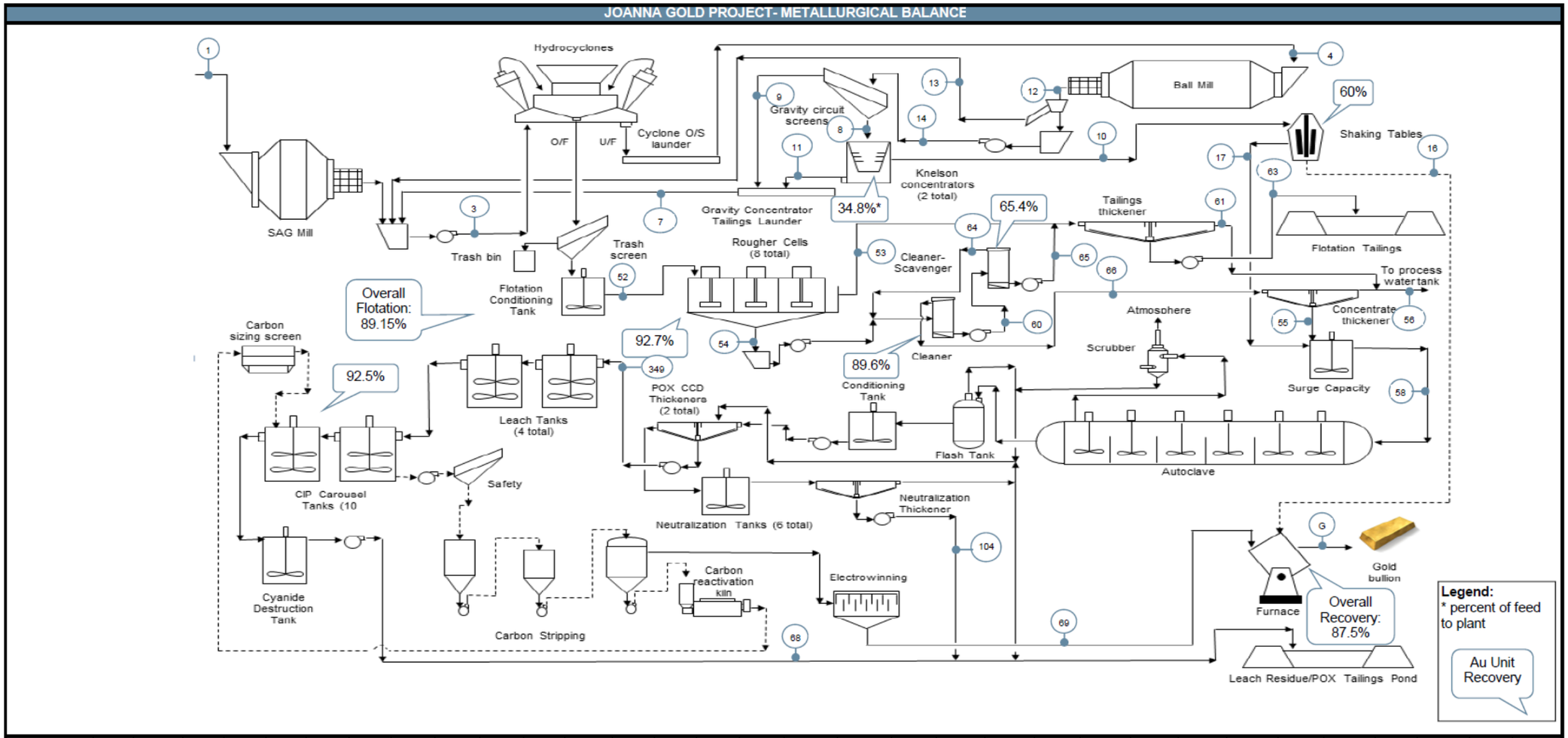
Stream	Annual (dry Mt/y)	Nominal (dry t/op. h)	Design (dry t/op. h)
Ore Processed	3.10	416.7	465.7
Flotation Concentrate	0.155	20.8	23.3
Flotation Tailings	2.95	395.7	443.2
Pressure Oxidation(POX) Tailings	0.110	14.8	16.5
Cyanidation Tailings	0.153	20.5	22.9

The plant utilization rate of 85% was based on the availability of the autoclave, which was specified by Sherritt who was responsible for the testwork, process flowsheet and plant design for the Pressure Oxidation (POX) area. The average gold head grade was derived from the block model and optimized pit design of the Joanna Hosco deposit.

17.2 General Flowsheet, Mass and Water Balances

The results of the metallurgical testwork as well as the aforementioned process design basis were used to develop a flowsheet and a mass and water balance, which are presented in Figure 17.1 and Figure 17.2. The water balance was first developed to establish process requirements and subsequently incorporated climatic factors such as rainfall and evaporation, which were

provided by Roche. The balances presented are based on a per-operating-hour basis (at 85% plant availability) and are for nominal conditions. These balances serve as a basis for design criteria development and equipment selection and sizing.



Description	Plant Feed	Cyclone Feed	BM Feed	Knelson Tails	Knelson Feed	Gravity Screen O/S	Knelson Conc	Knelson Tails	BM Discharge	Knelson Bypass	Gravity Screen Feed	Shaker Conc	Shaker Tails	Rougher Feed	Rougher Tails	Rougher Conc	Concentrate Thickener U/F	Concentrate Thickener O/F	POX Feed	Cleaner Tails	Tailings Thickener O/F	Tailings Thickener U/F	Scavenger Conc	Scavenger Tails	Cleaner Conc	CN Tails	Electro. Product	Neutraliz. Tails	Leach Feed	GOLD		
Stream	1	3	4	7	8	9	10	11	12	13	14	16	17	52	53	54	55	56	58	60	61	63	64	65	66	68	69	104	349	G		
Pulp (t/op. h)	425.0	2581.8	1320.6	1215.5	1052.5	44.0	0.9	1171.6	1320.6	353.9	966.7	0.0	0.2	1261.3	1104.5	163.0	34.7	71.7	35.5	100.5	578.5	608.8	34.4	70.1	104.1	41.5	0.0	29.7	40.9			
Solids (t/op. h)	416.7	1409.7	993.1	726.9	705.2	21.8	0.1	705.0	993.1	266.2	727.0	0.0	0.1	416.6	375.8	40.8	20.8	0.0	20.9	27.1	0.0	395.7	7.2	19.9	20.8	20.5	0.0	14.8	20.5			
Liquid (t/op. h)	8.3	1172.2	327.5	488.7	347.3	22.2	0.8	466.5	327.5	87.8	239.7	0.0	0.1	844.7	728.7	122.3	13.9	71.7	14.5	73.4	578.5	213.1	27.1	50.2	83.3	21.0	0.0	15.0	20.4			
% Solids	98.0	54.6	75.2	59.8	67.0	49.6	11.8	60.2	75.2	75.2	75.2	45.0	60.0	33.0	34.0	25.0	60.0	0.0	59.0	27.0	0.0	65.0	21.0	28.4	20.0	49.3	100.0	49.6	50.0			
Gold (oz/op. h)	16.9	37.3	26.3	13.3	18.6	0.6	5.9	12.8	26.3	7.0	19.2	3.5	2.4	11.0	0.8	10.2	9.8	0.0	12.2	1.1	0.0	1.2	0.7	0.4	9.8	0.9	11.3	0.0	11.9	14.8		
Gold (g/t)	1.26	0.82	0.82	0.82	0.82	0.82	1726	0.56	0.82	0.82	0.82	1.E+05	697	0.82	0.07	7.79	14.66		18.08	1.31		0.09	3.21	0.62	14.66	1.39		0.00	18.08			

Figure 17.1: Process Flowsheet Showing Mass and Process Water Balance

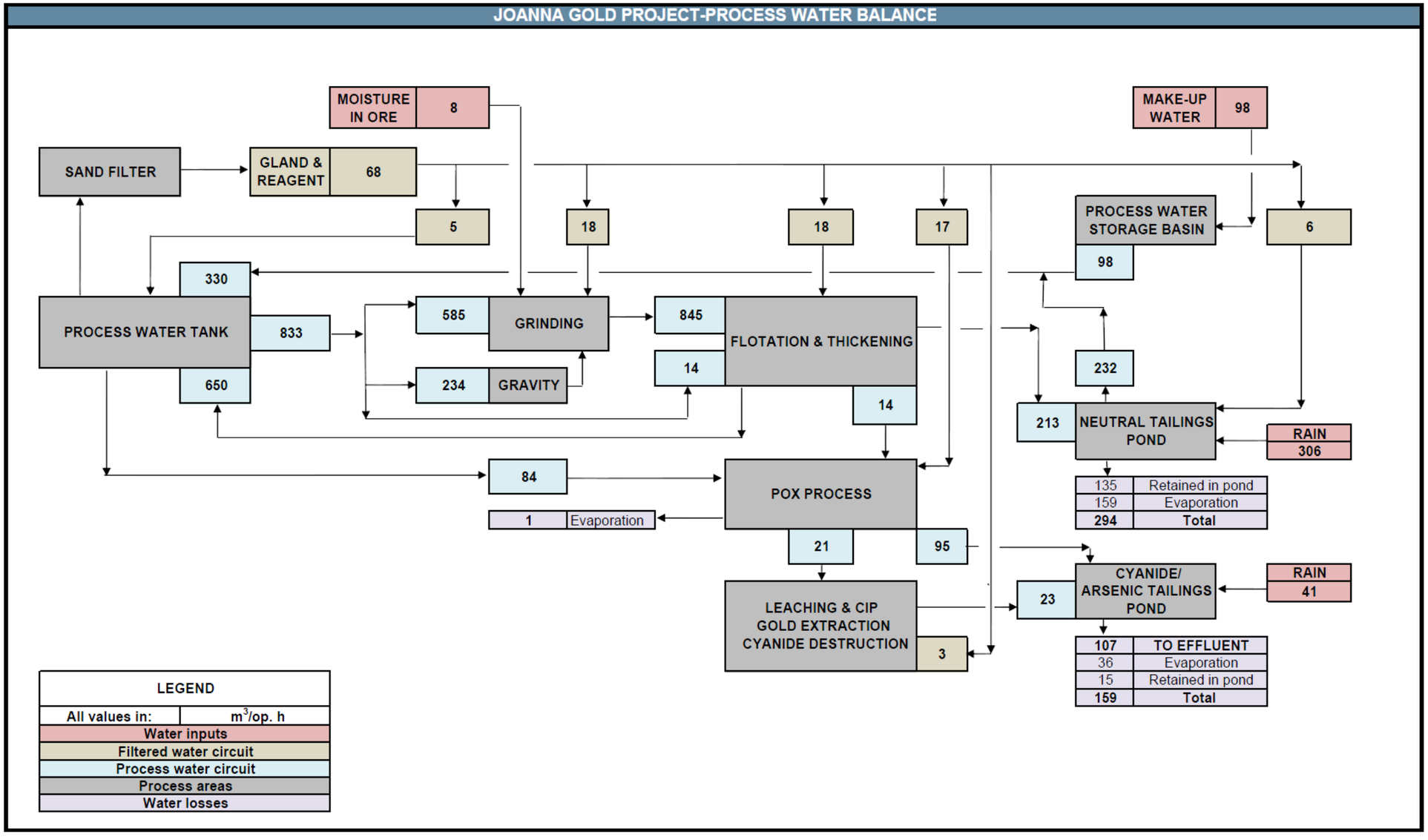


Figure 17.2: Global Water Balance

17.3 Process Description

17.3.1 Summary

The following is a summary of major processing steps and design criteria:

- The general location of the crusher, crushed ore storage silo, concentrator, tailings disposal areas, waste disposal areas and other infrastructure are presented in the site plan, presented in Section 18.
- Operations run 365 days per year with a total of 14 days for scheduled maintenance shutdown required for the autoclave. The overall plant utilization rate is in the order of 85% (310 days). The crusher utilization rate is in the order of 65%.
- Ore is crushed using a single jaw crusher. Crushed ore is stored in a concrete silo from which it is reclaimed with vibrating feeders onto a belt conveyor, which then feeds the SAG mill.
- The grinding circuit comprises a single pinion SAG mill with a variable speed drive operated in an open circuit, which feeds a ball mill with a fixed speed drive. The ball mill is operated in a closed circuit with a hydrocyclone cluster. The product of the ball mill circuit is then fed to a flotation circuit to be concentrated.
- A portion of the ball mill discharge is fed to a pair of gravity concentrating units operated in parallel, which recover gravity recoverable gold. The gravity circuit tailings are returned to the ball mill feed. The concentrate from the gravity circuit is upgraded using a shaking table (as a batch operation). The shaking table concentrate is directed to the melting furnace and the tailings are combined with the flotation concentrate, which in turn are sent to the Pressure Oxidation (POX) area.
- The ground ore from the grinding circuit is directed to the flotation area. The flotation circuit is made up of a roughing stage comprised of eight (8) mechanical tank cells in series, a cleaning column and a cleaner-scavenger column.
  - The flotation tails are thickened using a high rate thickener and pumped to the flotation tailings pond for final disposition. These tailings are classified as neutral and non-acid generating so no special design features are required for this tailings impoundment area.

- The flotation concentrate is thickened using a high rate thickener and is directed to the POX area.
- The POX area consists of the following steps:
- The first step in the POX process consists of pressure oxidation in the autoclave. In this stage, the concentrate slurry is exposed to a high pressure oxygen atmosphere, which results in a high degree of oxidation of the sulphide-bearing minerals in the concentrate.
- Oxygen is produced on-site with a vacuum pressure swing adsorption (VPSA) plant, which will be owned and operated by an industrial gas Vendor.
- The oxidized solids are washed in a series of two high rate thickeners in the counter current decantation (CCD) stage to separate it from the acidic solution generated in the POX stage. The washed concentrate constitutes feed to the cyanide leaching area.
- The aforementioned acidic solution is directed to the neutralization stage, which is made up of a series of stirred vessels that serve to neutralize the solution with additions of limestone and lime. The neutralization reaction causes the precipitation of dissolved metals to occur. The precipitated solids (neutralization tails) are thickened in a high rate thickener and are combined with the cyanide leaching tailings (after cyanide destruction) for final disposition.
- The oxidized concentrate is leached in a cyanide solution in a series of four (4) agitated and aerated tanks in order to dissolve the contained gold.
- From the leach tanks, the slurry is then fed to a series of ten (10) carousel-type carbon-in-pulp (CIP) tanks. At this stage, the slurry is brought into contact with activated carbon, which adsorbs the gold in solution.
- The gold-loaded activated carbon is sent to the stripping stage where the gold is desorbed from the activated carbon into a concentrated solution. The gold is then electrodeposited in an electro-winning cell. Lastly, the gold is melted in an induction furnace and cast into bars that constitute the final product from the Joanna gold extraction process.
- The leached slurry is sent from the CIP stage to a cyanide destruction stage comprised of an agitated vessel to which SO_2 gas, $CuSO_4$ and air are added. Following the cyanide destruction stage, this tailings slurry, combined with the neutralization slurry, is pumped to the POX tailings pond. These tailings contain a high level of As. The water also contains some residual cyanide, which must be destroyed before being released to the watershed. As a result, the POX tailings impoundment area, which contains the tailings and the water, requires a special design in order to prevent contact with other surroundings.

17.3.2 Primary Crushing and Ore Handling

The run-of-mine ore, which has a F_{80} of 600 mm, is transported to the crusher area using 100-ton capacity haul trucks. The trucks dump onto a static grizzly discharging into a 270-tonne capacity hopper. The grizzly retains material larger than 850 mm, and a hydraulic rock breaker is in place to break the oversize material. When the crusher is not available, the mining trucks dump into the ROM stockpile located nearby, as indicated on the site plan. This ore is subsequently reclaimed as required using a front-end loader and mining truck.

An apron feeder feeds material from the hopper onto a belt conveyor, which discharges into a chute and into the primary crusher, thus assuring a controlled and optimal feed rate to the crusher. The primary crusher is a jaw-type crusher with a feed opening measuring 1,250 mm x 950 mm and an installed power of 200 kW. Crusher product has a P_{80} of 110 mm. The crusher was sized assuming a utilization rate of 65%.

The crusher product is discharged onto a discharge belt conveyor, which is equipped with a magnet to remove metal scraps. During normal operation, this conveyor discharges into a chute, which in turn discharges onto the main conveyor, which feeds the crushed ore silo. A belt scale measures the feed rate to the silo. During periods when the main conveyor belt is not available or when the silo is full, crusher operations can be maintained by diverting crushed ore to an emergency stockpile via a transfer conveyor. Material going in and out of this stockpile is handled by loader. A hopper is located on the main conveyor to allow for reclaiming ore from the stockpile.

The crusher area is equipped with an overhead crane with a 20-tonne capacity and an air compressor for the operation of instruments. Dust is controlled in the primary crushing area using a baghouse-type dust collector.

The crushed ore silo has a live capacity of 2,360 tonnes which corresponds to approximately five and a half (5.5) hours of concentrator operation. It acts as a buffer between the crushing plant and the concentrator and assures a constant ore flow to the grinding mill.

Four (4) vibrating pan feeders reclaim material from the silo and feed it onto the mill feed conveyor. A belt scale monitors ore flow to the mill.

17.3.3 Grinding and Gravity Separation

The mill feed conveyor feeds material to the primary grinding mill, which consists of a single pinion Semi-Autogenous mill (SAG mill). A trommel screen at the mill discharge keeps oversized material in the mill so the mill is operated in open circuit. The SAG mill has a diameter of 9,144 mm and an effective grinding length (EGL) of 2,896 mm (30' D x 9.5' EGL) and has an installed power of 5,040 kW. The SAG mill receives feed material with an F_{80} of 110 mm and produces material with a P_{80} of 1,753 µm.

The SAG mill product is discharged into a pump box from which the slurry is pumped to the secondary grinding circuit. The circuit is comprised of a cluster of 500 mm hydrocyclones (3 operating and 2 installed spares), two parallel Knelson gravity concentrators and a ball mill having a diameter of 5,486 mm, an effective grinding length of 8,534 mm (18' D x 28' EGL) and an installed power of 5,040 kW. The ball mill is equipped with a trunnion magnet to remove metal scraps from the ball mill discharge.

The grinding area is equipped with an overhead crane with a capacity of 45 tonnes and an auxiliary hoist having a capacity of 25 tonnes.

The ball mill discharge product, which has a P_{80} of 473 µm, is combined with the SAG mill product in a pump box which feeds the hydrocyclone cluster. Approximately 73% of the ball mill product is directed by gravity to the Knelson concentrators, which are preceded by two screens. The portion not fed to the gravity concentrators is returned to the hydrocyclone feed pump box.

The Knelson gravity concentrators produce a concentrate and a tail. The tail is returned to the hydrocyclone feed pump box. The concentrate is flushed from the Knelson units approximately every 60 minutes during a period of about 2.5 minutes. The concentrate is collected in a stock tank with a 96-hour capacity. The stock tank has a conical shaped bottom and is designed to reduce the water content of the concentrate. The Knelson concentrate is further concentrated in a batch operation shaking table. The shaking table concentrate is directed to the gold room

induction furnace for melting. The shaking table tail is collected in a tank and is subsequently combined with the flotation concentrate in the holding tank ahead of the POX area.

The grinding circuit produces a final product through the hydrocyclone overflow, having a P_{80} of 90 µm, which constitutes the targeted nominal grind. The hydrocyclone overflow is directed by gravity to the flotation area conditioning tank. The hydrocyclone underflow is recycled to the ball mill.

The gravity area is equipped with an overhead crane with a capacity of 7.5 tonnes.

17.3.4 Flotation

The hydrocyclone overflow discharges into the flotation conditioning tank, an agitated vessel with a residence time of 2 minutes, to which is added the gold collector Aero 208.

From the conditioning tank, the slurry overflows into the bank of rougher cells, comprised of eight (8) mechanical tank cells of 100 m^3 each, which provide a total residence time of approximately 37 minutes. The gold collector, Aero 208, the frother, MIBC, and the sulphide collector, PAX, are added to the rougher cells.

The tails from the rougher stage overflow into a pump box and are pumped to the tailings thickener for dewatering prior to pumping to the tailings pond. The flocculant used in this thickener is Flomin 905MC. The thickener overflow flows by gravity to the process water tank. The concentrate from the rougher stage is collected in a launder and is combined with the cleaner-scavenger concentrate and is pumped to the cleaner column, which is 10 m high and 3.05 m in diameter and has a residence time of approximately 13 minutes. The frother, MIBC, and the sulphide collector, PAX, are added to the column.

The tails from the cleaner column are pumped to the cleaner scavenger column, which is 8.5 m high and 2.44 m in diameter and has a residence time of approximately 8.5 minutes. The frother, MIBC, and the sulphide collector, PAX, are added to the column. The concentrate from the cleaner scavenger flows by gravity to the rougher concentrate pump box and is recycled back to the first cleaner column. The tails of the cleaner scavenger overflow into a pump box

from where they are combined with the rougher tailings and pumped into the flotation tailings thickener mentioned earlier.

The final concentrate from the flotation circuit, produced by the first cleaner column, is directed to the concentrate dewatering thickener, which is 11 m in diameter. The flocculant used in this thickener is Flomin 905MC. The thickener overflow flows by gravity to the process water tank. The thickened concentrate is pumped to the flotation concentrate surge tank, ahead of the POX area. The surge tank has a residence time of 24 hours and will normally operate half full in order to provide 12 hours of buffer capacity. In the event of an emergency shutdown in the POX area, grinding and flotation operations could be maintained for up to 12 hours.

Blower air is forced through the flotation cells and columns. It is estimated that the overall gold recovery of the flotation circuit is 89.15%.

The flotation area is equipped with an overhead crane with a capacity of 15 tonnes.

17.3.5 Pressure Oxidation (POX)

17.3.5.1 Autoclave

The flotation concentrate is pumped from the surge tank to the pressure oxidation circuit which consists of a high pressure slurry feed system, autoclave vessel and agitators, conditioning tank, and vent gas handling system. The autoclave system also includes an agitator seal system, steam boiler (for start-up only), and high pressure cooling water (quench water) pumps for autoclave temperature control.

The autoclave is a refractory lined, multi-compartment, horizontal vessel and operates at 220°C and 3,000 kPa (gauge), with design retention time of 75 min. Oxygen required for the oxidation reactions in the autoclave is provided from an on-site vacuum pressure swing adsorption (VPSA) oxygen plant.

The purpose of the pressure oxidation system is to oxidize sulphides and arsenopyrite in the flotation concentrate, thereby rendering the gold amenable to subsequent recovery by cyanide

leaching. The reactions that occur in the autoclave are highly exothermic, so cooling is required to maintain the desired temperature. The temperature in each compartment is controlled through direct injection of process water as quench water.

The autoclave slurry from the final compartment is discharged into a flash tank where the steam and slurry are separated. The steam flows through the vent line to a scrubber system while the slurry drains into the conditioning tank. Basic iron sulphates, which would otherwise consume lime in the cyanide leach circuit, are decomposed in the conditioning tank. The conditioning tank discharge slurry is pumped to the CCD wash circuit.

17.3.5.2 Counter Current Decantation (CCD)

The counter current decantation (CCD) wash circuit, which is located outside of the main building, consists of two high rate thickeners, with repulp tanks, overflow tanks, and associated pumps.

The oxidized residue is sent to the CCD wash to remove sulphuric acid and metal sulphates from the oxidized slurry before it is sent to the cyanide leaching circuit. The CCD wash also aids in cooling the slurry. The washed slurry from the CCD is pumped to the cyanide leaching circuit while the wash solution is pumped to the neutralization circuit.

17.3.5.3 Neutralization

The neutralization circuit is located outside of the main building and adjacent to the CCD circuit. The two stage circuit consists of six (6) agitated tanks connected in series, a neutralization thickener and an overflow solution pump.

The solution generated from washing the oxidized concentrate is pumped to the neutralization circuit where the acid in solution is neutralized and the soluble metals in the solution, such as aluminum, arsenic and iron are precipitated. The neutralizing agents used in this process are limestone and lime. The final neutralized slurry exits the circuit at 65°C and a pH of 8. This slurry is thickened to 50% solids in a high rate thickener. A portion of the overflow is recirculated

to the CCD wash circuit and the remainder is combined with the thickener underflow and pumped to the tailings pumping station.

Limestone for neutralization is delivered to the site and is ground to the required size in a package limestone grinding plant. The ground limestone slurry is pumped to the first stage neutralization tanks at a controlled rate.

Lime is delivered to the site in pneumatic discharge tanks and stored in a silo. Lime is prepared in a slaking circuit and pumped to the second stage neutralization tanks at a controlled rate.

17.3.6 Cyanide Leaching and CIP and Cyanide Destruction

17.3.6.1 Leaching Circuit

Four (4) leaching tanks and an associated conditioning tank are located outdoors, adjacent to the CCD wash thickeners. The washed POX residue from the CCD area is pumped from the underflow of the second wash thickener to the cyanide leaching conditioning tank, which has a residence time of approximately 30 minutes. Slaked lime is pumped from the aforementioned lime storage and slaking system into the conditioning tank in order to adjust the pH of the slurry to approximately 10.5 to prevent HCN gas formation in the downstream leaching tanks.

The CCD slurry is then pumped from the conditioning tank to a series of four (4) agitated leaching tanks, which provide a total residence time of approximately 48 hours. Each tank has a diameter of 8.1 m. A solution of NaCN is pumped into the first leaching tank from the NaCN storage tank. Compressed air is fed into each tank using gas diffusers to optimize the gold leaching rate. The heights of the four (4) leaching tanks are 9.0 m, 8.7 m, 8.4 m and 8.1 m, which allows the slurry to overflow from one stage to the next. Each tank is equipped with a launder arranged in such a way that any one tank can be bypassed without having to shut down the entire train of leaching tanks. The overflow of the fourth and final leaching tank flows by gravity to the CIP carousel circuit, located inside the plant building.

17.3.6.2 CIP

The gold-containing solution is directed to the CIP (carbon in pulp) circuit (Kemix AAC pumpcell system) where the gold in solution is allowed to adsorb onto the carbon. The circuit includes ten (10) tanks (10 x 20 m^3) in series, one of which is on standby. Each tank contains one (1) tonne of activated carbon.

The pumpcell mechanism combines the functions of agitation, interstage screening, and slurry transfer in one unit. The suspension of the carbon and slurry mixture is maintained by the hydrofoil mixer. The interstage screen surface is kept free of carbon buildup by means of wiper blades attached to the rotating cage. The pumping action of the interstage screen generates a head differential, so that all the adsorption tanks can be placed at the same level and it becomes relatively simple to provide common feed and tailings launders to allow the plant to operate in a carrousel mode.

The CIP tanks are fed by a common launder, which, via a series of plug and gate valves, allows the feed slurry to be fed to any tank, depending on which tank is first in the carousel sequence at any particularly time. The first tank has the highest gold-on-carbon loading in the sequence. When the carbon in this tank has reached the required gold-on-carbon loading, it is isolated from the circuit and the second tank becomes the first, third becomes second and so on. The loaded carbon in the isolated tank is separated from the slurry by pumping the contents of the entire tank over a linear screen (loaded carbon screen) to separate the carbon from the slurry. The carbon is diverted to either acid wash or strip vessels. The screened slurry flows back by gravity to the CIP feed launder. The design carbon concentration in the CIP tanks is 50 g/L.

After the first pumpcell in the carrousel sequence is drained and its contents are sent to the stripping circuit, this pumpcell becomes the last cell in the carrousel sequence. Regenerated carbon is pumped to this tank via one of two carbon sizing screens located above the carousel circuit, depending on the location of the tank to be fed. To avoid carbon abrasion, reactivated carbon can be transferred to the pumpcell when the CIP tank is filled with half of its volume with the slurry tails of the upstream cell in the series. The sizing screen is used to separate any carbon fines from the carbon stream before it is introduced into the CIP circuit. The fines flow by gravity to the carbon fines settling tank in the regeneration area while coarse carbon flows by gravity to the appropriate CIP tank.

After the adsorption circuit, the CIP tailings stream flows by gravity to the carbon safety screen to recover fine carbon attrited or worn down in the CIP circuits. The slurry flowing through the safety screens discharges into a pump box from where it is pumped to the cyanide destruction tank, which is located outside, adjacent to the CIP area.
The CIP area is equipped with an overhead crane with a capacity of four (4) tonnes.

17.3.6.3 Cyanide Destruction

The cyanide destruction tank is an agitated vessel with a residence time of four (4) hours. The selected method of cyanide destruction is the Inco SO_2/Air process. The system uses SO_2 in combination with air, sparged into the tank, to destroy the cyanide in the tailings slurry. $CuSO_4$ is added, as required, to catalyze the cyanide destruction reaction. The target total cyanide content in the slurry at the discharge of the destruction system is less than twenty (20) ppm. The discharge of the cyanide destruction tank overflows into a pump box, from which it is pumped to the POX tailings pumping station. The tailings discharged from the cyanide destruction tank are combined with the neutralization tails for final disposal in the POX tailings pond.

17.3.7 Carbon Stripping and Regeneration

The carbon regeneration and stripping circuit has the capacity to treat one (1) t/d of carbon.

The loaded carbon from the CIP adsorption circuit is pumped to the loaded carbon screen. The oversize (loaded carbon) flows by gravity to the loaded carbon tank and onto the acid wash column.

The acid wash is required to remove any organic material or carbonates which may have been adsorbed on the activated carbon during the CIP process. A solution of 1% hydrochloric acid is circulated by pump from the diluted acid tank into the bottom of the acid washing vessel and from the top of the tank back to the diluted acid tank, while the pH is monitored. Once the acid wash cycle is completed, the spent acid is neutralized by adding caustic soda, drained, and pumped to the flotation tailings pump box for disposal.

An exhaust fan is connected to the diluted acid tank and acid wash vessel to remove fumes. After the acid wash treatment, the carbon is flushed with water and then rinsed again with a

small amount of caustic soda to ensure neutralization. The carbon is then pumped to the carbon stripping vessel.

In the stripping vessel, the gold is stripped from the carbon by circulating a strip solution at high temperature (130°C) and pressure (65 psig). The solution is heated up by an electric heater via a heat exchanger. Once the strip cycle is complete, the loaded strip solution is cooled using heat exchangers and is sent to the electrowinning cell (Section 17.3.8). The barren solution discharged from the electrowinning cell is returned to the barren strip solution tank. Caustic soda and cyanide are added as needed to the barren strip solution tank to adjust the chemistry and conductivity of the strip solution to the appropriate levels.

At the end of the elution cycle (8 hours to 10 hours), part of the barren solution is bled to the CIP circuit to control solution quality and to add reagents (i.e., caustic and cyanide solution), prior to the next stripping cycle. The carbon is cooled down and rinsed with fresh water in the stripping column.

The stripped and cooled carbon is then pumped to the stripped carbon dewatering screen where the water used to transport the carbon and any fine carbon are separated and sent to the carbon fines settling tank. The dewatered carbon from the screen is stored in a one (1) tonne capacity hopper in front of the kiln, which ensures a steady feed during the kiln operation. The electric carbon regeneration kiln has a capacity to process one (1) tonne of carbon per day. The kiln discharges the regenerated carbon into a quench tank filled with water to simultaneously cool and wet the carbon. From the quench tank, the regenerated carbon is pumped to a storage tank where it is combined with conditioned fresh carbon.

The carbon fines settling tank receives screened fine carbon from the carbon sizing screens and the stripped carbon dewatering screen. The carbon fines are then pumped to the carbon fines filter, a plate and frame type filter. The filtered carbon fines are stored in bags and sold for their gold content.

Fresh carbon is introduced into the system to make up for the loss of attrited carbon. The fresh carbon is conditioned prior to use in order to remove sharp edges and to thoroughly wet the

particles in the carbon attrition tank. The conditioned fresh carbon is then pumped to a storage tank where it is combined with the regenerated carbon. This mixture of carbon is then pumped to the carbon sizing screen, to remove fines, and is then introduced into the CIP adsorption circuit.

17.3.8 Electrowinning and Refining

In the gold room, gold is recovered from the pregnant solution using an electrowinning cell with a capacity of 1.4 m^3 (50 ft^3). The loaded stainless steel cathodes are periodically cleaned in-situ with a high-pressure washer. The sludge collected at the bottom of the cell is directed to a sludge pump feeding a recessed plate filter for dewatering. The filtered solids are discharged from the recessed plate filter onto trays and dried in the electric oven. The filtrate is fed back to the electrowinning discharge pump box along with the barren solution from the electrowinning cell and pumped to the barren strip solution tank.

The dry solids are cooled and mixed with an appropriate amount of flux for smelting in the induction furnace. Refined gold is poured from the furnace into a series of moulds and the slag is poured into a slag pot. The barren solution discharged from the electrowinning cells is collected and pumped back to the barren solution tank in preparation for the next strip cycle.

The electrowinning cell and the oven are vented to the atmosphere using a blower. The induction furnace is equipped with a cartridge dust collector and the exhaust is vented to the atmosphere using a fan.

The refinery area is equipped with an overhead crane with a capacity of one (1) tonne.

17.3.9 Reagents

Caustic Soda and HCl, used in the carbon stripping process, and the flotation reagents Aero 208 and MIBC, are delivered in totes. Pumps are connected directly to the totes, which are used to feed the reagent to the appropriate process areas. The individual feed lines are equipped with flow meters and control valves to ensure the appropriate dosage are achieved.

The flotation reagent PAX, as well as the flocculants Flomin 905MC and 4240SSH, as well as the cyanide destruction reagent $CuSO_4$, are delivered in solid form and are dissolved in batches in a mixing tank using filtered water. The batch is then pumped to a storage tank from where the reagent is pumped continuously to the appropriate process area.

Quick lime, used in the neutralization and cyanide leaching processes, is delivered by truck and is transferred pneumatically to the lime storage silo. A screw conveyor transfers the quick lime from the silo to the slaker, where it is wetted with filtered water. The slaked lime passes through grit separators and into the quick lime mixing tank, to which more filtered water is added to create slurry of the appropriate density. Grit is removed from the separator and disposed of using a screw conveyor. Parallel lines of slakers and grit separators are installed so that operations can continue in the event that one of these pieces of equipment fails. The quick lime slurry is pumped from the mixing tank continuously to the appropriate process areas.

A VPSA (vacuum pressure swing adsorption) oxygen plant is installed on site and operated by an industrial gas Vendor to supply high purity (92.5%) oxygen to the autoclave.

Sodium Cyanide (NaCN), used in the cyanide leaching and carbon stripping processes, and sulphuric acid, used in the cyanide destruction process, are delivered to site in solution in tanker trucks. The solutions are transferred from the tanker truck to a storage tank, from which they are distributed to the appropriate process areas by pump.

Limestone is used in the neutralization process. It is delivered to site by truck and stored in an outdoor stockpile. From the stockpile, the limestone is fed using a front-end loader to a hopper, which is equipped with a static grizzly to reject any large waste material. A vibrating feeder transfers limestone to a belt conveyor from the hopper. The belt conveyor feeds limestone to a ball mill operated in closed circuit with a hydrocyclone, and sump tank and a pump. Filtered water is added to the limestone grinding circuit in order to achieve the appropriate slurry density. The target product size of the limestone, exiting the grinding circuit by the hydrocyclone overflow, is P_{90} of 100 microns. This stream flows by gravity to the agitated limestone slurry distribution tank. From this tank, the limestone slurry is pumped to the neutralization circuit.

SO_2 is used in the cyanide destruction process. It is delivered to site by tanker truck and is stored outdoors, adjacent to the cyanide destruction system, in a pressurized storage tank.
Nitrogen gas is used in seal water system for the autoclave. Small gas cylinders of nitrogen gas are delivered to site and are replaced as needed.

17.3.10 Common Services

An air compressor is used to provide compressed process air for the plant, which is used for the operation of instruments, air addition to the flotation columns, the neutralization circuit, the cyanide leach tanks and the cyanide destruction system.

A set of five (5) blowers operated in parallel, supply air to the flotation rougher cells.

A steam boiler package will supply steam to the autoclave as required. Steam is only used to restart the autoclave, and is therefore used intermittently.

17.3.11 Process, Filtered and Fire Protection Water

The process water tank which has a diameter of 15 m and a height of 5 m, receives water from the flotation tailings and concentrate thickeners. Water is also reclaimed from the flotation tailings pond polishing basin and from the process water pond. The level in the process water tank is controlled by pumping water from one of these two sources. Process water is circulated by pump throughout the plant from the process water tank.

Water is pumped from the process water tank to the sand filter to remove fine particles, and stored in the filtered water tank. Filtered water is used for the preparation of reagents and in the pump gland seals. There are two gland seal water circuits; high pressure for the high pressure tailings pumps, and low pressure for all other pumps requiring seal water.

Water is also pumped from the process water tank to a cooling water circuit, used to cool motors and electrical components.

A small portion of the filtered water is fed to a water demineralization package, in order to provide demineralized water to the autoclave seal water system.

Domestic water will be sourced from an esker approximately 4.5 kilometers from the plant. It will be pumped to the plant and stored in the domestic water tank. From the domestic water tank, the water will be fed to the domestic water circuit, comprised of showers, emergency eye wash stations and lavatories.

The process water storage basin will also serve as the fire protection water reservoir. The fire protection water circuit is equipped with both an electric pump and a diesel pump in case of a power failure.

17.3.12 Major Process Equipment List

Table 17.3 presents the major process equipment list used as the basis for developing the Capital Cost Estimate.

Table 17.3: Major Process Equipment List

Item	Notes
Primary Jaw Crusher	200 kW
Crushed Ore Conveyor	400 HP, 36" belt, 275 fpm, 1809' long, 156' lift
Crushed Ore Silo	11m diameter x 20m high, 5.5hr live load
SAG Mill Feed Conveyor	100 HP, 36" belt, 250 fpm, 605' long, 46' lift
SAG Mill	5040 kW, 30'-0" x 9'-6" EGL single pinion with trommel screen
Ball Mill	5040 kW, 18'-0" x 28'-0" EGL single pinion with trommel magnet
Hydrocyclone Cluster	500 mm hydrocyclones (3 operating and 2 installed spares)
Gravity Concentrators	2 parallel screens and gravity concentrator units
Rougher Cells	8 mechanical cells, each 100 m^3 with a 150 HP agitator
Cleaner Columns	Cleaner column - 3.05 m diameter x 10 m high Cleaner scavenger column - 2.44 m diameter x 8.5 m high
Concentrate Dewatering Thickener	11 m diameter, 5 HP rake, high rate
POX package	One (1) Autoclave, 6 x 7 m diameter neutralization tanks and associated equipment
CCD Wash Thickeners	2 x 13 m diameter, 5 HP rake, high rate, covered
Neutralization Thickener	10 m diameter, 5 HP rake, high rate, covered
Leach Tanks	4 x 8.1m diameter tanks, cascade configuration (heights: 9.0m, 8.7m, 8.4m, 8.1m)
CIP Package	Carousel system, 10 tanks each 20 m^3 with 10 HP pump screen agitator
Carbon Stripping and Regeneration Package	Carbon fines filter, carbon reactivation kiln, acid wash vessel, carbon stripping vessel and associated Equipment
Gold Room Package	Electrowinning cell, induction melt furnace and associated equipment
Limestone Grinding Package	150 HP ball mill, hydrocyclone cluster, pump box and pump
Lime Storage Mixing and Distribution Package	Storage silo, slakers, mixing tank, pumps
Flotation Tailings Thickener	25 m diameter, 15 HP rake

18 PROJECT INFRASTRUCTURE

This section describes the major infrastructure required to support the Joanna Project.

18.1 General Joanna Site Plan

The general Joanna site plan is presented in Figure 18.1. In close proximity to the Property, there is a Hydro Quebec 120 kV power line to the north and to the south there is a CN railway and further south the Highway No.117, all running in an east-west direction. Site infrastructure generally lies between these lines with the exception of the POX tailings pond which lies directly to the north of the power line. The main site access road connects to Highway 117 to the south of the Property. The main site infrastructure includes the following:

- There are two mine pits included in the design; the Hosco pit, which is the main pit, and the much smaller West pit. The pits are oriented east-west directly to the north of the railway line. The design of pit shell was performed considering geotechnical information (Golder, 2009) and the position of the railway line, therefore the relocation of the rail line is not required. Once fully developed, the Hosco pit is designed to be approximately 2,190 m long by 660 m at its widest point and a maximum depth of 280 m. The West pit, approximately 200 m to the west of the Hosco pit is approximately 660 m long by 450 m wide and a maximum depth of 120 m.
- The mine development plan provides that topsoil be segregated and stored in a dedicated stockpile, which will be located to the north of the West pit. It is oriented in the east-west axis and will measure 1,400 km long by 500 m wide by 5 m in height once fully developed.
- The main waste pile is located to the north of the Hosco pit and is triangular in shape. The design of the waste pile took into consideration the prevailing geotechnical conditions in the area as well as the objective of minimizing the environmental impact. Once fully developed, the overall surface area will be approximately 163 hectares. The projected height is 70 m.
- A much smaller waste pile for material containing Nickel and Arsenic (Ni/As) will be located to the north of the main waste pile. It will measure approximately 550 m by 550 m by 45 m in height for an overall surface area of 29 hectares once fully developed. This material will be segregated due to environmental considerations, as described in Section 20 of this Report.

- The overburden pile will be located to the east of the main waste pile and will measure approximately 2 km by 700 m by 7.5 m in height. The characteristics of the overburden material limits the height of the pile. Therefore this pile occupies a relatively large footprint on the site plan (approximately 167 hectares).
- A network of mine roads, restricted in access to only authorized vehicles, connects the open pits to the crusher, waste piles, and the mine services area. These roads are generally 35 m in width, allowing for two-way traffic. Total length of the mine road network is approximately 3.5 km.
- The crusher building is located to the north-west of the Hosco pit, between the main waste pile and the topsoil pile. A mine road connects the open pit to the crusher building. The footprint of the crusher building is approximately 25 m by 16 m and lies within a larger cleared area. Design provides that a trailer facility having toilets and a lunch room area for approximately 15 people be located in proximity of the crusher to allow for mine operating personnel to take their breaks.
- An open area to the west of the crusher building has been provided for stockpiling ROM ore as well as for emergency stockpiling of crushed ore. Environmental considerations were made in design for this stockpile.
- A conveyor having a length of approximately 500 m feeds crushed ore to the storage silo. The silo has a live capacity of 2,360 t or 5.5 hours. It consists of a concrete structure measuring 11 m in diameter and 20 m in height. A service road runs the length of this conveyor.
- Crushed ore from the storage silo is conveyed to the SAG mill within the concentrator building by a conveyor having a length of approximately 200 m.
- The ore processing area incorporates the concentrator building as well as various other processing areas located outside. Grinding, gravity concentration, flotation, the autoclave, CIP carousel tanks, the carbon area and the gold room are all within a building. Thickeners, holding tanks, leaching tanks, lime storage, etc. are all located outside. The limestone storage area is outside and the limestone grinding area is located within an enclosed area. Employee facilities and offices for employees assigned to the concentrator are located within the concentrator building. The ore processing area is located directly to the north of the crusher building and to the west of the Ni/As waste pile and has a footprint of approximately 90 m by 90 m.

- The mine services area is located on a cleared pad adjacent to the processing area. The mine services area consists of the following:
  - The mine garage and truck wash station are megadome type structures having a surface area of approximately 1,200m^2. Considering the type and size of the mining fleet, the design provides that this facility be used only for performing normal maintenance. Major maintenance or overhauls will require that equipment components be transported to equipment vendor maintenance facilities in Val-d'Or.
  - One 100,000-liter reservoir for storing diesel is provided. Diesel fuel is delivered directly to the site by tanker trucks owned and operated by local fuel suppliers. The fueling station is only used to fill mobile mining equipment. In-pit equipment such as drills and shovels are filled directly by on-site fuel/lube truck.
  - Mine employee facilities, administration offices, core shack and laboratory as well as other service areas are designed using trailer-type facilities. A parking area for light vehicles and employee vehicles is provided in the mine services area. Also, a parking and staging area for mine vehicles, separate from that of light vehicles, is provided.
- Power will be supplied by Hydro Quebec, who will install a 120 kV line over a distance of approximately 2,000 m and tie into the site's main substation, located adjacent to the concentrator building.
- Given environmental considerations and tailings characteristics, process tailings will be disposed of within two separate tailings ponds. A detailed description is provided in Section 20.
  - Flotation tailings, representing about 95% of the tailings produced are considered neutral and non-acid generating. The flotation tailings pond is located in the northwest corner of the site, 1,400 m to the west of the concentrator and will have a surface area of 208 hectares and a height of 23.8 m when it is fully developed. Process water pumped with the tailings as well as surface water is collected within a polishing pond measuring approximately 650 m by 350 m.
  - Tailings generated from the POX and cyanide leaching areas have a high arsenic content and are disposed of separately in order to minimize the project's environmental footprint. The POX tailings pond is located approximately 1.1 km to the north of the concentrator and will have a surface area of 37 hectares and a height of 13 m when it is fully developed. Water pumped with the POX tailings as well as surface water is

collected within a polishing pond located directly to the west of the POX tailings pond and has a footprint of approximately 100 m by 100 m.

- The process water pond is located to the west of the concentrator building and has a footprint of approximately 17 hectares. This pond is used to store process water for periods of low precipitation. It provides make-up water for the process during periods when the flotation tailings polishing pond water level is low. It is also used as a fire protection water reservoir.
- The process water polishing pond serves as a collection point for all excess water or for water streams that cannot be used for the process. As such, this pond effluent is the single process water effluent from the site. It measures approximately 110 m by 140 m.
- The site access road is approximately 4 km long. It branches off of Highway 117 at the south of the property and travels north. It passes to the west of the topsoil pile, between the flotation tailings pond and the process water pond to the concentrator.
- Site access will be controlled by a guard house and a security gate at a location to be determined along the main site access road. The guard house will be a pre-fabricated structure.
- Tailings in slurry is pumped from the concentrator to the two tailings ponds. Surface water is collected from the various areas on site and directed to a designated basin for use as process water or for effluent to the local watershed. Potable water is obtained as required from a nearby esker. A piping network constructed mainly of HDPE pipe is designed for this purpose. Table 18.1 lists the various pipelines included in the design.

Table 18.1 – Water and Tailings Pipelines

Pipeline Description	Diameter (mm/in)	Length (m)
Water from the POX polishing pond is pumped by electric powered pumps to the process water polishing pond where the overflow is discharged to the local watershed.	150/6	2,500
Water from flotation tailings polishing pond is pumped by electric powered pumps to the concentrator process water tank.	350/14	3,600
Water from the process water pond is pumped by electric powered pumps to the concentrator process water tank via the pipeline from the flotation tailings polishing pond. This system serves as make-up water, as required, but	350/14	550

Pipeline Description	Diameter (mm/in)	Length (m)
also as a backup when the pumping system from the flotation tailings polishing pond is not available.		
Mine dewatering is done using diesel powered pumps within the pit. Pumping is intermittent and water is directed into the process water pond as it can be used as process water.	150/6	2,000
Surface water and run-off water from As/Ni waste rock pile is collected and pumped intermittently by diesel powered pumps and directed to the process water pond as this water can be used as process water.	150/6	1,000
Surface water and run-off water from the main waste rock and overburden pile is collected and pumped intermittently by diesel powered pumps. They normally discharge to the environment but can be directed to the process water pond as this water can be used as process water.	150/6	1,700
Surface water and run-off water from the crusher pad and the ROM ore stockpile is collected and pumped intermittently by diesel powered pumps and directed to the process water pond as this water can be used as process water.	150/6	1550
Surface water and run-off water from concentrator and mines services pad is collected and pumped intermittently by diesel powered pumps and directed to the process water pond as this water can be used as process water.	150/6	550
Potable water is pumped intermittently by diesel powered pumps from a nearby Esker to a potable water tank in the process plant area for distribution to the employee facilities around the plant. Areas of low water use will have bottled water accessible.	150/6	4,500
Flotation tailings are pumped by electric powered pumps from the plant to the flotation tailings pond.	450/18	2,800
POX tailings are pumped by electric powered pumps from the cyanide destruction process in the plant to the POX tailings pond.	150/6	3,200

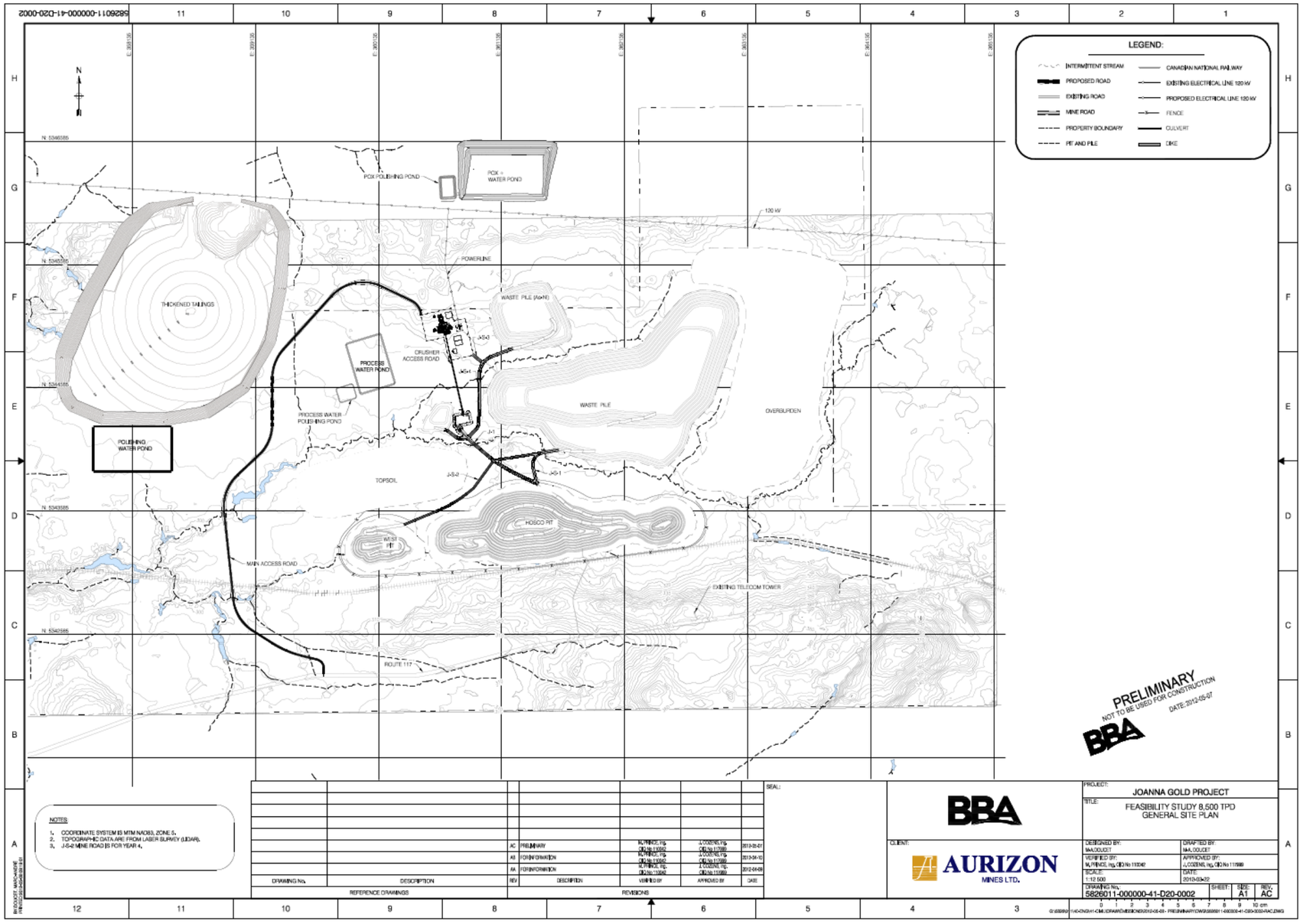


Figure 18.1 – Site Plan

18.2 Electricity

A load list was developed once site-wide electrical power requirements were estimated for process and mechanical equipment in the various areas of the plant and the site. Crushing and grinding power requirements were estimated based on ore grindability data obtained from the testwork. Power requirements for pumping, conveying, and other process equipment were estimated based on flow rates developed in the mass and water balance as well as from BBA's reference projects database. All mining equipment in the open pit are diesel powered, therefore there is no electric power loop going to the mine. Power requirements for infrastructure and service buildings were also estimated from BBA's reference projects. Table 18.2 presents the site-wide estimated power requirements as well as the estimated annual electrical consumption for the various sectors of the facility. As can be seen, the projected annual electrical consumption is in the order of 168 GWh and the power demand is estimated at 23.5 MW. These results have been used to estimate operating costs as presented in Section 21 of this Report. Furthermore, the load list was used to develop the preliminary electrical Single Line Diagram and to size major electrical equipment and components. The facility power requirements were provided to Hydro Quebec in order to allow them to plan for providing power to the facility on a timely basis, which will then be used to orient a study conducted by BBA to evaluate the options available for power transmission to the Joanna mine site.

A plan for providing emergency power to the facility in the case of a power failure was developed. Equipment requiring emergency power was identified and an emergency diesel powered generator of 1,500 kW is provided in the plant design.

Table 18.2 – Electrical Load List

PROJECT: JOANNA GOLD PROJECT
SUB-PROJECT: Revised Load List
PROJECT NUMBER: 5826011
DATE: 2012-04-27
REV.: 00

PREPARED BY: Sylvain Bélec

SECTOR	DESCRIPTION	CONNECTED LOAD	RUNNING LOAD	RUNNING POWER (1)	AVERAGE EFFICIENCY FACTOR (2)	LOAD FACTOR (3)	DIVERSITY FACTOR (4)	POWER DEMAND (5) = (1)/(2)*(3)*(4)	ANNUAL LOAD FACTOR (6)	ESTIMATED ANNUAL ENERGY CONSUMPTION (7) = (5)*(6)*365*24/1000
		(HP)	(HP)	(MW)				(MW)		(GWh)
4100	PRIMARY CRUSHING	504	504	0.3760	0.92	0.85	0.85	0.2953	0.9	2.33
4200	PRIMARY CRUSHING - CONVEYOR	410	410	0.3059	0.92	0.85	0.85	0.2402	0.9	1.89
4970	PRIMARY CRUSHING - AIR COMPRESSOR	60	60	0.0448	0.92	0.85	0.85	0.0352	0.9	0.28
5100	ORE HANDLING	210	210	0.1567	0.92	0.85	0.85	0.1230	0.9	0.97
6100	GRINDING	1924	1074	0.8012	0.92	0.85	0.85	0.6292	0.9	4.96
6100	GRINDING (SAG and BALL MILL)	13600	13600	10.1456	0.95	0.81	1	8.6932	0.9	68.54
6150	GRAVITY	200	193	0.1440	0.92	0.85	0.85	0.1131	0.9	0.89
6200	FLOTTATION	1377	1362	1.0161	0.92	0.85	0.85	0.7979	0.9	6.29
6310	PRESSURE OXIDATION	2257	1540	1.1488	0.92	0.85	0.85	0.9022	0.9	7.11
6320	COUNTERCURRENT DECANTATION WASH	350	245	0.1828	0.92	0.85	0.85	0.1435	0.9	1.13
6330	NEUTRALIZATION	385	315	0.2350	0.92	0.85	0.85	0.1845	0.9	1.45
6600	CYANIDE LEACHING CIRCUIT	476	419	0.3126	0.92	0.85	0.85	0.2455	0.9	1.94
6700	CARBON HANDLING AND STRIPPING	772	772	0.5759	0.92	0.85	0.85	0.4523	0.9	3.57
6750	GOLD ROOM	102	52	0.0388	0.92	0.85	0.85	0.0305	0.9	0.24
6800	REAGENTS	506	332	0.2477	0.92	0.85	0.85	0.1945	0.9	1.53
6940	MILL WORKSHOP	27	27	0.0201	0.92	0.85	0.85	0.0158	0.9	0.12
6970	SERVICES	2304	810	0.6043	0.92	0.85	0.85	0.4745	0.9	3.74
6980	PROCESS WATER	520	275	0.2052	0.92	0.85	0.85	0.1611	0.9	1.27
6990	WATER COOLING	315	160	0.1194	0.92	0.85	0.85	0.0937	0.9	0.74
7100	FLOTTATION TAILLING THICKENER	1792	941	0.7020	0.92	0.85	0.85	0.5513	0.9	4.35
	OXYGENE PLANT			4.4800	0.92	0.98	0.9	4.3125	0.9	34.00
	GENERAL OFFICE			0.1000	0.90	1.00	0.6	0.0667	0.9	0.53
	MINE GARAGE			2.5000	0.90	1.00	0.6	1.6667	0.9	13.14
	ALLOWANCE FOR HVAC	1000	1000	0.7460	0.90	1.00	0.6	0.4973	0.9	3.92
	NETWORK LOSS (2%)							0.4184	0.9	3.30
	TOTAL	29091	24301	25.2085			POWER DEMAND SUBTOTAL	21.3381		168.23
							TOLERENCE (±10%)	2.1338		0.00
							TOTAL POWER DEMAND	**23.47**		**168.23**

19 MARKET STUDIES AND CONTRACTS

Mines Aurizon Ltd already has a sales contract in place for its Casa Berardi operation, typical of, and consistent with standard industry practices. It is expected that terms contained within futures sales contract related to Joanna production would be of a similar nature to its current process.

20 ENVIRONMENT

20.1 Jurisdictions and Applicable Laws and Regulations

The legal framework for the construction and operation of the projected facilities is a combination of provincial, national, and municipal policies, regulations and guidelines. The design and the environmental management of the Project must be done in accordance with this legal framework.

20.1.1 Québec Procedure relating to the Environmental Assessment of the Project

20.1.1.1 Overview

Section 31.1 of the Environment Quality Act (EQA) states that "No person may undertake any construction, work, activity or operation, or carry out work according to a plan or program, in the cases provided for by regulation of the Government without following the Environmental Impact Assessment and review procedure and obtaining an authorization certificate from the Government."

Moreover, Section 2 of the Regulation respecting Environmental Impact Assessment and Review provides the list of projects subject to the Environmental Impact Assessment and review procedure, namely:

> *"(n.8) the construction of an ore processing plant for metalliferous ore or asbestos ore, where the processing capacity of the plant is 7,000 metric tons or more per day, or any other ore[1], where the processing capacity of the plant is 500 metric tonnes or more per day;*
>
> *(p) the opening and operation of a metals mine[2] or an asbestos mine that has a production capacity of 7,000 metric tonnes or more per day, or any other mine[3] that has a production capacity of 500 metric tonnes or more per day."*

1 Other than metalliferous, asbestos or uranium ore.

2 "Mine" means all the surface and underground infrastructures used for the extraction of ore.

3 Other than a metal, asbestos or uranium mine.

Under the Joanna Project, the milling rate is planned at 8,500 tpd; therefore, the evaluation process and review of environmental impacts will apply. Note also that the Québec Mineral Strategy, released on June 29, 2009, sets a recommendation lower than 3,000 tpd for the threshold to trigger an EIS.

According to the *Regulation on Assessment and Review of Environmental Impacts,* following the filing of the Project notice, the Ministry of Sustainable Development, Environment and Parks (MDDEP) sets 15 months as the maximum period within which the Minister must submit the application record to the Government for approval. It is, however, noted that the period of 15 months does not include the time taken by the ‘’proponent’’ to draft the EIA report, the time between requests for information by the MDDEP and transmission responses by the ‘’proponent’’, nor the time taken by the Government (Cabinet) to make its decision. Note that the MDDEP intends to reduce this time period to 12 months. Figure 20.1 shows the steps involved in the procedure.

For the Joanna Project, it is likely that public hearings will be required given the relative proximity of a biodiversity reserve and residential areas, and the fact that environmental groups have raised concerns about the mining industry. It should be noted that Aurizon Mines Ltd. has already initiated active and participative discussions with local stakeholders and that such an approach will be continuing as the Project progresses. This approach aims at establishing and maintaining a sustainable dialogue with stakeholders in order to identify specific issues associated with the Project.

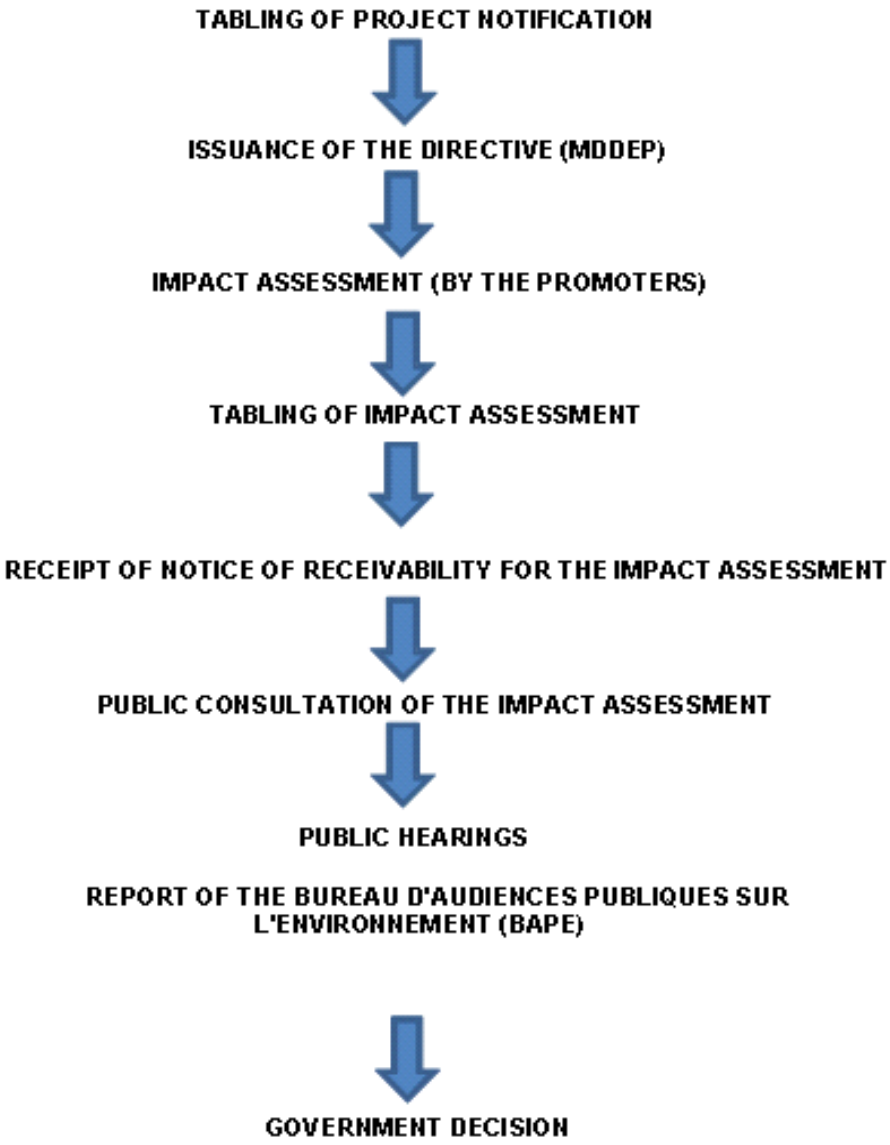


Figure 20.1: Steps in the Environmental Impact Assessment Procedure

Section 31.2 of the EQA states that "*Every person wishing to undertake the realization of any of the projects contemplated in Section 31.1 must file a written notice with the Minister describing the general nature of his project; the Minister, in turn, shall indicate to the proponent of the project the nature, the scope and the extent of the Environmental Impact Assessment statement that he must prepare.*"

A project notice was filed on September 2, 2010, with the MDDEP. A week later, following the study of the project notice, a Directive defining the required scope and contents of the Environmental Impact Assessment of the project was sent by the MDDEP to Mines Aurizon. The Directive specific to the Joanna Project received in September 2010 is identical to the document entitled *Directive for an Environmental Impact Assessment of a Mining Project* (in French only) (MDDEP, August 2005).

Given that infrastructure changes were made as part of the Feasibility Study, an addendum to the project notice will be submitted to the MDDEP. A revised version of the *Directive for an Environmental Impact Assessment of a Mining Project* was released by the MDDEP in January 2012; however no significant changes were made in this new Directive.

20.1.1.2 General Contents of an Environmental Impact Assessment Statement

Section 3 of the Regulation respecting *Environmental Impact Assessment and Review* defines the contents of an Environmental Impact Assessment statement:

a) A description of the project mentioning, in particular, the desired objectives, the site [...], the project timetable, any subsequent operation and maintenance activities, the amounts and characteristics of types of borrowed materials required, power sources, methods of management of waste or residue other than road construction residue, transportation activities inherent in the construction and subsequent operation of the project, any connection with land use planning and development plans, urban zoning plans or agricultural zoning and reserved areas within the meaning of the act to preserve agricultural land [...];
b) A qualitative and quantitative inventory of the aspects of the environment which could be affected by the project, such as fauna, flora, human communities, the cultural, archaeological and historical heritage of the area, agricultural resources and the use made of resources of the area;
c) A list and evaluation of positive, negative and residual impacts of the project on the environment, including indirect, cumulative, latent and irreversible effects [...];

d) A description of the different options to the project, in particular regarding its location, the means and methods of carrying out and developing the project, and all other variables in the project as well as reasons justifying the option chosen;
e) A list and description of measures to be taken to prevent, reduce or attenuate the deterioration of the environment, including [...] In particular, any equipment used or installed to reduce the emission, deposit, issuance or discharge of contaminants into the environment, any control of operations and monitoring, emergency measures in case of accident, and reclamation of the area affected.

20.1.1.3 Summary of the Environmental Impact Assessment Statement

Section 4 of the *Regulation respecting Environmental Impact Assessment and Review* indicates that an Environmental Impact Assessment statement prepared pursuant to Section 31.1 of the *Environment Quality Act* must be accompanied by a non-technical summary of the main elements and conclusions of the studies, documents or research. The summary is published separately.

20.1.1.4 Evaluation of the Environmental Impact Assessment (EIA)

A well-defined project is essential to producing an EIA report that takes the project to be carried out into account and that will be considered acceptable by the authorities early on in the procedure. Upon receipt of the EIA report, the MDDEP will determine its admissibility. This evaluation involves consultations with many government departments and agencies. Generally, the proponent should expect to receive questions and comments to be addressed before the EIA report can be determined admissible. Following the response to this first set of questions, a second set of questions may be issued. To avoid delays associated with this procedure, it is essential to produce an Environmental Impact Assessment that covers as specifically as possible every aspect raised in the Directive issued by the MDDEP.

20.1.1.5 Public Consultations

Section 31.3 of the EQA states that "*After receiving the Environmental Impact Assessment statement, the Minister shall* make *it public and indicate to the proponent of the project to initiate the stage of public information and consultation provided for by regulation of the Government.*"

Once the impact study is found to be admissible, the Minister will then direct the Bureau d'audiences publiques sur l'environnement (BAPE) to prepare the case for public consultation. This consultation process extends for 45 days (Section 11 of the *Regulation respecting Environmental Impact Assessment and Review*).

20.1.1.6 Public Hearings

Section 31.3 of the EQA also states that "*Any person, group or a municipality may, within the time prescribed by regulation of the Government, apply to the Minister for the holding of a public hearing in connection with such a project. Unless he considers such application to be frivolous, the Minister shall direct the Bureau to hold a public hearing and report its findings and its analysis thereof to him.*"

Public hearings are governed by the *Rules of Procedure relating to the Conduct of Public Hearings (Q-2, r. 45).*

Following the public hearings, the BAPE commission files its report with the Ministry of Sustainable Development, Environment and Parks. The commission is required to complete its mandate and file its report within four (4) months.

The Minister then has 60 days to publicly release the BAPE report.

20.1.1.7 Government Decision

On the basis of the BAPE report and the MDDEP's evaluation of the EIA, the Minister analyses the case and makes a recommendation to the Government. As specified in Section 31.5 of the EQA[4], the Government makes its decision by Decree: it authorizes the project, with or without

[4] Section 31.5: *Where the Environmental Impact Assessment statement is considered satisfactory by the Minister, it is submitted together with the application for authorization to the Government. The latter may issue or refuse a certificate of authorization for the realization of the project with or without amendments, and on such conditions as it may determine. That decision may be made by any committee of ministers of which the Minister is a member and to which the Government has delegated that power.*

changes and conditions, or rejects it. The maximum period between the publication of the BAPE report and the Government's decision is not specified either in the EQA or its regulations.

20.1.2 Federal Procedure

The objective of and various steps involved in the federal environmental assessment and review procedure are similar to those associated with the Québec procedure. In most cases, Québec's environmental review process is the only process that applies to projects to be conducted within its boundaries. In some cases, federal government representatives are also involved in an environmental review of a project.

Federal environmental review procedures are governed by the *Canadian Environmental Assessment Act* (1992, C. 37) and its four main regulations:

a) Law List Regulations (SOR/94-636);
b) Inclusion List Regulations (SOR/94-637);
c) Comprehensive Study List Regulations (SORS/94-638);
d) Exclusion List Regulations (SORS/94-639).

According to Section 5 of the Act, one of the following conditions is required for the application of the federal procedure:

e) A federal authority is the project proponent ;
f) A federal authority "has the administration of federal lands and sells, leases, or otherwise disposes of those lands or any interests in those lands," or;
g) A federal authority provides financial support, i.e., "makes or authorizes payments or provides a guarantee for a loan or any other form of financial assistance to the proponent";
h) A federal authority issues a permit or license, grants approval, or takes any other action for the purpose of enabling the project to be carried out in whole or in part.

From our understanding of the project, the first two conditions obviously do not apply, nor does the third one since neither a loan nor financial assistance has been or will be provided by a federal authority for the development of this project.

Many provisions of various federal acts and related regulations have a bearing on the last condition. The *Fisheries Act* is one judicial element that could be a trigger. Section 35 of the *Fisheries Act* specifies that:

> *"(1) No person shall carry on any work or undertaking that results in the harmful alteration, disruption, or destruction of fish habitat.*
>
> *(2) No person contravenes subsection (1) by causing the alteration, disruption, or destruction of fish habitat by any means or under any conditions authorized by the Minister or under regulations made by the Governor in Council under this Act."*

As mentioned in Section 20.7, all mine infrastructures are located within only one sub-watershed, namely Stitchman Creek. In fact, the POX Tailings Management Facility (POX-TMF) is located outside that area, but all waters coming from that facility will be managed in such a way that it will be pumped to the Process Water Polishing Pond (PWPP) and then discharge with the final effluent within the Stitchman Sub-Watershed. No water discharge is located outside the Stitchman Sub-Watershed. Therefore, within that area and where there was potential to locate any of the required surface infrastructures, a comprehensive Fisheries and Fish Habitats Assessment Program was performed to precisely locate all sections of the Stitchman Creek and its affluents, which are to be considered as fish habitats according to Canada's *Fisheries Act*.

The objective of such work was to limit the ecological footprint of the Project to areas located out of those habitats. Among other things, the use of thickened tailings was also prioritized in order to minimize the footprint of the Tailings Management Facility.

As depicted in Section 20.3, the performed Fisheries Assessment Program showed that the Stitchman Creek and its affluents, where considered as fish-bearing creeks, host species such as brook stickleback, pearl dace (about 99%) and catfish. None hosts fish species of importance to any ongoing productivity of commercial, recreational or Aboriginal fisheries. Such information might be of interest in line with the most recent Bill C-38 which enumerates factors to be taken into account before recommending to the Governor in Council that a regulation be made in

respect of Section 35 of the Fisheries Act. Among those are “the contribution of the relevant fish to the ongoing productivity of commercial, recreational or Aboriginal fisheries and whether there are measures and standards to avoid, mitigate or offset serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or that support such a fishery.” Such further analysis will have to be done with the Department of Fisheries and Oceans Canada (DFO) as part of the Environmental and Social Impact Assessment process. In any case, considering the proposed site general arrangement plan, only the location of the proposed Hosco and West pits will require the deviation of existing fish-bearing watercourses. Among those is a ditch implemented as part of the past operating Hosco Mine. In both cases, following preliminary discussion with the DFO, the deviation of those two watercourses is not expected to require a license under Section 35 of the *Fisheries Act*. Indeed, works proposed by Aurizon Mine and Roche Ltd to deviate those watercourses have for now been considered sufficient to mitigate/offset the impact on fish.

Finally, results depicted in Section 20.3.1.1 show that the most recent hydrogeology assessment works indicate that rocks within the pit area are lowly permeable and that the anticipated drawdown would not result in decreased water flows/levels within the nearby Stitchman Creek.

Within the context of the current study, fish habitat was not considered a trigger for the federal environmental review procedure. A project notice in compliance with the Project Proposal Guide Submitted to Fisheries and Oceans Canada for Analysis Under the Provisions of the Fisheries Act Respecting Fish Habitat Protection will need to be submitted to the Canadian Environmental Assessment Agency for a final decision.

The *Explosives Act can* also be a trigger for major mining projects. Indeed, for certain large-scale mining projects, it might be cheaper to manufacture explosives directly at the mine site. However, the manufacturing of explosives requires a license issued by Natural Resources Canada, which triggers a federal environmental assessment. In the case of the Joanna Project, explosives will be neither manufactured nor stored on site in large quantities.

Until 2010, when a trigger led to the application of the Canadian Environmental Assessment Act, the Act applied only to the trigger element and not the entire project. However, a ruling by the Supreme Court of Canada in January 2010, clarified the framework for the application of the Act. From now on, as a result of this ruling, when a trigger element leads to the application of the Canadian Environmental Assessment Act, the Act must be applied to the entire project and not just the trigger element.

If the Joanna Project is subject to the federal environmental assessment procedure, it would be subject to a comprehensive study under Section 10: "*The proposed construction, decommissioning or abandonment of a facility for the extraction of 200,000 m^3/a or more of ground water or an expansion of such a facility that would result in an increase in production capacity of more than 35 per cent" and Section 16 c) "The proposed construction, decommissioning or abandonment of a gold mine, other than a placer mine, with an ore production capacity of 600 t/d or more*" of the Comprehensive Study List Regulations

We should also mention that in the case where the federal procedure was to apply, provincial and federal governments have signed a cooperative agreement for the joint review of studies of environmental impacts. This agreement applies to projects located in Québec, south of the 49th parallel and will reduce delays related to the involvement of both levels of government. The agreement provides for the holding of joint public hearings.

Moreover, the federal government recently introduced recommendations made by a committee that is reviewing the *Canadian Environmental Assessment Act*. These recommendations are:

> *"Improving Timelines: The Standing Committee endorsed the concept of environmental assessments (EAs) being carried out by the "best-placed regulator", which at the federal level should be the Canadian Environmental Assessment Agency (CEA Agency). The Committee also recommended eliminating the consideration of "alternatives to" a project under the CEAA, as EAs should not include a review of the proposed project's business case. Finally, while the Committee noted the recent regulations under the CEAA that provide mandated timelines for comprehensive study-level assessments, the Committee*

recommended further amendments to the CEAA framework to provide for binding timelines for all EAs under the Act.

Reducing Duplication: The Standing Committee recommended that the CEA Agency should be empowered to determine that another jurisdiction's EA process fulfills the requirements of the CEAA and is thus "equivalent". The Committee suggested that specific pieces of provincial legislation deemed "equivalent" should be identified in a Schedule to the CEAA and that the CEAA should be amended to exempt projects subject to EAs under those provincial laws from the requirements of the CEAA. In addition, the Standing Committee recommended moving the CEAA away from a trigger-based approach to EA (i.e., if there is a "trigger" under Section 5 of the CEAA, then an EA is required) to a project list approach whereby projects requiring federal EAs are clearly enumerated in advance, subject to the discretion of the Minister of Environment to require an EA for a non-listed project.

Aboriginal Consultation: The Standing Committee recommended that Aboriginal consultation be more fully integrated into the EA process. The Committee recommended that the federal government work with the provinces, territories and Aboriginal groups to develop a single Aboriginal consultation process applicable to all EAs under the CEAA.

Certainty of Outcomes: While the Standing Committee noted that improving the EA process was important, it also recognized that improvements were needed to yield positive, measurable outcomes in federal EAs. To that end, the Committee recommended that EAs be conducted early in the planning stages of a project, which will facilitate cooperation with other jurisdictions and enhance the ability of Aboriginal groups to participate in project planning. The Committee recommended that EAs under the CEAA consider positive environmental effects of proposed projects, not just adverse environmental effects. Greater reliance on follow-up programs was also recommended by the Committee."

20.1.3 Schedule

A period of about 20 months is anticipated for the preparation of the Environmental Impact Assessment and obtaining the government decision. This period will include the time needed for the preparation of the Environmental Impact Assessment, the evaluation of the study by the authorities, the preparation of answers to questions, the holding of public hearings, the preparation and filing of BAPE's report, the minister's recommendation to the government and the release of the government decree.

Figure 20.2 illustrates the various stages involved in obtaining the Decree, along with the time required for each step.

20.2 Environmental Permitting

Even if the Project underwent Environmental Impact Assessment and was authorized by the Government pursuant to Section 31.5 of the Act, it would still be subject to Section 22 of the *Environmental Quality Act* and must therefore obtain a certificate of authorization as stated in Section 6 of the *Regulation Respecting the Administration of the Environment Quality Act.*

> ***"6.*** *Notwithstanding Sections 1 to 3 of this Regulation, any project arising from a project authorized by the Government pursuant to Section 31.5 of the Act is subject to the application of Section 22 of the Act."*

In addition to the mitigation measures set out as part of the environmental and social impact assessment, the final project design must comply with all applicable standards for the proposed infrastructure and equipment.

The issuance of the certificate of authorization should only be a formality, as the certificate issued pursuant to Section 31.5 of the Act binds the Minister as to where he exercises the powers provided in Section 22, as specified in Section 31.7 of the Act.

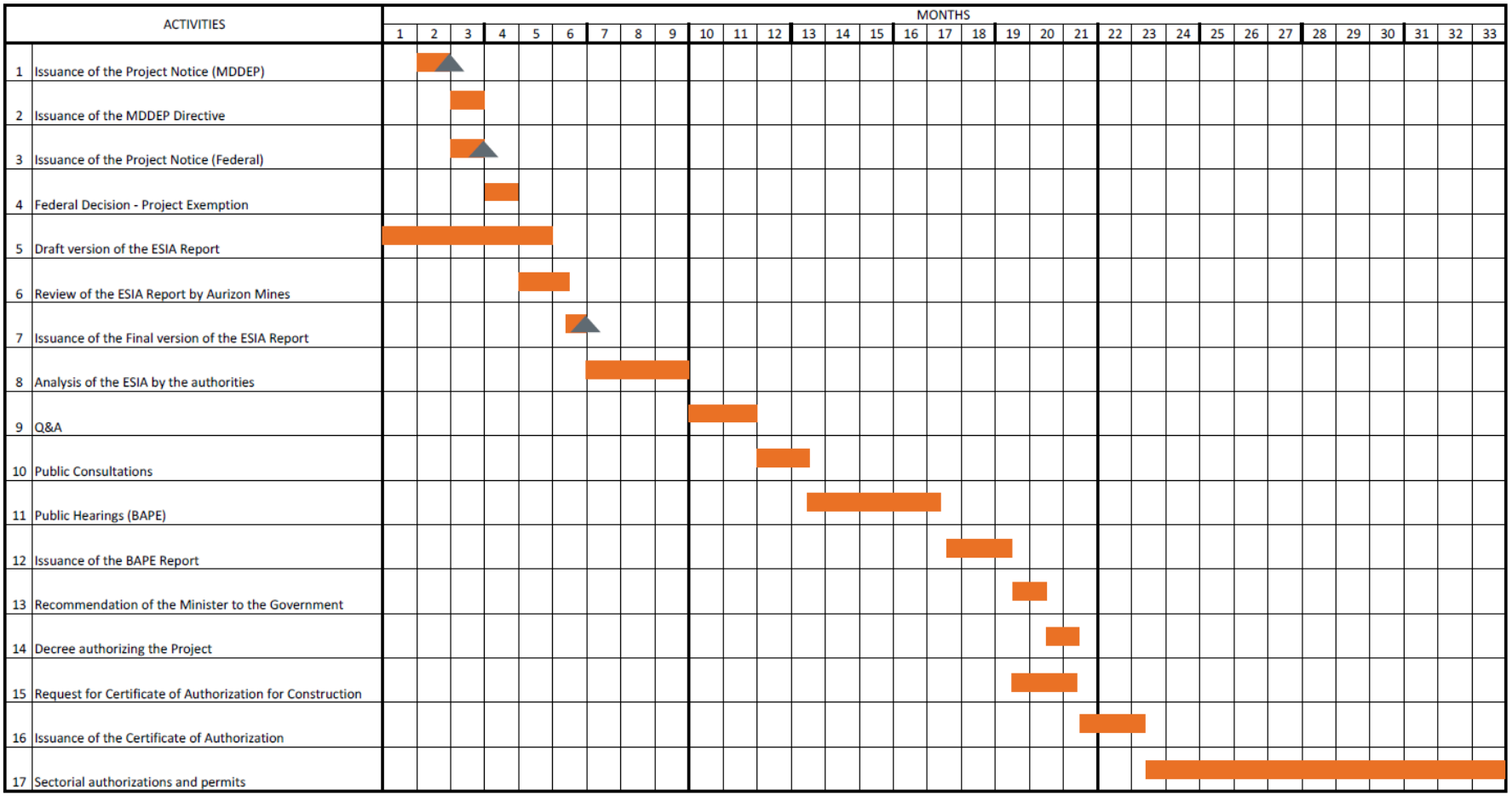


Figure 20.2: Schedule - Impact Assessment Process and Relevant Permitting

> **"*31.7.*** *Every decision rendered under Section 31.5 or 31.6 is binding on the Minister, where he subsequently exercises the powers provided in Section 22, 32, 55, 70.11 or in Division IV.2."*

In addition to the certificate of authorization required under Section 22 of the *Environment Quality Act*, the proponent must obtain permits, authorizations, approvals, certificates and leases required from the appropriate authorities that are described in the following sections.

The authorization application and permitting process is expected to take one year (Table 20.1.2). Applications may be filed concurrently with the construction work and should therefore not impact the critical project schedule.

20.2.1 Certificates of Authorization

Certificates of Authorization

To carry out the Joanna Project, one or more certificates of authorization (CAs) will be required from the MDDEP under Section 22 of the *Environment Quality Act* (EQA). A CA application includes a form that is attached to the documents and information set out in Sections 7 and 8 of the *Regulation Respecting the Application of the Environment Quality Act* (c. Q-2, r. 3). For mining activities, CA applications must also comply with Directive 019 requirements.

In addition, because the Joanna Project involves discharges into the aquatic environment, it will be necessary to complete the effluent discharge objectives application form (*Demande d'objectifs environnementaux de rejet (OER) pour les industries*) and attach it to the CA application. The CA application forms and all required documents must be sent to the MDDEP's Abitibi-Témiscamingue and Nord-du-Québec regional branch. The time required to analyse an application for a certificate of authorization directly depends on the complexity of the Project. Under the Declaration of Services to the Public, the Department is committed to providing an official response within 75 days following receipt of the application for a certificate of authorization or approval.

The number of CA applications to prepare will depend on the timeline for the Project activities and the associated work items. Dividing the Project into predefined items will enable a step-by-step implementation process.

Under the *Ministerial Order Concerning the Fees Payable Under the Environment Quality Act*, fees are payable by the enterprise seeking authorization under the Act. The fees established in the Ministerial Order are approximately $1,000 to $5,000 for each request for authorization under Sections 22, 31, 32, 48 and 70.8 of the Act.

Authorization to Establish Water Supply Intakes and Devices for the Treatment of Drinking Water and Wastewater

Authorization under Section 32 of the EQA is needed to establish drinking water and wastewater treatment facilities. Two forms must be completed and signed by the project engineer, and the required documents must be attached. The two forms are the application for authorization to carry out a water or wastewater system project (*Demande d'autorisation pour réaliser un projet d'aqueduc et d'égouts*) and the submission of applications for authorization for domestic wastewater treatment systems (*Présentation des demandes d'autorisation pour les systèmes de traitement des eaux usées d'origine domestique*). The required documents include administrative documents and a technical document signed by the project engineer. The application for authorization must be submitted to the MDDEP regional branch.

Authorization to Install Apparatus or Equipment to Prevent, Reduce or Cause the Cessation of the Release of Contaminants into the Atmosphere

Under Section 48 of the EQA, an application for authorization must be submitted for the installation of apparatus or equipment to prevent, reduce or cause the cessation of the release of contaminants into the atmosphere. The application for the authorization of an industrial project (*Demande d'autorisation pour un projet industriel*) must be completed and submitted to the MDDEP regional branch. The documents to be attached to the application are listed on the form.

20.2.2 Approval

Approval for the Location of the Process Concentration Plant and Mine Tailings Management Facility

Under Section 240 of the *Mining Act*, "Any person who intends to operate a mill for the preparation of mineral substances, a concentration plant, a refinery or a smelter shall, before commencing operations, have the site approved by the Minister or, where the Project is subject to the Environmental Impact Assessment and review procedure provided for in (Division IV.1 of Chapter I of the *Environment Quality Act* Chapter Q-2), by the Government." Section 241 of the same Act also states, "Every person responsible for the management of a concentration plant, refinery or smelter shall, before commencing activities, have the site intended as a storage yard for tailings approved by the Minister. The same applies to every holder of a mining right, owner of mineral substances or operator who intends to establish a mine tailings site." Consequently, a request for approval must be submitted to the Ministry of Natural Resources and Wildlife (MRNF) before activities begin at the Joanna Mine. This request must include the information and documents set out at Sections 124 and 125 of the *Regulation Respecting Mineral Substances other than Petroleum, Natural Gas and Brine*.

20.2.3 Attestation

Depollution Attestation

In accordance with Order in Council 515-2002 issued on May 1, 2002[5], the Joanna Mine project requires a depollution attestation from the MDDEP. This certificate, which is renewable every five (5) years, identifies the environmental conditions that must be met by the industrial establishment when carrying out its activities. The certificate compiles all the environmental requirements relating to the operation of an industrial establishment. The depollution attestation differs from the certificate of authorization issued under Section 22 of the *Environment Quality Act*. The latter is a statutory instrument issued prior to the implementation of a project or activity, whereas the former applies strictly to the operation of an industrial establishment. The steps in the depollution attestation process are described below.

Order in Council

The process for issuing a depollution attestation is implemented through the adoption by the Government of Québec of an order in council that subjects certain categories of industrial

[5] Respecting the application of subdivision 1 of Division IV.2 of Chapter I of the Environment Quality Act to the mineral industry and primary metal manufacturing sector.

establishments to Subdivision 1 of Division IV.2 of Chapter 1 of the *Environment Quality Act*. This Subdivision of the Act establishes the legal framework for the depollution attestation.

Application for a Depollution Attestation

The operator of an industrial establishment subject to an order in council must apply to the Ministry for a depollution attestation within a month of the start-up date. This application must be made using the form provided by the Ministry that identifies all of the required information.

First draft of depollution attestation:

The Ministry prepares and submits a first draft of the depollution attestation to the industrial establishment. The establishment then has 30 days to provide comments, as stipulated in the regulation.

Public Consultations

As stipulated in the regulation, the Ministry must publish a notice of public consultation in a daily or weekly newspaper within 90 days of the mailing date of the first draft of the depollution attestation. The Ministry must also make the request and attestation project accessible for public consultation. These consultations must take place over a period of at least 45 days. The establishment is also informed of the attestation project being submitted for public consultation.

Second Draft of Depollution Attestation

Following the public consultations, the Ministry reviews the comments that were received and prepares a second draft of the depollution attestation. The second draft is submitted to the industrial establishment, which has 30 days to provide comments.

Issuing of Depollution Attestation

The Ministry reviews the final comments provided by the industrial establishment and prepares the final version of the depollution attestation, which is issued to the industrial establishment for a period of five (5) years.

The establishment for its part is responsible for requesting a renewal of its depollution attestation six (6) months before it expires. The original certificate will remain in effect until a new certificate is issued.

20.2.4 Permits

Forest Management Permit for Mining Activities

Under Section 20 of the *Forest Act*, holders of mining rights can obtain forest management permits for mining activities in order to exercise their rights under the *Mining Act*. This permit authorizes the holder to cut timber on the land covered by the holder's mining rights for the construction of buildings or any other operations necessary to the holder's mining activities, in compliance with the *Forest Act* and its regulations. The applicant must have already obtained the right to operate the site for mining purposes, a right which is granted by the Mines Section of the MRNF. Prior to proceeding with timber cutting operations, the holders of mining rights must submit a written request to the MRNF forest management unit in order to obtain a permit for mining operations. The request can be for the clearing of a site for mining activities, the exploratory boring of a gravel bed or the clearing of a gravel or sand pit.

It is important to note that the holder of a forest management permit for mining activities must scale all timber harvested in forests in the domain of the State according to the standards prescribed by Government regulation as specified in Section 26 of the *Forest Act*. The holder is responsible for paying the prescribed dues as stipulated in Section 6 of the *Regulation Respecting Forest Royalties*.

High-Risk Petroleum Equipment Operating Permit

Under Section 120 of the *Safety Code*, "The owner of a petroleum equipment installation that includes at least one component that is high-risk petroleum equipment must obtain a permit for the use of all the high-risk petroleum equipment situated at the same address, until the equipment is removed from its respective place of use".

"High-risk" petroleum equipment is defined in Section 8.01 of the *Construction Code* as having one of the following characteristics:

- For underground storage systems:
  - Capacity of 500 or more litres, used to store gasoline or diesel;
  - Capacity of 4,000 or more litres, used to store heating oil and heavy fuel oil except for equipment used for heating a residential single-family dwelling;
- For above-ground storage systems:
  - Capacity of 2,500 or more litres, used to store gasoline;
  - Capacity of 10,000 or more litres, used to store diesel;
  - Capacity of 10,000 or more litres, used to store heating oil and heavy fuel oil except for equipment used for heating a residential single-family dwelling;
  - Storage tanks used to store gasoline, diesel, heating oil and heavy fuel oil for profit, regardless of their capacity.

The "Application for a Permit for the Use of High-Risk Petroleum Equipment" form must be completed and submitted to the Régie du bâtiment. The application must include all of the information and documents identified in Section 121 of the *Safety Code*. A permit is valid for 24 months. The issuing and renewal of a high-risk petroleum equipment permit is subject to compliance and performance monitoring under the provisions of the *Construction Code* and the *Safety Code*.

Explosives Permit

Under the *Act Respecting Explosives*, no person shall possess, store, sell or transport explosives unless he holds a permit for such purpose. Depending on the nature of their use, several permits are required for the possession of explosives for industrial or commercial purposes. Division II of the Regulation under the *Act Respecting Explosives* describes the different types of permits that are required. A general explosives permit entitles the holder to have explosives in his possession. Only the holder of a general permit can obtain a magazine, sales or transport permit. A magazine permit entitles the holder of a general permit to purchase and store explosives in a container or a building that complies with the regulations. A transport permit entitles the holder of a general permit to transport explosives.

To obtain these permits, the "Application for a General Explosives Permit" and "Application for a Sales, Magazine or Transport Permit" forms, available from the Sûreté du Québec, must be

completed. The required documents and fees must be submitted to the Sûreté du Québec. Permits are valid for a period of five (5) years.

20.2.5 Leases

Mining Leases

Under Section 100 of the *Mining Act*, "*no person may mine mineral substances, except surface mineral substances, petroleum, natural gas and brine, unless he has previously obtained a mining lease from the Minister […]".* In order to obtain a mining lease, a claim holder must establish the existence of indicators of the presence of a workable deposit. Applications must be submitted to the Registrar's Office or the regional office. The initial term of a mining lease is 20 years. The lease can then be renewed every 10 years for the duration of the mining operations. The procedure for obtaining a mining lease is described in the MRNF's online publication "Mining Leases and Concessions".

Non-Exclusive Lease for the Mining of Surface Mineral Substances

According to Section 109 of the Mining Act, "*a lessee or a grantee may use, for their mining activities, sand and gravel that is part of the domain of the State except where the land that is subject to the lease is already subject to an exclusive lease to mine surface mineral substances in favour of a third person*". The mining of sand and gravel located outside of mining leases requires a non-exclusive lease for the mining of surface mineral substances, under Section 140 of the *Mining Act*. The applicant must make a request for a non-exclusive lease by completing the "Application for Non-Exclusive Lease (BNE) for Mining Surface Mineral Substances" form and include the documents identified in Section 3 of the form.

Lease for the Occupation of the Domain of the State

Under Section 239 of the *Mining Act*, "*the holder of mining rights or the owner of mineral substances may, in accordance with the Act respecting the lands in the domain of the State (Chapter T-8.1), obtain that public lands be transferred or leased to him to establish a storage site for tailings, or a site for mills, shops or facilities necessary for mining activities*". Several components of the Joanna Project might be located outside of the lands covered by the mining leases. Since the project is located on public lands, the land in question will need to be leased under Section 47 of the *Act Respecting the Lands in the Domain of the State*.

20.3 Environmental Baseline Studies

Several comprehensive assessments were performed from 2007 to-date in order to define, in compliance with all applicable guidelines, policies, regulations and laws in Québec and Canada, the state of reference of the Joanna Project receiving environment. The following subsections aim at briefly presenting those works and associated results.

20.3.1 Physical Environment

A review of all existing information on local topography, regional hydrography (watershed) and geomorphology was done in order to precisely define the physical context in which the Joanna project will be implemented. Geotechnical investigations were also completed (see Section 20.6). The area is relatively flat as a result of past glacial activity which led to the deposition of a vast amount of clay (Ojibway-Barlow Lake). Soils are therefore essentially made of glaciolacustrine deposits (luvisols, gleysols). Those, combined with a flat topography and a relatively cold climate, have favoured the development of important peatlands. Surface clay deposits being subject to erosion and re-suspension, local and regional waters commonly have high turbidity. In some other areas, thin glacial (till) and/or glaciofluvial deposits are observed, on which grow tree species such as black spruce and jack pine. Soil quality analyses were also performed to assess baseline conditions. None of the sampled soils, which were located where surface infrastructures will be implemented, showed results exceeding applicable criteria for industrial use.

Groundwater quality was assessed by sampling water in existing wells used for drilling or as observation wells as part of the Hydrogeology Assessment Program. Results are presented in Section 20.3.1.1.6.

All waters from the mine site area run towards the Stitchman Creek. Downstream, waters flow to the Kinojevis River, a 140-km long river which flows south to the Ottawa River. Surface water quality was assessed from 2007 to-date by annual sampling stations implemented in order to cover the entire Stitchman Creek Sub-Watershed and downstream. Results were compared to applicable provincial and federal criteria and showed that surface waters are generally of good quality for the protection of aquatic life. However, some natural exceedances were noted for some metals, all of which are not surprising in an area with such mining potential. Sediment

quality was also assessed by annual sampling at those same stations. Results were also compared to applicable criteria and showed some exceedances in metal content, all of which, again, are not surprising in the area.

20.3.1.1 Hydrogeology Assessment Program

As for any typical mining project, the Joanna Mine Project is expected to influence the groundwater system in the vicinity of the project site. The magnitude and extent of the influence must be evaluated prior to the beginning of the project, since the site is located in an area where groundwater is used as a source of water supply, and because it may be linked to sensitive surface-water ecosystems.

20.3.1.1.1 Site Hydrogeological Investigation

In order to characterize the groundwater flow regime, detailed geotechnical and hydrogeological studies were conducted as part of the project in the pit area, and at the sites where the development of associated mining infrastructure is foreseen. Aurizon Mines mandated TechnoRem Inc. to perform a hydrogeological study of the Joanna Mine site for the present Feasibility Study (TechnoRem, 2010). In summary, the work carried out in 2010 consisted in:

- drilling of 12 deep boreholes in the bedrock;
- four (4) short-term and five (5) long-term pumping tests in the boreholes;
- four (4) variable-head hydraulic tests in the boreholes; and
- an exhaustive groundwater sampling and analysis program in the boreholes and surrounding private wells.

The objectives of the program were to collect data, primarily on the bedrock hydrogeological system, in order to model and predict mine dewatering rates and the impact of operations on residential water wells, an esker located 4 km east, and the Vaudray and Joannès Lakes Biodiversity Reserve.

A supplementary program was carried out in 2012 in order to collect additional data, this time on the spatial distribution and hydraulic properties of the surface geological deposits in the vicinity of the proposed pits. The work consisted in:

- drilling and installation of 9 shallow monitoring wells in the surface geological deposits;
- variable-head hydraulic tests (*slug tests*) in each new monitoring well;
- groundwater level survey.

The data collected on groundwater levels in the shallow deposits was intended primarily to improve the prediction of groundwater surface configuration for pit slopes design.

20.3.1.1.2 Hydrostratigraphy

The information available on the project area shows the occurrence of two (2) main hydrogeological systems. The first system, of a confined to semi-confined type, is encountered deep in the bedrock formations, while the second system is encountered near the surface, and presents potentially semi-confined to unconfined conditions.

Hydrogeological System in Bedrock

As previously described, the bedrock in the project area is part of the Superior Geological Province of the Canadian Shield (Abitibi and Pontiac sub-provinces). The rock formations consist in volcano-sedimentary rocks with some intrusive rocks (granitoids, mafic and ultramafic dykes and sills). At the project site, the main geological units consist in a greywacke including beds of siltstone, argillite, and arkose; and a greywacke including polygenetic conglomerate units. The sedimentary rocks have experienced metamorphism.

The regional Cadillac Fault crosses the project site at the pit location. Observations in drillholes intersecting the faults and mineralogy of the latter suggest that it does not act as a significant groundwater flow corridor.

The type of rock formations encountered at the project site are typically considered as low-permeability rocks. These rocks display little primary porosity in the matrix, but a more important secondary porosity associated with fracture and joint systems. Groundwater flow within the rock mass is controlled primarily by the fracture network patterns and connectivity. At the pit location, the surface of the bedrock is weathered over 10 to 20 m deep, but of good quality deeper.

The bedrock formations are the main aquifer[6] in the project area. They typically allow the installation of low-yield wells that can supply water for private homes. These conditions are found in the project area.

Hydrogeological System in Surface Geological Deposits

The project area is covered by glacial deposits from the Wisconsinian geological period, notably till deposits. Mostly in the southern area of the proposed pit, and also in the northern part of the project area, the bedrock outcrops, as numerous small hills. Additionally, glaciolacustrine deposits are widespread in the project area. They consist in successive layers of fine to coarse sediments overlaid by peat deposits. An esker is also reported some four (4) km east of the proposed pit. Because of the coarse nature of the sediments typically found in eskers, it is assumed that it acts as a source of recharge of Vaudray and Joannès lakes.

At the project location, geological data has allowed the determination of the site stratigraphy (TechnoRem, 2010). Typically, peat deposits of less than 2 m thick overlay an interbedded clayey silt to silty clay layer. This layer was intersected almost in all boreholes, and the thickness of these clay and silt deposits can reach up to 60 m in some areas. A slightly coarser layer of about 10 m thick, composed of silt and sandy silt, is often encountered beneath the clay and silt deposits. Little data is available on the spatial distribution of this layer. Lastly, a discontinuous coarser layer of silty sand, which directly overlies the bedrock, was found in some boreholes.

The shallow hydrogeological system is located in the more permeable layers of the surface geological deposits, encountered near the surface and the contact with the bedrock. The bulk porosity and hydraulic conductivity of sandy and silty deposits are often higher than those measured in bedrock formations. Groundwater flow within the deposits mass is controlled by the primary porosity of the matrix, which can allow storage of significant quantities of water.

In order to act as an aquifer, the sand and silt deposits must have significant thicknesses and extents. This does not seem to be the case in the vicinity of the project site. No private wells

[6] A saturated geological unit that produces an economically feasible quantity of water to a well or spring.

were reported to be installed in sand or silt deposits in the project area. However, the more permeable silt layers near the surface may play a significant role in the local surface hydrology, since they can be connected with surface water bodies, such as the Stitchman Creek.

In contrast, the clay layer near the surface acts as an aquitard[7]. When present, it confines and protects the bedrock aquifer.

20.3.1.1.3 Hydraulic Properties

The results of the various hydraulic tests performed at the site were used to estimate the main hydraulic properties of surface deposits and bedrock formations, i.e., the hydraulic conductivity (permeability), transmissivity and storage coefficient. The results confirm the low hydraulic potential of the sandy-silt and bedrock formations, and values are comparable to those generally reported in the literature for such units (Freeze and Cherry, 1979). Table 20.1 summarizes the results of the various hydraulic tests.

[7] A confining bed that retards, but does not prevent, the flow of water to or from an adjacent aquifer. It does not readily yield water to wells or springs, but can serve as a storage unit for groundwater.

Table 20.1: Hydraulic Properties of Bedrock Formations and Surface Geological Deposits (adapted from TechnoRem, 2010)

Bedrock Formations				
Statistical Parameter	**Transmissivity**	**Hydraulic Conductivity**	**Storage Coefficient**	**Radius of Influence**
	m^2/d	m/s	-	m
Maximum	29.8	3.2×10^{-6}	2.2×10^{-3}	570
Minimum	1.3	3.6×10^{-8}	1.2×10^{-4}	130
Geometric Mean	5.6	4.0×10^{-7}	3.0×10^{-4}	322
Surface Geological Deposits				
Maximum	-	5.2×10^{-4}	-	-
Minimum	-	3.8×10^{-6}	-	-
Geometric Mean	-	3.4×10^{-5}	-	-

The bedrock formations display hydraulic conductivities ranging between 3.6×10^{-8} m/s and 3.2×10^{-6} m/s. Transmissivity values are in the range of 1.3 m²/d to 29.8 m²/d. Storage coefficients are estimated between 1.2×10^{-4} and 2.2×10^{-3}. These results indicate that the bedrock formations are under confined to semi-confined conditions, and that the low transmissivity is representative of a small-capacity aquifer.

The hydraulic testing performed in the shallow wells has also allowed the evaluation of the hydraulic properties of the surface deposits. Hydraulic conductivity values ranging between 3.8×10^{-6} m/s and 5.2×10^{-4} m/s are estimated in the deposits. The variation in the results is probably related to the nature of the deposit tested, i.e. according to the variable proportions of clay, silt and sand, characteristic of these heterogeneous deposits.

20.3.1.1.4 Groundwater Flow Regime

The results from the hydrogeological investigations carried out at the project site were used to define the components of the groundwater flow (TechnoRem, 2010). Knowledge of the different groundwater flow components at the project site will allow the identification of sensitive areas, such as recharge and discharge zones, and the evaluation of the migration paths for potential contaminants in the groundwater.

Drilling activities performed in 2010 in the pit area showed that the depth of the water table was between 0.40 m and 2.04 m in the bedrock aquifer (TechnoRem, 2010). Data collected in the new shallow wells in 2012 have shown groundwater level depths ranging between 0.91 m and 2.39 m. These results suggest that the piezometric surface follows relatively well the topography of the site, with respect to the shallow and deep hydrogeological systems.

Data collected during the water level survey conducted in July and August, 2010 were used to prepare a preliminary groundwater piezometric map for the bedrock aquifer (TechnoRem, 2010). The piezometric map suggests that regional groundwater flow direction is mainly towards the west; following the natural course of the Stitchman Creek. At the project site, groundwater flow is concentric, and flows naturally from higher-elevation grounds surrounding the sites towards the proposed pits location. With such a pattern, the flow is presumably controlled by the groundwater level, where the recharge is done directly by precipitation in bedrock outcrop areas and/or permeable surface deposits setting topographic highs and hills.

20.3.1.1.5 Classification of Hydrogeological Units

The status of a groundwater resource and its relative value varies depending on a number of criteria related to its hydrogeological properties and potential uses. The MDDEP groundwater classification system is a tool to ensure the reconciliation of the uses of the territory with those of groundwater resources according to the following classes (MENV, 1999).

The main classes defined within this system are:

- Class I: Aquifer that is an irreplaceable source of water (single source of water supply or supplying a community);
- Class II: Aquifer that is a common source (II a) or a potential source (II b) of water because of its acceptable quality (drinking water with usual treatment) and sufficient quantities (transmissivity greater than 1 m^2/d);
- Class III: Aquifer that, although saturated with water, cannot constitute a water supply (III a and III b: poor quality, insufficient quantity or non-economic extraction).

According to these definitions, the bedrock aquifer at the mine site is a Class II aquifer that can be used to supply small wells. The source of water for the residents, mainly located south of the site along provincial road 117, is provided by bedrock private wells.

Surface geological deposits could represent aquifers of Class I or II, insofar as they are of sufficient thickness and extent to serve as a source of supply of interest. This is certainly not the case at the project site, but the esker located four (4) km west could be a Class I aquifer although it is beyond the area influenced by the project.

20.3.1.1.6 Geochemical Characteristics of the Groundwater

The geochemical characteristics of the groundwater were established from the results of an exhaustive sampling campaign and analytical program (TechnoRem, 2010). Groundwater samples were collected in private wells, as well as in the test and monitoring wells specifically installed for the current project.

The samples were analyzed for the following physico-chemical parameters: pH, temperature, electric conductivity, hardness, Eh Potential, dissolved oxygen, turbidity, colour, odour, major ions (Ca, Mg, Na, K, HCO_3, Cl, SO_4), minor ions (Fe, F, Mn, nitrates and nitrites), metals (Al, Sb, As, Ba, B, Cd, Cr, Cu, Hg, Pb, Se); and cyanides. Groundwater from private wells was also analyzed for microbiological parameters. The results of the chemical analyses are described, hereafter.

The results of the groundwater samples collected in all monitoring and test wells, five (5) at the site location, present concentrations below all criteria included in *Guideline 019* of the MDDEP for the mining industry, except for iron and arsenic that are exceeding the criteria in two (2) wells. All other physico-chemical parameters present concentrations well below the criteria.

Groundwater was also sampled in eleven (11) private wells. The results were compared against Québec drinking water regulation (*Règlement sur la qualité de l'eau potable – RQEP*) and Canadian standards. Groundwater respects the criteria for almost all parameters, except for iron and hardness. In one case, arsenic criterion for drinking water was exceeded; such exceedance is not surprising considering that other natural exceedances were observed in the past for

private wells in the region. Hardness is very high, which explains the use by owners of water-softener systems.

Groundwater samples collected in a well installed in the esker located four (4) km east of the site was also analyzed. The groundwater chemical content exhibits concentrations below the MDDEP drinking-water criteria (RQEP) for all parameters analyzed, except iron, manganese and hardness. In this case, the hardness is very low.

The groundwater sampled in the private wells and the wells drilled for the project is representative of the groundwater quality in the bedrock aquifer. The well in the esker presents the geochemical characteristics of a distinct hydrogeological system. Iron, manganese and hardness do not represent a hazard for human health, but are rather esthetical criteria. Only the occurrence of arsenic, detected in a few samples, represents a threat for health, and should be monitored if groundwater is intended to be used as a source of drinking-water supply.

20.3.1.1.7 Groundwater Vulnerability

The assessment of groundwater vulnerability to a potential contamination from human activities is an approach used to implement, if necessary, measures for the protection of the resource. One of the approaches suggested by the MDDEP to assess groundwater vulnerability is the DRASTIC method (Aller et al., 1989). The method is applied for the protection of an aquifer, which is the bedrock aquifer in the current project.

The vulnerability of groundwater in the bedrock aquifer depends highly on the nature and hydrogeological characteristics of the overlying surface deposits. Indeed, many of the parameters required to estimate the DRASTIC index are associated with the geological layer overlying the aquifer. There is currently little information on the spatial distribution of the hydraulic properties of the surface deposits. Accurate data on hydraulic characteristics and thickness of the deposits are required to calculate the DRASTIC index. The information collected at the site is primarily related to the bedrock, although it will be completed for surface deposits in the current year (2012). However, it is possible to obtain a first qualitative estimate of the vulnerability, namely when using a map showing the distribution of surface deposits.

Areas characterized at the surface by the occurrence of low-permeability clayey and silty deposits will typically present a lower vulnerability. A higher vulnerability will characterize areas of bedrock outcrops and those where the thickness of the deposits above the rock is low. A better knowledge of the spatial distribution of groundwater vulnerability will be obtained as a result of the complementary hydrogeological characterization work to be carried out in 2012, which will address geotechnical issues concerning hydraulic characteristics of surface deposits and groundwater depth.

20.3.1.1.8 Users of the Resource

The project site is located along Provincial road 117, some 20 km east of Rouyn-Noranda. About 50 private homes have been identified along the road within a 5-km radius around the project site. Some of them are located less than 1.5 km from the site.

Since there is no aqueduct in the vicinity of the project site, all private homes are assumed to be equipped with an individual well. Groundwater is used for these homes as the main drinking-water source, as well as for cooking, washing, and laundry. Several houses are equipped with a water-softener system.

As described earlier, the quality of the groundwater is generally good. High content in iron and manganese are measured in the groundwater, as well as a very high hardness.

20.3.1.1.9 Impact and Issues on Groundwater Resources

Mine Dewatering

The development of the mine will required the dewatering of the pits. The occurrence of pit deposits and potentially saturated silt and sand layers may represent a challenge for dewatering. Pumping activities will influence the groundwater system, notably the groundwater level elevation in the vicinity of the mining pits. The results will be to lower down the water table, which can affect nearby private wells and baseflow in surrounding streams and lakes.

In order to assess the impact of dewatering activities, mathematical modeling of the successive mine operation phases was performed to estimate the daily flow rate required to keep the pits

dry and to evaluate the area and extent of the zone affected by the groundwater drawdown (TechnoRem 2010 and 2012). The modeling results indicated that an average rate of 8,100 m^3/d should be maintained for operation of the mine. The radius of influence of the dewatering activities will potentially affect an area of up to approximately 1 km from the pit limits.

Impact of Operations on Residential Water Wells

Dewatering impact on residential wells is sensitive to stakeholders. According to the results of the hydrogeology study performed in 2010 and 2012 by TechnoRem, the operation of the Joanna mine should not impact residential water wells in the area, in terms of either water supply or quality.

Moreover, groundwater quality and level will be monitored to ensure residential wells are not affected by mining operations.

Impact of Operations on the Esker

A north-south trending esker is located 4 km east of the future pit location. It represents a potential link between the two protected land areas, a potential resource of freshwater, and the contingency plan for the water source of the Rouyn-Noranda community. As a result, the possible impact of the operation on the esker is a sensitive point for the stakeholders.

According to the results of the hydrogeological study (TechnoRem 2010 and 2012), the operation of the Joanna mine will have no impact on the esker.

- The esker is upstream from the operation;
- All infrastructure and future pit operations are currently located along a different watershed than the esker.

Impact of Operations on the Vaudray-and-Joannès-Lakes Biodiversity Reserve

The Vaudray-and-Joannès-Lakes Biodiversity Reserve is located about 1.2 km south of the future pit. As a result, the possible impact of the operation on this protected area is a sensitive point for stakeholders.

The hydrogeological study established the piezometric map for the project area. A watershed boundary separates Joannès Lake and the Joanna site. Joannès Lake and the Joanna site are therefore located in two separate watersheds and there is no hydraulic link between the two locations. The drawdown of the water table from mining operations will not impact the water level of Joannès Lake.

Impact on Groundwater Quality

In addition to a local change around the pit itself, no effect is expected on the groundwater quality. Dewatering activities will also act as a hydraulic trap that will prevent the migration from the site of any potential contaminant in the area influenced by pumping. Water from the pit will be treated before being returned into the environment.

Furthermore, a groundwater monitoring network will be implemented at the mining site in the vicinity of the potential source of contamination.

20.3.2 Biological Environment

A study area covering the entire Property was established in order to assess biological conditions prevailing prior to the implementation of the Project. Plant communities were mapped within that area; results indicate that the Property is essentially covered by softwood stands (about 55%; mainly black spruce moss stands) followed by wetlands (about 25%; mainly bogs). No special-status plant species were found within the Property, even if special attention was paid to their presence.

Most recently, the Government of Québec adopted Bill 71, namely the *Loi concernant des mesures de compensation pour la réalisation de projets affectant un milieu humide ou hydrique* (published in French only) which states that:

> *"[...] the Minister of Sustainable Development, Environment and Parks [MDDEP] can require the project proponent to implement compensation measures aiming at, among other things, restoring, protecting or valorizing a wetland, an aquatic or a terrestrial environment."* [Translation done by the author]

Therefore, Aurizon Mines Ltd will have to compensate for losses of wetlands caused by the implementation of its Joanna Project. Further discussions with the MDDEP are necessary in order to establish how those losses will be compensated, but to-date, several scenarios were proposed and some were deemed acceptable by the MDDEP. However, the precise evaluation of the selected scenario will be completed as part of the Environmental and Social Impact Assessment process.

A vast review of all existing information on wildlife in the study area was also completed. That led to the mapping of all legally-protected wildlife habitats (i.e. white-tailed deer yard, waterfowl gathering area, fish habitats, etc.). Additional field surveys were done to precisely map all fish habitats within the Property limits. A list of wildlife species occurring or potentially occurring within the study area was established; about 30 mammal species and 35 waterfowl and grouse species have the potential to be present within the Property limits according to this information. More detailed surveys with regards to forest birds are now ongoing. Among those species are special-status wildlife species with the potential to appear in the area, such as the bald eagle, peregrine falcon, short-eared owl, least weasel, two bat species and two micromammal species. It is however important to note that none of those species was ever observed in the area.

20.3.2.1 Fisheries and Fish Habitats

As previously mentioned, and in Section 20.7, all mine infrastructures are located within only one (1) sub-watershed, namely Stitchman Creek. In fact, the POX Tailings Management Facility (POX-TMF) is located outside that area, but all waters coming from that facility will be managed in such a way that it will be pumped to the Process Water Polishing Pond (PWPP) and then discharged with the final effluent within the Stitchman Sub-Watershed. No water discharge is located outside the Stitchman Sub-Watershed. Therefore, within that area and where there was the potential to locate any of the required surface infrastructures, a comprehensive Fisheries and Fish Habitats Assessment Program was performed to precisely locate all sections of the Stitchman Creek and its affluents which are to be considered as fish habitats according to Canada's *Fisheries Act*.

The objective of such work was to limit the ecological footprint of the Project to areas located outside those habitats. Among other things, the use of thickened tailings was also prioritized in order to minimize the footprint of the Tailings Management Facility.

As depicted in Section 20.1, the performed Fisheries Assessment Program showed that the Stitchman Creek and its affluents, where considered as fish-bearing creeks, host species such as brook stickleback, pearl dace (both totalling about 99%) and catfish. None host fish species of importance to any ongoing productivity of commercial, recreational or Aboriginal fisheries. Such information might be of interest in line with the most recent Bill C-38 which enumerates factors to be taken into account before recommending to the Governor in Council that a regulation be made in respect of section 35 of the *Fisheries Act*. Among those are "the contribution of the relevant fish [species] to the ongoing productivity of commercial, recreational or Aboriginal fisheries and whether there are measures and standards to avoid, mitigate or offset serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or that support such a fishery." Such further analysis will have to be done with the Department of Fisheries and Oceans Canada (DFO) as part of the Environmental and Social Impact Assessment process. Still, considering the proposed site general arrangement plan, only the location of the proposed Hosco and West pits will require the deviation of existing fish-bearing watercourses. No other encroachment in fish-bearing watercourses is required. Among those two (2) watercourses that will have to be deviated is a ditch implemented as part of the past operating Hosco mine. Following preliminary discussions with the DFO, the deviation of those two (2) watercourses would not require a license under Section 35 of the *Fisheries Act*. Indeed, works proposed by Aurizon Mine and Roche Ltd to deviate those watercourses have, for now, been considered sufficient to mitigate/offset the impact on fish.

Results depicted in Section 20.3.1.1 show that the most recent hydrogeology assessment works indicate that rocks within the pit area are lowly permeable, and that the anticipated drawdown would not result in decreased water flows/levels within the nearby Stitchman Creek.

20.3.3 Past Environmental Impact Assessment Works

Following the issuance by Aurizon Mines of a Pre-Feasibility Study (PFS), several works were completed in order to further assess the potential impact of the Project, as described in the PFS.

Among those were the impact on soils, surface water, sediments, groundwater, air quality, noise levels, vibrations, landscape, fisheries and fish habitats, terrestrial habitats and wetlands, amphibians and reptiles, waterfowls, prey and forest birds, and terrestrial wildlife. This information will be updated using the Project Description included in the present Study as part of the Environmental and Social Impact Assessment process. Most information will be used without any major modification, while others will have to be significantly updated to fit in with the actual Joanna Project.

20.4 Tailings Characteristics and Management

20.4.1 Considerations for Tailings Management

Directive 019 outlines MDDEP requirements with regards to mine tailings characteristics (low risk, leachable, acid-generating, cyanide-containing, and high risk).

The requirements specify leak-proofing measures to be applied to mine tailings management facilities in terms of the materials stockpiled, thereby ensuring groundwater protection. Figure 20.3 specifies the criteria to be considered in determining the leak-proofing measures.

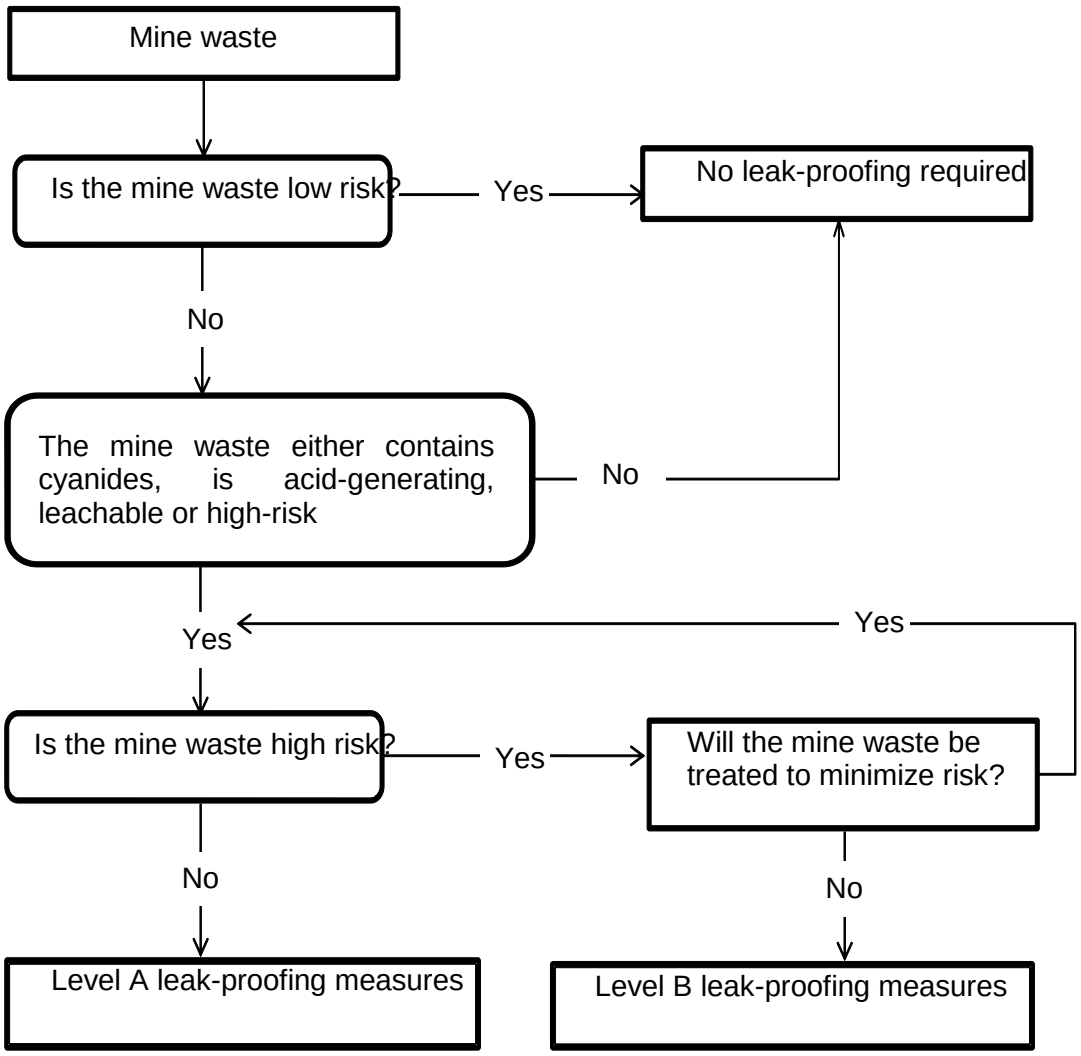
The results of total metal analyses and leaching tests conducted on ore show that it is acidogenic and leachable, especially for arsenic, which is found in high concentrations in the ore. The tailings produced from ore processing are therefore also acidic and leachable under Directive 019.

Processing this type of ore occurs in two stages. The first stage entails the flotation of sulphides associated with gold, and aims to produce a sulphide concentrate. The second stage involves treating sulphide concentrates by way of cyanidation to recover gold. Normally, the waste produced at each stage is combined and placed in tailings.

For the Joanna Project, the objective is to incorporate environmental protection measures in a context of sustainable development as soon as the initial phase of the Project is complete. It has, therefore, been decided to manage tailings separately. Since arsenic and sulphur are directly related to gold principally in the form of arsenopyrite, the goal is to produce de-sulphurized tailings through step flotation of sulphides (~95% of tailings). These tailings contain

little arsenic and sulphur and can therefore be disposed of in a conventional tailings facility, which can be restored by way of conventional methods. The other type of tailings (~5% of tailings) is leachable and requires protective measures specified under Directive 019.

The goal is to avoid having a small amount of tailings that represent an environmental risk from making all the tailings a risk to the environment. Laboratory tests have demonstrated the feasibility of recovering gold optimally by producing both desulphurized (flotation) and sulphurized (POX) tailings in a proportion of about 95% and 5% respectively.



Source: *Directive 019 for the mining industry - March 2012*

Figure 20.3: Criteria for Determining Leak-Proofing Measures to be Applied to Tailings Accumulation Areas

20.4.2 Flotation Tailings

20.4.2.1 Characterization of Flotation Tailings

Samples

Eleven samples of flotation tailings were collected and characterized at the SGS Lakefield pilot plant in 2012. The samples were generated following ore processing from various locations in the deposit. The mean values can therefore be calculated in order to obtain a good representation of the whole flotation tailings.

Acid Rock Drainage (ARD) Potential

The average sulfides and sulfur contents were 0.05% and 0.07%, respectively. The tailings are therefore *de facto* classified as non-ARD according to the Directive 019 criterion (0.3%). The average neutralization potential (NP) is 19.2 kg $CaCO_3$/t and the average net neutralization potential (NNP) is + 17.7 kg $CaCO_3$/t, which is slightly below the Directive 019 criterion (+ 20 kg $CaCO_3$/t) applying to mine waste with sulfur content > 0.3%. However, the average NP/AP ratio (13.1) is largely higher than the Directive 019 criterion (3.0).

Leaching Potential

The average concentrations measured in the leachates obtained with the TCLP, SPLP and CTEU-9 leaching tests were largely lower than the groundwater quality protection criteria for most of the parameters, with the exception of arsenic, aluminum and manganese.

The average arsenic concentrations measured with the TCLP test (0.22 mg/L), the SPLP test (0.24 mg/L) and the SFE test (0.33 mg/L) were lower than the groundwater protection criterion (0.34 mg/L). Flotation tailings are thus not classified as "leachable" for arsenic because the average concentration measured with the TCLP test was lower than the groundwater protection criterion.

The average aluminum concentrations measured with the TCLP test (0.83 mg/L) and the SPLP test (1.27 mg/L) were higher than the groundwater protection criterion (0.75 mg/L). However, the average aluminum concentrations obtained with the SFE test (0.66 mg/L) were lower than the groundwater protection criterion.

During the tests carried out at the *Unité de recherche et de service en technologie minérale de l'Abitibi-Témiscamingue* (URTSM) in 2011 on the same type of flotation tailings (sulfur content = 0.5%, NP = 22.6 kg$CaCO_3$/t), the aluminum concentration measured in the leachate from the TCLP test was 0.475 mg/L. However, leachates collected from the kinetic test (column) were always lower than 0.01 mg/L. The results from the kinetic test are more representative of the natural field conditions. Considering results from the kinetic test and the SFE static test, it can be concluded that flotation tailings are not likely to leach aluminum.

The average manganese concentrations measured with the TCLP test (2.2 mg/L) were slightly higher than the groundwater protection criterion (1.9 mg/L). However, the average concentrations obtained with the SPLP test (0.015 mg/L) and the SFE test (0.006 mg/L) were largely lower than the groundwater protection criterion.

During the tests carried out at the URTSM in 2011 on the same type of flotation tailings, the manganese concentration measured in the leachate from the TCLP test was 3.62 mg/L. However, leachates collected from the kinetic test (column) were always lower than 0.01 mg/L and these results are more representative of the natural field conditions. Considering the results from the kinetic test as well as those from the TCLP and SFE static tests, it can be concluded that flotation tailings are not likely to leach manganese.

20.4.2.2 Flotation Tailings Management Facility Design

Aurizon Mines decided to thicken the mine tailings so as to limit the area required for the tailings management facility, and to avoid encroaching upon streams considered to be a fish habitat.

As noted earlier, tailings resulting from the flotation step are considered low risk under Directive 019, which means no leak-proofing measures are required, as specified in Section 2.9.4 of Directive 019:

> *"Low-risk tailings accumulation areas do not require any leak-proofing measures for the protection of groundwater."*

The tailings management facility capacity was estimated at 38 Mt (40 Mt x 95%). At an estimated density of 1.35 t/m³, this represents a volume of 28.2 Mm³.

Geotechnical studies of the site will need to be conducted to specify dike location and design.

The main characteristics of the Flotation Tailings Management Facility are:

Approximate capacity	29.3 Mm^3
Surface area	2.1 Mm^2
Maximum height of dikes	23.8 m
Length of dikes	5,765 m
Slope of dikes	2.5 H :1 V
Volume of dike to be built	5.9 Mm^3
Crest width of dikes	6 m
Freeboard	N/A
Tailings deposit slope	3%

The area will be cleared. Stumps and surface vegetation will be removed where the dikes are to be installed, and a geotextile liner put in place to prevent particles from being transported under the dikes.

The dikes will be built with crushed neutral waste rock. These permeable dikes will allow water to filter through and be collected in ditches around the tailings pond. Additional studies are required to validate the use of mine tailings and particle sizes required to meet filtering criteria.

The drainage ditch encircling the tailings pond will be connected to a 450,000 m^3 pond. Pond water will be recirculated to the concentrator. The pond will also be equipped with an emergency spillway.

Given the lack of adequate geotechnical data, a dike slope of 2.5: 1 was considered, but this slope will also need to be validated at the detailed engineering stage by geotechnical studies.

To minimize construction costs, the Flotation Tailings Management Facility will be developed in three stages. Given the lack of adequate geotechnical data, the downstream construction method was selected. Construction costs for that infrastructure are estimated at $53.5 million. Table 20.2 summarizes construction costs for each step.

Table 20.2: Flotation Tailings Management Facility Cost Estimate

DESCRIPTION	QUANTITY	UNIT	UNIT PRICE	TOTAL COST
Step 1 (years 0 - 4)				
Deforestation - timber rights	Technominex prices		234 455.00 $	234 455.00 $
Deforestation - chipping	Technominex prices		34 500.00 $	34 500.00 $
Grubbing	27	ha	3 000.00 $	81 000.00 $
Geotextile 918	271 180	m²	3.00 $	813 540.00 $
Crushed waste rock (0 -150 mm) for dike construction	1 664 970	m³	8.50 $	14 152 245.00 $
Drainage ditch with no geomembrane and no riprap	1 575	m.l.	6.00 $	9 450.00 $
Drainage ditch with no geomembrane but with type 1 riprap	3 250	m.l.	25.00 $	81 250.00 $
Excavated material for SS polishing pond	450 000	m³	5.00 $	2 250 000.00 $
			Step 1 cost	17 656 440.00 $
Step 2 (years 4 - 8)				
Grubbing	3	ha	3 000.00 $	9 000.00 $
Geotextile 918	30 900	m²	3.00 $	92 700.00 $
Crushed waste rock (0 -150 mm) for dike construction	1 626 841	m³	8.50 $	13 828 148.50 $
			Step 2 cost	**13 929 848.50 $**
Step 3 (years 8 - 12)				
Grubbing	4.3	ha	3 000.00 $	12 900.00 $
Geotextile 918	43 000	m²	3.00 $	129 000.00 $
Crushed waste rock (0 -150 mm) for dike construction	2 561 072	m³	8.50 $	21 769 112.00 $
			Step 3 cost	21 911 012.00 $
			TOTAL	**53 497 300.50 $**

20.4.3 Tailings Produced by POX Process

The gold, arsenic and sulphur concentrate produced in the flotation stage will be treated using the POX process. Cyanide will be directly destroyed in the concentrator to reduce the cyanide concentration to less than 20 ppm prior to discharge in the POX Tailings Management Facility.

20.4.3.1 Characterization of POX Tailings

Samples Description

Four (4) samples of POX tailings (liquid and solid) have been collected at the Sherritt International Corporation installation in Alberta and characterized at Maxxam and Corem laboratories in 2011. Two (2) samples have been submitted to a neutralisation process in which part of the sulfide is oxidized to sulphate.

Liquid Phase of the Tailings

Liquid samples showed high mean concentrations of arsenic (4.9 mg/L), molybdenum (2.43 mg/L), cobalt (0.30mg/L) and copper (0.20 mg/L).

Solid Phase of the Tailings

Metal content

The mean arsenic content is especially high (26 500 mg/kg). Other metals showing mean content exceeding Criteria A of the *Politique de protection des sols et de réhabilitation des terrains contaminés* (PPSRTC) includes cobalt (182 mg/kg), copper (633 mg/kg) and nickel (617 mg/kg).

Acid Rock Drainage (ARD) Potential

The mean sulfides and sulfur contents were 2.98% and 1.48%, respectively. The mean acidification potential is 46.9 kg$CaCO_3$/t and the average neutralization potential (NP) is 79.5 kg$CaCO_3$/t. The average net neutralization potential (NNP) is + 32.6 kg$CaCO_3$/t, which is above the Directive 019 criterion (+ 20 kg$CaCO_3$/t). However, the average NP/AP ratio (1.7) is lower than the Directive 019 criterion (3.0). According to Price (2009) classification, this waste would be in the uncertainty zone.

Leaching Potential

The average arsenic concentrations measured with the TCLP test (4.14 mg/L) is below the criteria (5,0) found in Appendix II, Table 1, of Directive 019 and is therefore not classified as "high risk". However, the concentrations obtained with the SPLP test (5.48 mg/L) and the CTEU-9 test (8.03 mg/L) are higher than the criteria.

The average nickel concentrations measured with the TCLP test (11.4 mg/L) is also very high, but there are no criteria for this parameter in Appendix II, Table 1, of Directive 019.

Directive 019 stipulates that, for a given parameter, if the content is higher than Criteria A of the PPSRTC, and if the TCLP leachate shows concentrations higher than the groundwater quality protection criteria, the sample is classified as "leachable". Considering the groundwater quality protection criteria of 0.34 mg/L for arsenic and 0.22 mg/L for nickel, it can be concluded that POX tailings is a "leachable" mining waste.

Moreover, arsenic content (26 500 mg/kg) is largely higher than Criteria C of the PPSRTC (50 mg/kg) and arsenic concentrations in the liquid phase (4.9 mg/L) are largely higher groundwater quality protection criteria (0.34 mg/L). In this context, this material must be managed as a high risk mining waste.

These tailings will therefore be disposed of in a separate tailings management facility that meets level B leak-proofing criteria indicated in Directive 019.

20.4.3.2 POX Tailings Management Facility Design

To comply with Level B leak-proofing criteria specified in Directive 019, the bottom and walls of the POX Tailings Management Facility shall be protected by two (2) synthetic watertight liners.

The required POX Tailings Management Facility capacity is estimated at 3.75 Mt (40 Mt x 5% x 1.88 [increased tonnage factor for autoclave process]). At a density of 1.2 t/m^3, this represents a volume of 3.1 Mm^3.

The infrastructure is located in an area for which no detailed topographical information is available, and where very little geotechnical drilling has been done. A topographical survey of the area and additional geotechnical studies would help optimize its location and management.

The main characteristics of the POX Tailings Management Facility are:

Approximate capacity	3.1 Mm^3
Surface area	0.37 Mm^2
Maximum height of dikes	13 m
Length of dikes	2,437 m
Slope of dikes	2.5 H : 1V
Volume of dike to be built	1.7 Mm^3
Crest width of dikes	4 m
Freeboard	2 m
Tailings deposit slope	N/A

Its effluent will undergo a ferric sulphate treatment to lower arsenic concentrations to below the Directive 019 criterion (0.2 mg/l). The treated water will then flow through the Process Water Polishing Pond before being discharged into the Stitchman Creek.

The POX Tailings Management Facility will be fenced in to prevent animals from drinking the pond water.

It will be developed in a single step. Construction costs are estimated at $31.2 million. Table 20.3 provides a summary of costs.

Table 20.3: Tailings Mangement Facility Construction Cost Estimate

DESCRIPTION	QUANTITY	UNIT	UNIT PRICE	TOTAL COST
POX				
Deforestation - timber rights	Technominex prices		42 975.00 $	42 975.00 $
Deforestation - chipping	Technominex prices		138 000.00 $	138 000.00 $
Excavation of 0.5 metres of overburden	183 360	m^3	5.00 $	916 800.00 $
Crushed waste rock (0 -150 mm) for dike construction	1 710 800	m^3	8.50 $	14 541 800.00 $
Geotextile 918	268 300	m^2	3.00 $	804 900.00 $
PEHD-type, 2-mm geomembrane (2 membranes	536 600	m^2	12.00 $	6 439 200.00 $
Geocomposite Transnet 220-2-6	78 000	m^2	8.50 $	663 000.00 $
FT draintube	380 600	m^2	10.00 $	3 806 000.00 $
Clean pebbles	3 500	m^3	50.00 $	175 000.00 $
Pipeline	4 500	m	20.00 $	90 000.00 $
Observation wells and fittings	8	UNIT	10 000.00 $	80 000.00 $
Draining sand (300 mm thick)	110 016	m^3	16.50 $	1 815 264.00 $
Ferric sulphate treatment of arsenic (considered 3 mg/l As content) including building	Lump sum		1 500 000.00 $	1 500 000.00 $
Wildlife fencing	2450	m.l.	85.00 $	208 250.00 $
			Total	31 221 189.00 $

20.5 Waste Rock, Overburden and Top Soil Management

20.5.1 Waste Rock Characterization

20.5.1.1 Methodology

Comprehensive Geochemical Characterization

A comprehensive geochemical characterization of waste rock and ore has been carried out in 2011. A total of 154 samples of waste and 15 samples of ore have been selected by geologists working for Mines Aurizon Ltd. The sampling locations were selected using a grid in order to obtain a representative proportion of samples from the various geological types. The number of samples for the various geological types were as follow: Cadillac fault (20), amphibolite (2), wacke + amphibolite (2), wacke (120), conglomerate (5), wacke + conglomerate (3) and mafic intrusion (2). Near 80 % of the samples are from the wacke type. Amphibolite and Cadillac fault samples represent nearly 15 % of the samples.

All the samples from the characterization campaign have been subjected to the following tests:

- Element content for Al, Sb, Ag, As, Ba, Be, Bi, B, Cd, Co, Cu, Cr, Sn; F, Mn, Hg, Mo, Ni, Pb, Se, Th, U and Zn by partial digestion using aqua regia;
- U.S. EPA-1311 (TCLP) leaching test;
- U.S. EPA-1312 (SPLP) leaching test;
- Shake Flask Extraction (SFE) leaching test (ASTM, D3987);
- Acid Rock Drainage (ARD) potential by the Modified Sobek method.

X-ray diffraction (for mineralogical identification) and X-ray fluorescence following borate digestion (for total content of SiO_2, Al_2O_3, Fe_2O_3, MnO, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, Cr_2O_3 and V_2O_5) have also been carried out on 21 of the 154 waste samples.

Other less important characterization programs have been previously carried out. In 2010, kinetic tests (column) have been carried out at the *Unité de recherche et de service en technologie minérale de l'Abitibi-Témiscamingue* (URSTM) on three (3) composite samples of waste showing high to very high arsenic content: two (2) wacke type composites with total arsenic content of 4 800 mg/kg and 620 mg/kg, and an amphibolite type composite with a total arsenic content of 532 mg/kg. Arsenopyrite (FeAsS) has been identified as the main mineral

containing arsenic, with Gersdorffite (NiAsS) sometimes present in spatial relation with the Cadillac Fault. The objective of this program was to develop a better understanding of the geochemical behavior of arsenic in waste. Finally, three (3) field test pads with large waste tonnage have been monitored since 2008.

20.5.1.2 Mineralogical Description and Metal Contents

Mineralogical Description and Metal Contents (total digestion)

Amphibolite and Cadillac fault samples showed high contents of chlorite, talc, micas, amphibole (hornblende) and carbonate. Wacke and conglomerate samples showed higher quartz and feldspath contents and lower carbonate contents. High total calcium and magnesium contents have been observed for amphibolites and Cadillac fault samples, which indicate the presence of carbonates.

Metal Contents (partial digestion) and Soil Criteria

Table 20.4 shows mean metal contents for the various waste types and the ore as well as Criteria A, B and C of the *Politique de protection des sols et de réhabilitation des terrains contaminés* (PPSRTC). Average content of arsenic, cobalt, chromium, copper, mercury, molybdenum and nickel generally exceeded Criteria A.

20.5.1.3 Acid Rock Drainage (ARD) Potential

Table 20.5 outlines the summary of results for the acid rock drainage (ARD) potential study. The main conclusions are:

- Average sulfur content for amphibolite (0.14 %), mafic intrusion (0.17 %), wacke + amphibolite (0.26 %) and wacke + conglomerate (0.30 %) that were lower than the Directive 019 criterion (0.3 %). These types of waste are *de facto* classified as non-ARD generating according to Directive 019;

Table 20.4: Mean Metal Content for The Various Waste Types

Parameters (mg/kg)	Criteria A*	Criteria B*	Criteria C*	Amphibolite (n=2)	Wacke + Amphibolite (n=2)	Cadillac fault (n=20)	Wacke (n=120)	Wacke + Conglomerate (n=3)	Conglomerate (n=5)	Mafic intrusion (n=2)	Ore (n=15)
Al	-	-	-	18000	19500	25180	20512	17333	18800	15500	16267
Sb	-	-	-	0.425	1.00	0.32	<0,5	<0,5	-	<0,5	1
Ag	2	20	40	<0,5	<0,5	0.85	<0,5	<0,5	<0,5	<0,5	<0,5
As	6	30	50	*6*	*18*	*331*	*114*	*223*	*267*	1.4	*2293*
Ba	200	500	2000	76.25	350	118	238	287	337	14.5	193
Be	-	-	-	<1	<1	<1	<1	<1	<1	<1	<1
Bi	-	-	-	<10	<10	10.67	<10	<10	<10	<10	13
B	-	-	-	3	<2	3.28	1.4	<2	<2	10	2
Cd	1.5	5	20	1.5	3.25	1.81	1.29	1.3	7.7	1.8	1
Cr	85	250	800	**86**	**188**	*1061*	185.0	117	144	19	134
Co	15	50	300	**16**	**33**	**68**	25	20	24	26	23
Cu	40	100	500	40	70	54	47.4	66	72	120	49
Sn	5	50	300	5	27.5	4	4.47	6	4	5	3
Mn	770	1000	2200	770	885	965	391.0	337	530	475	414
Hg	0.2	2	10	<0,01	<0,01	0.01	<0,01	-	0.27	<0,01	1
Mo	2	10	40	5.75	<1	0.78	1.49	1.17	1.4	<1	20
Ni	50	100	500	**60**	*75*	**422**	97.8	61	73	25	61
Pb	50	500	1000	50	<5	32.68	5.61	-	5.1	<5	8
Se	1	3	10	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5	<0,5
Tl	-	-	-	<5	<5	<5	<5	<5	<5	<5	5
Th	-	-	-	2	2	2	5.7	7	4	0.8	-
U	-	-	-	0.425	0.63	0.47	1.3	2	1.09	<0,5	1
Zn	110	500	1500	110	305	48	67.11	58	56	50	66

*PPSRTC criteria

0.0	Exceed Criteria A
0.0	Exceed Criteria B
0.0	Exceed Criteria C

Table 20.5: Summary of Acid Rock Drainage (ARD) Potential for Waste Samples

	Number of Samples				Average Acid Rock Drainage Potential				
Rock Type	Nb	Proportion (%)	ARD According to Directive 019	ARD According to Price (2009)	% S	NP/AP	NP-AP ($kgCaCO_3/t$)	According to Directive 019	According to Price (2009)
Amphibolite	2	1,3	0	0	0,14	32,4	137,5	Not ARD	Not ARD
Cadillac Fault	20	13	3	3	0,43	12,8	145,8	Not ARD	Not ARD
Wacke + Amphibolite	2	1,3	1	(1*)	0,26	2,17	9,3	Not ARD	Not ARD
Wacke	120	78	42	60 (43*)	0,33	2,61	16,7	ARD	Not ARD
Conglomerate	5	3,25	1	0	0,33	3,71	28,3	Not ARD	Not ARD
Wacke + Conglomerate	3	2	1	(2*)	0,30	1,75	7,2	Not ARD	UZ*
Mafic Intrusion	2	1,3	0	0	0,17	4,65	19,4	Not ARD	Not ARD
Total	154	100,0	48	66 (46*)	0,34	4,49	37,1	Not ARD	Not ARD
Total (xcluding Amphibolite and Cadillac Fault)	130	85,8	45	60(46*)	0,33	2,65	17,1	ARD	Not ARD

* Uncertainty Zone

- Average sulfur content for the Cadillac fault (0.43 %), wacke (0.34 %), and conglomerate (0.33 %) exceeding the Directive 019 criterion. Waste from these geological types are not *de facto* classified as non-ARD generating;
- Wastes from the Cadillac fault are non-ARD generating because the net neutralisation potential (NNP) (+ 145.8 kgCaCO3/t) is higher than the Directive 019 criterion (+ 20 kgCaCO3/t) and the neutralization potential/acidification potential ratio (NP/AP) (12.8) is higher than the Directive 019 criterion (3.0);
- Waste from conglomerate are non-ARD generating because the NNP (+ 28.3 kgCaCO3/t) and the NP/AP ratio (3.71) are higher than the Directive 019 criteria;
- Waste from wacke is potentially ARD generating because the sulfur content (0.33%), the NNP (+ 16.7 kgCaCO3/t) and the NP/AP ratio (2.61) do not respect the corresponding Directive 019 criteria.

However, it should be noted that the Directive 019 criteria are based on the William Price report (1997) entitled Guidelines and Recommended Methods for Prediction of Metal Leaching and Acid Rock Drainage at Mine sites in British Columbia, Draft. BC Ministry of Employment and Investment. According to the author, this document is obsolete.

In 2009, Mr. Price prepared an updated version of this report entitled “Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials” (NEDEM Report 1.20.1) for the Canadian Mining Association, Natural Resources Canada, and the governments of British Columbia, Ontario and the Yukon.

Based on the literature available since 1997, Price (2009) concluded that sulfur content and NNP are useless to predict the ARD, and only the NP/AP ratio must be used for this purpose. He proposed the following classification: rocks with NP/AP > 2 are not potentially ARD; rocks with NP/AP < 1 are potentially ARD and rocks with 1 < NP/AP < 2 are in the uncertainty zone.

According to Price’s classification, only wacke + conglomerate waste type (< 2 % of waste tonnage) could be potentially ARD, since the NP/AP ratio (1.75) is in the upper portion of the uncertainty zone.

During the kinetic tests performed over 52 weeks on two (2) composite samples of wacke and one (1) composite sample of amphibolite, the pH of the leachates remained between 7.5 and 8.5. The URSTM concluded that the three samples were not potentially ARD-generating. It should be noted that one (1) composite sample of wacke showed a sulfur content of 0.88 %, which is significantly higher than the average content of the 120 samples of wacke studied in the comprehensive characterization (0.33%). Moreover, the pH measured in the leach pads remained between 7 and 8.

In conclusion, results from the kinetic tests (column and leach pads) confirm that waste from the wacke geological type are not potentially ARD as predicted by the Price (2009) classification. All other waste is also non ARD-generating based on the Directive 019 classification.

20.5.1.4 Leaching Potential

Table 20.6 outlines the summary of results for all waste types and leaching tests (TCLP, SPLP and SFE). For all parameters and waste types, leachate concentrations obtained with the TCLP test were lower than the criteria found in Appendix II, Table 1, of Directive 019. All waste types are therefore not classified as "high risk".

According to Directive 019, a sample is classified as "low risk" if, for a given parameter, the content is lower than Criteria A of the PPSRTC, or if the TCLP leachate shows concentrations lower than the Québec groundwater quality protection criteria. If the TCLP leachate shows concentrations higher than the groundwater quality protection criteria, the sample is classified as "leachable".

The main results from the various tests and the main geological types are:

Cadillac Fault: the average concentration of aluminum for the TCLP test was 1.11 mg/L, exceeding the groundwater protection criterion (0.75 mg/L). However, the average concentrations were lower than the criterion for the SPLP test (0.26 mg/L) and the SFE test (0.64 mg/L). There are no criterion for aluminum in the PPSRTC.

The average concentrations of manganese for the TCLP test was 10.1 mg/L, exceeding the groundwater protection criterion (1.9 mg/L). However, the average concentrations were lower

than the criterion for the SPLP test (0.014 mg/L) and the SFE test (< 0.005 mg/L). The average manganese content is higher than the Criteria A of the PPSRTC.

Finally, arsenic concentrations in leachates for the TCLP test (0.07 mg/L), the SPLP test (0.23 mg/L) and the SFE test (0.10 mg/L) are lower than the groundwater protection criterion (0.34 mg/L). However, these concentrations are higher than the Québec drinking water criterion (0.01 mg/L). The average concentration for the SPLP test also exceeds the effluent discharge criterion of Directive 019 (0.2 mg/L). The average content is higher than criteria C of the PPSRTC.

Amphibolite: the average arsenic concentrations in leachates for the TCLP test (0.85 mg/L), the SPLP test (0.79 mg/L) and the SFE test (0.48 mg/L) are all higher than the groundwater protection criterion (0.34 mg/L), the Québec drinking water criterion and the effluent discharge criterion of Directive 019. The average content is higher than Criteria C of the PPSRTC.

Table 20.6: Summary of Leaching Test Results

	Groundwater protection criteria	Amphibolite (n =2)			Wacke + Amphibolite (n=2)			Cadillac fault (n=20)			Wacke (n=120)			Wacke + Conglomerate (n=3)			Conglomerate (n=5)			Mafic intrusion (n=2)			Ore (n=15)		
Parameter	mg/l	mg/l			mg/l			mg/l			mg/l			mg/l			mg/l			mg/l			mg/l		
		TCLP	SPLP	SFE	TCLP	SPLP	SFE	TCLP	SPLP	SFE	TCLP	SPLP	SFE	TCLP	SPLP	SFE	TCLP	SPLP	SFE	TCLP	SPLP	SFE	TCLP	SPLP	SFE
Al	0.75	0.65	0.25	0.2	1.35	0.4	0.35	1.11	0.26	0.64	1.82	1.18	0.78	1.77	1.20	1.00	1.14	0.92	0.78	1.40	1.00	0.85	1.21	1.14	0.96
Sb		0.01	< 0,006	< 0,006	0.01	< 0,006	< 0,006	0.007	0.003	< 0,006	0.04	0.004	0.004	0.005	< 0,006	< 0,006	0.005	0.006	< 0,006	< 0,006	< 0,006	-	0.02	0.01	0.003
Ag	0.00042	<0,0006	<0,0006	< 0,0006	<0,0006	<0,0006	< 0,0006	0.004	0.0003	< 0,0006	0.001	0.003	0.0006	< 0,0006	< 0,0006	< 0,0006	0.001	< 0,0006	< 0,0006	< 0,0006	< 0,0006	< 0,0006	<0,0006	<0,0006	<0,0006
As	0.34	**0.85**	**0.79**	**0.48**	**0.86**	**1.5**	**0.83**	0.07	0.230	0.10	0.03	0.06	0.04	0.03	0.08	0.05	0.06	0.06	0.03	< 0,01	< 0,01	< 0,01	**0.52**	**0.68**	**0.67**
Ba	0.47	0.34	< 0,02	< 0,02	**0.60**	< 0,02	< 0,02	0.36	< 0,02	0.03	**0.49**	0.01	0.01	**0.61**	< 0,02	< 0,02	**1.02**	< 0,02	< 0,02	< 0,01	< 0,02	< 0,02	0.38	< 0,02	< 0,02
Be		< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,02	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	0.01	<0,01	< 0,01	< 0,01	<0,01	<0,01	< 0,01	< 0,01	< 0,01	< 0,01	<0,01	<0,01	<0,01
Bi		< 0,050	< 0,050	< 0,050	< 0,050	< 0,050	< 0,050	< 0,050	< 0,050	< 0,050	< 0,050	< 0,050	< 0,050	<0,05	< 0,050	< 0,050	<0,05	<0,050	< 0,050	< 0,050	< 0,050	< 0,050	<0,050	<0,050	<0,050
B		<0,7	<0,7	< 0,7	<0,7	<0,7	< 0,7	< 0,7	< 0,7	< 0,7	< 0,7	< 0,7	< 0,7	<0,7	< 0,7	< 0,7	<0,7	<0,7	< 0,7	< 0,7	< 0,7	< 0,7	<0,7	<0,7	<0,7
Cd	0.00084	<0,001	<0,001	<0,001	<0,001	<0,001	<0,001	0.01	0.001	<0,001	0.001	<0,001	0.001	<0,001	< 0,001	< 0,001	0.002	<0,001	<0,001	< 0,001	< 0,001	< 0,001	<0,001	<0,001	<0,001
Cr	0.85	0.02	< 0,01	< 0,01	0.05	< 0,01	< 0,01	0.04	< 0,01	< 0,01	0.02	< 0,01	< 0,01	0.01	< 0,01	< 0,01	0.01	<0,01	< 0,01	< 0,01	< 0,01	< 0,01	0.02	<0,01	<0,01
Co		0.07	<0,01	< 0,01	0.07	<0,01	< 0,01	0.06	<0,01	< 0,01	0.01	< 0,01	0.005	<0,01	< 0,01	< 0,01	<0,01	<0,01	< 0,01	0.01	< 0,01	< 0,01	<0,01	<0,01	<0,01
Cu	0.0059	<0,007	<0,007	<0,007	<0,007	<0,007	<0,007	< 0,001	0.01	<0,007	0.002	0.013	0.001	<0,007	< 0,002	< 0,01	<0,007	0.004	< 0,002	< 0,001	< 0,002	< 0,002	<0,007	<0,007	0.05
Sn		< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	<0,01	< 0,01	< 0,01	<0,01	<0,01	< 0,01	< 0,01	< 0,01	< 0,01	<0,01	<0,01	<0,01
F	4	0.35	0.25	< 0,2	0.4	0.20	< 0,2	0.46	0.66	< 0,2	0.75	0.35	0.06	0.33	0.27	< 0,2	0.28	0.24	< 0,2	0.20	0.25	< 0,1	0.39	0.49	< 0,1
Mn	1.9	**9.11**	0.007	< 0,005	<0,01	0.006	< 0,005	**10.11**	0.01	< 0,005	2.25	0.003	0.003	0.55	0.004	0.006	5.85	0.004	< 0,005	1.45	< 0,005	< 0,005	3.75	<0,005	<0,005
Hg		0.0001	< 0,00013	< 0,00013	< 0,00013	< 0,00013	< 0,00013	0.001	0.001	< 0,00013	0.000	0.001	0.0001	0.0005	0.001	< 0,00013	0.0003	< 0,00013	< 0,00013	0.0003	0.0014	< 0,00013	<0,00013	0.0003	<0,00013
Mo		<0,01	<0,01	< 0,01	<0,01	<0,01	< 0,01	<0,01	<0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	<0,01	<0,01	<0,01
Ni	0.22	**0.69**	< 0,01	< 0,01	**1.10**	0.0175	< 0,01	0.19	0.02	< 0,01	0.06	0.02	0.005	0.01	< 0,01	< 0,01	0.02	< 0,01	< 0,01	0.02	< 0,01	< 0,01	0.04	<0,01	<0,01
Pb	0.025	0.01	0.01	< 0,01	0.01	0.01	< 0,01	<0,5	< 0,01	< 0,01	0.03	0.006	< 0,01	0.01	0.01	< 0,01	0.01	0.01	< 0,01	< 0,01	< 0,01	< 0,01	0.02	<0,01	<0,01
Se	0.02	<0,01	< 0,01	< 0,01	<0,01	< 0,01	< 0,01	<0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	< 0,01	<0,01	<0,01	<0,01
Th		<0,01	< 0,001	< 0,001	0.001	< 0,001	< 0,01	0.001	< 0,001	< 0,001	0.002	< 0,001	0.0005	0.001	< 0,001	< 0,001	0.001	< 0,001	< 0,001	< 0,001	< 0,001	< 0,001	0.001	<0,001	<0,001
U		<0,005	<0,005	< 0,005	<0,005	<0,005	< 0,005	0.003	< 0,005	< 0,005	0.01	< 0,005	0.003	0.005	< 0,005	< 0,005	0.005	< 0,005	< 0,005	< 0,005	< 0,005	< 0,005	<0,005	<0,005	<0,005
Zn	0.055	<0,05	< 0,05	< 0,05	0.04	< 0,05	< 0,05	0.05	< 0,05	< 0,05	0.06	0.03	0.03	0.03	< 0,05	< 0,05	0.04	< 0,05	0.06	0.06	< 0,05	< 0,05	<0,05	<0,05	0.025

(0.0)	Exceed Criteria A and Groundwater protection criteria
(0.0)	Exceed Groundwater protection criteria

The average concentrations of nickel for the TCLP test were 0.69 mg/L, exceeding the groundwater protection criterion (0.22 mg/L). However, the average concentrations were lower than the analytical method detection limit for the SPLP test (< 0.01 mg/l) and the SFE test (< 0.005 mg/L). The average nickel content is higher than Criteria A of the PPSRTC.

The average concentrations of manganese for the TCLP test were 9.11 mg/L, exceeding the groundwater protection criterion (1.9 mg/L). However, the average concentrations were lower than the criterion for the SPLP test (0.007 mg/L) and the SFE test (< 0.005 mg/L). The average manganese content is higher than Criteria A of the PPSRTC.

Finally, the average concentrations of aluminum were lower than the groundwater protection criteria for the three (3) tests.

Wacke: the average concentrations of aluminum for the TCLP test (1.9 mg/L), the SPLP test (1.78 mg/L) and the SFE test (1.78 mg/L) exceed the groundwater protection criterion (0.75 mg/L). There is no criterion for aluminum in the PPSRTC.

The average concentrations of manganese for the TCLP test were 2.25 mg/L, exceeding the groundwater protection criterion (1.9 mg/L). However, the average concentrations were lower than the criterion for the SPLP test (0.003 mg/L) and the SFE test (0.003 mg/L). The average manganese content is higher than Criteria A of the PPSRTC.

The average concentrations of barium for the TCLP test were 0.49 mg/L, slightly exceeding the groundwater protection criterion (0.47 mg/L). However, the average concentrations were lower than the criterion for the SPLP test (0.01 mg/L) and the SFE test (0.01 mg/L). The average manganese content is lower than Criteria A of the PPSRTC.

Finally, the arsenic concentrations in leachates for the TCLP test (0.07 mg/L), the SPLP test (0.23 mg/L) and the SFE test (0.10 mg/L) are lower than the groundwater protection criterion (0.34 mg/L). However, these concentrations are higher than the Québec drinking water criterion. The average concentration for the SPLP test also exceeds the effluent discharge criterion of Directive 019. The average content is higher than Criteria C of the PPSRTC.

Table 20.7 provides a summary of PPSRTC criteria and groundwater quality protection criteria (TCLP test) exceedances for the various types of waste rock and ore.

Table 20.7: Summary of PPSRTC Criteria and Groundwater Quality Protection Criteria (TCLP Test) Exceedances for the Various Types of Waste Rock and Ore

Waste Type	> Groundwater Protection Criteria and > Criteria A from PPSRTC	> Groundwater Protection Criteria but < Criteria A from PPSRTC	> Groundwater Protection Criteria but no Criteria A from PPSRTC
Amphibolite	As, Mn, Ni	-	Al
Wacke + Amphibolite	As, Ba, Ni	-	-
Cadillac Fault	Mn	-	Al
Wacke	Ba	Mn, Pb	Al
Wacke + Conglomerate	Ba	-	Al
Conglomerate	Ba	Mn	Al
Mafic Intrusion	-	-	Al
Ore	As	Mn, Pb	Al

Natural Aluminum, Barium and Manganese Leachability: the kinetic tests are more representative of the natural site conditions than the static tests (TCLP, SPLP and SFE). Static tests require fine grinding, low liquid/solid ratio and, in some cases (TCLP and SPLP), the addition of acid. In fact, static tests usually overestimate the leaching potential of materials.

In this project, concentrations of aluminum, barium, nickel and manganese measured in the kinetic tests (columns and leach pads) were always lower than the corresponding groundwater quality protection criterion. In this context, it can be concluded that these metals are not likely to leach from any of the types of waste rock.

Real Leachability of Arsenic: arsenic has been observed in relatively high concentrations in all leaching tests performed on amphibolite and wacke + amphibolite samples. The column test performed on an amphibolite composite sample has also shown a significant concentration of arsenic. The latter has also been observed in significant concentrations in leachates from the

SPLP test for the Cadillac fault samples. In this context, it is safe to state that amphibolites, wacke + amphibolites and Cadillac fault types of waste could potentially leach arsenic.

This first waste stockpile would show an average sulfur content of 0.4% for an AP of 12.3 $kgCaCO_3/t$ and a NP of 158.2 $kgCaCO_3/t$. The NNP and NP/AP ratio would be + 145.8 $kgCaCO_3/t$ and 12.8, respectively. This dump will clearly be non-ARD.

This second waste dump would present an average sulfur content of 0.33%. The NNP and NP/AP ratio would be + 17.1 $kgCaCO_3/t$ and 2.65, respectively. This dump will clearly be non-ARD as per Price's (2009) classification. The percentage of sulfur, the NNP and the NP/AP ratio are also very close to the Directive 019 criteria.

20.5.2 Waste Rock Management

The waste rock found on the Joanna Property consists of: wacke, conglomerate, mafic intrusions, amphibolite and the Cadillac fault.

According to the waste rock geochemical characterization results, the wacke, conglomerate and mafic intrusions are low-risk under Directive 019, whereas the amphibolites and Cadillac fault are considered leachable for arsenic.

Waste rocks will therefore be stockpiled in separate piles. The mine plan includes waste rock segregation.

Low-risk waste rock (wacke, conglomerate and mafic intrusions) account for about 85-90% of all waste rocks. The area used for stockpiling this rock will simply be cleared. It will be encircled by a drainage ditch connected to two polishing ponds to settle suspended solids. These ponds will drain into two different branches of the Stitchman Creek so as to maintain adequate water levels in the stretches considered to be fish-bearing watercourses. The cost of constructing this stockpile is assessed at $0.37 million. Construction costs are detailed in Table 20.7.

Leachable waste rock (amphibolite and Cadillac fault) represent about 10-15% of all waste rocks. The area where this waste rock is stockpiled will be protected with a geo-membrane to prevent groundwater contamination.

Once the area is cleared, a 30-cm thick layer of draining sand will be placed in the waste rock accumulation areas. A geo-membrane will then be installed and covered with another 30-cm layer of draining sand. A one (1)-metre layer of crushed waste rock (0-150 mm) will then be installed to protect the geo-membrane.

The stockpile will be encircled by a geo-membrane-lined drainage ditch that is connected to a retention basin, from which the water will be pumped into the process water basin either for recirculation to the concentrator or treatment before being released into the environment. The cost of constructing this stockpile is assessed at $9.2 million. Construction costs are detailed in Table 20.8.

Table 20.8: Construction Cost Estimte fot Various Stockpiles

DESCRIPTION	QUANTITY	UNIT	UNIT PRICE	TOTAL COST
Neutral waste rock stockpile				
Deforestation - timber rights	Technominex prices		90 255.00 $	90 255.00 $
Drainage ditch with no geomembrane but with type 1 riprap	1 900	m.l.	25.00 $	47 500.00 $
Drainage ditch with no geomembrane but with type 3 riprap	2 600	m.l.	75.00 $	195 000.00 $
Excavated material for SS polishing pond	8 000	m³	5.00 $	40 000.00 $
Total				372 755.00 $
Waste rock (As and Ni) stockpile				
Deforestation - timber rights	Technominex prices		16 575.00 $	16 575.00 $
Deforestation - chipping	Technominex prices		51 750.00 $	51 750.00 $
Draining sand (300 mm thick) (two layers)	173 800	m³	16.50 $	2 867 700.00 $
PEHD-type 2-mm geomembrane	289 630	m²	12.00 $	3 475 560.00 $
Crushed waste rock (0 - 150 mm) 1 metre deep to cover bottom	289 630	m³	8.50 $	2 461 855.00 $
Drainage ditch with PEHD-type, 2-mm geomembrane but no riprap	1 300	m.l.	110.00 $	143 000.00 $
Drainage ditch with geomembrane and type 1 riprap	750	m.l.	135.00 $	101 250.00 $
Excavated material for retention basin prior to pumping	2 000	m³	5.00 $	10 000.00 $
PEHD-type, 2-mm membrane for retention basin prior to pumping	4 700	m²	12.00 $	56 400.00 $
918 geotextile for retention basin pror to pumping	4 700	m²	3.00 $	14 100.00 $
				9 198 190.00 $
Low-grade ore stockpile				
Deforestation - timber rights	Technominex prices		6 300.00 $	6 300.00 $
Draining sand (300 mm thick) (2 layers)	27 350	m³	16.50 $	451 275.00 $
PEHD-type, 2-mm geomembrane	45 600	m²	12.00 $	547 200.00 $
Crushed waste rock (0 - 150 mm) 1 metre deep to cover bottom	45 600	m³	8.50 $	387 600.00 $
Drainage ditch with PEHD-type, 2-mm geomembrane but no riprap	300	m.l.	110.00 $	33 000.00 $
Drainage ditch with PEHD-type, 2-mm geomembrane and type 1 riprap	775	m.l.	135.00 $	104 625.00 $
Excavated material for retention basin prior to pumping	1 000	m³	5.00 $	5 000.00 $
PEHD-type 2-mm geomembrane for retention basin prior to pumping	3 000	m²	12.00 $	36 000.00 $
Geotextile 918 for retention basin prior to pumping	3 000	m²	3.00 $	9 000.00 $
Total				1 580 000.00 $
Overburden stockpile				
Deforestation - timber rights	Technominex prices		83 760.00 $	83 760.00 $
Drainage basin with no geomembrane but with type 2 riprap	2 650	m.l.	50.00 $	132 500.00 $
Drainage basin with no geomembrane but with type 1 riprap	550	m.l.	25.00 $	13 750.00 $
Drainage basin with no geomembrane and no riprap	50	m.l.	6.00 $	300.00 $
Excavated material for SS finishing pond	30 000	m³	5.00 $	150 000.00 $
Total				380 310.00 $
Topsoil stockpile				
Deforestation - timber rights	Technominex prices		27 020.00 $	27 020.00 $
Drainage basin with no geomembrane and no riprap	150	m.l.	6.00 $	900.00 $
Drainage basin with no geomembrane but with type 1 riprap	375	m.l.	25.00 $	9 375.00 $
Drainage basin with no geomembrane but with type 2 riprap	2 900	m.l.	50.00 $	145 000.00 $
Excavated material for 2 SS finishing ponds	8 000	m³	5.00 $	40 000.00 $
Total				222 295.00 $

20.5.3 Low-Grade Ore Stockpile

A low-grade ore stockpile will be set up near the crusher. Since the ore is considered acidogenic and leachable under Directive 019, the low-grade ore stockpile pad will be similar to the leachable waste rock stockpile one. The construction cost of this stockpile is assessed at $1.6 million. Construction costs are detailed in Table 20.8.

20.5.4 Topsoil and Overburden Stockpiles

The topsoil and overburden that is removed to allow for pit operations will be stockpiled in two (2) separate piles and used for site restoration.

The areas used to stockpile these materials will be cleared and encircled by drainage ditches connected to polishing ponds to settle suspended solids. The water in the ponds will flow by gravity to Stitchman Creek. The construction cost of the topsoil and overburden stockpiles is assessed at $0.22 million and $0.38 million respectively. Construction costs are detailed in Table 20.8.

20.6 Site Geotechnical Assessment

20.6.1 Summary of Field Work

In 2007, a field campaign was conducted by Géolab. It consisted of 9 borings in the area of the Hosco pit. Since 2010, a total of 191 borings and 133 exploration pits were completed on the Property by Laboratoires d'Expertises de Québec Ltée. These different probes allowed for a better understanding of both the soil distribution as well as the groundwater conditions. These probes and the results from both field and laboratory tests have allowed the definition of potential geotechnical issues. Other field campaigns will be required in order to get a clearer picture of the soil distribution, as well as certain geotechnical properties of the soil and rock.

20.6.2 Properties and Nature of Materials

The probes identified the presence of the following materials:

- Surface organic soil (Lichen, peat, topsoil)
- Cohesive soils (clay)

- Granular soils (silt, sand, and gravel)
- Bedrock

The probes also indicated that the thickness of the different layers of soil varies from one area to another on the Property. In fact, in some places, such as the northern part of the Property close to the Hydro-Québec power lines, the bedrock is flush with the surface. In other places however, such as the area south of the West pit, the soil layer can attain approximately 30 meters in thickness. Figure 20.4 shows the distribution of the different layers of soil encountered during the campaigns held in 2010 and 2011.

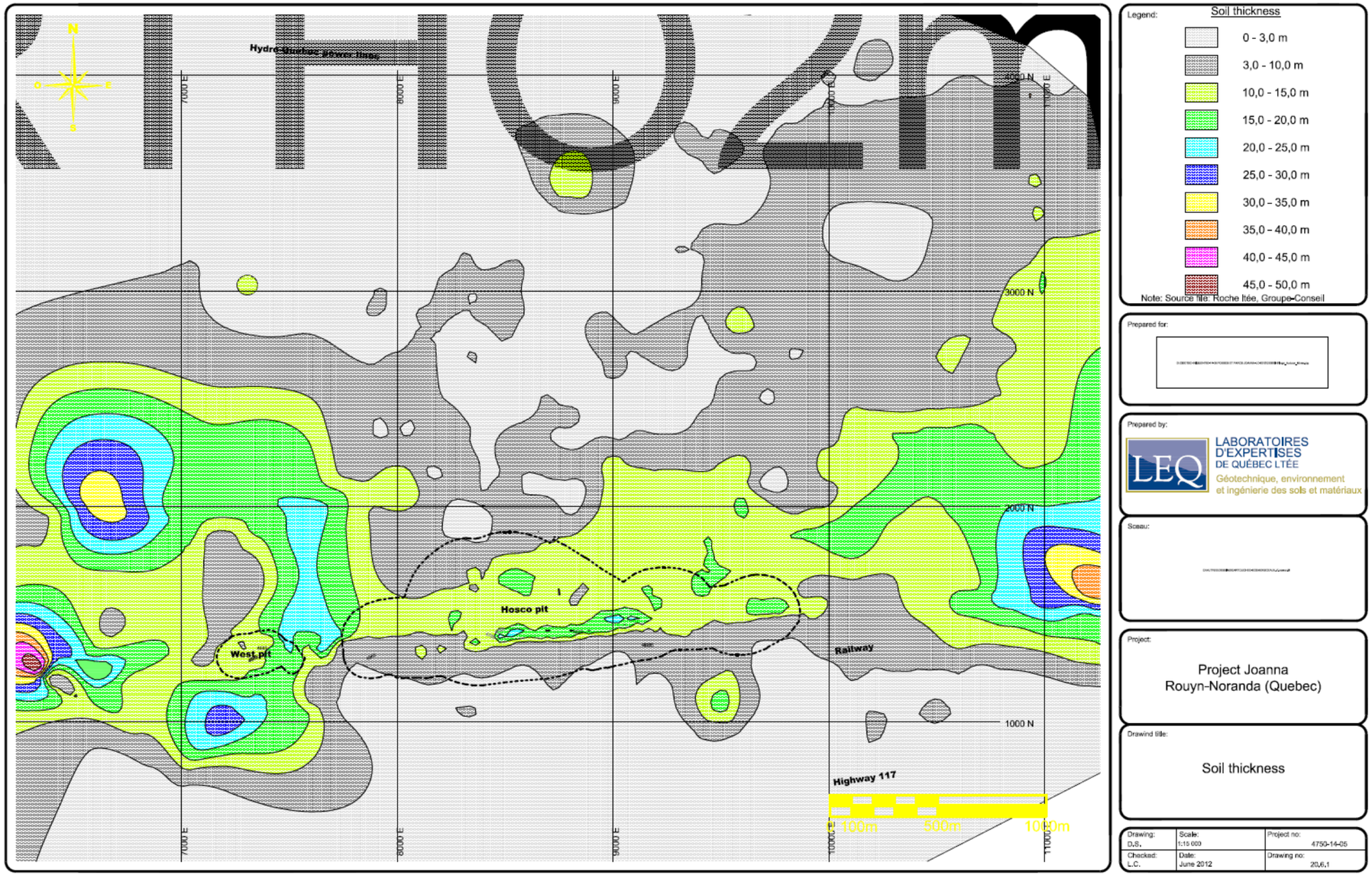


Figure 20.4: Distribution of the Different Layers of Soil Encountered during the Campaigns Held in 2010 and 2011

The cohesive soils generally have a medium to stiff consistency, except in the area of the Hosco pit, where they are rather soft to very soft, especially in the northwestern sector of the West pit. It is important to note that laboratory tests showed that these cohesive soils are sensitive to disturbance.

As for the granular soils, they consist mainly of fine soils (silts). They generally have a loose to medium density. The bedrock consists mainly of siltstone and greywacke. The quality of the rock is generally fair to good, based on the rock quality designation values.

20.6.3 Groundwater

With respect to the groundwater level, the different readings made in the boreholes and exploration pits indicate that it is located at the surface, or less than one meter below the surface.

20.6.4 Bearing Capacity

In general, the northern area, between the Stichman Creek and the Hydro-Québec power lines has an adequate bearing capacity for the construction of buildings with superficial foundations. Nevertheless, the northwestern extremity is preferable, as the presence of rock close to the surface would allow for the heavy loads associated with the concentrator, crusher and other buildings of sort. Figure 20.5 shows the different areas of the Property where these types of foundations are possible.

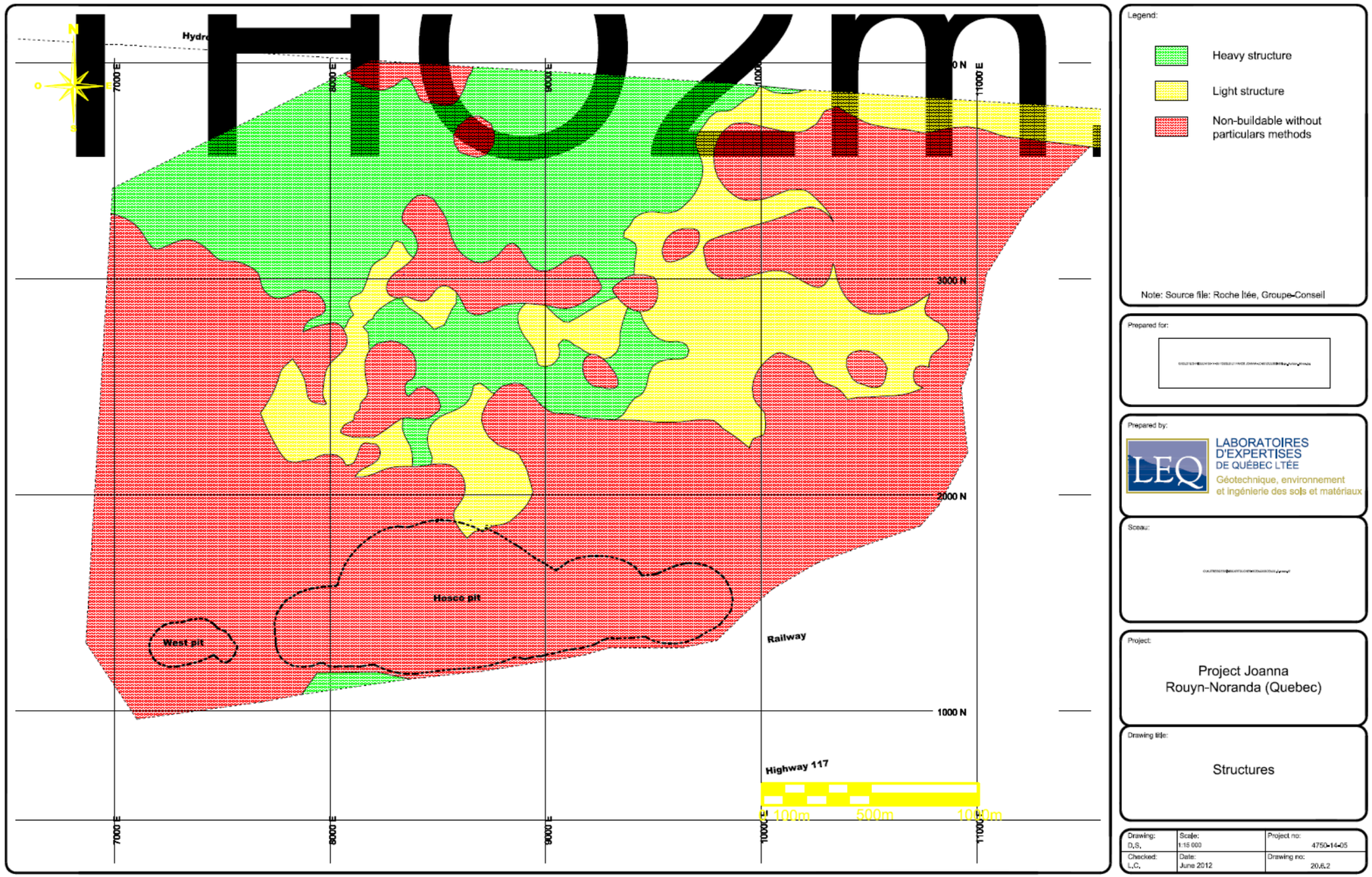


Figure 20.5: Foundation Types Required by Site Area

20.6.5 Pit Excavation

Preliminary analysis at the Hosco and West pits indicates that excavation slopes in the overburden should be within 1.0 vertically for 4.0 horizontally (1.0V: 4.0H) and 1.0V: 5.5H depending on the location. Figure 20.6 shows the approximate position of these preliminary slopes. A more specific analysis will be required to further specify the pit slopes.

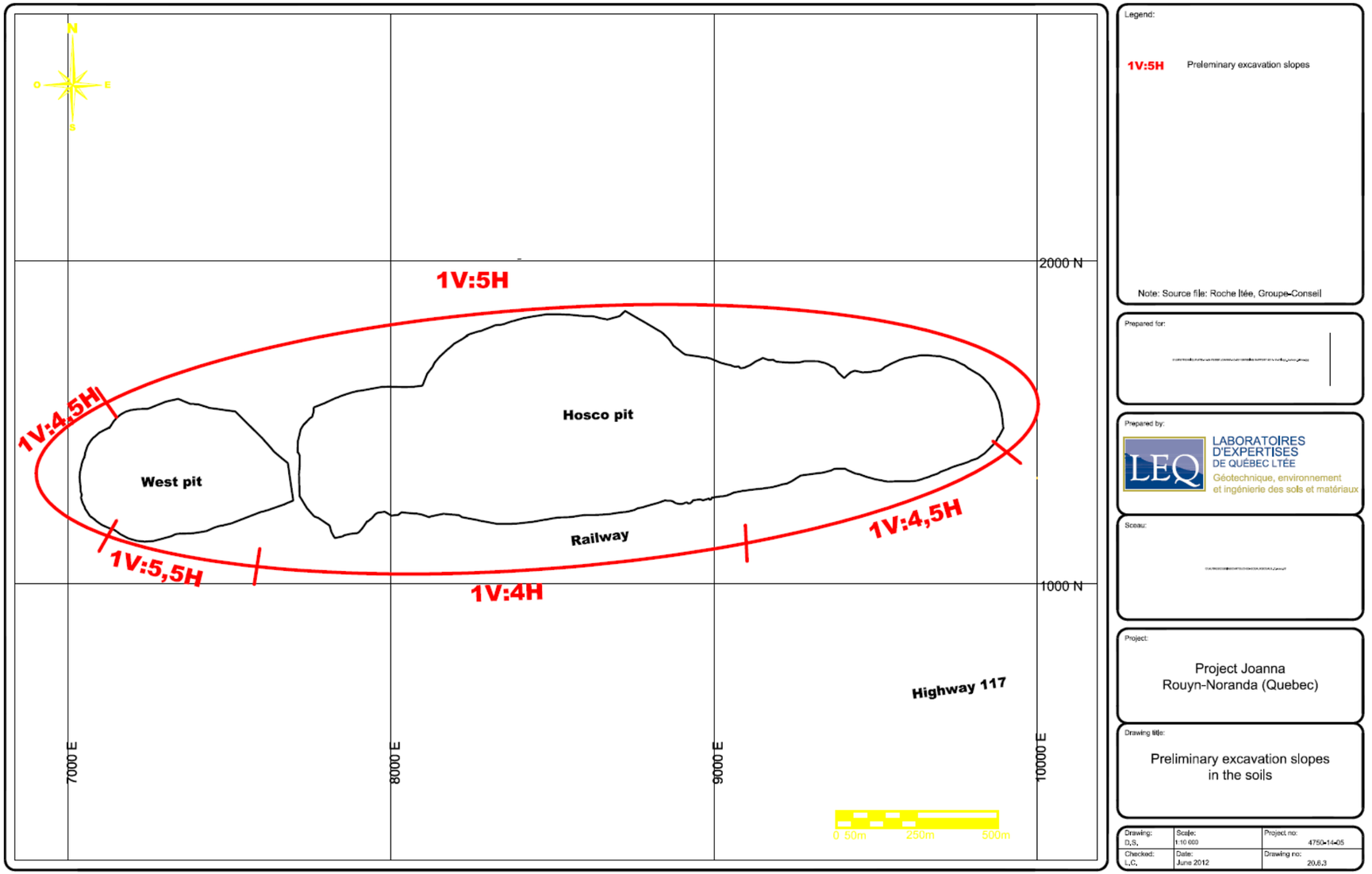


Figure 20.6: Overburden Excavation Slopes

The presence of the groundwater table and the sensitivity to disturbance of the clayey soils will need to be considered for the excavation of the pits. Certain field tests revealed that these soils have a tendency to be liquid-like after their excavation and transportation. This particularity complicates their excavation and storage on other parts of the Property.

Another particularity is the presence of the CN railway situated south of the West pit. The drawdown of the groundwater table associated with the excavation of the pit might lead to ground settlement beneath the railway. Also, the pit excavation might require some additional slope stability analysis. This point will need to be evaluated at the detailed engineering stage, including a dynamic stability analysis of the soft soil in that area.

20.6.6 Waste and Overburden Dumps

Because of the changing nature and geotechnical properties of the soil beneath the footprint of the waste and overburden piles, the construction of the dumps will be limited in height in some areas. Construction of the dumps in stages might also be required, as well as some monitoring in order to ensure safety against bearing soil failure. This aspect is not problematic, but will limit the final height of the dumps.

20.6.7 Neutral Tailings and POX Ponds

Two (2) potential sectors were considered for these ponds: the western sector of the Property for the flotation tailings pond, and the sector directly to the north of the Hydro-Québec power lines for the POX pond. Some geotechnical borings have been done to verify the compatibility of these sites. Additional geotechnical studies will be required at the detailed engineering stage in order to complete the dike and pond conceptions.

20.7 Water Management Plan

20.7.1 Mine and Process Water

On the Site General Arrangement Plan (see Figure 20.7), all surface infrastructures are located as well as associated ditches (pink) and pipelines (orange) which will be used to adequately manage water on the mine site. Tables 20.9 to 20.11 depict data on which the following Water Management Plan is based.

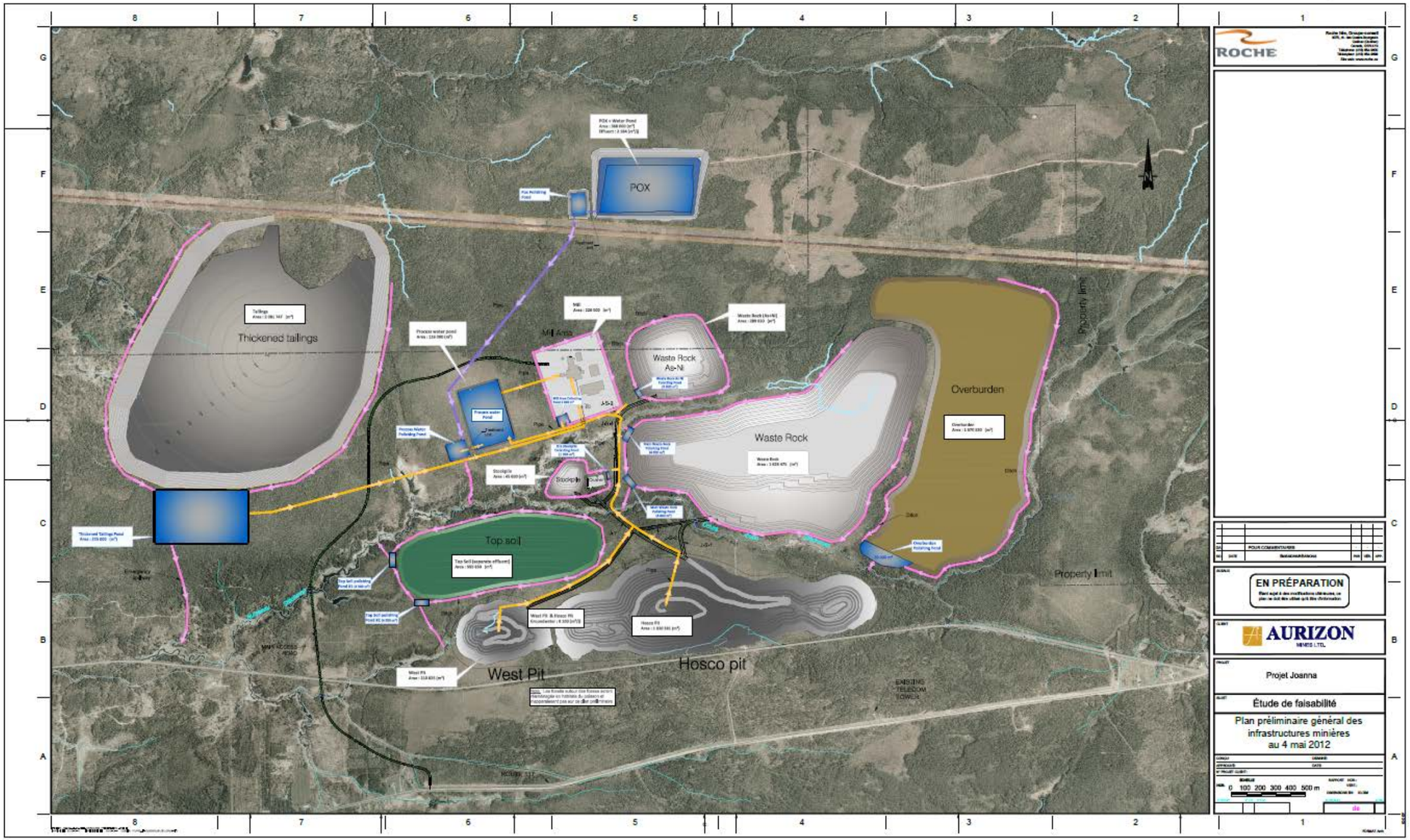


Figure 20.7: - Site General Arrangement Plan

Table 20.9: Final Effluent and Other Water Discharges

Infrastructures	Area (m^2)	Annual Rainfall[1] (m^3/d)	Evapotranspiration[2] (m^3/d)	Net Runoff (m^3/d)	Groundwater[3] (m^3/d)	Effluent (m^3/d)	FINAL EFFLUENT[4]
Totally or partially considered in the final effluent							
Thickened Tailings	2 081 747	5 036	1 259	3 777	-	1 477	
Thickened Tailings Pond	225 000	544	218	327	-		
West Pit	210 825	510	102	408	8 100	10 463	
Hosco Pit	1 010 381	2 444	489	1 955			m^3/d 13 308.54
Waste Rock (As+Ni) + Pond	291 620	705	212	493	-	493	m^3/h 554.52
Ore Stockpile + Pond	46 600	113	34	79	-	79	m^3/s **0.15**
Mill Area + Pond	226 500	548	112	436	-	436	
Process Water Pond + Polishing Pond	116 000	281	112	168	-	168	
POX Tailings + Polishing Pond	542 116	-	-	-	-	2 184	
Other discharges							
Overburden + Pond	1 700 520	4 114	1 039	3 075	-	3 075	
Main Waste Rock + Pond	1 636 475	3 959	1 190	2 769	-	2 769	n/a
Top Soil + Pond(s)	603 830	1 461	368	1 093	-	1 093	

[1] Annual mean total precipitation at the Kinojevis River Weather Station is 883 mm.

[2] Based on average runoff for different land types on a mine site in Northern Quebec (annual mean total precipitation = 798 mm). See below.

[3] Hydrogeological analysis perform by TechnoRem (2010).

[4] Make-up water is estimated at 83 m^3/h (1,992 m^3/d).

Table 20.10: Average Runoff and Evapotranspiration on Various Land Types

	Tailings Area, Overburden,	Waste Rock, Ore Stockpile	Open Pits, Built Areas	Natural Areas (lands, lakes)
Annual Average Runoff	75%	70%	80%	60%
Annual Average Evapotranspiration	25%	30%	20%	40%

Table 20.11: Design Criteria According to Québec's Directive 019 (MDDEP)

Flood Event	Infrastructures
1: 2 000	POX Tailings Pond
1: 1 000	Process Water Pond, POX Polishing Pond
1 : 100	Overburden, Main Waste Rock, Top Soil, Waste Rock As-Ni, Mill Area, Ore Stockpile, all ditches, Process Water Polishing Pond, Thickened Tailings Pond

Weather data from the Kinojevis River Station was used to estimate rain and snowfall in the area. The annual average rainfall is estimated at 648 mm, while snowfall is 235 cm annually. Total precipitation is therefore estimated annually at 883 mm on average. Evapotranspiration rates were used considering land types (tailings, waste rock, open pit, etc.; Table 20.10). Those rates were determined following numerical simulations performed for a different mine project located in a region with similar weather conditions

The Project Area is mainly flat and globally badly-drained with several peatlands (bogs) occupying major parts of the Property. All mine infrastructures are located within only one (1) sub-watershed, namely Stitchman Creek. In fact, the POX Tailings Management Facility (POX-TMF) is located outside that area, but all waters coming from that facility will be managed in such a way that they will be pumped to the Process Water Polishing Pond (PWPP) and then discharge with the final effluent within the Stitchman Sub-Watershed. No water discharge is located outside the Stitchman Sub-Watershed. The POX-TMF effluent will be controlled and is estimated at 91 m^3/d. Owing to its characteristics, that effluent cannot be re-introduced in the flotation section of the mill. No effluent will be directly released into the environment from the POX-TMF Pond since its effluent will be pumped to the PWPP prior to discharge with the final effluent.

Following Québec's Directive 019 issued by the MDDEP (*Ministry of Sustainable Development, Environment and Parks*), only one final effluent is to be considered (at 0.15 m^3/s). According to Directive 019, final effluent is defined as an effluent which needs no further treatment before being discharged into the environment. Equipment will be implemented to measure pH and flow. Also, the final effluent will be monitored as specified in Directive 019. The final effluent discharge criteria prescribed in Directive 019 consists of maximum limit values. The environmental discharge objectives (EDO) to be imposed by the MDDEP may well be far more stringent than those specified under Directive 019 and the Metal Mining Effluent Regulations (MMER). The EDOs will be based on water volumes discharged as well as water quality, the minimum annual flow, and usages of the receiving stream.

Based on the evaluation depicted in Table 20.9, the final effluent flow will be similar to the maximum flow measured during the 2010 low-flow period (July/August 2010) in the section of the Stitchman Creek where it is to be discharged, i.e. it is well within the natural range of that

creek. Additional hydrological samples are now being collected to adequately assess any potential impact on the environment.

Three other water discharges were also considered:

- Top Soil Pile: two (2) discharge points (two (2) basins), totalling 1,093 m³/d;
- Overburden Pile: one (1) discharge point, at 3,075 m³/d;
- Main Waste Rock Pile: two (2) discharge points (two (2) basins), totalling 2,769 m³/d.

These will help maintain sufficient water levels and flow within the upstream sections of Stitchman Creek so that fish habitats will most likely not be impacted by the proposed Water Management Plan, i.e. they will mitigate the impact of the proposed infrastructures on local natural drainage.

On the other hand, it is planned to collect water running off the As-Ni Waste Rock Pile, the Ore Stockpile and the Mill Area by pumping it from a collecting basin towards the Process Water Pond (using pipelines).

Mine water will also be pumped to the Process Water Pond. Mathematical modelling of the successive mine operation phases was performed to estimate the daily flow rate required to keep the pit dry. Based on the hydrogeological study performed by TechnoRem (2010), due to low hydraulic conductivity, the pit cannot be kept dry with dewatering wells. Therefore, infiltration and precipitation water will be accumulated in a catch basin at the bottom of the pit and then pumped to the Process Water Pond. The modelling indicated that an average rate varying from 2,500 m³/d to 8,100 m³/d should be maintained for operation of the mine, as the pit excavation will advance. The latter rate was considered for this Water Management Plan.

Water from the Flotation Tailings Pond will be recirculated to the mill. Make-up water required at the mill is estimated at 83 m³/h (1,992 m³/d). In order to guarantee this input during the first years of mill operation, the first construction phase will include the implementation of the Process Water Pond (350,000 m³), i.e. prior to production at the mine so that enough water can be accumulated for use as process water.

All water accumulated in the Process Water Pond that will not have been recirculated to the mill will be discharged with the final effluent following treatment.

20.7.1.1 Treatment Units

Two (2) water treatment units will be implemented – one at the end of the POX-TMF and another at the end of the Process Water Pond – to ensure their effluent complies with all applicable criteria/guidelines. The following information was considered:

- Volume:
  - 2,400 m³/d at 3.0 mg As/L for the POX-TMF;
  - 3,000 m³/d at 0.5 mg As/L for the Process Water Pond.
  - Objectives : 0.1 mg As/L;
- Assumptions :
  - Total Suspended Solids (TSS) content is unknown (might influence type of treatment, assays, etc.);
  - Water quality analyses and assays will help refine the proposed treatment;
  - The proposed treatment is similar to what has been proposed by ASDR Environment in one (1) other project (similar flow and As content).

The following treatment unit was considered:

- Treatment Line
  - Water flows from the POX-TMF treatment unit to the adjacent 5,000 m³ basin;
  - 5,000 m³ basin (POX Polishing Pond) with a 2-days retention time;
  - Water pumped from the POX-TMF Polishing Pond to the PWPP;
  - Water flows from the Process Water Pond treatment unit to the PWPP;
  - Gravity flowing (ditch) from the PWPP to the Stitchman Creek.
- Treatment Unit:
  - Ferrous sulphate (12%): up to 96 L/h (ref. ASDR);
  - Polymers: up to 22,5 L/h (ref. ASDR);
  - POX-TMF:
    - → Mixing is done along a pipeline; water is then pumped to geotubes which act as filters. The supernatant, which complies with applicable criteria/guidelines, then flows towards the 5,000 m³ basin. Geotubes are periodically cleaned and replaced.

The effluent of the PWPP is to be considered as the final effluent following Québec's Directive 019. The final effluent will comply with all provincial (Directive 019) and federal (CCME, MMER) criteria/guidelines.

20.8 Rehabilitation and Mine Closure Plan

The *Mining Act* (RSQ, c. M-13.1) is another important piece of provincial legislation concerning the management of mining activities in the Province of Québec. "The *object of this Act is to promote prospecting, mineral exploration, and development and operation of underground reservoirs, taking into account other possible uses of the land in the territory"* (s.17).

Section 232.1 of the Act states that "*Land rehabilitation and restoration work must be carried out, in accordance with the plan approved by the Minister. The obligation shall subsist until the work is completed or until a certificate is issued by the Minister under Section 232.10*."

The land rehabilitation and restoration work to be conducted must be planned and approved by the MRNF (Ministry of Natural Resources and Wildlife). According to Section 232.2 of the Act, "*Every person to whom Section 232.1 applies must submit a rehabilitation and restoration plan to the Minister for approval before commencing mining activities*."

Hence, as part of the project, a rehabilitation plan will have to be prepared (and approved by the MRNF). The rehabilitation and restoration plan should be elaborated in accordance with the provincial *Guidelines for Preparing a Mining Site Rehabilitation Plan* and *General Mining Site Rehabilitation Requirements* (MRNF and MDDEP, 1997) which provide the proponents with the rehabilitation requirements. The financial feasibility of the project will have to account for the cost of all work required for the rehabilitation of the mining site.

Moreover, Bill 14 "An Act respecting the Development of Mineral Resources in keeping with the Principles of Sustainable Development" will increase the financial guarantee from 70% to 100% of projected costs for the work required under the rehabilitation and restoration plan. According to this Bill, the guarantee must be paid in three (3) annual instalments. The first instalment corresponds to 50% of the total amount of the guarantee and must be paid within 90 days following receipt of the approval of the plan. The second and third instalments each represent 25% of the guarantee.

20.8.1 General Principles

The main objective of mine site rehabilitation is to restore the site to a satisfactory condition by:

a) Eliminating unacceptable health hazards and ensuring public safety;
b) Limiting the production and circulation of substances that could damage the receiving environment and, in the long-term, trying to eliminate maintenance and monitoring;
c) Restoring the site to a condition in which it is visually acceptable to the community;
d) Reclaiming the areas where infrastructures are located (excluding the accumulation areas) for future use.

Specific objectives are to:

a) Restore degraded environmental resources and land uses;
b) Protect important ecosystems and habitats with rare and endangered flora and fauna, which favours the re-establishment of biodiversity;
c) Prevent or minimize future environmental damage;
d) Enhance the quality of specific environmental resources;
e) Improve the capacity of eligible organizations to protect, restore and enhance the environment; and
f) Undertake resource recovery and waste avoidance projects, and prevent and/or reduce pollution.

The general guidelines of a rehabilitation plan include:

a) Favouring progressive restoration to allow for rapid re-establishment of biodiversity;
b) Implementing a monitoring and surveillance program;
c) Maximizing recovery of previous land uses;
d) Establishing new land uses;
e) Promoting habitat rehabilitation using operational environmental criteria;
f) Ensuring sustainability of restoration efforts.

The mine site rehabilitation plan focuses on land reclamation, reclamation of the tailings area and water basins, and surface drainage patterns to prevent erosion. The successful completion of a rehabilitation plan will ensure that the project incurs a minimum of disturbance. Site inspections will be carried out before the Property is returned to the Government.

The rehabilitation concept for the current project is described below and complies with the requirements set out in the *Guidelines for Preparing a Mining Site Rehabilitation Plan* and *General Mining Site Rehabilitation Requirements* and current legislation.

20.8.2 Mining Site Rehabilitation Plan Concept

Progressive Rehabilitation and Restoration

Progressive restoration is always favoured in order to rapidly achieve the objectives of the rehabilitation program, promote early habitat re-establishment and increase biodiversity.

Progressive restoration will be possible as waste rock piles (or sectors in it) reach their maximum capacity. The same applies to the overburden and topsoil piles, provided the materials can be used.

Final Rehabilitation and Restoration

The final rehabilitation and restoration plan concept is summarized as follows:

Tailings Accumulation Cells and Waste Rock Piles

a) Exposed surfaces of the accumulation areas (tailings accumulation cells, waste rock and overburden piles) will be covered with a layer of topsoil/overburden and re-vegetated.
b) Flotation Tailings Management Facility and the accumulation areas will be covered with a layer of topsoil/overburden and seeded so that vegetation will cover and stabilize the surfaces and prevent erosion.
c) Given that tailings from the POX process are considered high risk under Directive 019, rehabilitation should help eliminate the oxidation and arsenic release process. The POX Tailings Management Facility will therefore be covered with two (2) synthetic, watertight membranes (or a 1.2-m layer of clay and one (1) synthetic, watertight membrane) plus a 60 cm layer of draining sand. The entire area will be covered with 15 cm of overburden and then re-vegetated.
d) Waste rock accumulation areas will be contoured to flatten them out and to further stabilize them against erosion, hence promoting re-vegetation, that is, by creating berms and a gentler slope than the natural slope on a rock pile, in compliance with MRNF safety criteria. The surface of the dump will remain unchanged. No rock pile stability work will therefore be required. The restoration will involve covering the waste rock pile with a layer of about 40 cm

of overburden as a substratum for re-vegetation, seeding plant species and self-sustaining nutrients. The restoration plan should be discussed with all the stakeholders and approved by MDDEP. Given the proximity of the airport and runway position, NAVCAN will be consulted regarding the rehabilitation principles, especially concerning height and types of vegetation.

e) The leachable waste rock pile may leach arsenic. The pile will be covered with a layer of clay 1.2 m thick (monolayer system) to halt the oxidation process, and will then be re-vegetated.

f) Water in the ponds will be treated before being released into the environment. The sludge from the various ponds will be recovered and disposed of in the POX Tailings Management Facility. The ponds will be filled in and re-vegetated.

g) Industrial wastewater treatment unit will be dismantled.

Haul Roads

A one (1) metre layer of material will be removed from the haul roads and disposed of on the leachable waste rock pile. The surface will be scarified and re-vegetated.

Industrial Complex and Buildings

a) No building will be left in place. Whenever possible, buildings will be sold with the equipment they contain, completely or partially. During the dismantling process, beneficiation/recycling of construction material will be maximized. Remaining waste will be disposed of in an appropriate site. Owing to high arsenic levels, materials that may be contaminated with arsenic will first be tested and, if applicable, decontaminated or disposed of at an authorized site or, if permitted, placed in the POX tailings pond.

b) All equipment and machinery will be sold or recycled off-site.

c) Explosives magazine and related facilities will be dismantled.

d) The drinking water supply and domestic wastewater treatment facilities will either be transferred to a competent administrative authority or dismantled.

e) Infrastructure relating to the supply and distribution of electricity will be dismantled if of no use to other parties.

All underground services (power lines, pipelines, water and sewer pipes, etc.) shall remain in place since they are unlikely to cause any environmental damage. Openings and access to such pipelines, however, shall be sealed.

Open Pit

Surface exploitation of materials is common in Québec. Many open pits created to extract materials or ore are therefore found throughout the province. Unlike quarries that are essentially developed on rock outcrops, ore deposits are located below the surface of the soil, which means that pits are filled with groundwater. In many open pit mines, water rises up to the overburden; this is particularly true when there is a thick layer of overburden, as in the area of the Joanna pit.

Once mining activities cease, the pit will gradually fill up to its equilibrium level with rainfall and groundwater. The pit overburden slope around the pit will have been set out safely from the beginning in order to avoid any risk to workers. No special work in this regard will be required upon cessation of mining activities.

To permanently close the pit access roads, two (2)-meter high embankments will be built along with an equivalent crest line using blocks of waste rocks. A ditch two (2) metres wide and one (1) metre deep will be excavated in front of the embankment.

Discussions should be undertaken as required with the relevant authorities, environmental groups and research centers who could take part in developing the final pit restoration plan...

Environmental Aspects

Drainage

- Whenever possible, a surface water drainage pattern will be re-established to conditions similar to the original hydrological system.

Topsoil Management

- During site construction and ore body stripping, overburden and topsoil will be salvaged separately and used for re-vegetation purposes. Slopes of the overburden storage area and flat surfaces will be seeded and re-vegetated.

Waste Management

- Demolition waste will be:
  - Decontaminated when required;
  - Recycled when cost-effective;
  - Buried in an appropriate site.
  - All non-contaminated waste will be sent to an appropriate site.

Hazardous Materials

- Facilities containing petroleum products, chemicals, solid waste, hazardous waste, and/or contaminated soil or materials will be dismantled and managed according to regulatory requirements.
- All hazardous waste will be managed according to existing laws and regulations and will be transported off-site.

Characterization Study

The *Land Protection and Rehabilitation Regulation*, which came into force on March 27, 2003, applies several provisions concerning land protection in the new Section IV.2.1 of the *Environmental Quality Act*. The term *land* also includes groundwater and surface waters. The Regulation sets limit values for a range of contaminants, and specifies the categories of targeted commercial or industrial activities. The mining industry is one of the categories governed by the Regulation.

For the mining industry, this generally entails undertaking a site characterization study within six (6) months following the termination of mine operations. In cases where contamination exceeding the criteria set in the Regulation is observed, a rehabilitation plan specifying the environmental protection measures to be undertaken must be submitted to the MDDEP for approval.

Waste rocks and mine tailings are not soils and are not covered by the Regulation. The characterization study will address the areas that are likely to have been contaminated by human activities, specifically the handling of petroleum products.

Schedule and Costs

Final restoration of the mine site will be completed within three (3) years of the end of commercial production.

Mine closure and restoration costs are estimated at $27.1 million. Restoration costs are summarized in Table 20.12.

Table 20.12: Site Restauration Cost Estimate

DESCRIPTION	QUANTITY	UNIT	UNIT PRICE	TOTAL COST
Flotation Tailings Management Facility				
Overburden fill (15 cm)	407 989	m³	5.00 $	2 039 945.25 $
Hydroseeding	2 719 927	m²	0.80 $	2 175 941.60 $
Tree seedlings (1,500 plants/ha)	285	ha	1 000.00 $	284 892.70 $
Backfilling stormwater retention basins with overburden	129 000	m³	5.00 $	645 000.00 $
Process water basin				
Excavated material (20 cm from bottom of basin) for disposal in POX tailings management facility	26 600	m³	5.00 $	133 000.00 $
Push dikes into basin	Lump sum		7 000.00 $	7 000.00 $
Backfilling with overburden (15 cm)	19 950	m³	5.00 $	99 750.00 $
Hydroseeding	133 000	m²	0.80 $	106 400.00 $
Tree seedlings (1,500 plants/ha)	13	ha	1 000.00 $	13 300.00 $
POX process water basin				
Excavated material (20 cm from bottom of basin) for disposal in POX tailings management facility	5 300	m³	5.00 $	26 500.00 $
Push dikes into basin	Lump sum		4 200.00 $	4 200.00 $
Backfilling with overburden (15 cm)	3 975	m³	5.00 $	19 875.00 $
Hydroseeding	26 500	m²	0.80 $	21 200.00 $
Tree seedlings (1,500 plants/ha)	27	ha	1 000.00 $	26 500.00 $
Process water polishing pond				
Excavated material (20 cm from bottom of basin) for disposal in POX tailings management facility	3 637	m³	5.00 $	18 185.00 $
Push dikes into basin	Lump sum		2 800.00 $	2 800.00 $
Backfilling with overburden (15 cm)	2 728	m³	5.00 $	13 638.75 $
Hydroseeding	18 185	m²	0.80 $	14 548.00 $
Tree seedlings (1,500 plants/ha)	2	ha	1 000.00 $	1 818.50 $
POX Tailings Management Facility				
Clay cover (1.2 m thick)	440 064	m³	5.00 $	2 200 320.00 $
PEHD-type, 2-mm geomembrane	366 720	m²	12.00 $	4 400 640.00 $
Geotextile 918	733 440	m²	3.00 $	2 200 320.00 $
Backfilling with overburden (15 cm)	55 008	m³	5.00 $	275 040.00 $
Hydroseeding	366 720	m²	0.80 $	293 376.00 $
Waste rock stockpile (As and Ni)				
Clay cover (1.2 m thick)	348 000	m³	5.00 $	1 740 000.00 $
Backfilling with overburden (20 cm layer)	58 000	m³	5.00 $	290 000.00 $
Hydroseeding	290 000	m²	0.80 $	232 000.00 $
Waste rock water basin (As and Ni)				
Excavated material (20 cm from bottom of basin) for disposal in POX tailings management facility	400	m³	5.00 $	2 000.00 $
Backfilling with overburden	9 500	m³	5.00 $	47 500.00 $
Hydroseeding	2 000	m²	0.80 $	1 600.00 $

DESCRIPTION	QUANTITY	UNIT	UNIT PRICE	TOTAL COST
Tree seedlings (1,500 plants/ha)	0.2	ha	1 000.00 $	200.00 $
Neutral waste rock stockpile				
Backfilling with overburden (40 cm layer)	651 400	m³	5.00 $	3 257 000.00 $
Hydroseeding	1 628 500	m²	0.80 $	1 302 800.00 $
Tree seedlings (1,500 plants/ha)	163	ha	1 000.00 $	163 000.00 $
Waste rock water basin				
Excavated material (20 cm from bottom of basin) for disposal in POX tailings management facility	1 600	m³	5.00 $	8 000.00 $
Backfilling with overburden	10 200	m³	5.00 $	51 000.00 $
Hydroseeding	8 000	m²	0.80 $	6 400.00 $
Tree seedlings (1,500 plants/ha)	1	ha	1 000.00 $	800.00 $
Low-grade ore stockpile				
Excavated material (1 m) for disposal on waste rock (As-Ni) stockpile	290 000	m³	5.00 $	1 450 000.00 $
Hydroseeding	290 000	m²	0.80 $	232 000.00 $
Low-grade ore stockpile water basin				
Excavated material (20 cm) from bottom of basin) for disposal on waste rock (As-Ni) stockpile	200	m³	5.00 $	1 000.00 $
Backfilling with overburden	3 000	m³	5.00 $	15 000.00 $
Hydroseeding	1 000	m²	0.80 $	800.00 $
Tree seedlings (1,500 plants/ha)	0.1	ha	1 000.00 $	100.00 $
Access road				
Excavated material (1 m) for disposal on waste rock (As-Ni) stockpile	200 000	m³	5.00 $	1 000 000.00 $
Hydroseeding	200 000	m²	0.80 $	160 000.00 $
Tree seedlings (1,500 plants/ha)	20	ha	1 000.00 $	20 000.00 $
Overburden				
Hydroseeding	1 669 940	m²	0.80 $	1 335 952.00 $
Backfilling water retention basin with overburden	31 500	m³	5.00 $	157 500.00 $
Topsoil				
Hydroseeding	603 833	m²	0.80 $	483 066.40 $
Backfilling water retention basin with overburden	8 000	m³	5.00 $	40 000.00 $
Drainage ditch				
Backfilling ditches	12 865	m.l.	5.00 $	64 325.00 $
Total				**27 086 234.20 $**

20.8.3 Monitoring Program and Post-Closure Monitoring

20.8.3.1 Physical Stability

The physical stability of the tailings dams and the waste rock piles will be assessed, and signs of erosion will be noted. This surveillance will be conducted on an annual basis for a minimum of five (5) years following closure of the mine.

20.8.3.2 Environmental Monitoring

Geotechnical and hydrogeological monitoring of tailings ponds and overburden piles will be carried out for a five (5)-year period.

Monitoring of water quality (surface and groundwater) will continue for five (5) years after completion of the restoration work. This monitoring period will be 20 years for the POX Tailings Management Facility.

20.8.3.3 Agronomic Monitoring

The agronomic monitoring program is designed to assess the effectiveness of the re-vegetation done as part of mining rehabilitation efforts.

To document the success of the re-vegetation efforts over the accumulation areas, agricultural monitoring will be undertaken following the establishment of a vegetative cover on the areas subject to the progressive restoration program. Monitoring will be conducted annually for three (3) years following the implementation of vegetation.

Once the mine site closes, the restoration plan will be implemented and the vast majority of the site will be re-vegetated.

Re-vegetation success will be monitored for five (5) years. Re-seeding will be carried out, as required, in areas where re-vegetation is deemed unsatisfactory.

20.9 Community and First Nations Relations

In 2008, Aurizon updated the company's mission to focus on global development and set out 12 guiding principles aiming to consolidate economic, environmental and social considerations.

Since a positive pre-feasibility study was obtained in November 2009, in participation with Joanna's stakeholders, Aurizon has engaged in an ongoing information, consultation and feedback approach. Continuous information activities were undertaken involving local population, regional stakeholders and governmental agencies using fact sheets distribution, presentations, plenary sessions and public open houses. Interests and concerns were addressed during specific meetings, in thematic workshops and through existing land users' committees. Alist of stakeholders is presented in Table 20.13.

Joanna's project lies within the Algonquin Nation territory. In an effort to build and uphold a long-term relationship with the Anishinabeg, Aurizon appointed an Algonquin Community Facilitator/Consultant. The primary objective is to enable members of Pikogan, Lac Simon, Timiskaming and Long Point First Nations to voice their opinions and concerns trough public presentations and ongoing community-based relational committees.

Table 20.13: List of Stakeholders

Ecologist Groups	Action boréale de l'Abitibi-Témiscamingue – ABAT
	Coalition pour que le Québec ait meilleure mine !
	Regroupement des groupes écologistes du Québec – RQGE
Environmental Groups	Organismes de bassins versants du Témiscamingue/Abitibi-Jamésie
	Société des eaux souterraines en Abitibi-Témiscamingue – SESAT
Socio-environmental Groups	Conseil régional en environnement de l'Abitibi-Témiscamingue – CREAT
	Groupe éco citoyen de Rouyn-Noranda – GECO
	Regroupement d'éducation populaire de l'Abitibi-Témiscamingue – RÉPAT
Socio-economical Groups	Conférence régionale des élus de l'Abitibi-Témiscamingue – CRÉAT
	McWatters Residents and Vaudray-Joannès and Bousquet Lakes Riparian
	Vaudray-Joannès Lakes Association of Residents – Board of Directors
	Vaudray-Joannès Lakes Association of Residents – Joanna Mining Project Committee
Municipal Authorities and Services	Mayors and Municipal Councillors - Abitibi Region
	Rouyn-Noranda Land Management Service
	Rouyn-Noranda Watershed Management Service
	Rouyn-Noranda Regional Airport Service
Governmental Authorities and Services	Provincial and Federal Authorities - Abitibi Region
	Ministère du développement durable de l'environnement et des parcs – MDDEP
	Direction des affaires régionales de l'Abitibi-Témiscamingue – MRNF
	Direction de la restauration des sites miniers – MRNF
	Ministère des transports du Québec - MTQ Regional office
	Environnement Canada
	Secrétariat aux affaires autochtones – SAA
Economical Groups	Centre local de développement de Rouyn-Noranda – CLDRN
	Chamber of commerce - Abitibi Region
Mining Industry	Association minière du Québec – AMQ
	Association de l'exploration minière du Québec – AEMQ
	Réseau d'expertise en innovation minière – MISA
Scientific Research Institutions	Unité de recherche et de services en technologie minérale - URSTM
	Institut des sciences de l'environnement – UQAM
	Université du Québec en Abitibi-Témiscamingue – UQAT
	Geological Survey of Canada – GSC

21 CAPITAL AND OPERATING COSTS

The Joanna Gold Project scope covered in this Feasibility Study (FS) is based on the construction of a greenfield facility having a nominal ore processing capacity of 8,500 t/d. The Capital and Operating Cost estimates related to the mine, concentrator, and the site infrastructure, have been developed by BBA. Costs related to environmental and tailings management and site Closure Plan have been developed by Roche. BBA consolidated cost information from all sources. Table 21.1 presents a summary of total estimated initial and sustaining capital costs for the Project. All Cost estimates are expressed in constant April 2012 Canadian Dollars, with an exchange rate at par with the US dollar.

Table 21.1: Total Estimated Initial and Sustaining Capital Costs (M$ CAD)

Category	Initial	Sustaining
Mining	39.0	24.6
Concentrator and Site Infrastructure	301.6	1.0
Environmental and Tailings Management	81.5	44.6
TOTAL	**422.2**	**70.2**
Closure and Restoration	0	27.1

The total initial capital cost including direct costs, indirect costs and contingency has been estimated to be in the order of $422.2M. Mining costs include costs of pre-stripping of waste and overburden required as part of the mine construction phase, which have been estimated at $21.8M. To support continued operations, sustaining capital required over the life of the operation has been estimated to be in the order of $70.2M. An additional amount estimated by Roche in the order of $27.1M is expected to be put in escrow for site closure and restoration related to the permanent closure of waste rock stockpiles and tailings ponds. Of this amount, $20.3M will be deposited during the initial period. The total of Sustaining Capital plus Closure and Rehabilitation Costs is in the order of $97.3M.

Table 21.2 presents a summary of total estimated average operating costs presented in Canadian Dollars (CAD$) per tonne of ore milled and per ounce of gold produced. Operating costs were estimated based on the average cost over the life of the mine. Operating costs exclude pre-stripping costs that have been capitalized.

Table 21.2: Total Estimated Average Operating Cost

Category	$/t Milled	$/oz Gold
Mining	11.92	337
Concentrator and Infrastructure	12.37	349
Sales, General & Administration	0.59	17
Environmental and Tailings Management	0.45	13
TOTAL	**25.32**	**716**
Transportation and Refining	0.11	3
Royalties (varies according to gold price)	0.95	27

The total estimated operating costs are in the order of $25.32 per tonne of ore milled and $716 per ounce of gold produced. Transportation and refining costs as well as royalties paid are presented separately.

21.1 Basis of Estimate and Assumptions

The Capital Cost Estimate pertaining to the processing areas and Joanna site infrastructure within the BBA scope was performed by a professional estimator in BBA's estimation team. Capital costs for the mine were estimated by BBA's mining group. Capital costs for the tailings ponds, environmental management, waste rock disposition areas and site rehabilitation and closure costs were developed by Roche.

21.1.1 Type and Class of Estimate

The Cost Estimate Classification System as defined by BBA, maps the phases and stages of asset cost estimating following these guidelines:

- Provides a common basis of the concepts involved with classifying project cost estimates, regardless of the type of facility, process or industry the estimates relate to;
- Fully defines and correlates the major characteristics used in classifying cost estimates so that companies may unambiguously determine how their practices compare to the guidelines;
- Uses a measured degree of project definition and degree of engineering completion as the primary characteristic to categorize estimate classes; and;
- Reflects generally accepted practices in the cost engineering profession.

The present Capital Cost Estimate pertaining to this Feasibility Study carries an accuracy of +/-15%. Generally, engineering is estimated to be developed in the order of 10-15% and project definition is in the order of 50%. The following list indicated the degree of development of the major study elements:

- Detailed Project Scope Description;
- Detailed Mine Plan and Mine Fleet Requirements;
- Plant Location and Site Plan (50% developed);
- Geotechnical Study; (50 % completed) Detailed Plant Production / Facility Capacity and Description;
- Detailed Process Flow Diagram and Mass and Water Balances;
- Preliminary Building Sizing;
- Preliminary Foundation and Structural Steel Design and MTO's;
- Detailed Process and Mechanical Equipment List;
- Advanced General Arrangements of Major Equipment;
- Preliminary Piping and Valve Lists and MTO's;
- Detailed Electrical Power Requirements and Single Line Diagram;
- Preliminary Environmental and Permitting Overview;
- Project Master Schedule;
- Work Breakdown Structure.

21.1.2 Dates, Currency and Exchange Rates

This Capital Cost Estimate is expressed and presented in constant April 2012 Canadian Dollars (CAD$). The exchange rate was assumed at par; $1 CAD = $1 US.

21.1.3 Construction Labour Rates and Labour Productivity Factors

For the purpose of defining the “Work Week”, all estimated costs for labour are based on ten (10) hours per day, through five (5) days per week, plus two and a half (2.5) hours per week considered as occasional overtime, for a total of fifty-two and a half (52.5) hours per week. There is no allowance for an accelerated construction scheme (night shift, or week end work). The present estimate is structured and based on the philosophy that contracts will be awarded to reputable contractors on a lump-sum basis, within a normal construction labour availability.

The hourly Crew Rates used in the present estimate have been derived from Quebec construction collective agreements dated April 29, 2012. Crew Rates include a mix of skilled, semi-skilled and unskilled labour for each trade, as well as the fringe benefits on top of the gross wages. The supervision by the foremen and surveyors is built into the direct costs. The indirect costs consist of items such as small tools, consumables, supervision by the general foremen, management team, contractors on site temporary construction facilities, mobilization / demobilization, contractor’s overhead and profit. They also include the cost related to the transportation of the employees to and from their residence to the construction site, as well as room and board (if required, applicable at 35% of the work force) for the duration of the construction activities. The Construction Equipment rates are based on the tariff proposed by “La Direction Générale des Acquisitions du Centre de Services Partagés du Québec”, as per edition dated April 1, 2012 and by the “Ontario Provincial Standard Specifications (OPSS)” schedule of rental rates for construction equipment. The cost of fuel (diesel) associated to the construction equipment in the present estimate is estimated at 1 $/litre and considers reimbursements of certain taxes.

A specific Crew Rate was developed for activities related to the construction and installation of equipment within the POX area for hours estimated by Sherritt, based on their experience. This rate insured that there was no duplication of cost components already incorporated within Sherritt’s estimate.

Construction Project Performance is an important concern for project owners, constructors, and cost management professionals. Project cost and schedule performance depend largely on the quality of project planning, work area readiness, preparation and the resulting productivity of the work process made possible in project execution. Labour productivity is often the greatest risk factor and source of cost and schedule uncertainty to owners and contractors alike.

Many terms are used to describe productivity in the construction industry: performance factor, production rate, unit person-hour rate and others. Traditionally, productivity has been defined as the ratio of input / output, i.e., the ratio of the input of an associated resource (usually expressed in person-hour) to real output (in creating economic value). To restate this definition for use in the construction industry: labour productivity is the physical progress achieved per person-hour, e.g., person-hours per linear meter of conduit laid or person-hours per cubic meter of concrete poured. The two most important measures of labour productivity are:

- the effectiveness with which labour is used in the construction process;
- the relative efficiency of labour, doing what it is required, at a given time and place.

Labour productivity factors have been developed for each discipline and applied as a productivity loss factor to the base man-hours developed for each discipline.

21.1.4 General Direct Capital Costs

This Capital Cost Estimate is based on the construction of a greenfield facility having a nominal capacity to process 8,500 t/d of ore. The design of the crusher area, the crushed ore silo area and the concentrator area has been developed by BBA and has largely been based on BBA's experience on other projects. General Arrangement drawings developed in this Study have been used to estimate quantities and generate MTO's for all commodities. Equipment costs have been estimated using vendor prices based on budget proposals for major process equipment and some mechanical equipment packages. Labour rates have been estimated as described previously in this section. Related infrastructure has been estimated by BBA based on the site plan developed. The Capital Cost Estimate for areas requiring special environmental considerations (waste stockpiles and tailings ponds) have been developed by Roche.

BBA has developed this Capital Cost Estimate based on the following assumptions and estimation methodology:

- Mining equipment quantities and costs have been developed by BBA's mining group based on the mine plan developed in this Study. Equipment costs were estimated from vendor budget proposals and from BBA's recently updated database of vendor pricing.
- Pre-stripping costs including overburden and waste rock removal, incurred in the pre-production period, have been capitalized. This Capital Cost Estimate is based on pre-stripping tonnage as defined in the mine plan and includes costs associated with the mining and hauling of overburden, waste rock and ore. It is assumed that overburden removal will be contracted out.
- The mining equipment fleet is based on 100 t capacity haul trucks. This size truck allows for undertaking major overhauls in a local vendor maintenance facility, which greatly reduces on-site requirements for the mining equipment maintenance facility. Therefore, based on this, the garage facilities consist of a mega-dome type enclosure for performing regular maintenance of the mining equipment. An allowance of six (6) million dollars is projected for the installation related to servicing mining equipment. This amount was agreed to be between Aurizon and BBA representatives.
- Mining roads were estimated by BBA's mining department, based on using waste rock from the pre-stripping operation and an assumed unit cost from an outside contractor to crush and screen rock for the application.
- Tasks dedicated to tree clearing including royalties were estimated by Technominex Services. First class excavation represents 122,300 m^3, while second class is estimated at 118,000 m^3. Backfilling materials are evaluated around 320,000 m^3 and it was assumed that all backfill constituents will be available from gravel pits, esker or other sources located within a radius of 5 km.
- The capital cost for the crusher, crushed ore silo and concentrator areas were estimated based on engineering quantities and MTO's developed by the various disciplines. Major process equipment was sized by BBA and budget prices were obtained from vendors or from recently updated BBA database.

- Other mechanical and related equipment quantities and sizes were developed by BBA engineering. Major equipment packages were developed and budget prices were obtained from vendors.
- Site electrical includes all infrastructure to connect to the local power grid, to build a new main substation and to distribute electrical power to the main site areas. Costs of major electrical components identified on the single line diagram were estimated using BBA's electrical engineering group based on recent vendor prices.
- Unit costs for commodities such as concrete, structural steel, metal works and architectural finishes were estimated from local suppliers and BBA's internal database. Concrete quantities are estimated at 16,800 m^3, and structural steel elements represent 3,100 MT.
- Sherritt Technologies developed the flowsheet and major equipment list for the POX area and provided the Capital Cost Estimate for this area. BBA consolidated the cost estimate within its cost estimate and made sure that construction Crew Rates were adjusted as required to avoid gaps and duplications. The total direct cost estimate associated with the Sherritt POX area is in the order of $28.7M.
- The Capital Cost Estimate for Environmental and Tailings Management were developed by Roche. Quantities and unit costs for each of the areas (waste stockpiles, tailings ponds, processing areas, etc.) were developed based on prices obtained from vendors and from experience on other similar projects.
- The cost estimate for Site Rehabilitation and Mine Closure were developed by Roche. Quantities and unit costs for each of the areas (waste stockpiles, tailings ponds, processing areas, etc.) were developed based on prices obtained from vendors and from experience on other similar projects. These costs cover only items related to environmental closure of waste stockpiles and tailings ponds. They do not include costs for dismantling infrastructure which are assumed by BBA to be neutral when considering salvage value of equipment and structures.

21.1.5 Indirect Costs

Indirect Costs for areas under BBA's responsibility for capital cost estimation were factored by BBA based on experience on other projects as described below. It is important to note that these factors were estimated based on the assumption that Aurizon Mines will execute the project using an internal PCM (Procurement and Construction Management) team rather than contracting a major PCM firm. Total indirect costs amount to 20% of direct costs.

- Owner's costs are estimated as a percentage of total direct costs. For this Study, BBA used 4%. Owner's costs include items such as Owner's team salaries and expenses, insurance, authorization certificates and permits, geotechnical and surveying costs, laboratory testwork, etc.;
- Engineering services including procurement, commissioning, and training activities performed by Sherritt Technologies, are estimated at ± 8.25% of direct costs;
- Temporary construction facilities were estimated as 2.5% of direct costs;
- Operations of temporary construction facilities are capitalized and were estimated at 1.25% of direct costs;
- Cost of sub-consultants and other third parties were estimated at 0.3% of direct costs;
- Costs for pre-operational acceptance of mechanical equipment were estimated at 0.2% of direct costs;
- Costs for plant mobile equipment used during construction were estimated at 0.15% of direct costs;
- Other indirect costs and add-ons were estimated at 0.5% of direct costs;
- Costs for spare parts, freight, vendor's reps and other such items were estimated at 8.5% of equipment value;
- For the estimates provided by Roche, indirect costs were included within the direct costs.

21.1.6 Contingency

Contingency provides an allowance to the Capital Cost Estimate for undeveloped details within the Scope of Work covered by the estimate. Contingency is not intended to take into account items such as labour disruptions, weather related impediments, changes in the scope of the Project from what is defined in the Study, nor does contingency take into account price escalation or currency fluctuations. A contingency of 10% of the sum of direct and indirect costs

has been attributed to the Capital Cost Estimate developed in this Study for areas estimated by BBA. A contingency of 20% has been attributed to the Capital Cost Estimate developed by Roche for the Environmental and Tailings Management areas under their responsibility. No contingency was included in the cost estimate for rehabilitation and mine closure developed by Roche.

21.1.7 Exclusions

The following items are not included in this Capital Cost Estimate:

- Inflation and escalation, the estimate is in constant April 2012 Canadian Dollars;
- Costs associated with protection against currency fluctuations;
- All taxes, duties and levies;
- Working capital;
- Project financing costs.

21.2 Estimated Capital Costs

The initial capital cost for the Project is estimated to be in the order of $422.2M. In addition, an initial amount in the order of $20.3M needs to be allocated as sustaining capital, as required by Provincial regulations, for restoration and site rehabilitation. Sustaining capital requirements to support operations over the LOM have been estimated to be in the order of $70.2M. This excludes a further amount estimated at $6.8M required to be put in escrow for site rehabilitation, subsequent to the initial payment mentioned earlier. These costs are used as the basis for the financial analysis of the Project.

21.2.1 Mining Capital Costs

The Capital Cost Estimate pertaining to the mining operation, as developed by BBA, includes costs for mining equipment, pre-stripping costs that have been capitalized, and costs related to the construction of mine roads. Total initial capital cost for the mine is estimated at $39.0M. Sustaining capital required to support the mine plan and ongoing operations is estimated to be in the order of $24.7M.

Costs for the construction of the initial mine roads, estimated at $1.5M, were developed based on using waste rock from the pre-stripping operation that would be crushed and screened by a contractor. Costs for clearing have been estimated based on a proposal from a contractor. Costs for removal of overburden during the pre-production period were assumed at $2.50/t based on past experience. Pre-stripping costs were estimated using the mining operating cost model and were capitalized. The total initial capital cost associated with clearing, overburden removal, and pre-stripping, is in the order of $21.8 M. Costs for mining equipment were developed from vendor budget proposals. Initial costs for mining equipment were estimated at $15.8M. Sustaining capital for additions to the fleet as well as for replacement of certain equipment over the LOM was estimated at $24.4M.

21.2.2 Processing Plant and Site Infrastructure Capital Costs

The estimated capital cost for the various processing areas and major site infrastructure is in the order of $301.6M. Included in site infrastructure are the following major items:

- Mine garage and mine equipment maintenance facility;
- Employee facilities;
- Site access road;
- Main electric substation;
- Civil works including all site pipe works;
- Communications and telecom system;
- Domestic sewage treatment system.

For the processing plant and major site infrastructure, sustaining capital was estimated to be in the order of $1M and is mainly associated with costs for adding pumping capacity for pit dewatering.

21.2.3 Environmental and Tailings Management Capital Costs

The capital cost for Environmental and the Tailings Management Facility was developed by Roche. The total initial capital cost was estimated to be in the order of $81.5M. Sustaining capital estimated to be in the order of $44.6M is required in Years 4 and 8 of the mine life, mainly to increase the capacity of the flotation tailings pond disposal area.

21.2.4 Rehabilitation and Mine Closure Capital Costs

The cost for site rehabilitation and mine closure was developed by Roche. These costs are mainly related to the permanent closure of tailings ponds and stockpiles. They do not include costs for dismantling of the processing plant and site infrastructure.

Proposed regulatory guidelines require that Financial Assurance be posted up front as part of the Rehabilitation and Closure Plan. For the purpose of this Study, it was assumed that $20.3M will be escrowed by Aurizon in the pre-production year and $ 6.8M in the first year of operation. This cost is considered a sustaining capital cost and is treated as such in the Financial Analysis.

21.3 Operating Costs

The average operating cost over the life of the mine has been estimated at $25.32/t of ore milled or $716/oz of gold produced. This excludes transport and refining of gold produced at Joanna as well as contractual royalties to be paid by Aurizon.

Mining operating costs vary from year to year based on the mine plan. Mining pre-stripping costs incurred in the pre-production year (PP) have been estimated using the mining operating cost model but have been capitalized, and therefore are excluded from operating costs.

These costs were based on 41.1 Mt of ore milled and 1,453,510 oz of gold produced over the LOM.

21.3.1 Mining Operating Costs

The average mining operating cost is estimated at $2.23/t mined (including overburden), $11.92/t of ore milled or $337/oz of gold produced. The major cost elements consist of equipment operating costs (which include cost of fuel and cost of maintenance for owned equipment as well as costs for equipment rental for equipment which is required occasionally), blasting, labour and overburden removal (assumed contracted out). The relative percentage of each of these mining operating cost items (over the life of mine), are also indicated in Figure 21.1

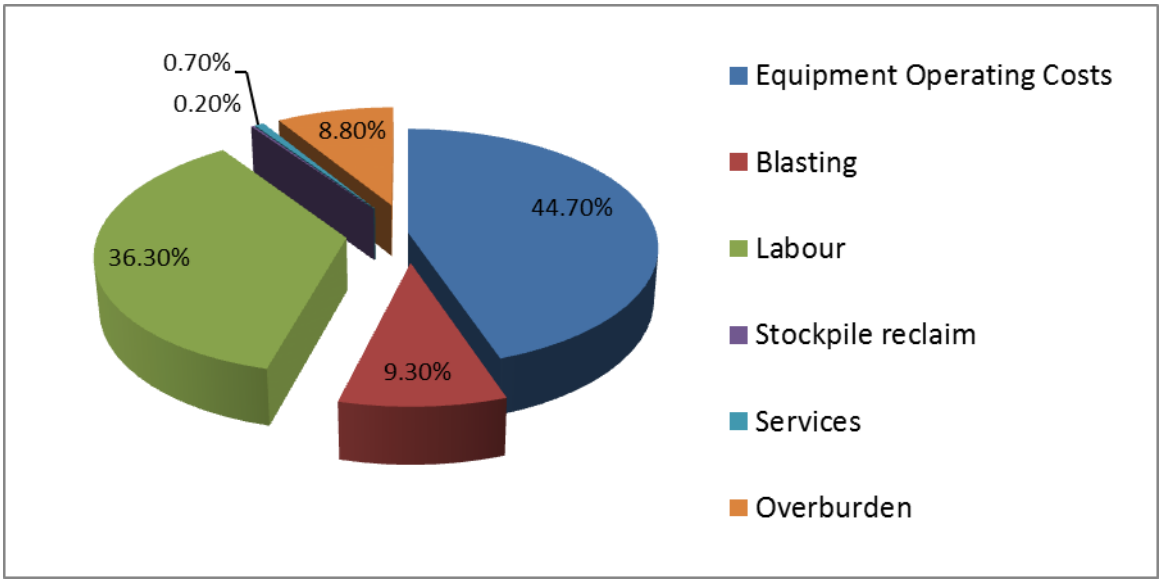


Figure 21.1: Mining OPEX Breakdown over the LOM

The major mining OPEX cost components are as follows:

EQUIPMENT MAINTENANCE

Equipment maintenance costs have been estimated by BBA based on experience, historic data on similar projects and vendor information. Maintenance costs include the costs of repairs, spare parts, consumables, etc., and are compiled on a maintenance cost per hour of operation basis for each equipment type. It should be noted that equipment maintenance costs exclude the cost of maintenance personnel, which are accounted for in mining labour costs.

EQUIPMENT FUEL

Diesel fuel is used to operate all mining equipment. Fuel consumption was estimated for each year of operation based on vendor equipment specifications and equipment utilization on a dollar per operating hour basis. The price of diesel fuel was estimated at $0.90, which includes rebates of certain taxes. Fuel is readily available from local suppliers in the Rouyn area.

BLASTING

Blasting costs for ore and waste rock have been estimated based on parameters and powder factors presented in Section 16 of this Report. Blasting unit costs were estimated at $0.228/t for ore and $0.184/t for waste rock based on a budget price obtained for bulk emulsion. Blasting costs also include fuel for mixing, detonators, boosters and connectors.

LABOUR

Labour requirements and ensuing costs have been determined on an annual basis to support the mine plan developed in this Study. Labour rates have been estimated from data supplied by Aurizon Human Resources Department based on local labour conditions. Labour headcounts have been estimated as previously presented in Section 16 of this Report.

OTHER

Other OPEX items are as follows:

- Costs for additional handling of ore stockpiled when reclaimed were estimated at $0.81/t;
- Cost for removal of overburden by contractor was estimated at $2.50/t;
- An allowance was taken for costs related to mine dewatering. An annual cost of $250,000 was assumed.

21.3.2 Concentrator, Processing and Infrastructure Operating Costs.

The annual and unit operating costs for processing of the Joanna ore were derived from nominal operating conditions. The estimated processing cost is in the order of $12.37/t of ore milled and includes costs related to site infrastructure. Averaged over the LOM, this represents a cost of $349.73/oz of gold produced. This includes all processing from crushing to the production of unrefined gold bars inclusively, as well as the costs of labour and maintenance of equipment. Figure 21.2 presents a breakdown of the operating cost estimate by cost category.

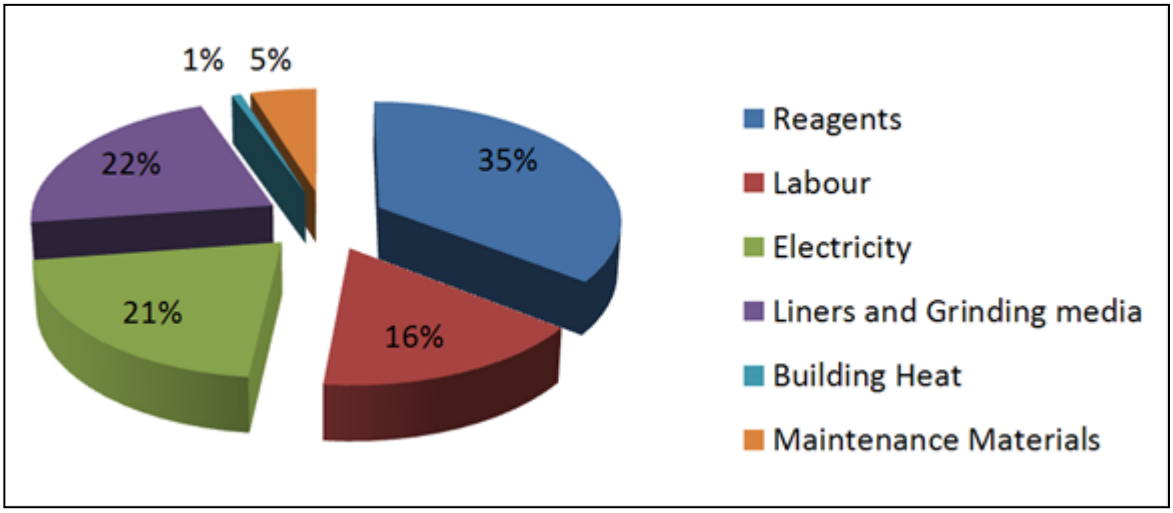


Figure 21.2: Joanna Concentrator Operating Cost By Category

REAGENTS

The cost of reagents is estimated to be in the order of $4.38/t of ore milled. Almost 60% of reagents (which include limestone, lime and oxygen) are associated with the POX area. The consumption of reagents in the various process areas was estimated based on test work results, and unit costs of reagents were based on vendor budgetary quotations and from Aurizon's Casa Berardi operation.

LABOUR

Table 21.3 presents a detailed breakdown of the average headcount over the LOM. Labour rates have been estimated from data supplied by Aurizon Human Resources Department based on local labour conditions.

The estimated cost of labour for the crusher and concentrator is in the order of $2.02/t of ore milled, which represents about 16% of the total processing costs.

Table 21.3: Joanna Concentrator Workforce

Position	Number
Salaried Personnel	
Shift Foreman	4
Concentrator Superintendant	1
Chief Metallurgist	1
Metallurgist	1
Laboratory Supervisor	1
Analyst	1
General Foreman	1
Subtotal	**10**
Hourly Personnel	
Crusher Operator	4
Control Room Operator	4
Flotation Operator	4
POX Sector Attendant	4
CIP/Carbon Regeneration Attendant	4
Assistant Operator	2
Laboratory Attendant	4
Laboratory Attendant (Wet Area/Leco)	2
Laboratory Attendant (Sample Prep.)	2
Metallurgy Technician	3
Gold Room Attendant	1
Mechanic-Fixed Machines	6
Electro-Technician	4
Welder	1
Subtotal	**45**
Total	**55**

ELECTRICITY

Electric power consumption was estimated by BBA and a load list was developed for the Joanna site previously presented in Section 18 of this Report. The power consumption for the grinding mills, which represents approximately 40% of the total site power consumption, was estimated based on ore grindability data and grinding circuit modelling. The power consumption for the oxygen plant was estimated based on oxygen requirements by a vendor which provided technical and commercial information within a budget proposal. The power consumption for the remainder of the concentrator and for site infrastructure was estimated by BBA based on the mechanical equipment selection and sizing. The total annual power consumption for the concentrator and site infrastructure is estimated to be 168.2 GWh at a demand of 23.5 MW.

LINERS AND GRINDING MEDIA

Liner and grinding media consumptions were estimated by BBA based on the ore's abrasion index adjusted to experience of other operating plants within BBA's database. Unit costs were obtained from vendors. The cost of liners and grinding media for the crusher and the grinding mills was estimated to be $2.68/t of ore milled.

BUILDING HEATING AND SERVICES

For this Study, it was assumed that the buildings would be heated using natural gas, which is readily available from a main pipeline passing in the vicinity of the Property. A portion of the heating requirements in the concentrator building is provided by the heat generated within the POX area which produces significant amounts of heat during operations.

OXYGEN

A budget proposal was received from two oxygen plant vendors. For this type of application, vendors have proposed a VPSA type oxygen plant of the required capacity to support operations, including a limited quantity of liquid oxygen emergency back-up. Vendors will build, own and operate the oxygen plant and Aurizon will supply electrical power and clean water as well as a designated area ready to receive the equipment. Commercial terms would include a long-term supply contract (for the LOM) with fixed monthly take-or-pay rate.

MAINTENANCE MATERIAL

The annual cost of maintenance materials was estimated using a typical factor of about 2% of equipment value and represents about $0.59/t of ore milled. This excludes maintenance labour costs which have already been accounted for in labour costs.

21.3.3 Administration

Administration costs consist mainly of on-site personnel required to support operations as well as some incremental corporate Aurizon costs to support the Joanna operation. The estimated total annual administration cost was estimated to be in the order of $1.82M. Of this, $1.53M is attributed to on-site administrative personnel as indicated in Table 21.4. This represents $0.49/t milled. Labour unit costs are 'all-in' and include base wages, fringe benefits, social costs to the employer and other bonuses and premiums in line with local conditions and other Aurizon operations.

Table 21.4: Administration Manpower Estimate

Position	Number
Resident Operations Director	1
Human Resources Superintendant	1
Management Secretary	1
Office Assistant	1
Training Coordinator	1
Health and Safety Superintendant	1
Accounting Technician	1
Procurement Coordinator	1
Warehouse Attendant	2
Nurse	1
Information Technology Technician	1
Total	**11**

21.4 Environment

Costs related to Environmental and Tailings Management were estimated by Roche. They amount to a fixed annual cost of $0.49/t of ore milled. They include such items as:

- Maintenance of dikes, berms, pits and weirs;
- Chemicals and labour for arsenic treatment;
- Electricity;
- General environment follow-up:
  - Effluent water;
  - Underground water;
  - Geotechnical measurements;
  - Biological measurements;
  - Noise measurements;
  - Dust measurements;
  - Reports.

21.5 Other Non-OPEX Cost Items

Non-Opex cost items that are included in the Financial Analysis include costs to transport and refine the gold produced at the Joanna site, as well as royalties to be paid to '90569 Canada Inc.' and Iamgold Qc.

21.5.1 Transportation and Refining

Transportation and refining costs were estimated by BBA at $3.00/oz of gold. This is a typical amount developed based on experience.

21.5.2 Royalties

Aurizon has a contractual obligation to pay royalties to 90569 Canada Inc. Royalties are calculated based on 2% of net revenues (revenues less transport and refining costs). For this Study, the royalty was based on a gold price of $1,350/oz. The obligation toward 90569 Canada Inc. is for Aurizon to pay the amount of $500,000 upon completion of a feasibility study and receipt of decision by a financial institution to finance development of the property to commercial production. This amount is subsequently discounted on the first royalties due, once operations commence.

22 ECONOMIC ANALYSIS

The economic evaluation of the Joanna Gold Project was performed by BBA using the discounted cash flow model on a before-tax basis. The Capital and Operating Cost Estimates input into the Financial Analysis model were based on the mine plan developed in this Study to process 8,500 t/d of ore. The Internal Rate of Return (IRR) on total investment was calculated based on 100% equity financing. The Net Present Value (NPV), based on a discounting rate of 0% and 5% (Base Case) resulting from the net cash flow generated by the Project was also calculated. The payback period is also indicated as a financial measure. A sensitivity analysis was also performed and presented. The following assumptions were made for the financial analysis:

- A production life of **13.4 years** for the Hosco deposit, based on the resource estimate;
- A gold price of **$1350/oz** based on a 36 month moving average to the end of May 2012;
- Production startup 44 months after initiation of environmental permitting process;
- No escalation or inflation factor has been taken into account (constant 2012 $);
- US Dollar at par with Canadian Dollar.

Table 22.1 presents the undiscounted cash flow projection for the Project. BBA assumed that initial capital disbursement takes place over a two year period (year -1 and 0) prior to production startup. Production starts in Yr 1.

Table 22.1: Joanna Gold Project Cash Flow Model

	Construction		Operation Year														
	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	TOTAL
Production ore (Mt)		0.40	2.30	3.31	3.10	3.09	3.10	3.04	3.05	3.09	3.09	3.10	3.07	3.06	3.04	1.25	**41.1**
Gold recovered ('000 oz)			79.89	112.78	121.71	131.35	127.37	99.79	92.58	102.27	106.78	114.73	104.17	103.27	107.97	48.83	**1 453.5**
REVENUE (M$)																	
Revenue from Gold Sale			107.85	152.25	164.31	177.32	171.95	134.72	124.99	138.07	144.15	154.89	140.63	139.42	145.76	65.93	**1 962.2**
OPERATING COSTS (M$)																	
Mine (exc. pre-prod.)			35.52	38.73	40.76	42.74	38.16	38.51	43.01	38.84	38.20	37.03	36.26	30.51	24.19	7.35	**489.82**
Mill			29.00	38.32	38.34	38.22	38.33	37.65	37.70	38.17	38.21	38.31	37.97	37.86	37.61	22.55	**508.3**
Administration			1.82	1.82	1.82	1.82	1.82	1.82	1.82	1.82	1.82	1.82	1.82	1.82	1.82	0.61	**24.3**
Environmental			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	0.46	**18.3**
Refining and transport			0.24	0.34	0.37	0.39	0.38	0.30	0.28	0.31	0.32	0.34	0.31	0.31	0.32	0.15	**4.4**
Royalties			2.15	3.04	3.28	3.54	3.43	2.69	2.49	2.76	2.88	3.09	2.81	2.78	2.91	1.32	**39.2**
TOTAL OPERATING COST			70.1	83.6	85.9	88.1	83.5	82.4	86.7	83.3	82.8	82.0	80.6	74.7	68.2	32.4	**1 084.2**
NET REVENUE (M$)																	
Net Revenue			37.74	68.63	78.37	89.23	88.44	52.37	38.30	54.80	61.33	72.92	60.07	64.76	77.53	33.50	**878.0**
CAPITAL COSTS (M$)																	
Concentrator & Infrastructure	75.4	226.2				0.50				0.50							**302.6**
Mine		39.0	3.6	6.0		3.2	4.9	1.5	3.3	2.1							**63.7**
Environment	20.4	61.2				18.3				26.3							**126.2**
Rehabilitation	13.5	6.8	6.77														**27.1**
Upfront Royalites	0.5		-0.50														
TOTAL CAPITAL COST	**109.8**	**333.2**	**9.91**	**6.05**		**22.01**	**4.85**	**1.50**	**3.34**	**28.85**							**519.5**
CASHFLOW AND NPV (M$)																	
Net Cashflow	**-109.8**	**-333.2**	**27.83**	**62.58**	**78.37**	**67.23**	**83.59**	**50.87**	**34.96**	**25.95**	**61.33**	**72.92**	**60.07**	**64.76**	**77.53**	**33.50**	**358.5**
Cumulative net cashflow	**-109.8**	**-443.0**	**-415.21**	**-352.62**	**-274.25**	**-207.03**	**-123.44**	**-72.57**	**-37.62**	**-11.66**	**49.67**	**122.59**	**182.66**	**247.42**	**324.96**	**358.45**	
NPV @ 5% discount rate	**-109.8**	**-317.3**	**25.24**	**54.06**	**64.47**	**52.67**	**62.38**	**36.15**	**23.66**	**16.73**	**37.65**	**42.63**	**33.45**	**34.34**	**39.16**	**16.11**	**111.6**
Cumulative NPV	**-109.8**	**-427.2**	**-401.93**	**-347.87**	**-283.39**	**-230.72**	**-168.34**	**-132.19**	**-108.53**	**-91.80**	**-54.15**	**-11.51**	**21.94**	**56.28**	**95.44**	**111.55**	

A discount rate is applied to the cash flow to arrive at a NPV and payback period. The NPV calculation was done at 0% (undiscounted Cash Flow) and at 5% discount rate (Base Case). The discount rate selected is a reflection of the cost of capital and the project risk. Table 22.2 presents the results of the financial analysis for the Project based on the assumptions and cash flow projections previously presented.

Table 22.2: Base Case Financial Analysis Results

IRR		8.66%
Discount Rate	**NPV (M$)**	**Payback (Years)**
0%	358.5$	8.2
5%	111.6$	10.3

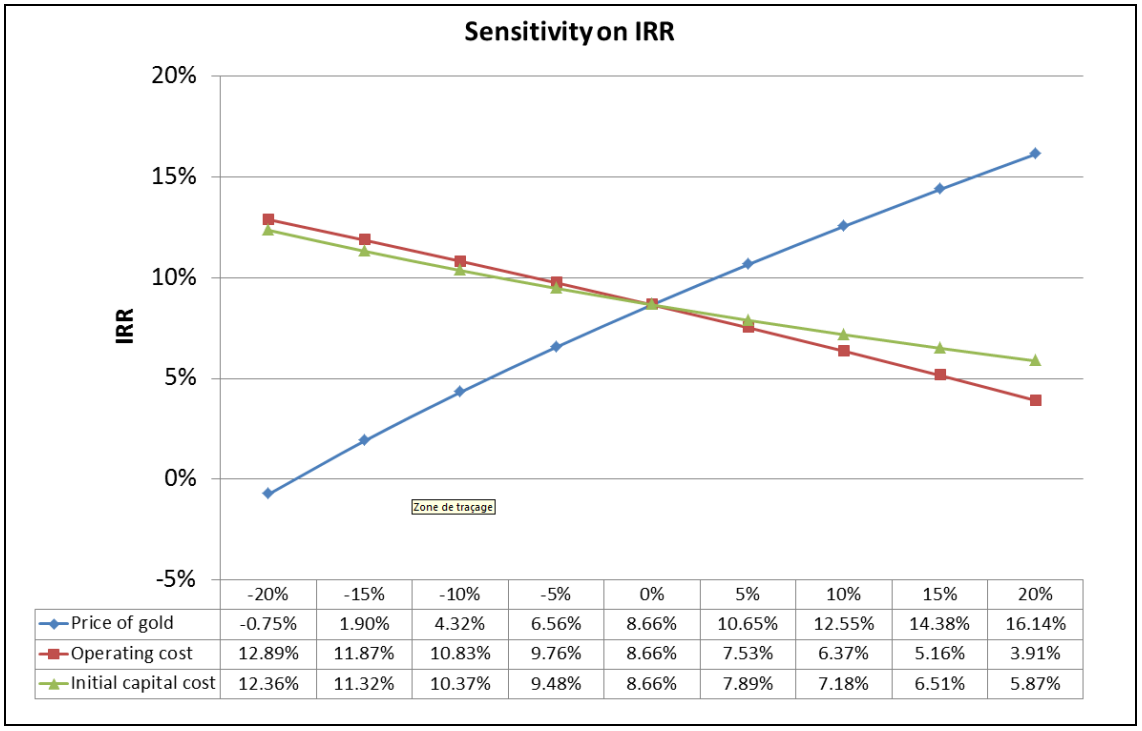
As can be seen, the Project is forecasted to provide an IRR of 8.66% (before tax). At the Base Case discount rate of 5%, NPV is $111.6 M. The Payback period is 10.3 years after the start of production.

22.1 Sensitivity Analysis

A sensitivity analysis was done on parameters that are deemed to have the biggest impact on project financial performance. The following sensitivity analysis was performed and results are presented in Figure 22.1 and Figure 22.2:

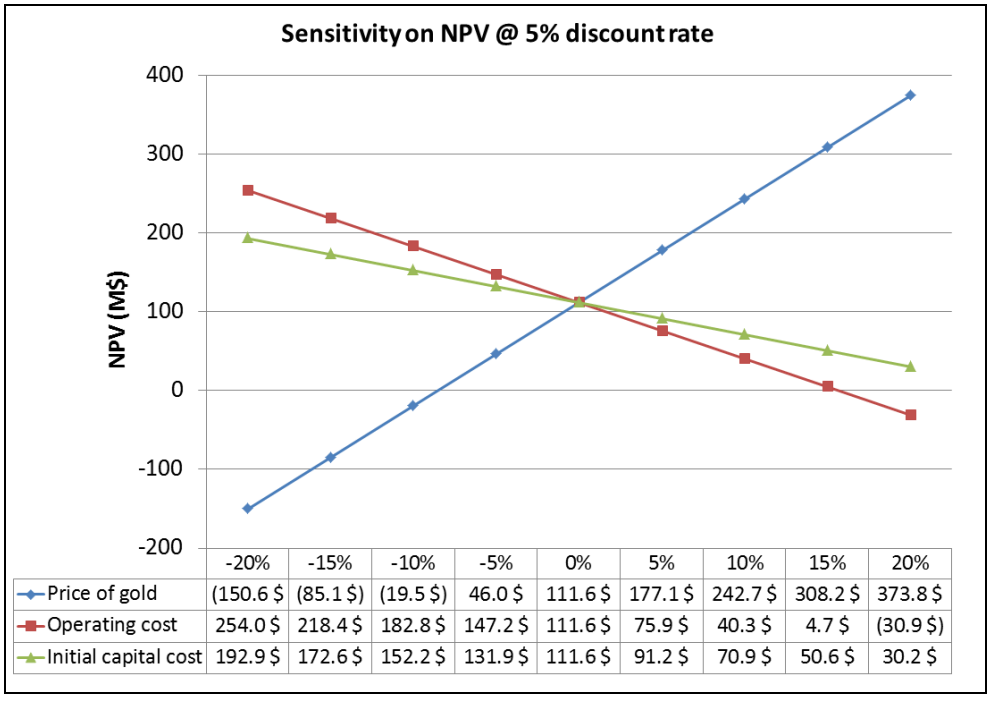
- Base Case CAPEX +/- 20% of initial capital cost;
- Base Case Price of Gold +/- 20%;
- Base Case OPEX estimate +/- 20% per year.

Figure 22.1and Figure 22.2 show a graphical representation of the sensitivity analysis. As is indicated, IRR and NPV at a 5% discount rate are more sensitive to variations in operating cost than to variations in initial capital cost. Variations in the gold price affect the IRR and NPV to a much greater extent than variations in operating costs and capital costs.



	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Price of gold	-0.75%	1.90%	4.32%	6.56%	8.66%	10.65%	12.55%	14.38%	16.14%
Operating cost	12.89%	11.87%	10.83%	9.76%	8.66%	7.53%	6.37%	5.16%	3.91%
Initial capital cost	12.36%	11.32%	10.37%	9.48%	8.66%	7.89%	7.18%	6.51%	5.87%

Figure 22.1: Sensitivity on IRR (%)



	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Price of gold	(150.6 $)	(85.1 $)	(19.5 $)	46.0 $	111.6 $	177.1 $	242.7 $	308.2 $	373.8 $
Operating cost	254.0 $	218.4 $	182.8 $	147.2 $	111.6 $	75.9 $	40.3 $	4.7 $	(30.9 $)
Initial capital cost	192.9 $	172.6 $	152.2 $	131.9 $	111.6 $	91.2 $	70.9 $	50.6 $	30.2 $

Figure 22.2: Sensitivity on NPV (M$)

A further sensitivity analysis, present in Table 22.3 was performed to assess the impact of a higher dilution rate on project return. It was determined that an increase in the dilution rate from 6.5% to 15% would result in a reduction in NPV (5% discount rate) of 28.8 M$ and a reduction of IRR of -1.2%.

Table 22.3: Sensitivity Analysis on Dilution Rate

Dilution Rate	Delta NPV @ 5% (k$)	Delta IRR
6.5% (Base Case)	0 $	0%
10%	(16 412 $)	-0.7%
15%	(28 750 $)	-1.2%

22.2 Risk Management

22.2.1 Scope

BBA's approach to Risk Management in the context of this Study is to identify major risks associated with this Project in terms of the following categories:

- Technology and technical risk;
- Risk related to meeting the targeted project schedule;
- Risk affecting CAPEX;
- Risk affecting OPEX;
- Risk affecting Profitability;
- Risk affecting Closure Costs.

Risk Management is a continuous process that is performed over the full life-cycle of a project; therefore, Risk Management is only complete when the project is complete. Consequently, the data and information presented in this Report is a snapshot of the project major risk profile as understood on the effective date of this Report. It will be noted that because of the continuous nature of the Risk Management process, many open risk issues exist at this time. A review of the Risk Register will show that not all risks have been fully evaluated nor are they

accompanied by well-defined mitigation plans or actions since these are to be updated regularly.

22.2.2 Risk Assessment Methodology

Risk identification is the process of examining the various project areas and each critical process in order to identify and document any associated potential risks. A risk review meeting was held at BBA's offices on April 10th, 2012 with the participation of BBA, Roche and Aurizon.

For this Study, risk assessment was conducted on a qualitative basis. The Probability of Occurrence was assessed and qualified on three levels. The Manageability of the risk, which indicates how effectively the risk may be mitigated, was also qualified on three levels. These are indicated in Table 22.4. The consequence associated with the risk was qualified in the Risk Register but only as a description.

Table 22.4: Classification of Probability and Manageability

Label	Probability	Manageability
High	Likely to Occur	Direct control with a clear solution
Medium	May Occur	No control with a solution or some control with no clear solution
Low	Unlikely to Occur	No control and no clear solution

From this table it is understood that a risk with a High Probable Consequence and a Low Manageability should receive a high priority for mitigation activities and resources.

Table 22.5 presents a summary of the risk items identified and the risk category for each risk item and should be used in the next engineering phase of the Project.

Table 22.5: Risk Register

Revision No :	3
Document Owner :	Angelo Grandillo
Date :	May 24, 2012

Risk Register
JOANNA Gold Project

Legend

Probability	Manageability
High: Likely to occur	**High:** Direct control/Clear solution
Med: May occur	**Med:** Cont/No sol or No cont/Clear sol
Low: Unlikely to occur	**Low:** No control/No clear solution

N°	Risk Title	Risk Description	Category	Consequence (Description)	Probability	Manageability	Mitigation	Comment
1.	Environmental Permitting	1.1. Process for obtaining provincial and federal permits longer than planned	Schedule	1.1.1. Delay in construction	Medium	Low	-Improve quality of ESIA document to limit questions from authorities and therefore limit delay	
2.	Stakeholders and First Nations	2.1. Attempt to modify project	Schedule	2.1.1. Delayed construction and start up	Low	Medium	-Establish good communication with community and stake holders	
			CAPEX	2.1.2. Increase in CAPEX	Low	Medium		
3.	Electric Power Line	3.1. Hydro Quebec delay in providing timely connection to power grid	Schedule	3.1.1. Delayed plant start up	Low	High	-Hydro Québec has completed study and will provide power from beginning of construction	
4.	Tailings Management	4.1 High density flotation tailings thickening not achievable	Capex/Schedule	4.1.1 May change footprint of the tailings pond 4.1.2 Will not be able to operate without a revised permit	Low	High	-Testwork already done by vendor shows that required % solids levels are achievable with a high-wall/high-rate thickener	
		4.2 High density tailings pumping and stacking not technically feasible	CAPEX/Schedule	4.2.1 May change footprint of the tailings pond 4.2.2 Will not be able to operate without a revised permit	Medium	High	-Rheology testwork needs to be performed prior to final design	
		4.3 Potential contamination of surface waters	CAPEX/OPEX /Closure	4.3.1 Redesign of water management infrastructure 4.3.2 Increased water treatment costs 4.3.3 Increase rehabilitation costs	Low	Medium	-Comply with all legal requirements (Directive 019, etc.) -Perform geochemical tests (ARD/ML potential) -Design provides for water from tailings ponds to be treated if required prior to release to environment	
5.	Hydrogeology	5.1 Potential contamination of surface waters (quality)	OPEX/Closure	5.1.1 Increase in monitoring and rehabilitation costs	Low	High	-Comply with all legal requirements (Directive 019, etc.). -Provide adequate design to treat water prior to release to environment	
		5.2 Fish habitat (quantity)	OPEX/Closure	5.2.1 Increase in monitoring and rehabilitation costs 5.2.2 May require additional compensation	Low	High	-Comply with all legal requirements (Directive 019, etc.). -Provide adequate design to treat water prior to release to environment	
6.	Hydrology and Water Management	6.1 Insufficient year-round water availability to ensure continuous operations	CAPEX	6.1.1 Interruption of production	Low	High	-Design retention basin with sufficient capacity to ensure adequate water supply during the dry season	
		6.2 Potential contamination of surface waters (quality)	OPEX/Closure	6.2.1 Increase in monitoring and rehabilitation costs	Low	High	Comply with all legal requirements (Directive 019, etc.). Provide adequate design to treat water prior to release to environment	
		6.3 Fish habitat (quantity)	OPEX/Closure	6.3.1 Increase in monitoring and rehabilitation costs	Low	High	-Comply with all legal requirements (Directive 019, etc.). -Provide adequate design to treat water prior to release to environment	
7.	Pit Slope Stability	7.1 Overburden stability	Resource/OPEX	7.1.1 Shallower slopes resulting in reduced resource 7.1.2 Bigger pit footprint resulting in high strip ratios	Low	Medium	Perform further drilling prior to final design	Inherent risk related to overburden properties
		7.2 Bedrock stability	Resource/OPEX	7.2.1 Shallower slopes resulting in reduced resource 7.2.2 Bigger pit footprint resulting in high strip ratios	Medium	Medium	Perform further drilling prior to final design	Inherent risk related to bedrock rock mechanics

Revision No :	3
Document Owner :	Angelo Grandillo
Date :	May 24, 2012

Risk Register
JOANNA Gold Project

Legend

Probability
High: Likely to occur
Med: May occur
Low: Unlikely to occur

Manageability
High: Direct control/Clear solution
Med: Cont/No sol or No cont/Clear sol
Low: No control/No clear solution

N°	Risk Title	Risk Description	Category	Consequence (Description)	Probability	Manageability	Mitigation	Comment
8.	Waste Rock Management	8.1 Potential contamination of surface waters	CAPEX/OPEX /Closure	8.1.1 Increase CAPEX 8.1.2 Increase OPEX (water treatment) 8.1.3 Increase rehabilitation costs	Low	Medium	-Comply with all legal requirements (Directive 019, etc.) -Perform geochemical tests (ARD/ML potential)	
		8.2 Higher quantity of Ni/As waste rock generated	CAPEX	8.2.1 Larger footprint of Ni/As waste rock pile	Low	Medium	-Incorporate Ni/As in the block model to validate quantities -Optimize segregation of Ni/As rock from regular waste in the mining operation	
9.	Site Wide Geotechnical	9.1 Poorer than expected soil/rock condition	CAPEX	9.1.1 Heavier foundations required	Medium	High	-Perform more drilling prior to final design -Relocate affected infrastructure to a more favorable area	
		9.2 Rail too close to the pit	CAPEX/Resource	9.2.1 Loss of resource	Medium	High	Relocate railway	
10	Procurement of Long Lead Items	10.1. If vendors are busy, lead times will be longer than expected	Schedule	10.1.1. Delayed construction and plant start up	Medium	High	-Define a pre-purchase strategy as part of project early works -Identify alternate sourcing	
11	Construction	11.1. Longer than expected construction schedule	Schedule	12.1.1. Delayed construction and plant start up	Medium	Medium	-Early discussions with potential contractors to secure labour force	
		11.2 Construction delays resulting from inexperience in installation of autoclave	CAPEX/Schedule	12.2.1 Delayed construction and plant start up	Medium	Medium	Identify key qualified personnel and secure their services early in the process	
		11.3 Inability to perform construction management with owner's team	CAPEX	12.3.1 Full service EPCM contractor will be required	High	High	Owner's to hire a sufficient number of qualified personnel	
12	Site Restoration	12.1 Management of contaminated site/buildings	Closure Costs	12.1.1 Increase rehabilitation costs (disposal of contaminated material)	High	High	Establish a management plan	
13	Health and Safety and Hygiene	13.1 Risks related to exposure to As bearing dust	CAPEX/OPEX	13.1.1 Effects to personnel health and safety	High	High	-Perform a detailed HAZOP prior to final design -Personal protection equipment -Dust abatement equipment -Customized work practices	
14	Processing	15.1 Ore hardness higher than expected	CAPEX/OPEX	14.1.1. Throughput bottle-neck in grinding area	Medium	High	-Perform more grindability tests prior to final design -Size the SAG mill more conservatively at the expense of increased CAPEX	
		15.2 Variability in concentrate amenability to POX/Leaching	Profitability	14.2.1. Lower than expected gold recovery	Medium	Medium	Autoclave and leach circuit are designed with conservative residence times	Inherent risk to project
		15.3 Plant availability lower than expected	Profitability	14.3.1 Lower than expected gold production	Low	High	Install required redundancy	Inherent availability related to autoclave

23 ADJACENT PROPERTIES

As illustrated in Figure 23.1, the Joanna property is surrounded by claims owned mostly by Agnico Eagle Mines Ltd., Iamgold-Québec Management Inc., Xstrata Plc., and Breakwater Resources Ltd. No resources are delimited on those close properties while Iamgold property owns the former Rouyn-Merger orebody.

More details on adjacent properties can be found in Geostat's report of November 2009.

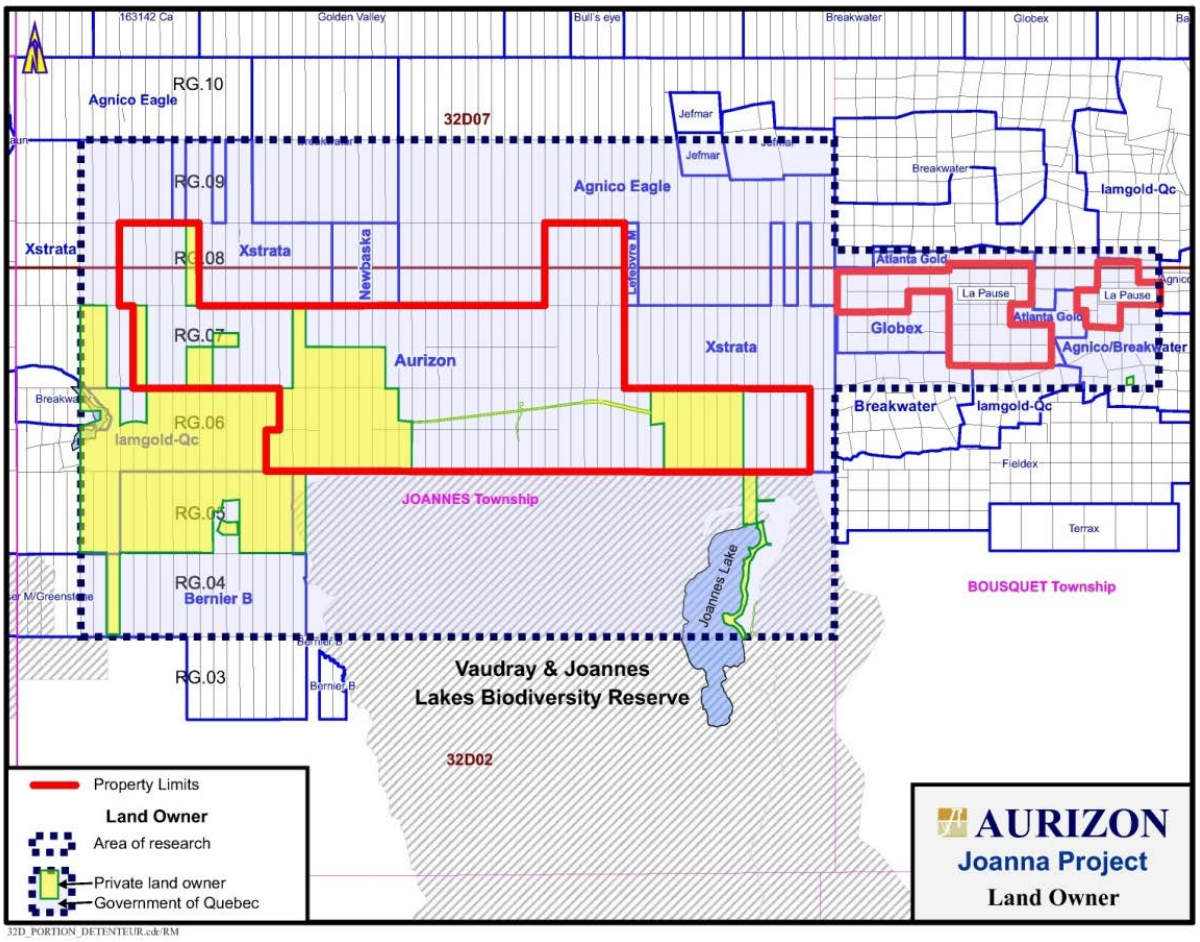


Figure 23.1: Adjacent Properties

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation Schedule

The implementation schedule for the Joanna Project is presented in Figure 24.1. It was developped based on the following assumptions:

- The schedule begins with Aurizon Mines issuing the project notice to the authorities;
- The environmental permitting process is projected to last 21 months;
- The construction period is assumed to last 20 months and will not begin until the environmental permit is received.
- The purchase order for long lead items will be issued so as not to impact the construction schedule. As of the effective date of this report, the main long lead items are as follows:
  - Autoclave delivery time is 18 months;
  - Oxygen plant delivery time is 24 months.

Based on the preliminary schedule developed, it is expected that plant operations will begin about 44 months after the environmental permitting process is initiated.

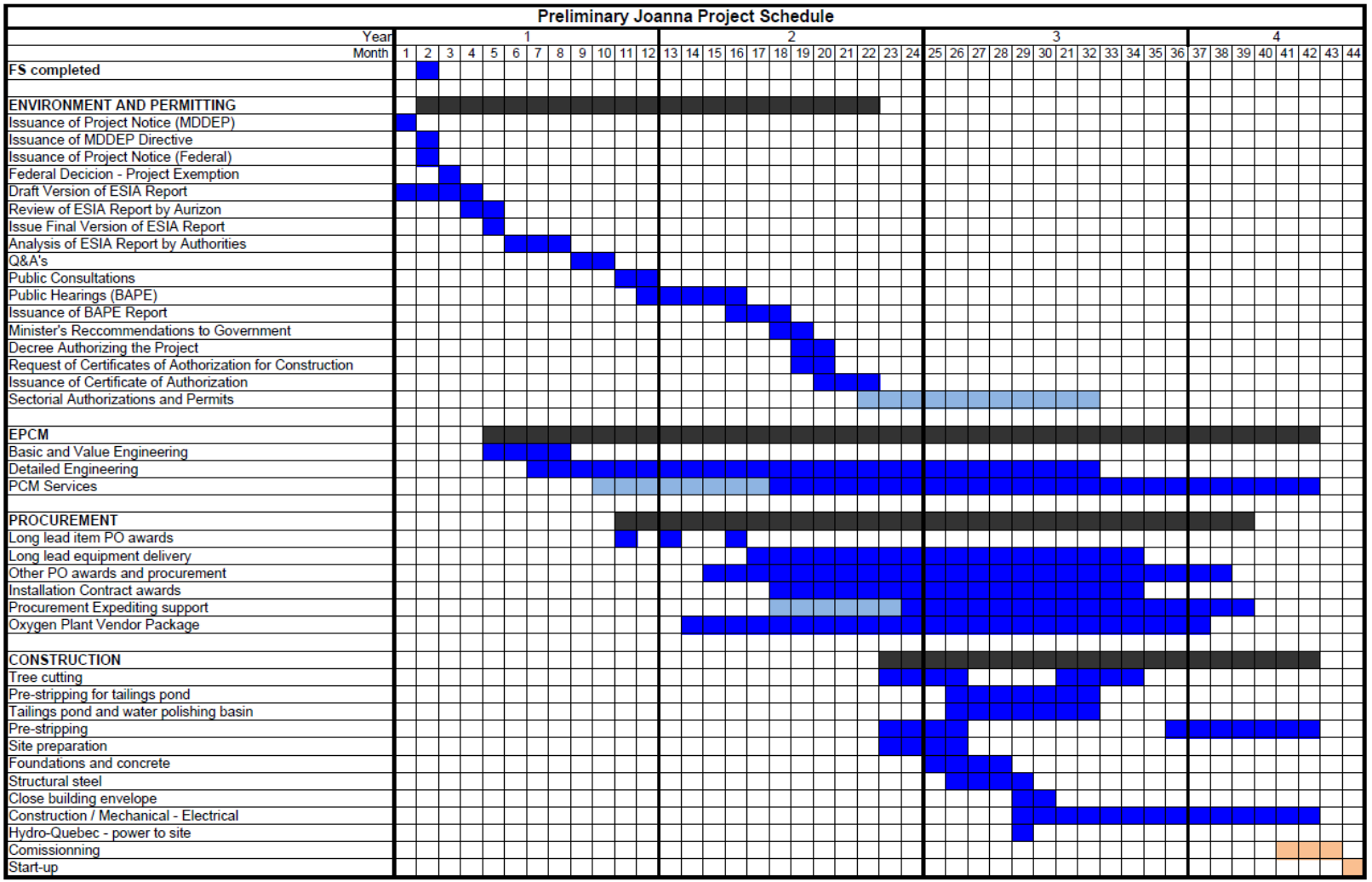


Figure 24.1: Project Implementation Schedule

25 INTERPRETATION AND CONCLUSIONS

Much progress has been made in the development of the Joanna Project since the publishing of the Pre-Feasibility Study in December 2009. Additional drilling has allowed for a significant increase of mineral reserves (in the order of 67%) and in-pit mineral reserves have now been established at 1.45 million ounces of gold recoverable in the Hosco deposit. During the course of the Feasibility Study, a detailed analysis of numerous mine plan options was performed by BBA and the most cost effective plan was retained as the Base Case mine plan for this Study. Extensive metallurgical testwork highlighted the complexity of the Joanna deposit mineralization and resulted in reorienting the oxidation step in the Process Flowsheet from the Albion Process to the Pressure Oxidation (POX) Process. Furthermore, all environmental aspects and impacts were studied in great detail and allowed for the development of a solid environmental management plan for tailings and waste disposal. The increase in the level of engineering detail developed during the course of the Feasibility Study allowed for significant de-risking of the Project when compared to the Pre-Feasibility Study. As such, the capital cost and operating cost estimates associated with the Project have been developed to a much better degree of accuracy.

25.1 SGS Geostat Interpretations and Conclusions on Mineral Resources

As part of the independent verification program, SGS Geostat conducted a review of the exploration methodology used by Aurizon Mines including sampling methodology, sample preparation and analysis, and QA/QC protocol. Data verification consisting of validation of the digital database used for the Project and collection of independent duplicate samples from recent mineralized core samples drilled by Aurizon Mines was also completed as part of the verification program.

SGS Geostat completed an updated mineral resource estimate of the Hosco deposit based on additional surface diamond drilling done by Aurizon Mines since the last mineral resource estimate of June 2011. The results of this updated mineral resource estimate, as reported in the NI 43-101 Technical Report published in March 2012, constitute the current resource estimate for the Hosco deposit and were used in the development of this Feasibility Study NI 43-101 Technical Report. The current mineral resource estimate for the Hosco deposit is defined,

based on two distinct mining scenarios, in-pit and underground, using distinct Base Case gold cut-off grade although only the in-pit resources were considered for this Feasibility Study. Also, as described in the June 13, 2011 Company news release, the mineral resources of the Heva deposit were not updated at that time but were restated using a modified cut-off grade for the mineral resources located below 300 m from the surface. The restatement of the Heva deposit mineral resource disclosed on June 13, 2011 was made to better reflect the reasonable prospect of economic extraction in relation to the advanced stage of exploration of the Hosco deposit located to the east. The restated mineral resource estimate for the Heva deposit disclosed in June 2011 is used in this Report. As described in the Company's June 13, 2011 news release, as the updated resource outline of the Hosco deposit disclosed in the news release crossed over the Alexandria claim boundary, all the resources of the Alexandria sector were restated. However, the Alexandria resource estimate was not updated in June 2011. The restated mineral resource estimate for the Alexandria sector disclosed by the Company on June 13, 2011 is used in this Report.

This Feasibility Study only considers resources classified as reserves for the Hosco deposit. It should be understood that mineral resources which are not mineral reserves do not have demonstrated economic viability.

The final updated mineral resources for the Hosco deposit are as follows: The in-pit mineral resources using a Base Case cut-off grade of 0.33 g/t Au totals 30,690,000 tonnes grading 1.33 g/t Au for 1,311,000 oz Au in the measured category; 27,150,000 tonnes grading 1.18 g/t Au for 1,033,000 oz Au in the indicated category with an additional 7,050,000 tonnes grading 1.18 g/t Au for 267,000 oz Au in the inferred resources category. The U/G mineral resources using a Base Case cut-off grade of 2.0 g/t Au totals 50,000 tonnes grading 2.65 g/t Au for 5,000 oz Au in the indicated category with an additional 590,000 tonnes grading 2.54 g/t Au for 48,000 oz Au in the inferred resources category. Table 25.1 and Table 25.2 summarise the mineral resources for the Hosco deposit using cut-off grades of 0.33 g/t Au (Base Case), 0.5 g/t Au and 1.0 g/t Au for the in- pit scenario and cut-off grades of 2.0 g/t Au (Base Case), 2.5 g/t Au and 3.0 g/t Au for the underground scenario.

Table 25.1: September 2011 Updated Mineral Resources for the Hosco Deposit (In-Pit)

Grade Cut-off (Au g/t)	Category	Tonnage* (t)	Grade (g/t)	Au metal** (oz)
0.33 (Base Case)	Measured (M)	30,690,000	1.33	1,311,000
	Indicated (I)	27,150,000	1.18	1,033,000
	Total (M+I)	57,840,000	1.26	2,344,000
	Inferred	7,050,000	1.18	267,000
0.5	Measured (M)	29,460,000	1.37	1,295,000
	Indicated (I)	26,410,000	1.21	1,024,000
	Total (M+I)	55,870,000	1.29	2,319,000
	Inferred	6,950,000	1.19	266,000
1.0	Measured (M)	20,370,000	1.62	1,058,000
	Indicated (I)	15,130,000	1.51	736,000
	Total (M+I)	35,500,000	1.57	1,794,000
	Inferred	4,520,000	1.39	201,000

Updated: September 22, 2011- *Rounded to nearest 10k- **Rounded to 1k
Relative density used: 2.75t/m3
CIM Definitions were followed for mineral resources
Mineral resources which are not mineral reserves do not have demonstrated economic viablilty
Historical production of 9,704 oz has not been removed from mineral resources

Table 25.2: September 2011 Updated Mineral Resources for the Hosco Deposit (Underground)

Grade Cut-off	Category	Tonnage (t) (t)	Grade (g/t)	AuMetal** (oz)
2.0 (Base Case)	Indicated (I)	50,000	2.65	5,000
	Inferred	590,000	2.54	48,000
2.5	Indicated (I)	20,000	3.33	2,000
	Inferred	180,000	3.39	19,000
3.0	Indicated (I)	10,000	3.90	1,000
	Inferred	100,000	4.00	12,000

Updated: September 22, 2011- *Rounded to nearest 10k- **Rounded to 1k
Relative density used: 2.75t/m3
CIM Definitions were followed for mineral resources
Mineral resources which are not mineral reserves do not have demonstrated economic viablilty

The current NI 43-101 compliant mineral resources for the Joanna Property, which include mineral resources for the Heva and Alexandria deposits of the Property, are summarized in Table 25.3.

Table 25.3: Current Mineral Resources for the Joanna Property

Sector	Resource Depth	Resource Category	Cut-off Grade (g/t)	Tonnage* (t)	Au Grade (g/t)	Au Metal** (oz)
Hosco	In-pit	Measured	0.33	30,690,000	1.3	1,311,000
Hosco	In-pit	Indicated	0.33	27,150,000	1.2	1,033,000
Heva	Above 4,700m Elev.	Indicated	0.5	4,410,000	1.9	270,000
Joanna	**In-pit/Above 4,700m Elev.**	**Total M + I**		**62,250,000**	**1.3**	**2,614,000**
Hosco	In-pit	Inferred	0.33	7,050,000	1.2	267,000
Heva	Above 4,700m Elev.	Inferred	0.5	7,680,000	1.7	421,000
Alexandria	Above 4,700m Elev.	Inferred	0.5	980,000	1.2	37,000
Joanna	**In-pit/Above 4,700m Elev.**	**Total Inferred**		**15,710,000**	**1.4**	**725,000**
Hosco	U/G	Indicated	2.0	50,000	2.6	5,000
Joanna	**U/G - Below 4,700 m Elev.**	**Total Indicated**		**50,000**	**2.6**	**5,000**
Hosco	U/G	Inferred	2.0	590,000	2.5	48,000
Heva	Below 4,700 m Elev.	Inferred	2.0	650,000	2.8	59,000
Joanna	**U/G - Below 4,700 m Elev.**	**Total Inferred**		**1,240,000**	**2.7**	**107,000**

*Rounded to nearest 10k - **Rounded to nearest 1k

Updated: September 22, 2011

CIM definitions were followed for mineral resources

Mineral resources which are not mineral reserves do not have demonstrated economic viability

Historical production of 9,704 oz (Hosco) and 10,700 oz (Heva) has not been removed from mineral resources

The updated mineral resources of the Hosco deposit were estimated from two distinct block models: one for high grade gold mineralization and one for the low grade mineralization, most of which is located in the surroundings of the high grade model. The high grade block model has been interpolated from 2 m interval composite data constrained within 3D wireframe solids defined from the channel and drillhole mineralized intercepts. The low grade block model was estimated from composite data of similar length located outside the defined high grade wireframe solids. Both block models are defined by a block size of 8 m (E-W) by 5 m (N-S) by 8 m (vertical) and cover an area located with sections 7050mE to 10,000mE of the Project to a maximum depth of more than 980 m below surface. The interpolation of the block grades was completed using OK methodology with multiple passes using search ellipsoids that increased in size from one interpolation pass to another. The final updated mineral resources correspond to the estimated blocks from both block models located below the bedrock-overburden interface as well as outside known barren late intrusive units observed in the deposit area. The updated mineral resources were finally classified into measured, indicated and inferred using an automated classification process followed by a manual smoothing to produce coherent mineral resource categories. A bulk density of 2.75 t/m^3 was used to calculate the final tonnages of the mineral resources based on the volumetric estimates of the block models.

SGS Geostat validated the core sampling and QA/QC procedures used by Aurizon Mines as part of an independent verification program and concluded that the drill core handling, logging and sampling protocols, including the insertion control samples into the sample stream for the Project, are at conventional industry standard and conform to 'generally accepted' best practices. SGS Geostat is confident that the system is appropriate for the collection of data suitable for the estimation of a NI 43-101 compliant mineral resources.

As part of the verification program, SGS Geostat validated the Project's digital database and conducted independent sampling of mineralized core duplicates from recent drillholes done by the Company. The author and SGS Geostat are of the opinion that the database provided for the current resource estimate is valid and the data is acceptable for estimation of mineral resources.

25.2 Mining (Mineral Reserves)

The reserves for the detailed pit design have been calculated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM Standards on Mineral Resources and Reserves) in August, 2000. The mineral reserves (with dilution and ore loss) contained within the Main pit and West pit amount to 41.1 Mt of ore at an average grade of 1.26 g/t Au using a cut-off grade of 0.5 g/t Au, for a total of 1.66 Moz of in-situ gold and 1.45 Moz of recovered gold. Total waste, including rock, inferred material and overburden, is 184.6 Mt for a stripping ratio of 4.49. The detailed mineral reserve estimate is shown in Table 25.4.

Table 25.4: Mineral Reserve Estimate (Cut-off 0.5 g/t Au)

	Main Pit		West Pit		TOTAL			
Material	Material (k tonnes)	Grade (g/t Au)	Material (k tonnes)	Grade (g/t Au)	Material (k tonnes)	Grade (g/t Au)	Au koz (in-situ)	Au koz (recovered)
Proven Reserves	27 419	1.29	817	1.71	28 236	1.30	1 185	1 040
Probable Reserves	11 356	1.11	1 496	1.47	12 851	1.15	476	414
TOTAL	**38 775**	**1.24**	**2 313**	**1.56**	**41 087**	**1.26**	**1 660**	**1 454**
Waste	150 587		11 733		162 320			
Overburden	18 338		3 973		22 311			
TOTAL	**168 925**		**15 706**		**184 631**			
Strip Ratio	**4.36**		**6.79**		**4.49**			

Note: Mill Recovery Equation = $-0.0467\ Au^2 + 0.1745\ Au + 0.7299$

Conventional open pit mining methods will be used. The mining equipment selection is based on the use of 100-ton class haul trucks, hydraulic excavators, production drills and various ancillary equipments. Mine operations were designed to support an average daily production rate of 8,500 tonnes of ore. Initial production will start from a smaller pit and will be extended in two additional pushbacks; mining of the West Hosco pit is expected to begin in Year 4. The overburden consists of silt and clay and varies in thickness between six (6) and 15 metres. It is planned that the overburden will be removed during winter seasons.

The pits were designed with a triple benching arrangement and include a 15-metre geotechnical safety berm at every 120 metre vertical height. Based on the results of rock mechanic studies, (Golder Associates, June 2009), the recommended inter-ramp pit slope will be 53° for the hanging wall (north), 49° for the footwall (south) as well as both east and west sectors.

25.3 Metallurgy and Ore Processing

Gold mineralization in the Hosco deposit is associated with disseminated sulphides, mostly arsenopyrite (main gold bearing mineral) and pyrrhotite. As the material is partially refractory, the selection of an oxidation method before the cyanidation of a flotation concentrate became necessary to improve and optimize the level of gold recovery. The consistency of the results obtained at the pilot-plant level, combined with higher recoveries, led to the selection of the conventional Pressure Oxidation (POX) technology utilizing an autoclave as the preferred method of oxidation for the Hosco ore. This technology is well established and proven for this application. Sherritt Technologies performed all testwork and process development for the POX area of the plant.

Metallurgical testwork provided data for the following:

- The ore competency is considered to be hard to very hard with respect to the SGS database, thus the grinding energy requirement is an important design and operating factor.
- Determining gold deportment, gold recovery at each processing step and the derivation of the gold recovery curve as a function of gold head grade. Nominal overall gold recovery was estimated at 87.5% for the average LOM head grade of 1.26 g/t of gold.

- Sizing of major process equipment as well as ancillary equipment. This information was used for estimating the project capital costs.
- Determining reagent and consumables consumptions for the various processing steps. Also, power consumption was estimated based on ore grindability data and on sizing of ancillary equipment, including the oxygen plant. This information was used for estimating project operating costs.

The testwork results and ensuing process flowsheet provide a solid basis for processing the Joanna ore.

25.4 Environmental and Permitting

The development of the Joanna Project poses certain significant challenges with respect to environmental management. These challenges are brought about mainly by the presence of arsenic bearing minerals in the ore body. Much effort has been made to design the operation in order to allow for segregating and separating higher arsenic grade waste and tailings from that which is lower arsenic grade material. This allowed for reduced footprints of the arsenic waste rock pile and tailings pond as well as for improved water management. Despite these efforts, capital costs associated with design of the waste piles, tailings ponds and water management system are a significant contributor to overall capital costs. Although environmental design was performed for diligent management of waste, tailings and water, the environmental approval process for the Project may require further design features, over and above those required by the regulation, which may add further costs to the Project.

25.5 Conclusions

Based on the overall results obtained in this Feasibility Study, the following conclusions can be drawn:

- The Joanna Project Hosco deposit consists of a low grade, arsenic bearing refractory gold mineralization.
- Mining at a rate of 8,500 t/d classifies this operation as a low tonnage operation.
- Ore processing requires crushing, grinding, gravity, flotation, pressure oxidation, cyanide leaching and CIP gold extraction.

- The arsenic content of the ore has significant repercussions on environmental management, namely in the design of the higher grade arsenic bearing waste rock and tailings. This also adds to the complexity of managing water and effluent on site. Despite incorporating diligent design techniques to minimize environmental risks, the mere presence of arsenic bearing waste rock and tailings on site may affect the Environmental Permitting process.
- The pre-production (initial) capital cost for the Project was estimated at $422.2M. Site closure and restoration costs were estimated at $27.1M. Sustaining capital required over the life of the operation has been estimated to be in the order of $70.2M.
- The average operating cash cost is estimated at US$716 per ounce of gold and $25.32 per tonne milled. This excludes transportation, refining and royalties.
- The financial analysis, using a price of gold of US$1,350 per ounce indicates a pre-tax Net Present Value ("NPV") (using a 5% discount rate) of $112 million with a pre-tax internal rate of return ("IRR") of 8.7%. The undiscounted payback period is 8.2 years.

The results of the Feasibility Study indicate that although Project economics are positive (based on the financial assumptions used), the potential financial returns are however marginal and small fluctuations in gold price (and to a lesser extent capital and operating costs) can significantly impact the economic viability of the Project.

At this time, given the exploration potential elsewhere on the property, Aurizon Mines may want to consider further development of the Heva deposit and evaluate if a joint Hosco/Heva mining operation would improve project economics. Furthermore, considering that the Environmental Permitting process for this Project may be more complex and longer than planned, Aurizon Mines may want to consider proceeding with obtaining permits and authorizations for the Project so that if and when project economics become more favorable, they can proceed quickly with execution.

26 RECOMMENDATIONS

Based on the results obtained in this Feasibility Study, it is recommended that Aurizon Mines perform the following activities as part of the final design development for the Joanna Project.

26.1 Drilling Recommendations

The drilling programs completed by Aurizon since the last reported mineral resource estimate successfully outlined additional resources in the measured and indicated categories located mostly below the open pit shell, previously defined in the Pre-Feasibility Study, but including a new small mineral resources core located near the surface between sections 7225mE and 7425mE. Based on the Project's updated Hosco deposit mineral resources and discussions with Aurizon personnel, SGS Geostat recommends the following exploration work in the Hosco and other sectors of the Property:

1) Delineation drilling – Hosco West sector:
   - Increase the inferred resources (50 m X 50 m drill pattern) from surface to 200 m below the surface between sections 7000mE and 7800mE;
   - Complete the conversion of inferred resources into measured and indicated resources (25 m X 25 m drilling pattern) to 200 m below surface around significant results;
   - Delineation drilling: 15,000 m (2.1M$ budget).
2) Exploration and delineation drilling – Heva sector:
   - Increase inferred resources (100 m X 100 m drill pattern) from the surface to a depth of 200m below the surface between sections 4200mE and 5000mE and between sections 5500mE and 7000mE; 14,000m of drilling
   - Complete the conversion of inferred resources into indicated resources around significant results (50 m X 50 m drill pattern) from the surface to a depth of 200m below the surface between sections 4200mE and 7000mE; 30,000m of drilling.
   - Exploration and delineation drilling: 44,000 m (6.2M$ budget).
3) Exploration drilling – Hosco sector:
   - Build a comprehensive structural model of the higher grade mineralization identified in the South and North Zones to help the planning of deeper drilling, with the objective of defining potential underground mineral resources.

- Increase inferred resources (100 m X 100 m drill pattern) between 300 m to 600 m below the surface to test higher grade for potential underground mineral resources.
- Exploration drilling: 12,000 m (1.7M$ budget).

4) Exploration drilling – Alexandria:
   - Investigating the continuity at depth and in the extension of showings discovered in 2009-2010 during the prospection and drilling program;
   - Compilation of historical works (drilling, geophysics survey, and mapping) with the objective to generate new targets in the overall Alexandria sector;
   - Exploration drilling: 10,000 m (1.4M$ budget).
5) Exploration drilling and field works – Henriksen:
   - Investigating the continuity in the extension of significant results from the 2008 drilling program;
   - Increase understanding of the gravimetric survey (2010) in order to generate new drilling targets;
   - Surface mapping and sampling in “oreille ouest” sector to understand structural and mineralization controls of the area;
   - Exploration drilling (5000 m) and field works (0.8M$ budget).

26.2 Testwork and Process Recommendations

There may be some opportunities for optimizing grinding or for increased gold recovery, which can be assessed by performing further testwork as part of the final design. This would help create a more robust design. The following activities are recommended:

- Considering the relatively small quantity of grindability samples tested, it is recommended that more samples be taken and additional full JK Drop Weight tests and SMC tests be performed to validate the chosen mill sizes and power. Samples should be taken to match the mine plan phases in order to validate ore hardness in the first months and years of operation in a more detailed manner.
- Further chemical analysis (QEMSCAN) of the ore to better under the mineralogy of the deposit.
- Investigate the possibility of using flash flotation to increase recovery through a reduction in over grinding of the sulphide.

26.3 Environmental and Geotechnical Recommendations

- Realize kinetic tests on representative composite samples of wacke waste rock type. The arsenic content (partial digestion) of the composite samples must be ± 115 mg/kg. The results should confirm the non-ARD potential and the non-leaching potential of wacke;
- In order to proceed with detailed engineering, additional geotechnical studies must be carried out at the location of all infrastructures and along the CN railway. This program must include drilling, surveys as well as laboratory analysis and soil dynamic stability studies.
- Realize geotechnical surveys at the location of the dams in order to confirm the slope of the dams for the tailings and the various ponds;
- Realize topographic surveys in the projected area for the POX tailings north of the hydroelectric line to optimize infrastructure location;
- Confirm location of the POX tailings using detailed topography and geotechnical surveys;
- Data collected at the project site has allowed the definition of the local hydrogeological setting and the identification of groundwater protection issues. It was found that the relationship between some surface water bodies, and shallow and deep groundwater systems that could potentially be influenced by dewatering activities could be better documented, more specifically, in the Stitchman Creek area where a thin overburden layer was encountered. Detailed work, including monitoring well installation, hydraulic testing and groundwater sampling and analysis, is currently carried out to complete existing information. Regular monitoring of stream flow in the Stitchman Creek and water levels in some shallow and deep monitoring wells could also be performed in order to document seasonal fluctuations;
- Pursue discussions with Québec Ministry of Sustainable Development, Environment and Parks concerning compensation for wetlands destruction;
- Presentation of the existing fish habitat compensation program to Canada Ministry of Fisheries and Oceans for the ditch and the small stream which will be diverted by the establishment of the open-pit. This program will allow confirmation that those works will not trigger the federal environmental assessment procedure;

- Given the new location and size of the infrastructure, update of noise and visual modelling will have to be carried out in due course;

26.4 Estimated Budget for Further Work

It is BBA's understanding that Aurizon Mines does not expect to move forward with the development of the Joanna Project at this time. BBA also understands that, despite this decision, Aurizon Mines will allocate sums in the vicinity of $4.2M for further drilling in the Hosco and Heva areas.

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