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TECHNICAL REPORT ON THE MINERAL RESOURCES ESTIMATE AND PREFEASIBILITY STUDY FOR THE CROINOR PROJECT (according to Regulation 43-101 and Form 43-101F1)

Project Location

Pershing Township, Province of Québec, Canada
(NTS: 32C/03)
(UTM 349800E; 5330200N)
(Zone 18, NAD 83)

Prepared for

Blue Note Mining Inc
1080 Côte du Beaver Hall, Suite 2101
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Effective Date: 19 December 2011
Signature Date: 29 February 2012

CERTIFICATES OF AUTHORS

I, Sylvie Poirier, Eng (OIQ no.112196; PEO no.100156918) do hereby certify that:

1) I am a Consulting Engineer of: InnovExplo Inc, *560-B 3e Avenue*, *Val-d'Or, Québec, Canada, J9P 1S4.*

2) I graduated with a Bachelor's degree in mining Engineering from École Polytechnique (Montréal, Québec) in 1994.

3) I am a member of the Ordre des Ingénieurs du Québec (OIQ, no.100156918), the Professional Engineers of Ontario (PEO no.112196), and the Canadian Institute of Mines (145365).

4) I have worked as an engineer for a total of seventeen (17) years since graduating from university. My mining expertise was acquired while working for Lafarge Canada and for Placer Dome and McWatters at the Sigma mine, as well as for Natural Resources Canada on a special research initiative program on narrow-vein mining. I have been a consulting engineer for InnovExplo Inc since September 2008.

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

6) I am the author of Author of sections 3, 5.4, 15, 16, 18, 19, 21 (except for 21.1.9, 21.1.11 and 21.2.9), 22, and 24, and co-author of sections 1, 2, 25, 26, and 27 of the report titled "Technical Report on the Resources Estimate and Prefeasibility Study for the Croinor Project (according to Regulation 43-101 and Form 43-101F1)", effective date of December 19, 2011 and signature date of February 29, 2012 ("the Technical Report"). I visited the Croinor property as part of a field visit on April 20 and May 28, 2010 at which time I had no other involvement with the property.

7) I have never had any prior involvement with the property that is the subject of the Technical Report.

8) I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report misleading.

9) I am independent of the issuer applying all of the tests in section 1.5 of Regulation 43-101.

10) I have read Regulation 43-101 respecting standards of disclosure for mineral projects and Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.

11) [1] I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: 19 December 2011
Signature Date: 29 February 2012

(signed and sealed on the original)
Sylvie Poirier, Eng
InnovExplo

[1] If an issuer is using this certificate to accompany a technical report that it will file only with the exchange, then the exchange recommends that this paragraph be included in the certificate.

I, Carl Pelletier, Geo. (OGQ no.384; APGO no.1713) do hereby certify that:

1) I am a Consulting Geologist of: InnovExplo Inc, *560-B 3e Avenue*, *Val-d'Or, Québec, Canada, J9P 1S4.*

2) I graduated with a Bachelor's degree in Geology from the Université du Québec à Montréal (Montréal, Québec) in 1992 and I initiated a Master's degree at the same university for which I completed the course program but not the thesis.

3) I am a member of the Ordre des Géologues du Québec (OGQ, no.384), the Association of Professional Geoscientists of Ontario (APGO no.1713), and the Canadian Institute of Mines, Harricana Section.

4) I have worked as a geologist for a total of 18 years since graduating from university. My mining expertise has been acquired in the Silidor, Sleeping Giant, Bousquet II, Sigma-Lamaque and Beaufor mines, and my exploration experience has been acquired with Cambior and McWatters. I have been a consulting geologist for InnovExplo Inc since February 2004.

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

6) I am responsible for supervising the preparation of the report titled "Technical Report on the Resources Estimate and Prefeasibility Study for the Croinor Project (according to Regulation 43-101 and Form 43-101F1)", effective date of December 19, 2011 and signature date of February 29, 2012 ("the Technical Report"). I am responsible for sections 4 (except for 4.3 and 4.4), 5 (except for 5.4), 6, 7, 8, 9, 10, 11, 12 and 23, and co-author of sections 1, 2, 25, 26, and 27. I have conducted several field visits on the Croinor Project between September 2004 and May 2005 and again in 2010.

7) I supervised the preparation of an earlier report titled "Independent Technical Report on the Mineral Resource Estimate of The Croinor Project)" dated April 11, 2005.

8) I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report misleading.

9) I am independent of the issuer applying all of the tests in section 1.5 of Regulation 43-101.

10) I have read Regulation 43-101 respecting standards of disclosure for mineral projects and Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.

11) [1] I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: 19 December 2011
Signature Date: 29 February 2012

(signed and sealed on the original)
Carl Pelletier, BSc, Geo
InnovExplo

[1] If an issuer is using this certificate to accompany a technical report that it will file only with the exchange, then the exchange recommends that this paragraph be included in the certificate.

I, Karine Brousseau, Eng. (OIQ, no.121871) do hereby certify that:

1) I am a Consulting Engineer of: InnovExplo Inc, *560-B 3e Avenue*, *Val-d'Or, Québec, Canada, J9P 1S4.*

2) I graduated with a Bachelor's degree in Geology Engineering (B.Sc.) from Université Laval (Sainte-Foy, Québec) in 1998.

3) I am a member of Ordre des Ingénieurs du Québec (OIQ, no. 121871).

4) I have over 12 years of experience as a geologist in the exploration and mining industry. My exploration expertise has been acquired with SOQUEM, the Ministry of Natural Resources of Québec (Geology Branch), and Cambior. I have been a consulting geologist for InnovExplo Inc since September 2005.

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

6) I am the author of Section 14.0 of the report titled "Technical Report on the Resources Estimate and Prefeasibility Study for the Croinor Project (according to Regulation 43-101 and Form 43-101F1)", effective date of December 19, 2011 and signature date of February 29, 2012 ("the Technical Report"). I did not visit the Croinor property.

7) I have never had any prior involvement with the property that is the subject of the Technical Report.

8) I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report misleading.

9) I am independent of the issuer applying all of the tests in section 1.5 of Regulation 43-101.

10) I have read Regulation 43-101 respecting standards of disclosure for mineral projects and Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.

11) [1] I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: 19 December 2011
Signature Date: 29 February 2012

(signed and sealed on the original)
Karine Brousseau, Eng.
InnovExplo

[1] If an issuer is using this certificate to accompany a technical report that it will file only with the exchange, then the exchange recommends that this paragraph be included in the certificate.

I, Rodrigue Ouellet, MScA, Eng (OIQ no.37361) do hereby certify that:

1) I am a Consulting Engineer of: Golder Associates Ltd, 375 Avenue Centrale, Bureau 102, Val-d'Or, Québec, Canada, J9P 1P4.

2) I graduated with a Bachelor's degree in Geological Engineering from the Université du Québec à Chicoutimi in 1982. In addition, I obtained a Master's Degree in Earth Sciences from the Université du Québec à Chicoutimi in 1986, and a Certificate in Environmental Science from the Université du Québec à Montréal in 2006.

3) I am a member of the Ordre des Ingénieurs du Québec (OIQ, no.37361).

4) I have worked as a geological engineer for a total of 25 years since graduating from university. My expertise in the mining industry has been acquired with Falconbridge Copper Ltd, Minnova Inc, Metal Mining Corporation, Inmet Mining Corporation, and Aur Resources Inc. My expertise in environmental mining has been acquired with Golder Associates Ltd on several mining environment projects. I have been a consulting engineer for Golder Associates Ltd since January 2005.

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

6) I am responsible for calculating costs and for writing sections related to environmental aspects, and I am the author of sections 4.3, 4.4, 20, 21.1.9, 21.1.11, and 21.2.9 of the report titled "Technical Report on the Resources Estimate and Prefeasibility Study for the Croinor Project (according to Regulation 43-101 and Form 43-101F1)", effective date of December 19, 2011 and signature date of February 29, 2012 ("the Technical Report"). I visited the Croinor site as part of a field visit on April 12, 2010. I was involved in the permitting process of the project in 2009 and 2010.

7) I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report misleading.

8) I am independent of the issuer applying all of the tests in section 1.5 of Regulation 43-101.

9) I have read Regulation 43-101 respecting standards of disclosure for mineral projects and Form 43-101F1, and the Technical Report has been prepared in accordance with that regulation and form.

10) [1] I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: 19 December 2011
Signature Date: 29 February 2012

(signed and sealed on the original)
Rodrigue Ouellet, MScA, Eng

[1] If an issuer is using this certificate to accompany a technical report that it will file only with the exchange, then the exchange recommends that this paragraph be included in the certificate.

I, Jocelyn Bouchard, Eng. (OIQ no.128654) do hereby certify that:

1) I am an Engineer employed as Senior Mineral Processing Engineer by GENIVAR at 1600, Rene-Levesque Boulevard, Montreal, Québec.

2) I received a Bachelor's Degree in Engineering from Université Laval (Québec City, Québec) in 2001, as well as a Master's (2004) and Ph.D. (2007) Degree from the same university.

3) I am a registered member of the Ordre des Ingénieurs du Québec (OIQ member no.128654) and Professional Engineers Ontario (License Number 100141948).

4) I have over 7 years of experience as an engineer in the mining industry. My experience has been acquired mostly with Université Laval, Iamgold, Xstrata and GENIVAR. I have been working for GENIVAR since March 2010 as Principal Mineral Processing & Process Control Engineer.

5) I have read the definition of "qualified person" set out in Regulation 43-101 ("R 43-101") standards for disclosure for mineral projects and certify that by reason of my education, affiliation with a professional association (as defined in R 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of R 43-101.

6) I am responsible for Sections 13 and 17 of the report titled "Technical Report on the Resources Estimate and Prefeasibility Study for the Croinor Project (according to Regulation 43-101 and Form 43-101F1)", effective date of December 19, 2011 with signature date of February 29, 2012 ("the Technical Report").

7) I am "independent" (as such term is defined in Section 1.5 of R 43-101) of Blue Note Mining Inc.

8) I have never had any prior involvement with the property that is the subject of the Technical Report.

9) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

10) I have read R 43-101, Appendix 43-101A1 and the Technical Report which has been prepared in compliance with that instrument and form.

11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: 19 December 2011
Signature Date: 29 February 2012

(signed and sealed on the original)
Jocelyn Bouchard, PhD, Eng
GENIVAR

[1] If an issuer is using this certificate to accompany a technical report that it will file only with the exchange, then the exchange recommends that this paragraph be included in the certificate.

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1.0 SUMMARY

At the request of Mr Léon Methot, Chairman, President & Chief Executive Officer of Blue Note Mining Inc ("Blue Note Mining" or "the issuer"), InnovExplo has been retained to complete a Technical Report ("the report"), an update of the Mineral Resource Estimate, and an update of the preliminary feasibility study for the Croinor Project in compliance with Regulation 43-101 and Form 43-101F1. Sylvie Poirier, Eng., was assigned to the project and visited the property on April 20 and May 28, 2010.

In addition to the principal author, Sylvie Poirier, Eng. (InnovExplo), the other qualified persons responsible for the preparation of this report were: Carl Pelletier, BSc, Geo (InnovExplo); Karine Brousseau, Eng (InnovExplo); Jocelyn Bouchard, PhD, Eng (GENIVAR); and Rodrigue Ouellet, MScA, Eng (Golder Associates). In addition, Tafadzwa Gomwe, PhD, GIT (InnovExplo), participated in the preparation of this report under the supervision of Carl Pelletier.

The Mineral Resource Estimate herein was performed by Karine Brousseau, Eng. and Tafadzwa Gomwe, PhD, GIT, under the supervision of Carl Pelletier, BSc, Geo. Sylvie Poirier is responsible for the prefeasibility study. The environmental studies in the report were completed by Golder Associates Ltd (Golder) and the review of the custom milling option for the Croinor ore was completed by Genivar Inc (Genivar).

Property Description and Ownership

The Croinor property is approximately 70 km northeast of the city of Val-d'Or, Québec, in the Pershing Township on NTS map sheet 32C/03 (Figs. 4.1 and 4.2). The approximate UTM coordinates for the geographic centre of the property are 349800E and 5330200N (Zone 18, NAD83).

The Croinor property comprises 367 mining titles covering a total area of 7,674 ha and one mining lease covering approximately 90 ha. All claims and the mining lease are registered to X-Ore Resources Inc.

On January 19, 2010, Blue Note Mining Inc proceeded with the acquisition of X-Ore Resources Inc by the amalgamation of two companies, X-Ore Resources Inc ("X-Ore") and 9216-4706 Québec Inc, a wholly-owned subsidiary of Blue Note Mining. Blue Note owns 50% in the Croinor Project.

On July 19, 2010, Blue Note Mining and First Gold Exploration Inc ("First Gold") announced that they had entered into a binding agreement regarding the acquisition by Blue Note Mining of First Gold's 50% interests in the Croinor Project.

On January 16, 2012, Blue Note Mining and Critical Elements Corporation ("Critical Elements"; formerly First Gold) reported they had agreed to extend the term of the binding agreement announced on July 19, 2010 providing for the acquisition by Blue Note Mining of all of Critical Elements' interests in the Croinor Project.

Pershing Township has been a target for exploration work since the early 1930s. The Croinor deposit was discovered in 1940, probably by a prospector named Fred Thompson. Subsequently, several companies conducted exploration on the property. The work can be subdivided into four main periods:

- 1944-1948: Surface and underground drilling, shaft sinking (three compartments), and the development of 2,020 metres of drift on four levels by Croinor Pershing;

- 1979-1989: Rehabilitation of the shaft, driving of a ramp and surface plus underground drilling by Onaping Resources and then Sullivan Mines followed by Cambior;
- 1996-1997: Open pit mining and custom milling of over 51,000 tonnes by Goldust;
- 1998-2003: Trenching, surface drilling and completion of a 20,000-tonne bulk sample by Exploration Malartic-Sud.

As a result of these previous work, the Croinor deposit is serviced by a ramp measuring 300 m long by 4 m high by 4.5 m wide (4m x 4.5m) that extends to level 125 (38m), and by a 3-compartment shaft extending 195 m deep. Development was completed on four (4) levels: 496 metres on level 125; 560 metres on level 250; 233 metres on level 375; and 730 metres on level 500. Approximately 320 metres of raise development was also completed. The Croinor mine is currently flooded to the portal entrance.

Geology and Mineralization

The Croinor property is located in the eastern part of the Archean Abitibi Greenstone Belt in the southern Superior Province of the Canadian Shield. The Abitibi Greenstone Belt is one of the most extensive continuous expanses of low metamorphic grade Archean volcanic and sedimentary rocks on Earth (Card and Poulsen, 1998). It also happens to be one of the richest mining regions in the world and has produced large amounts of gold, copper, zinc, silver and iron from the Timmins, Kirkland Lake, Rouyn-Noranda, Val-d'Or, Matagami and Chibougamau mining districts.

The Croinor property is underlain by two major lithological packages—the dominantly volcanic Assup Domain in the north and the dominantly sedimentary Garden Island Domain in the south. The Pershing Batholith overlaps the southwest end of the property. The property is transected by the Garden Island Deformation Corridor in a NW-SE direction, which overprints and partially follows the contact between the Assup and Garden Island domains.

The Croinor deposit is located 3 km northeast of the Pershing Batholith and 2 km northeast of the Garden Island Deformation. It is hosted by the synvolcanic Croinor Sill. This dioritic sill is from 60 to 120 metres thick and hosted within volcanic rocks of the Assup Domain. The deposit is characterized by gold-rich lenses consisting of quartz-carbonate-tourmaline-pyrite veins, altered pyritic host rock material, and/or tectonic breccia (pyritic host fragments within a quartz-carbonate-tourmaline-pyrite vein).

The mineralized lenses at Croinor range from 60 to 120 metres long (Chénard and Turcotte, 2003). The lenses can generally be followed from one section to another (10-metre sections) over lateral distances varying from several tens of metres up to 600 metres. To date, about forty (40) gold-rich lenses have been identified. Gold-hosting structures within the dioritic Croinor Sill consist of a series of narrow breaks and reverse shears with dips of 25° to 60° to the north. These structures display en-echelon patterns and are variably oriented between N290°-N330°. They can cause displacements of a few centimetres to tens of metres.

Mineral Resource Estimate

The Mineral Resource Estimate detailed in this report herein was performed by Karine Brousseau, Eng. and Tafadzwa Gomwe, PhD, GIT, under the supervision of Carl Pelletier, BSc, Geo.

InnovExplo is of the opinion that the current Mineral Resource Estimate can be classified as Measured, Indicated and Inferred resources based on the density of the processed data, the search ellipse criteria, and the specific interpolation parameters. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves. The resources were estimated using different gold cut-off grades and a minimum width of 1.8 metres (true width). The selected cut-off of 4 g/t was used to determine the mineral potential of the deposit for the underground mining option. Resource Estimate for the Croinor deposit (54 lenses). InnovExplo is of the opinion that the current Mineral Resource Estimate can be used for a pre-feasibility study. The following tables display the results of the In Situ Mineral.

Results of the In Situ Mineral Resource Estimate at different cut-off grades (Table 14.7)

Measured Resources				Indicated Resources				Measured + Indicated Resources			
Cut-off	Tonnes	Au g/t	Oz Au	Cut-off	Tonnes	Au g/t	Oz Au	Cut-off	Tonnes	Au g/t	Oz Au
> 6 g/t	46 800	10.98	16 500	> 6 g/t	355 800	12.13	138 800	> 6 g/t	402 600	12.00	155 300
> 5 g/t	59 400	9.81	18 700	> 5 g/t	447 300	10.78	155 000	> 5 g/t	506 700	10.66	173 700
> 4 g/t	80 500	8.41	21 800	> 4 g/t	599 600	9.18	176 900	> 4 g/t	680 100	9.08	198 700
> 3 g/t	112 400	7.00	25 300	> 3 g/t	848 300	7.51	204 700	> 3 g/t	960 700	7.45	230 000
> 2 g/t	165 200	5.55	29 500	> 2 g/t	1 192 200	6.05	231 700	> 2 g/t	1 357 400	5.99	261 200
> 1 g/t	257 600	4.08	33 800	> 1 g/t	1 889 600	4.35	264 100	> 1 g/t	2 147 200	4.31	297 900
> 0.6 g/t	297 700	3.64	34 800	> 0.6 g/t	2 283 700	3.73	274 100	> 0.6 g/t	2 581 400	3.72	308 900
> 0.3 g/t	335 200	3.28	35 300	> 0.3 g/t	2 685 400	3.24	279 800	> 0.3 g/t	3 020 600	3.25	315 100

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by Regulation 43-101, are Carl Pelletier, B.Sc., Geo. and Karine Brousseau, Eng. (InnovExplo Inc), and the effective date of the estimate is November 4, 2011.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Results are presented undiluted and in situ. The estimate includes 54 gold-bearing lenses.
- Resources were compiled using cut-off grades of 0.3 g/t, 0.6 g/t, 1.0 g/t, 2.0 g/t, 3.0 g/t, 4.0 g/t, 5.0 g/t and 6.0 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 2.8 g/cm^3 was used in zones and waste.
- A minimum true thickness of 1.8 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High-grade capping was done on raw data and established at 70.0 g/t Au for diamond drill hole assays and 55.0 g/t Au for underground chip sample assays.
- Compositing was not done over entire drill hole lengths. Instead, compositing was done on drill hole and chip sample sections that fall within the mineralized zone envelopes (composite = 1 metre).
- Resources were evaluated from drill hole and underground chip samples using an I/D6 interpolation method in a block model.
- The Measured, Indicated and Inferred categories were defined using different search ellipsoid parameters.
- The Inferred reclassified category is the result of isolated blocks or series of blocks showing no spatial continuity in terms of grade and/or density of information that were reclassified from Indicated to Inferred.
- The Indicated reclassified category is the result of blocks or series of blocks showing good spatial continuity in terms of grade and/or density of information that were reclassified from Inferred to Indicated.
- Ounce (troy) = Metric Tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation 43-101.
- Other than the drill hole CR-11-414 to CR-11-419, that are not included in the current resource estimate because they had pending assays at that time the cut-off date (mid-November) of the Mineral Resource Estimate,InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource Estimate

Results of the In Situ Mineral Resource Estimate at different cut-off grades (Table 14.8)

Inferred Resources			
Cut-off	**Tonnes**	**Au g/t**	**Oz Au**
> 6 g/t	84 500	12.02	32 700
> 5 g/t	102 400	10.90	35 900
> 4 g/t	160 100	8.56	44 100
> 3 g/t	227 800	7.03	51 500
> 2 g/t	358 300	5.36	61 800
> 1 g/t	564 600	3.94	71 500
> 0.6 g/t	664 700	3.47	74 200
> 0.3 g/t	726 300	3.21	75 100

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by Regulation 43-101, are Carl Pelletier, B.Sc., Geo. and Karine Brousseau, Eng. (InnovExplo Inc), and the effective date of the estimate is November 4, 2011.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Results are presented undiluted and in situ. The estimate includes 54 gold-bearing lenses.
- Resources were compiled using cut-off grades of 0.3 g/t, 0.6 g/t, 1.0 g/t, 2.0 g/t, 3.0 g/t, 4.0 g/t, 5.0 g/t and 6.0 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate, mining cost).
- A fixed density of 2.8 g/cm^3 was used in zones and waste.
- A minimum true thickness of 1.8 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High-grade capping was done on raw data and established at 70.0 g/t Au for diamond drill hole assays and 55.0 g/t Au for underground chip sample assays.
- Compositing was not done over entire drill hole lengths. Instead, compositing was done on drill hole and chip sample sections that fall within the mineralized zone envelopes (composite = 1 metre).
- Resources were evaluated from drill hole and underground chip samples using an I/D6 interpolation method in a block model.
- The Measured, Indicated and Inferred categories were defined using different search ellipsoid parameters.
- The Inferred reclassified category is the result of isolated blocks or series of blocks showing no spatial continuity in terms of grade and/or density of information that were reclassified from Indicated to Inferred.
- The Indicated reclassified category is the result of blocks or series of blocks showing good spatial continuity in terms of grade and/or density of information that were reclassified from Inferred to Indicated.
- Ounce (troy) = Metric Tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource Estimate.

The Mineral Resource Estimate was made using 3D block modelling and the inverse distance power six (1/D6) interpolation method for a 1,570-m strike-length corridor of the Croinor property, from 910mW to 640mE (local grid, oriented on a 291.9167° Azimuth) and down to a vertical depth of 545 metres below surface. This method was selected in order to facilitate the comparison with the previous estimate of O'Dowd (2009), which used the polygonal cross-sectional method. Although the choice of a higher power value makes the two methods more similar, it does not make them equivalent. The following table compares the two mineral resource estimates using a cut-off grade of 5 g/t Au.

Comparison of the 2011 and 2009 mineral resource estimates for the Croinor deposit (Table 14.9)

	Measured + Indicated Resources			Inferred Resources		
Cut-off > 5 g/t Au	Tonnes	Au g/t	Oz Au	Tonnes	Au g/t	Oz Au
2011	506 700	10.66	173 700	102 400	10.90	35 900
2009	814 200	9.11	238 400	n.a	n.a	n.a
Difference	-307 500	1.55	-64 700	102 400	10.90	35 900

After the resource estimation by O'Dowd in 2009, Blue Note Mining drilled 65 drill holes for a total of 15,390 m. In order to quantify the effect of these holes on the mineral resources, InnovExplo performed a second calculation of the block model without the 2010-2011 drill holes. A total of 47 drill holes and 2,444 assays were removed from the database and the block model was recalculated using the same 3D solids and the same parameters.

The changes in resource estimation methodology and geological interpretation for the current 2011 Mineral Resource Estimate resulted in a decrease of 307,500 tonnes in the Measured+Indicated category compared to the 2009 estimate. On the other hand, the addition of data and results from the 2010 and 2011 diamond drilling programs increased Indicated resources by 16,209 tonnes, and Inferred resources by 37,990 tonnes.

This exercise demonstrates that the 2010-2011 drilling programs had a positive impact on the mineral resources, and that the decrease in Mineral Resources is mainly caused by the use of a different estimation method.

Inferred Resources have been calculated for the current Mineral Resource Estimate, but were not calculated in the previous estimate (O'Dowd, 2009). This was made possible by the additional drilling in 2010 and 2011 along the lateral extensions and at depth, demonstrating the continuity of mineralized zones outside the area of known resources. This provided sufficient information to establish the geological and grade continuities of these zones. Inferred resources in the Croinor deposit are all in the immediate vicinity of the Indicated resources. The majority form a fringe around the Indicated resources, up to a maximum distance of 70 m, in which there are not enough drill hole intersects to categorize the resources as Indicated. It should be relatively easy to convert all or parts of these Inferred resources to Indicated by adding definition drilling.

Mineral Reserve

InnovExplo designed the underground mine plan based on the current Mineral Resource. The proposed approach uses sublevel long-hole retreat and room and pillar, with 75% of the ounces from long-hole and 25% from room and pillar. InnovExplo was able to demonstrate the economic viability of the proposed extraction and processing of the Measured and Indicated resources found within the mine plan. Mineral reserves were classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Reserves for the current Project incorporate appropriate mining dilution and mining recovery allowances for the selected mining method. The following Table presents the estimated Proven and Probable reserves, which total 120,883 oz after applying the appropriate mining recovery and dilution factor for the selected method.

Mineral Reserve Estimate (Table 15.1)

Category	tonnes	g/t	ounces
Proven	68,849	6.23	13,789
Probable	498,023	6.69	107,094
Total reserve	566,872	6.64	120,883

In order to determine the resources to be converted to reserves, we used the Mineable Shape Optimizer (MSO), a Datamine software application. According to specified stope parameters, MSO generates individual stope shapes from the block model.

Two mining methods appear to be most convenient for the Croinor deposit: long-hole retreat and room and pillar. In order to select the most appropriate mining method, two MSO runs were completed on the block model using the following parameters for both methods. A small block size was selected in order to better reflect the narrow vein nature of the deposit.

Long-hole mining method:

- Cut-off grade value: 3.7 g/t;
- Minimum mining width of 1.8 m (stope thickness);
- Sublevel interval of 13 m (vertical height);
- Spacing interval of 5 m (stope length);
- Mining dilution of 0.4 m on the hanging wall and 0.2 m on the footwall;
- Minimum slope wall angle of 45 degrees.

Room and pillar mining method:

- Cut-off grade value: 5.4 g/t;
- Minimum mining width of 1.8 m (stope thickness);
- Maximum mining width of 3 m (stope thickness);
- Spacing interval of 5 m x 5 m (stope size along strike);
- Maximum slope wall angle of 45 degrees.

Processing

Genivar was commissioned by InnovExplo and Blue Note Mining to review custom milling options in the context of a prefeasibility study for the Croinor gold project. Only four gold ore processing concentrators located within a 120-km radius could potentially process the Croinor ore: Beacon Gold, Aurbel Gold, Sigma-Lamaque Complex and Camflo. This study assumes that the ore will be processed at the Camflo mill. In addition to having successfully processed Croinor ore in the past (hence mitigating the technical risk), the plant is likely to have an availability for new custom feed fitting the life of mine requirements for Croinor and has been in the custom milling business for years. Ore previously mined from the Croinor open pit operations was processed at the Camflo mill and based on the results of those runs, a gold recovery of 97.5% has been used in the present prefeasibility study.

Environment

The last period of mine operations at the Croinor site ended in May 2005. In order to facilitate the resumption of mining operations, Golder undertook a review of environmental permitting and licensing requirements. A Certificate of Authorization (CofA) request document for the operation of the Croinor mine was prepared by Golder and submitted to the Ministry of Sustainable Development, Environment and Parks (Ministère du Développement durable, de l'Environnement et des Parcs; MDDEP) in February 2010. Following the analysis of the CofA document, 2 requests for additional information were received from the MDDEP in March and June 2010. Replies to these requests were submitted in May and June 2010, respectively. The delivery of a CofA by the MDDEP for operating the Croinor mine was delivered in September 2010.

Capital and Operating Cost

Most of the capital cost was estimated using quotes from equipment suppliers and contractors. In some cases, comparable installations at other projects were used. The capital cost estimate is accurate within ±20%.

The pre-production costs are estimated at $20.68 million, including $2.4 million of capitalized operating costs net of production revenue received during the pre-production period. Sustaining capital is estimated at $17.9 million, excluding $0.66 million for final closure costs.

Capital expenditure breakdown (Table 21.1)

Description	Pre-production	Sustaining	Total cost
Capitalized operating cost	$16,363,677		$16,363,677
Capitalized revenue	($13,950,520)		($13,950,520)
Dewatering and rehabilitation	$1,249,609		$1,249,609
Development	$4,537,911	$10,760,313	$15,298,224
Ventilation equipment	$340,075		$340,075
Mine dewatering	$442,718	$56,614	$499,331
Surface installation and equipment	$2,081,591	$670,628	$2,752,219
Electrical distribution	$6,029,064	$1,232,000	$7,261,064
Mobile equipment	$2,955,638	$4,903,494	$7,859,132
Environment	$371,596	$258,162	$629,758
Contractor demobilization	$255,642		$255,642
Total capital expenditures	**$20,677,000**	**$17,881,210**	**$38,558,210**

Operating costs are estimated in 2011 Canadian dollars with no allowance for escalation. The total life-of-mine operating cost and average unit operating costs are summarized in the following table. The overall unit operating cost is $164 per tonne of milled ore. InnovExplo estimated mine operating costs using budget quotes from contractors and suppliers.

Summary of total life-of-mine operating costs (Table 21.12)

Description	Total cost	Unit cost	
Definition drilling	$2,332,270	4.40 $/t	20.44 US$/oz
Stope development	$16,039,477	30.25 $/t	140.55 US$/oz
Mining	$19,372,502	36.54 $/t	169.76 US$/oz
Blue Note Mining staff	$10,847,925	20.46 $/t	95.06 US$/oz
Contractor (indirect cost)	$10,885,100	20.53 $/t	95.38 US$/oz
Surface services	$189,508	0.36 $/t	1.66 US$/oz
Energy cost	$4,482,168	8.45 $/t	39.28 US$/oz
Milling and transportation	$22,055,439	41.60 $/t	193.27 US$/oz
Environment	$779,739	1.47 $/t	6.83 US$/oz
Total:	**$86,984,128**	**164 $/t**	**762 US$/oz**

To supply electric power to the site, a new 26.5 km 25 kV three-phase overhead power line is planned. In this study, this new power line is assumed to be held and owned by Hydro-Québec. Further discussions between the client and Hydro-Québec will be required to define the project.

Financial Analysis

An after-tax model was developed for the Croinor Project. All costs are in 2011 Canadian dollars with no allowance for inflation or escalation.

The Croinor Project is subject to the following taxes: Québec mining rights; Federal and provincial taxes.

The income tax rate is 26.9% (2012 federal and Québec tax rate) and the mining tax rate is 16% (2012); this latter rate will be sanctioned as proposed in the May 2011 bill.

It is assumed that Blue Note Mining Inc ("Blue Note Mining"), a federal corporation, and X-Ore Resources Inc ("X-Ore"), a Québec corporation, will proceed with a vertical amalgamation. X-Ore and Blue Note Mining are allowed to do so, but only following the continuance of X-Ore under the Canada Business Corporations Act, or of Blue Note Mining under the Business Corporations Act (Québec). This vertical amalgamation will allow the available non-capital losses of Blue Note Mining and X-Ore to be used by the resulting corporation to offset any future income derived from its mining activities.

The economic valuation of the project was performed using the Internal Rate of Return (IRR) and Net Present Value (NPV) methods. The IRR on an investment is defined as the rate of interest earned on the unrecovered balance of an investment. The NPV method converts all cash flows for investments and revenues occurring throughout the planning horizon of a project to an equivalent single sum at present time at a specific discount rate. The discount rate used in the analysis is 7%. According to the NPV method, a positive NPV represents a profitable investment where the initial investment plus any financing interests are recovered.

The following parameters were considered in the financial analysis:

- An average gold price of US$1495/oz and an exchange rate of 1.03 CAD/1USD, which correspond to the Bloomberg consensus estimate of December 19, 2011. It also considers that preproduction would be initiated in April 2012. The following table provides details of the Bloomberg base case consensus forecasts.

Bloomberg base case consensus forecasts as of December 19, 2011 (Table 22.1)

	2012	2013	2014	2015
Gold price ($US/oz)	1,834	1,893	1,572	1,506
Exchange rate (CAD/USD)	1.03	1.01	1.04	1.01

The resulting main parameters and cash flow analysis for the entire project are presented in the following Table.

Cash flow analysis summary (Table 22.2)

Parameters	Results
Proven & Probable mineral reserves	566,429 t @ 6.64 g/t
Total contained gold reserve	120,833 oz
Mine life (including 18-month pre-production)	5 years
Daily mine production	425 tpd ramping up to 675 tpd in Year 4
Gold recovery	97.5%
Annual gold production	21.259 to 41,578 oz
LOM recovered gold	117,956 oz

Parameters	Results
Average cash operating cost	$164/tonne
Average cash operating cost	US$762/oz
Capital cost (including $17.9M sustaining/working capital)	$38.6 million
Total cost per ounce	US$1032/oz
Total gross revenue	$182 million
Total operating cost	$86.9 million
Total project cost	**$125.5 million**
Total operating cash flow (before tax & royalties)	**$47.2 million**
Estimated mining and income taxes	**$12.5 million**
Net cash flow	**$31 million**
Pre-tax NPV (7% discount)	**$30.6 million**
Pre-tax IRR	**57%**
After-tax NPV (7% discount)	**$21 million**
After-tax IRR	**44%**
Payback period	**38 months**
Pre-production period (including 36,201t of production)	**18 months**

The parameters in the sensitivity analysis were chosen based on their potential impact on the outcome of the economic evaluation. The parameters were divided into two different groups: production parameters and economic parameters.

The following production parameters were analyzed:

- Grade (g/t)
- Gold price (US$/oz)

The following economic parameters were analyzed:

- Total net revenue (REVENUE)
- Operating expenditure (OPEX)
- Capital expenditure (CAPEX)

Sensitivity calculations were performed on the project's after-tax NPV, IRR and total cash flow by applying a range of variation (±25%) to the parameter values. The sensitivity analysis demonstrates thatn the Croinor Project is highly sensitive to changes in grade, gold price and revenue. It is moderately sensitive to changes in OPEX and CAPEX.

Conclusion and Recommendation

Other than the drill hole CR-11-414 to CR-11-419, that are not included in the current resource estimate because they had pending assays at that time the cut-off date (mid-November) of the Mineral Resource Estimate, InnovExplo is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that would materially affect the Mineral Reserve Estimate. InnovExplo considers the present prefeasibility study to be reliable and thorough, based on quality data, reasonable hypotheses and parameters compliant with Regulation 43-101 and CIM standards with regard to Mineral Reserve and Resource estimates.

The results from this Mineral Reserve Estimate and Prefeasibility Study demonstrate that the Croinor Project is technically and economically viable and InnovExplo recommends that Blue Note Mining continues to advance the project towards production.

InnovExplo proposes an exploration program that includes follow-up drilling. The drilling program will augment the information on the Indicated Resources before mining begins. The program should also include the Inferred zone in an effort to convert those resources to Indicated. Included in the program should be a follow-up on the extensions to the east and west and at depth to increase the extent of the Inferred Resources. A simplified sketch (Fig. 26.1) shows the current understanding and location of the Indicated and Inferred zones to be targeted by drilling. The sketch also shows significant intercepts located beyond the eastern and western limits as defined by the 2011 drilling program.

The following recommendations pertain to the Mineral Resources and Mineral Reserves. InnovExplo provides the following recommendations:

- Continue infill and down-plunge exploration drilling aimed at expanding the current resources.
- Update the Mineral Resource Estimate with further drilling.
- Pursue discussions with Hydro-Québec to finalize a contract and define the scope of work to advance the design of the electric power line extension to the feasibility study stage and initiate related permitting requests.
- Continue to work on general permitting for the project.
- Complete additional engineering work to optimize the mine design and production schedule;
- Evaluate the possibility of applying ore sorting technology at the Croinor site.
- Prepare contract documents for the activities to be contracted for the work.
- Pursue negotiations to finalize agreements for custom milling and ore transportation.

To advance the project, InnovExplo estimates an overall budget of approximately $2,185,000 as presented in the following Table. A brief breakdown of the approximate cost is as follows:

Proposed Work Program and Budget (Table 26.1)

Item	Cost
Exploration and definition drilling program	
Definition drilling : (approx. 8,000 m at $110/m)	$880,000
Exploration drilling: (approx. 10,000 m at $110/m)	$1,100,000
Property-scale exploration, to locate diorite and verify surface outcrops	$100,000
Subtotal exploration:	**$2,080,000**
Preparation to advance project towards production	
Electric power line extension feasibility study	$20,000
General permitting for the project	$25,000
Complete additional engineering work to optimize the mine design and production schedule	$30,000
Evaluate the possibility of applying ore sorting technology at the Croinor site	$15,000
Prepare contract documents for the activities to be contracted for the work	$15,000
Subtotal preparation work:	**$105,000**
Grand Total:	**$2,185,000**

2.0 INTRODUCTION

At the request of Mr Léon Methot, Chairman, President & Chief Executive Officer of Blue Note Mining Inc ("Blue Note Mining" or "the issuer"), InnovExplo has been retained to complete a Technical Report ("the report"), an update of the Mineral Resource Estimate, and an update of the preliminary feasibility study for the Croinor Project in compliance with Regulation 43-101 and Form 43-101F1. This report is addressed to Blue Note Mining, a Canadian exploration company listed on the TSX Venture Exchange under the symbol BNT. InnovExplo is an independent mining and exploration consulting firm based in Val-d'Or (Québec).

The Croinor property is located approximately 70 km northeast of Val-d'Or, Québec.

2.1 Terms of reference

The issuer requested an update of the mineral resource estimate and pre-feasibility study dated June 22, 2011, to include the recent drilling information from the 2010-2011 diamond drilling programs and to use a 3D block model for the estimation method. The mineral resource estimate and pre-feasibility study have been updated accordingly and are presented herein.

The objectives of this prefeasibility study are to:

- Determine the best project design from multiple options;
- Determine the most appropriate mining method according to the geometry and grade of the Croinor deposit;
- Establish the basic design for most of the facilities, and the infrastructure needed to access, develop and mine the mineralized zones;
- Estimate the capital cost;
- Estimate the operating cost;
- Estimate the cost and timeframe for the pre-production and production period;
- Analyze the financial aspects of the project;
- Establish the ore reserves;
- Make recommendations for additional work to be done in order to allow for a detailed engineering design.

2.2 Principal Sources of Information

InnovExplo's review of the Croinor Project was based on published material, as well as data, professional opinions and unpublished material submitted by Blue Note Mining.

InnovExplo estimated costs using quotes from contractors and suppliers, as well as data provided in the 2010 volumes of Mining Cost Service (with a subscription to Cost Data Update Services) published by CostMine, a division of InfoMine USA Inc.

InnovExplo has reviewed the data provided by the issuer and/or by its agents. InnovExplo has also consulted other information sources, such as government management databases for assessment work and the status of mining titles.

InnovExplo conducted a review and appraisal of the information used in the preparation of this report, as well as in its conclusions and recommendations, and believes that such information is valid and appropriate considering the status of the project and the purpose for which the report is prepared. The authors have fully researched and documented the conclusions and recommendations made in the report.

2.3 Qualified Persons and Inspection on the Property

In addition to the principal author, Sylvie Poirier, Eng. (InnovExplo), the other qualified persons responsible for the preparation of this report were: Carl Pelletier, BSc, Geo (InnovExplo); Karine Brousseau, Eng (InnovExplo); Jocelyn Bouchard, PhD, Eng (GENIVAR); and Rodrigue Ouellet, MScA, Eng (Golder Associates). In addition, Tafadzwa Gomwe, PhD, GIT (InnovExplo), participated in the preparation of this report under the supervision of Carl Pelletier. The qualified persons are responsible for the following sections of the report:

- Sylvie Poirier: author of sections 3, 5.4, 13, 15, 16, 18, 19, 21 (except 21.1.9, 21.1.11 and, 21.2.9), 22, 24; co-author of sections 1, 2, 25, 26, 27;
- Carl Pelletier: author of sections 4 (except 4.3 and 4.4), 6, 7, 8, 9, 10, 11,12, 23; co-author of sections 1, 2, 25, 26, 27;
- Karine Brousseau: author of section 14; co-author of sections 25 and 26;
- Rodrigue Ouellet: author of sections 4.3, 4.4, 20, 21.1.9, 21.1.11 and 21.2.9;
- Jocely Bouchard: author of section 17.

Sylvie Poirier, Carl Pelletier, and Tafadzwa Gomwe visited the property on multiple occasions: Sylvie Poirier on April 20 and May 28, 2010; Carl Pelletier from 2005 to 2011; and Tafadzwa Gomwe in 2010 and 2011.

2.4 Units and Currencies

All currency amounts are stated in Canadian Dollars ($) or US dollars ($US). Quantities are stated in metric units, as per standard Canadian and international practice, including metric tons (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, and grams (g) or grams per metric ton (g/t) for gold grades. Wherever applicable, imperial units have been converted to the International System of Units (SI units) for consistency. A list of abbreviations used in this report is provided in Appendix I.

3.0 RELIANCE ON OTHER EXPERTS

The authors, Qualified and Independent Persons as defined by Regulation 43-101, were contracted by the issuer to study technical documentation relevant to the report, perform a resource estimate and a prefeasibility study on the Croinor deposit, and recommend a work program if warranted. The authors have reviewed the mining titles, their status, any agreements and technical data supplied by the issuer (or its agents), and any public sources of relevant technical information.

Many of the geological and technical reports for projects in the vicinity of the Croinor property were prepared before the implementation of National Instrument 43-101 in 2001 and Regulation 43-101 in 2005. The authors of such reports appear to have been qualified and the information prepared according to standards that were acceptable to the exploration community at the time. In some cases, however, the data are incomplete and do not fully meet the current requirements of Regulation 43-101. The authors have no known reason to believe that any information used in the preparation of this report is invalid or contains misrepresentations.

The authors relied on reports and opinions from the following persons for information that is not within InnovExplo's fields of expertise.

- Information about the mining titles and option agreements was supplied by Blue Note Mining staff. InnovExplo is not qualified to express any legal opinion with respect to the property titles or current ownership and possible litigation;
- The environmental study and associated information was provided by Rodrigue Ouellette, MScA, Eng, of Golder Associates;
- The evaluation for custom milling in the Val-d'Or mining camp was done by GENIVAR;
- The preliminary electrical requirements and cost estimates were provided by Louis Valade, president of Meglab Inc;
- The after-tax cash flow estimation was completed by Lucie Chouinard, MFisc, CA, of Samson Bélair/Deloitte & Touche;
- The preliminary ventilation network design was provided by Nathalie Gauthier, Eng, mining engineer for InnovExplo;
- The geomechanical assessment was done by Jane Alcott, MASc, Eng, of InnovExplo;
- InnovExplo's technical support for the planning work was provided by Kathleen Bonenfant, mine technician;
- MSO consulting work was done by Nelson Senhorinho, PhD, of CAE Mining;
- The linguistic editing was performed by Venetia Bodycomb, MSc, of Vee Geoservices.

The authors believe the information used to prepare the report and formulate its conclusions and recommendations is valid and appropriate considering the status of the project and the purpose for which the report is prepared.

The authors, by virtue of their technical review of the project's exploration potential, affirm that the work program and recommendations presented in the report are in accordance with Regulation 43-101 and CIM technical standards.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Croinor property is approximately 70 km northeast of the city of Val-d'Or, Québec, in Pershing Township on NTS map sheet 32C/03 (Figs. 4.1 and 4.2). The approximate UTM coordinates for the geographic centre of the property are 349800E and 5330200N (Zone 18, NAD83).

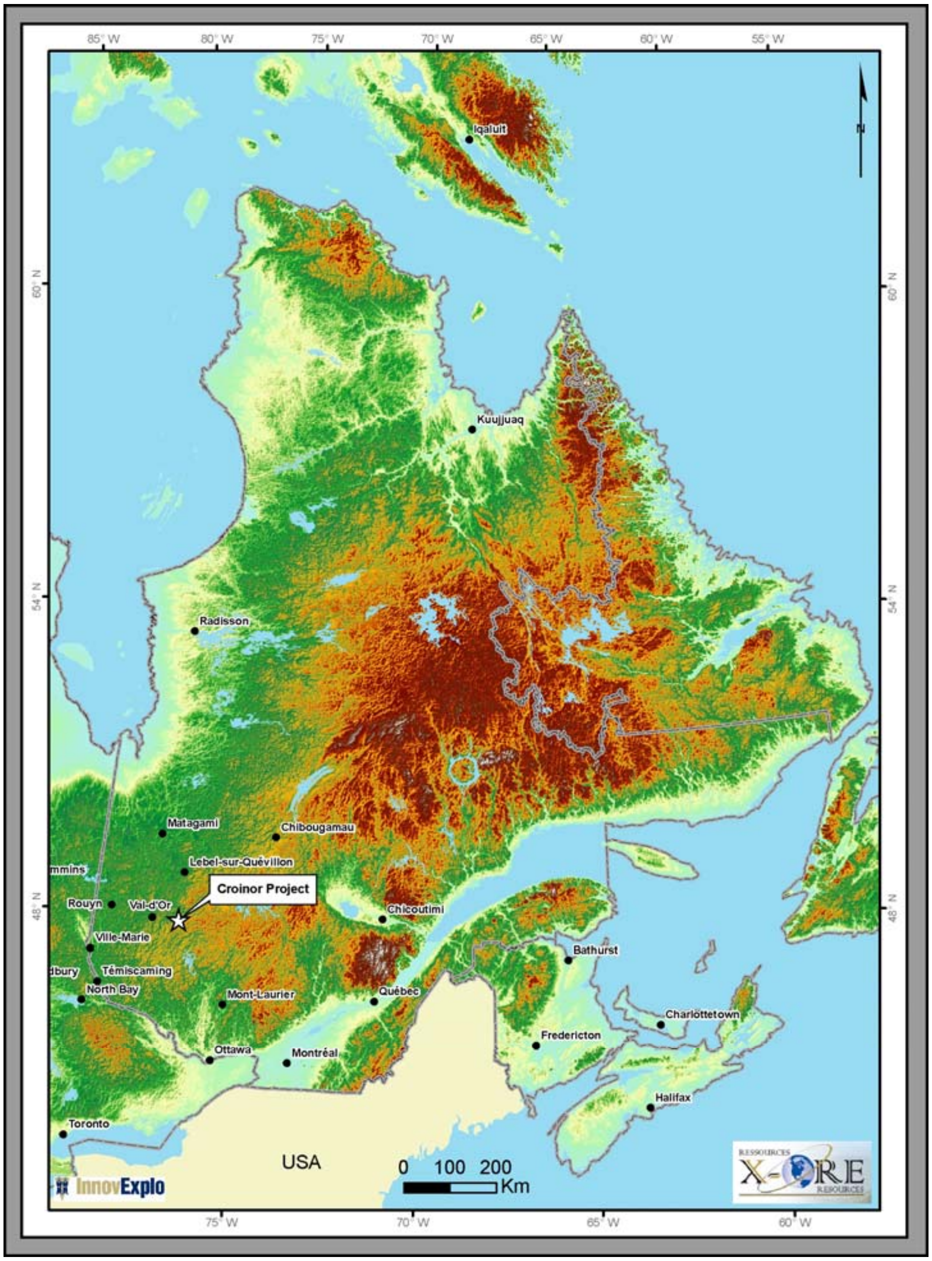


Figure 4.1 – Location of the Croinor property in the province of Québec

4.1 Claim Status

The Croinor property comprises 367 mining titles covering a total area of 7,674 ha and one mining lease covering approximately 90 ha (Fig. 4.2). Table 4.1 lists the claims and their details, and Table 4.2 provides details of the mining lease. All claims and the mining lease are registered to X-Ore Resources Inc.

On January 19, 2010, Blue Note Mining Inc proceeded with the acquisition of X-Ore Resources Inc by the amalgamation of two companies, X-Ore Resources Inc ("X-Ore") and 9216-4706 Québec Inc, a wholly-owned subsidiary of Blue Note Mining. Blue Note owns 50% in the Croinor Project.

On July 19, 2010, Blue Note Mining and First Gold Exploration Inc ("First Gold") announced that they had entered into a binding agreement regarding the acquisition by Blue Note Mining of First Gold's 50% interests in the Croinor Project. That agreement includes the Croinor mining lease and the claims listed in Table 4.3.

On January 16, 2012, Blue Note Mining and Critical Elements Corporation ("Critical Elements"; formerly First Gold) reported they had agreed to extend the term of the binding agreement announced on July 19, 2010 providing for the acquisition by Blue Note Mining of all of Critical Elements' interests in the Croinor Project.

Under the terms of the agreement, Blue Note Mining has until March 31, 2012, or such other later date as mutually agreed by Blue Note Mining and Critical Elements, to make a final payment of $2,250,000 to complete the transaction. In addition, Blue Note Mining shall issue 17.5 million common shares to be held in escrow, for release at a rate of 500,000 shares per month over 35 months from the date of closing. The transaction includes Critical Elements' 71% ownership in the Matchi-Manitou property.

The 50/50 Croinor joint operatorship shall remain in effect until the date of closing. In the meantime, Critical Elements shall be exempted from participating in any further cash calls until such date.

Prior to this binding agreement, four (4) different option agreements were signed regarding claims that now constitute the entire Croinor property, and those agreements still affect the respective claims. Two of those four option agreements affect the mining lease (BM 862) on which the Croinor Project is located. The agreements are summarized below in Table 4.4; figures 4.3 to 4.6 illustrate which claims are subject to royalties.

Other than what is discussed in the different option agreements, no liens or charges seem to be registered against the Croinor property. All lands seem to be in good standing according to the GESTIM database (Québec's claim management system).

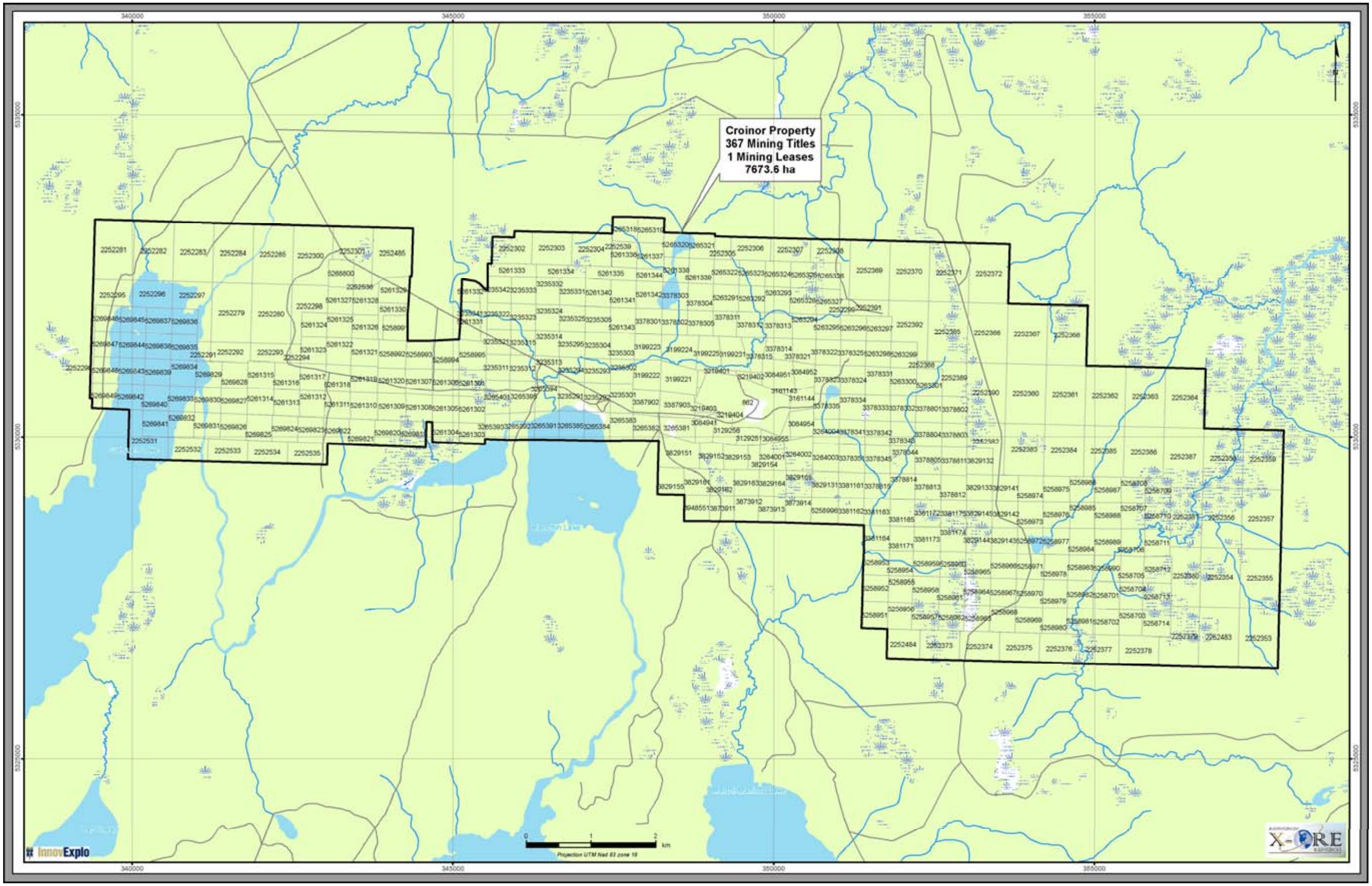


Figure 4.2 – Claim map of the Croinor property

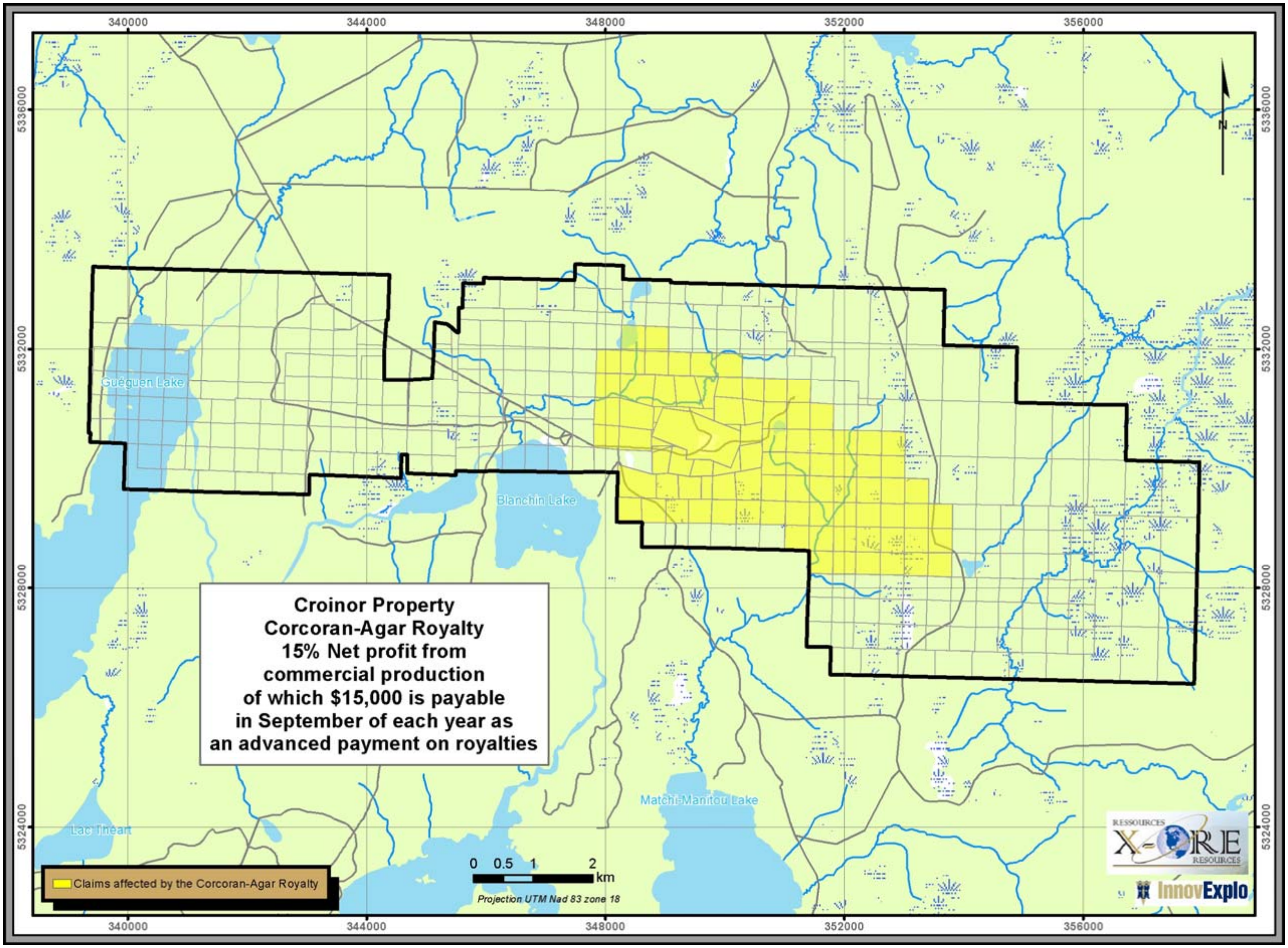


Figure 4.3 – Claims affected by the Corcoran-Agar Royalty

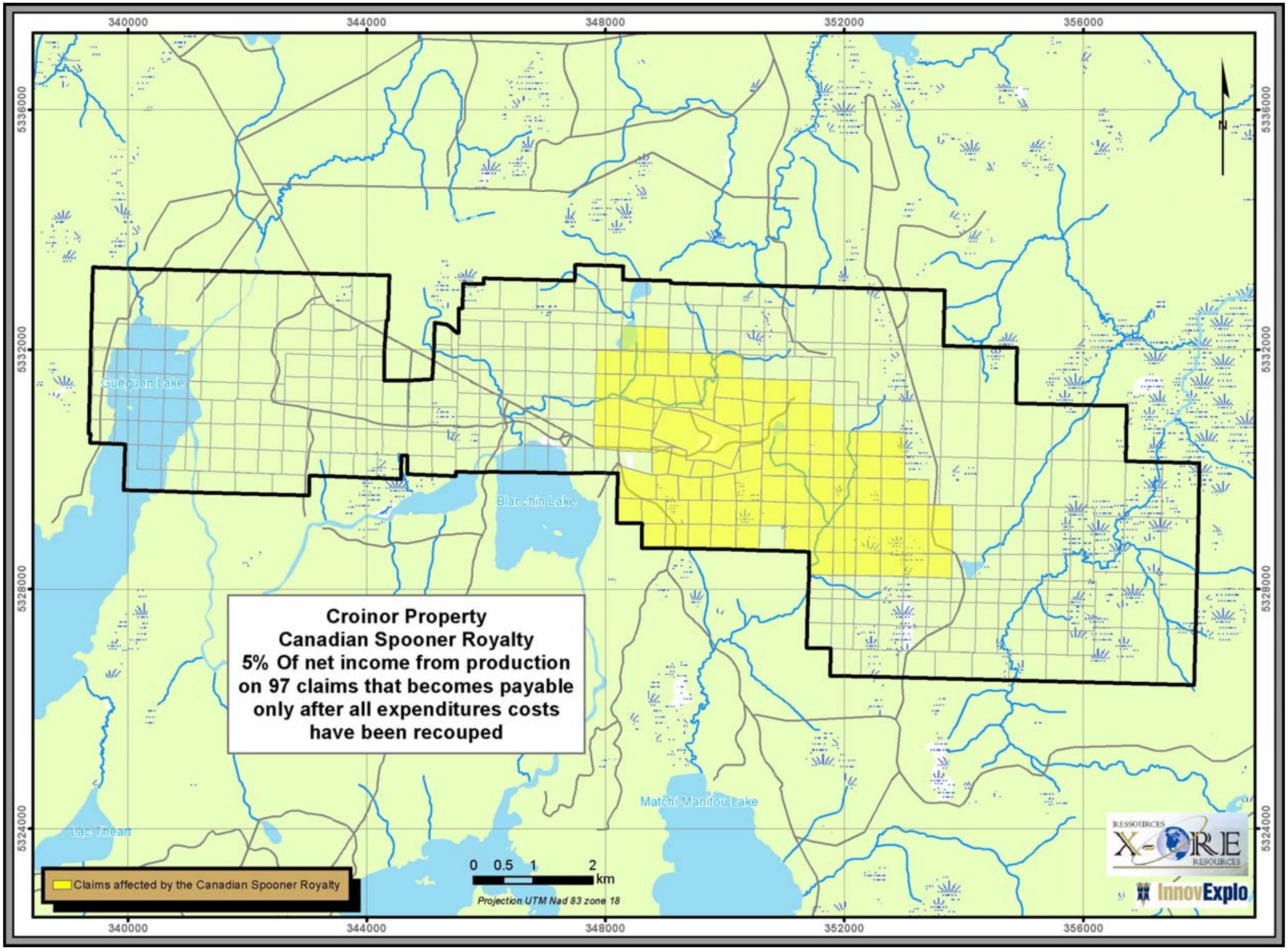


Figure 4.4 – Claims affected by the Canadian Spooner Royalty

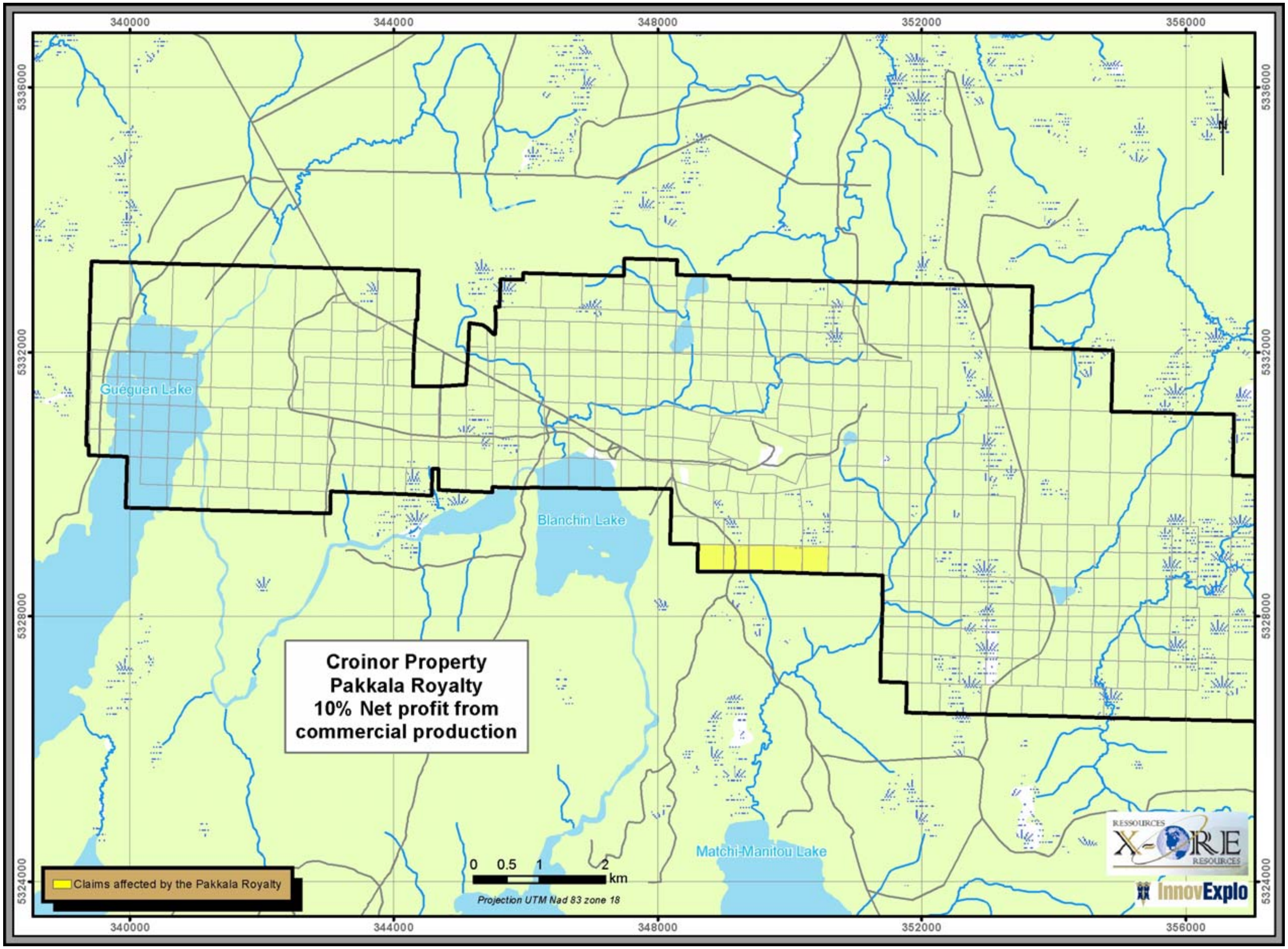


Figure 4.5 – Claims affected by the Pakkala Royalty

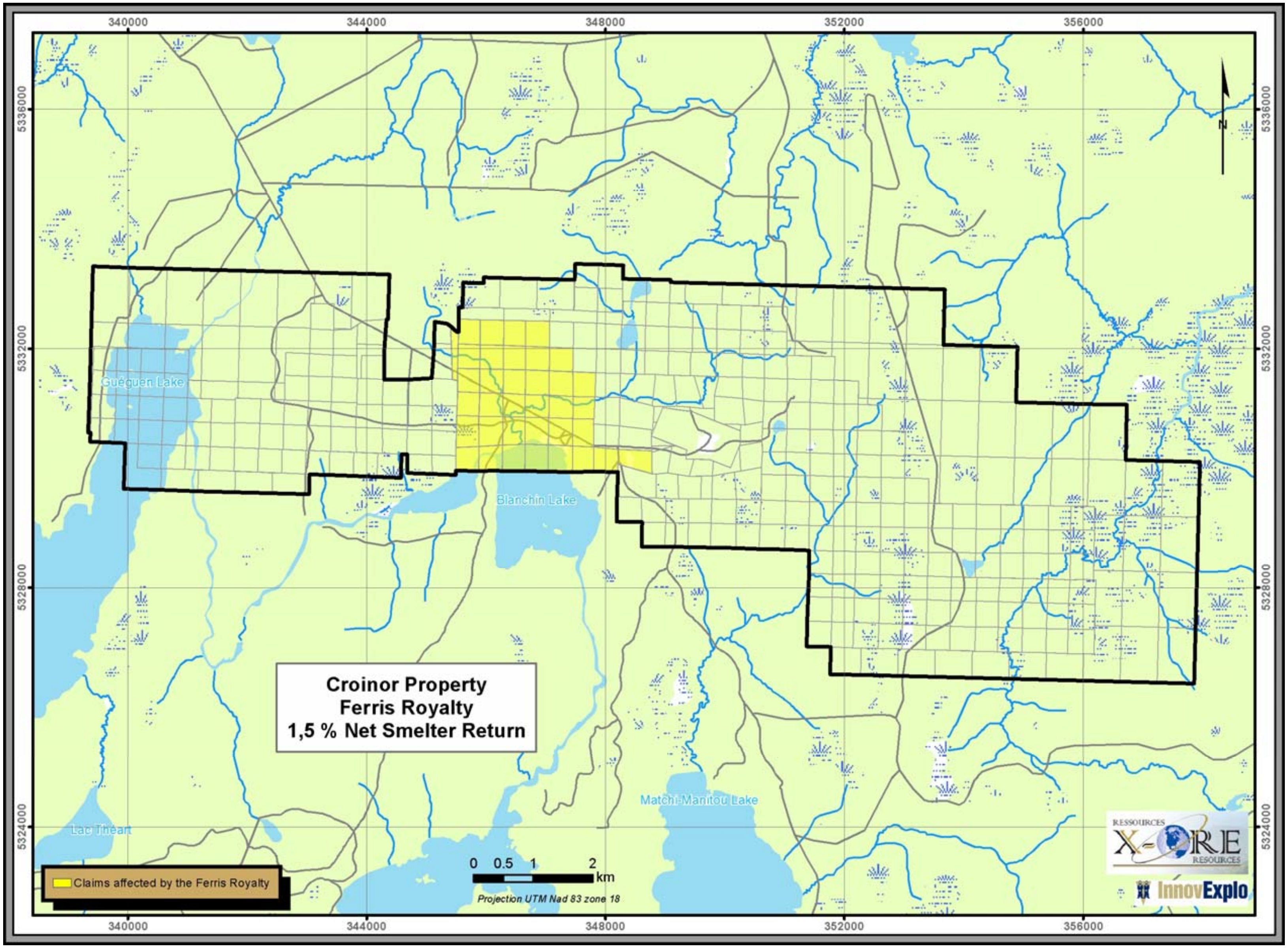


Figure 4.6 – Claims affected by the Ferris Royalty

Table 4.1 – List of mining titles comprising the Croinor property

Title	Type	NTS	Status	Area (ha)	Registration Date	Expiry Date	Work Required	Credit	Owner
2252279	CDC	32C03	Active	57.48	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252280	CDC	32C03	Active	57.48	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252281	CDC	32C03	Active	57.47	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252282	CDC	32C03	Active	57.47	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252283	CDC	32C03	Active	57.47	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252284	CDC	32C03	Active	57.47	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252285	CDC	32C03	Active	57.47	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252290	CDC	32C03	Active	3.78	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252291	CDC	32C03	Active	6.48	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252292	CDC	32C03	Active	18.65	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252293	CDC	32C03	Active	17.45	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252294	CDC	32C03	Active	4.64	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252295	CDC	32C03	Active	27.71	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252296	CDC	32C03	Active	25.71	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252297	CDC	32C03	Active	36.93	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252298	CDC	32C03	Active	32.24	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252299	CDC	32C03	Active	3.23	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252300	CDC	32C03	Active	55.11	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252301	CDC	32C03	Active	43.63	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252302	CDC	32C03	Active	28.00	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252303	CDC	32C03	Active	31.31	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252304	CDC	32C03	Active	31.53	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252305	CDC	32C03	Active	8.92	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252306	CDC	32C03	Active	21.89	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252307	CDC	32C03	Active	22.04	2010-10-01	2012-09-30	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252308	CDC	32C03	Active	30.22	2010-10-01	2012-09-30	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252353	CDC	32C02	Active	57.52	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252354	CDC	32C02	Active	57.51	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252355	CDC	32C02	Active	57.51	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252356	CDC	32C02	Active	57.50	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252357	CDC	32C02	Active	57.50	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252358	CDC	32C02	Active	57.49	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252359	CDC	32C02	Active	57.49	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252360	CDC	32C02	Active	57.48	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252361	CDC	32C02	Active	57.48	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252362	CDC	32C02	Active	57.48	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252363	CDC	32C02	Active	57.48	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252364	CDC	32C02	Active	57.48	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252365	CDC	32C02	Active	57.47	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252366	CDC	32C02	Active	57.47	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252367	CDC	32C02	Active	57.47	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252368	CDC	32C02	Active	57.47	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252369	CDC	32C02	Active	57.46	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252370	CDC	32C02	Active	57.46	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252371	CDC	32C02	Active	57.46	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252372	CDC	32C02	Active	57.46	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252373	CDC	32C02	Active	28.19	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252374	CDC	32C02	Active	28.11	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252375	CDC	32C02	Active	28.03	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252376	CDC	32C02	Active	27.95	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252377	CDC	32C02	Active	27.93	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252378	CDC	32C02	Active	27.95	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252379	CDC	32C02	Active	49.10	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252380	CDC	32C02	Active	42.12	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252381	CDC	32C02	Active	45.52	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252382	CDC	32C02	Active	30.46	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252383	CDC	32C02	Active	49.77	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252384	CDC	32C02	Active	49.52	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252385	CDC	32C02	Active	49.27	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252386	CDC	32C02	Active	49.03	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252387	CDC	32C02	Active	55.78	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252388	CDC	32C02	Active	1.66	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)

Title	Type	NTS	Status	Area (ha)	Registration Date	Expiry Date	Work Required	Credit	Owner
2252389	CDC	32C02	Active	22.69	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252390	CDC	32C02	Active	56.54	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252391	CDC	32C02	Active	15.07	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252392	CDC	32C02	Active	42.03	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252483	CDC	32C02	Active	57.52	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252484	CDC	32C02	Active	23.24	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252485	CDC	32C03	Active	49.13	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252531	CDC	32C03	Active	31.21	2010-10-04	2012-10-03	1,200.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252532	CDC	32C03	Active	22.22	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252533	CDC	32C03	Active	21.77	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252534	CDC	32C03	Active	21.32	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252535	CDC	32C03	Active	20.86	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252536	CDC	32C03	Active	0.35	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252537	CDC	32C03	Active	0.01	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252538	CDC	32C03	Active	0.06	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
2252539	CDC	32C03	Active	0.37	2010-10-04	2012-10-03	500.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3084941	CL		Active	16.50	1970-07-06	2013-03-29	1,000.00 $	1,365.00 $	X-Ore Resources inc (80225) 100 % (liable)
3084951	CL		Active	22.06	1970-07-06	2013-03-29	1,000.00 $	30,292.00 $	X-Ore Resources inc (80225) 100 % (liable)
3084952	CL		Active	16.00	1970-07-06	2013-03-29	1,000.00 $	21,737.00 $	X-Ore Resources inc (80225) 100 % (liable)
3084954	CL		Active	15.76	1970-07-06	2013-03-29	1,000.00 $	165,071.00 $	X-Ore Resources inc (80225) 100 % (liable)
3084955	CL		Active	9.22	1970-07-06	2013-03-29	1,000.00 $	3,780,770.00 $	X-Ore Resources inc (80225) 100 % (liable)
3129251	CL		Active	9.93	1970-02-13	2013-03-29	1,000.00 $	498,534.00 $	X-Ore Resources inc (80225) 100 % (liable)
3129255	CL		Active	0.01	1970-02-13	2013-03-29	1,000.00 $	991,416.00 $	X-Ore Resources inc (80225) 100 % (liable)
3129256	CL		Active	15.36	1970-02-13	2013-03-29	1,000.00 $	673,371.00 $	X-Ore Resources inc (80225) 100 % (liable)
3161143	CL		Active	1.58	1971-05-10	2013-03-29	1,000.00 $	627,135.00 $	X-Ore Resources inc (80225) 100 % (liable)
3161144	CL		Active	12.34	1971-05-10	2013-03-29	1,000.00 $	220,329.00 $	X-Ore Resources inc (80225) 100 % (liable)
3199221	CL		Active	16.00	1971-12-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3199222	CL		Active	16.00	1971-12-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3199223	CL		Active	16.00	1971-12-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3199224	CL		Active	16.00	1971-12-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3199225	CL		Active	16.00	1971-12-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3199231	CL		Active	16.00	1971-12-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3219401	CL		Active	5.00	1972-05-30	2013-03-29	1,000.00 $	229,939.00 $	X-Ore Resources inc (80225) 100 % (liable)
3219402	CL		Active	5.24	1972-05-30	2013-03-29	1,000.00 $	93,894.00 $	X-Ore Resources inc (80225) 100 % (liable)
3219403	CL		Active	4.12	1972-05-30	2013-03-29	1,000.00 $	202,410.00 $	X-Ore Resources inc (80225) 100 % (liable)
3219404	CL		Active	2.10	1972-05-30	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235291	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	1,247.00 $	X-Ore Resources inc (80225) 100 % (liable)
3235292	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	373.00 $	X-Ore Resources inc (80225) 100 % (liable)
3235293	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235294	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235295	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235301	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235302	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235303	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235304	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235305	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235311	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235312	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235313	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235314	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235315	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235321	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235322	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235323	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235324	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235325	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235331	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235332	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235333	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235341	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3235342	CL		Active	16.00	1972-06-08	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3264001	CL		Active	16.00	1972-09-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3264002	CL		Active	16.00	1972-09-25	2013-03-29	1,000.00 $	48,132.00 $	X-Ore Resources inc (80225) 100 % (liable)

Title	Type	NTS	Status	Area (ha)	Registration Date	Expiry Date	Work Required	Credit	Owner
3264003	CL		Active	16.00	1972-09-25	2013-03-29	1,000.00 $	45,807.00 $	X-Ore Resources inc (80225) 100 % (liable)
3264004	CL		Active	16.00	1972-09-25	2013-03-29	1,000.00 $	50,813.00 $	X-Ore Resources inc (80225) 100 % (liable)
3265381	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265382	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265383	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265384	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265385	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265391	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265392	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265393	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265394	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265395	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3265401	CL		Active	16.00	1972-10-25	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378301	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378302	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378303	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378304	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378305	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378311	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378312	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378313	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378314	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378315	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378321	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378322	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378323	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378324	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378325	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378331	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	8,031.00 $	X-Ore Resources inc (80225) 100 % (liable)
3378332	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378333	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	13,844.00 $	X-Ore Resources inc (80225) 100 % (liable)
3378334	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378335	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	247,493.00 $	X-Ore Resources inc (80225) 100 % (liable)
3378341	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	10,182.00 $	X-Ore Resources inc (80225) 100 % (liable)
3378342	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378343	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378344	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378345	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	1,731.00 $	X-Ore Resources inc (80225) 100 % (liable)
3378351	CL		Active	16.00	1973-07-09	2013-03-29	1,000.00 $	10,632.00 $	X-Ore Resources inc (80225) 100 % (liable)
3378801	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378802	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378803	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378804	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378805	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378811	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378812	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378813	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3378814	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	2,505.00 $	X-Ore Resources inc (80225) 100 % (liable)
3378815	CL		Active	16.00	1973-07-17	2013-03-29	1,000.00 $	4,636.00 $	X-Ore Resources inc (80225) 100 % (liable)
3381161	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	7,419.00 $	X-Ore Resources inc (80225) 100 % (liable)
3381162	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	4,732.00 $	X-Ore Resources inc (80225) 100 % (liable)
3381163	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3381164	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3381165	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3381171	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3381172	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	3,818.00 $	X-Ore Resources inc (80225) 100 % (liable)
3381173	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	2,799.00 $	X-Ore Resources inc (80225) 100 % (liable)
3381174	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	5,796.00 $	X-Ore Resources inc (80225) 100 % (liable)
3381175	CL		Active	16.00	1973-08-09	2013-03-29	1,000.00 $	4,768.00 $	X-Ore Resources inc (80225) 100 % (liable)
3387902	CL		Active	16.00	1973-11-06	2013-03-29	1,000.00 $	806.00 $	X-Ore Resources inc (80225) 100 % (liable)
3387903	CL		Active	16.00	1973-11-06	2013-03-29	1,000.00 $	9,784.00 $	X-Ore Resources inc (80225) 100 % (liable)
3829131	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829132	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)

Title	Type	NTS	Status	Area (ha)	Registration Date	Expiry Date	Work Required	Credit	Owner
3829133	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829141	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829142	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829143	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829144	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829145	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	5,745.00 $	X-Ore Resources inc (80225) 100 % (liable)
3829151	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829152	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829153	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829154	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829155	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3829161	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	1,979.00 $	X-Ore Resources inc (80225) 100 % (liable)
3829162	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	2,367.00 $	X-Ore Resources inc (80225) 100 % (liable)
3829163	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	3,663.00 $	X-Ore Resources inc (80225) 100 % (liable)
3829164	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	3,663.00 $	X-Ore Resources inc (80225) 100 % (liable)
3829165	CL		Active	16.00	1979-07-04	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3873911	CL		Active	16.00	1980-01-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3873912	CL		Active	16.00	1980-01-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3873913	CL		Active	16.00	1980-01-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3873914	CL		Active	16.00	1980-01-17	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
3948551	CL		Active	16.00	1980-10-29	2013-03-29	1,000.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258701	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258702	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258703	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258704	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258705	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258706	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258707	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258708	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258709	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258710	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258711	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258712	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258713	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258714	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258951	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258952	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258953	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258954	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258955	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258956	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258957	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258958	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258959	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258960	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258961	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258962	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258963	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258964	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258965	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258966	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258967	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258968	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258969	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258970	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258971	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258972	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258973	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258974	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258975	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258976	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258977	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258978	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)

Title	Type	NTS	Status	Area (ha)	Registration Date	Expiry Date	Work Required	Credit	Owner
5258979	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258980	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258981	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258982	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258983	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258984	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258985	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258986	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258987	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258988	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258989	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258990	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258991	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258992	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258993	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258994	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258995	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5258996	CL		Active	16.00	2000-09-07	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261301	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261302	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261303	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261304	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261305	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	10,300.00 $	X-Ore Resources inc (80225) 100 % (liable)
5261306	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261307	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	15,171.00 $	X-Ore Resources inc (80225) 100 % (liable)
5261308	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261309	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	16,911.00 $	X-Ore Resources inc (80225) 100 % (liable)
5261310	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	14,318.00 $	X-Ore Resources inc (80225) 100 % (liable)
5261311	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261312	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261313	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261314	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261315	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261316	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261317	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261318	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261319	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	28,027.00 $	X-Ore Resources inc (80225) 100 % (liable)
5261320	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261321	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	3,117.00 $	X-Ore Resources inc (80225) 100 % (liable)
5261322	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261323	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261324	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261325	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261326	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261327	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261328	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261329	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261330	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261331	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	2,470.00 $	X-Ore Resources inc (80225) 100 % (liable)
5261332	CL		Active	8.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261333	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261334	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261335	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261336	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261337	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261338	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261339	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261340	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261341	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261342	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261343	CL		Active	16.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5261344	CL		Active	8.00	2001-08-13	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263291	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)

Title	Type	NTS	Status	Area (ha)	Registration Date	Expiry Date	Work Required	Credit	Owner
5263292	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263293	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263294	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263295	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263296	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263297	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263298	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263299	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263300	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5263301	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265318	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265319	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265320	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265321	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265322	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265323	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265324	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265325	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265326	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265327	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5265328	CL		Active	16.00	2002-11-18	2013-03-29	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5268800	CL		Active	16.00	2005-12-29	2013-12-28	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269819	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269820	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269821	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269822	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269823	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269824	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269825	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269826	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	21,260.00$	X-Ore Resources inc (80225) 100 % (liable)
5269827	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269828	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269829	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269830	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269831	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269832	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269833	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269834	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269835	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269836	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269837	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269838	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269839	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269840	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269841	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269842	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269843	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269844	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269845	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269846	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269847	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269848	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)
5269849	CL		Active	16.00	2004-01-13	2014-01-12	750.00 $	- $	X-Ore Resources inc (80225) 100 % (liable)

Table 4.2 – Mining lease included in the Croinor property

Title	NTS	Status	Area (ha)	Registration Date	Expiry Date	Annual Rent	Owner
BM 862	32C03	Valid	89.72	2004-07-06	2024-07-05	3,678.52$	X-Ore Resources inc (80225) 100 % (liable)

Table 4.3 – Mining titles included in the binding agreement with Critical Elements

Title	Type	NTS	Status	Area (ha)	Registration Date	Expiry Date	Work Required	Credit	Owner
3084951	CL		Active	22.06	1970-07-06	2013-03-29	1,000.00$	30,291.99$	X-Ore Resources inc (80225) 100 %
3084955	CL		Active	9.22	1970-07-06	2013-03-29	1,000.00$	3,780,770.00$	X-Ore Resources inc (80225) 100 %
3129251	CL		Active	9.93	1970-02-13	2013-03-29	1,000.00$	498,533.90$	X-Ore Resources inc (80225) 100 %
3129255	CL		Active	0.01	1970-02-13	2013-03-29	1,000.00$	991,416.36$	X-Ore Resources inc (80225) 100 %
3129256	CL		Active	15.36	1970-02-13	2013-03-29	1,000.00$	673,370.90$	X-Ore Resources inc (80225) 100 %
3161143	CL		Active	1.58	1971-05-10	2013-03-29	1,000.00$	627,135.10$	X-Ore Resources inc (80225) 100 %
3219401	CL		Active	5.00	1972-05-30	2013-03-29	1,000.00$	229,939.45$	X-Ore Resources inc (80225) 100 %
3219402	CL		Active	5.24	1972-05-30	2013-03-29	1,000.00$	93,893.71$	X-Ore Resources inc (80225) 100 %
3219403	CL		Active	4.12	1972-05-30	2013-03-29	1,000.00$	202,410.90$	X-Ore Resources inc (80225) 100 %
BM 862	ML		Valid	89.72	2004-07-06	2024-07-06			X-Ore Resources inc (80225) 100 %

Table 4.4 – List of various royalties affecting the Croinor property

<table>
<tr><th colspan="2" rowspan="2">PROPERTY</th><th colspan="2">Royalty</th><th rowspan="2">Description</th></tr>
<tr><th>Name</th><th>Percentage</th></tr>
<tr><td rowspan="4">CROINOR</td><td rowspan="2">Affecting the mining lease BM 862</td><td>Successors of Fred D. Corcoran and Denis R. Agar</td><td>7.5% each</td><td>Net profit from commercial production of which $15,000 is payable in September of each year as an advance payment on royalties</td></tr>
<tr><td>Canadian Spooner Resources Inc.</td><td>5%</td><td>Of net income from production on 97 claims that becomes payable only after all expenditures costs have been recouped</td></tr>
<tr><td rowspan="2">Not affecting the mining lease BM 862</td><td>Verner Pakkala</td><td>10%</td><td>Net profit from commercial production.</td></tr>
<tr><td>A.N. Ferris</td><td>1.5%</td><td>Net Smelter Return</td></tr>
</table>

4.2 Mineral Royalties

The Croinor property is subjected to the Corcoran-Agar Royalty, which consists of 15% applied on net profit from commercial production of which $15,000 is payable in September of each year as an advance payment on royalties.

Another royalty affecting the Croinor property is the Canadian Spooner Royalty. Considering that any expense (of whatever nature) incurred on the property since 1983 would have to be recouped by the operator, it is highly unlikely that any amount would have to be paid with respect to the agreement regarding this royalty.

4.3 Environmental Obligations

4.3.1 Québec Environmental Quality Act and Legislation

The Québec Environmental Quality Act (EQA) requires that a Certificate of Authorization (CofA) be obtained for an industrial activity that may result in the release of contaminants into the environment.

The CofA application for operating the Croinor mine was prepared by Golder and submitted to the Ministry of Sustainable Development and Parks (*Ministère du Développement durable, de l'Environnement et des Parcs*; MDDEP) in February 2010. Following the analysis of the CofA application, two requests for additional information were received from the MDDEP in March and June 2010. Replies to these requests were submitted in May and June 2010, respectively. The MDDEP delivered the CofA for operating the Croinor mine in September 2010.

The CofA application was prepared in compliance with Québec's *Directive 019*. As part of the permitting process, a request for environmental objectives of rejects (OER: *Objectifs environnementaux de rejets*) was submitted to the MDDEP in May 2010. A request for additional information about the OER application was received in July 2010 and a reply was submitted to the MDDEP. The MDDEP released the OER in August 2010.

CofA applications will have to be submitted to the MDDEP for the sewage treatment system and the drinking water well. Documentation for these applications will be prepared and submitted to the MDDEP when a positive decision is made to reopen the mine.

4.3.2 Canadian Metal Mining Effluent Regulation

Environmental Effects Monitoring Studies (EEMS) are required by the Canadian Metal Mining Effluent Regulation (MMER). The initial EEMS plan for Croinor was prepared and submitted to Environment Canada (EC) in 2005 by Alliance Environment. The EEMS process was aborted following the cessation of mining operations in May 2005.

The EEMS process will have to be re-initiated when the CofA is obtained from the MDDEP and the project receives the go-ahead.

4.3.3 Québec Mining Act

The Québec Mining Act (QMA) requires that a mine site rehabilitation plan and a crown pillar stability study be submitted to the Ministry of Natural Resources and Wildlife (*Ministère des Ressources naturelles et de la Faune*; MRNF) when the project is initiated.

As per the QMA, the rehabilitation plan must be updated every 5 years. The last plan for Croinor was prepared and submitted in 2004 and the project is therefore due for an update. The plan is used to establish the financial guarantee that must be delivered to the MRNFP to cover reclamation of the site. When reclamation is completed, the Ministry will reimburse the financial guarantee. To date, X-Ore Resources has deposited a financial guarantee totalling $105,178. Golder prepared and updated the rehabilitation plan, which will be submitted to the MRNF once the project commences. Drilling for the crown pillar study was completed in June 2010. Field measurements and lab testing on the drill core were carried out in July. The report was delivered in April 2011.

4.3.4 Other permits

The following permits will be required for the development and operation of the Croinor mine:

- Wood cutting
- Gravel pit
- Construction
- Explosive
- Fuel storage
- Beaver dam dismantling

The application for these various permits will be prepared and submitted to the concerned agencies when a positive decision is made to reopen.

4.4 Permits status

The last period of mining at the Croinor site ended in May 2005. In order to facilitate the resumption of mining operations, Golder undertook a review of environmental permitting and licensing requirements. Table 4.5 summarizes the status and actions for obtaining the required permits.

Table 4.5 – Permitting status

PERMITS AND LICENCES			
Permit / Licence name	**Issuing Agency**	**Status**	**Comments**
Certificate of Authorization (CofA) for operating a mine	MDDEP	Completed	Delivered in September 2010
Request for Environmental Objectives for Rejects (OER)	MDDEP	Completed	Delivered in August 2010
CofA for septic installations	MDDEP	To initiate	Soil testing, design and CofA application to initiate once project receives go-ahead
CofA for drinking water well	MDDEP	To initiate	Drilling of well and CofA application to initiate once project receives go-ahead
Environmental Effects Monitoring Studies (EEMS)	Environment Canada	To initiate	Process to be initiated once project CofA for operation is obtained and project receives go-ahead
Rehabilitation plan	MRNFP	Completed	To be submitted to MNRW when project is started
Crown pillar study	MRNFP	Completed	Published April 2011
Wood cutting permit	MRNFP	To initiate	Documentation to be prepared and submitted to MNRW once project receives go-ahead
Gravel pit	MRNFP	To initiate	Only if required; application to be submitted once project receives go-ahead
Construction permit	Town of Senneterre	To initiate	Permit application document to be prepared and submitted once project receives go-ahead
Explosives permit	Natural Resources Canada	To initiate	Permit application document to be prepared and submitted once project receives go-ahead
Fuel storage permit	Régie du bâtiment du Québec	To initiate	Permit application document to be prepared and submitted once project receives go-ahead
Permit to dismantle beaver dams	MRNFP	To initiate	Permit application document to be prepared and submitted once project receives go-ahead

MDDEP: Ministère du Développement durable, de l'Environnement et des Parcs (Ministry of Sustainable Development, Environment and Parks)**; MRNFP:** Ministère des Ressources naturelles et de la Faune (Ministry of Natural Resources and Wildlife)

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Croinor property is situated in Pershing Township, roughly 55 km east of Val-d'Or (75 km by road), and is part of the municipality of Senneterre.

From Val-d'Or, access is afforded by the Trans-Canada Highway (Route 117) from the village of Louvicourt. Then, at the intersection of St-Felix River and Route 117, approximately 8 km south of Louvicourt, a gravel road leads north over a distance of 25 km to Lake Blanchin. This lake extends across the southern property limit (Fig. 5.1). Existing roads to the site need upgrading to support vehicle travel to and from the site, including the offsite transportation of ore for processing. A 13-km segment of the gravel road requires major maintenance. It is also possible to access the project from Senneterre via a gravel road leading south. Numerous trails (snowmobile, ATV) and roads that lead to hunting camps and cottages on Lake Blanchin run through the project area. Ponds and several creeks also cut through the area. Lake Blanchin and Vauquelin Bay represent the most important waterways. The nearest power line is linked to the former Chimo mine 20 km south of the property.

5.2 Climate

The region is under the influence of a continental climate marked by dry, cold winters and hot, damp summers. The average temperature for July is 17.1°C, whereas January temperatures hover around -17°C. Mean annual temperatures are 12°C. The registered record temperatures were a low of -43.9°C and a high of 36.1°C. There is an average of 209 days of frost. Historical records of annual precipitation rates indicate a mean rainfall of 954 mm in the Val-d'Or region. Most of the rain is in September with an average of 101.5 mm, although the heaviest individual rainfalls (precipitation per 24-hour period) have been recorded in July, including one of 68 mm. Snow accumulates from October to May with a peak between October and March. The average accumulation for this period, expressed as a water equivalent, is 54 mm.

5.3 Local Resources

The city of Val-d'Or (population 35,000) is the closest major service community. Both skilled and general workforces are readily available. Several operations and mills are currently active in the area. Suppliers, contractors, consulting firms and competent workers are available locally. An outfitter is also located about 10 km south of the Croinor property.

5.4 Infrastructure

The Croinor deposit is serviced by a ramp measuring 300 m long by 4 m high by 4.5 m wide (4 m x 4.5 m) extending to level 125 (38 m), and by a 3-compartment shaft 195 m deep. Development was completed on four (4) levels: 496 m on level 125; 560 m on level 250; 233 m on level 375; and 730 m on level 500. Approximately 320 m of raise development was also completed. The Croinor mine is currently flooded to the portal entrance. A photograph of the portal is presented on Figure 5.2 and the underground existing infrastructure is illustrated in Figure 5.3.

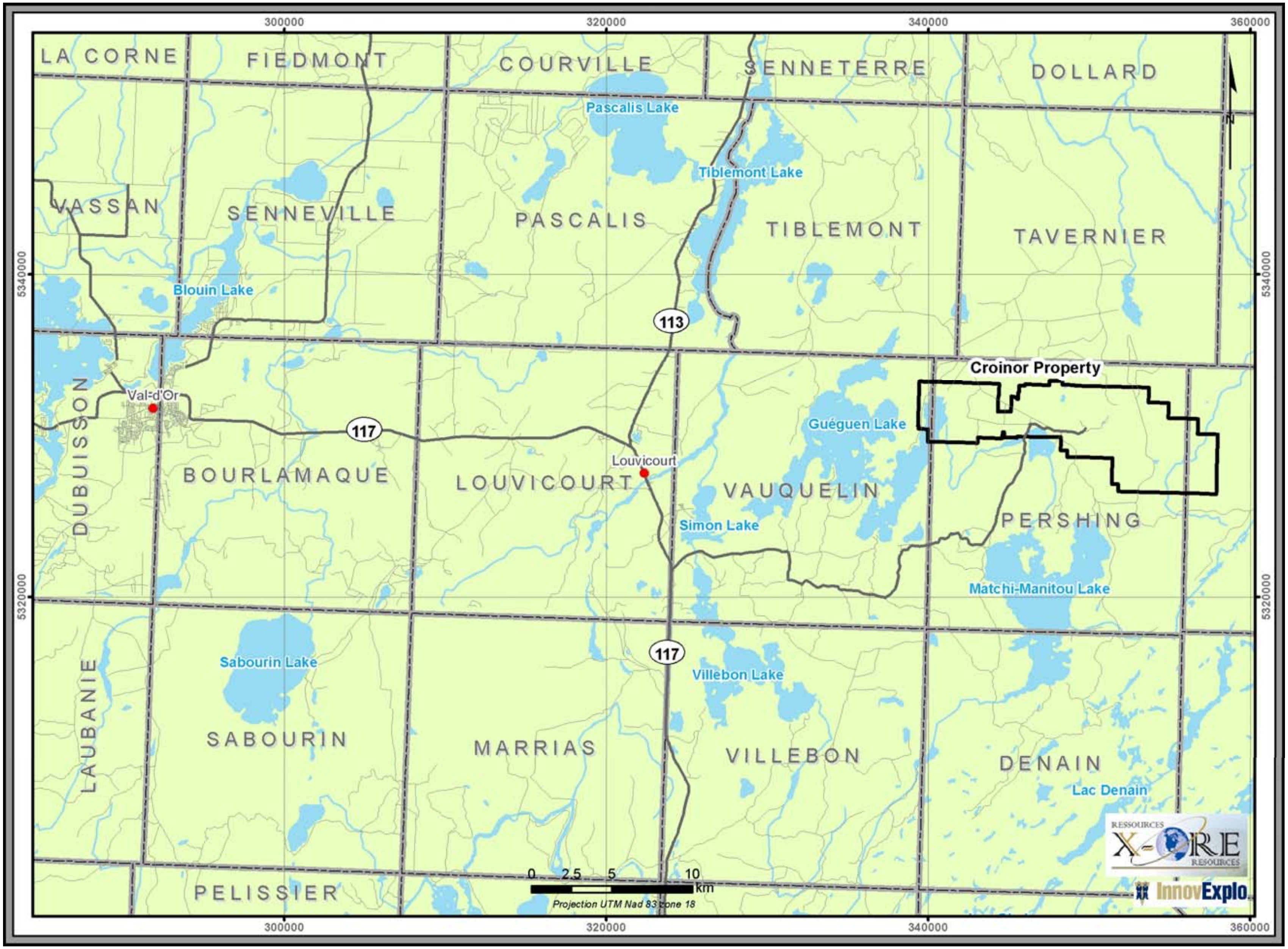


Figure 5.1 – Topography and accessibility of the Croinor property

Figure 5.2 – Portal entrance

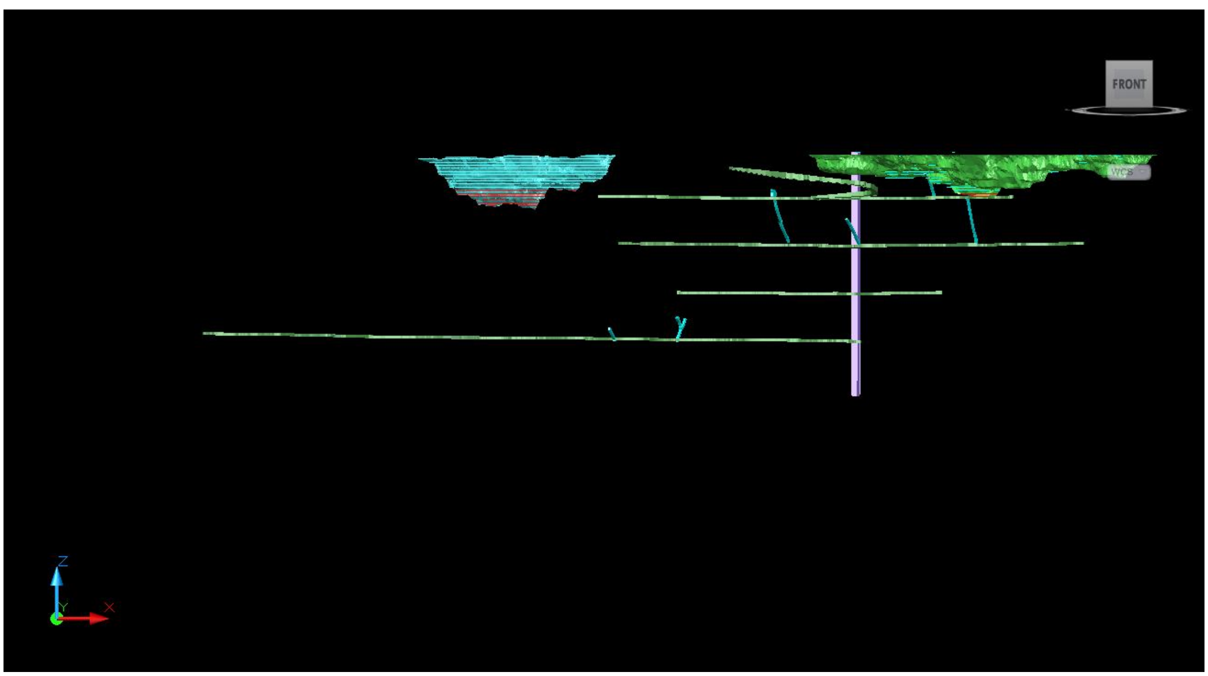


Figure 5.3 – Existing Croinor infrastructure

A single arch-type garage (Fig. 5.4) was erected on a concrete slab and is still present. The structure is in good condition and would need to be refurbished to be used for equipment maintenance. A concrete pad is also present next to the garage.

Figure 5.4 – Croinor garage

A settling pond (60 m x 60 m) is present on the site (Fig. 5.5), but it would require cleaning and stabilization work before it could be used again.

Figure 5.5 – Settling pond

5.5 Physiography

The property is marked by a low relief consisting mainly of poplar and evergreen forests, swamps and glacial drift. Several outcropping areas are found on the property. The property is predominantly covered by swamps in the western and northeastern parts. Water from the property generally drains into a creek flowing into Lake Blanchin. The latter drains into Marquis River and then into Vauquelin Bay of Guégen Lake, and from there into Simon Lake. Waters from Simon Lake flow north by Villebon Creek, traverse Sleepy Lake and Tiblemont Lake, and finally flow into Bell River.

6.0 HISTORY

Pershing Township has been explored since the early 1930s. The Croinor deposit was discovered in 1940, probably by a prospector named Fred Thompson. Several companies subsequently conducted exploration on the property. The work can be subdivided into four main periods:

- 1944-1948: Surface and underground drilling, shaft sinking (three compartments), and the development of 2,020 metres of drift on four levels by Croinor Pershing Mines Ltd;
- 1979-1989: Rehabilitation of the shaft, driving of a ramp, and surface plus underground drilling by Onaping Resources Ltd followed by Sullivan Mines Inc and then Cambior Inc;
- 1996-1997: Open pit mining and custom milling of more than 51,000 tonnes by Goldust Ltd;
- 1998-2003: Trenching, surface drilling, and completion of a 20,000-tonne bulk sample by Exploration Malartic-Sud Inc.

Table 6.1 summarizes the previous work on the Croinor property, and Table 6.2 summarizes the production history of the project. Table 6.3 lists the results of all previous resource estimates for the deposit since 1948.

Table 6.1 – Summary of previous work on the Croinor property

Year	Company	Work	Results	References
1939-43	Berthiaume option: Consolidated Mining & Smelting Company	• Geology report • Diamond drilling • Longitudinal projection	• Discovery of Croinor deposit	GM 3072, 7138-A, 7138-B, 7138-C, 7138-D, 8496-A, 8496-B, 8500
1944-48	Midd Pershing Mines Ltd; Croinor Pershing Mines Ltd	• **11,215 m** of drilling (surface) • Sinking of 195-m shaft • **5,588 m** of underground drilling	• 457-m mineralized zone grading 9.9 g/t Au with average width of 4.1 m	GM 124, 125, 372, 7137, 8066, 8089, 8092, 8093, 8101, 8499-A, 8499-B; MB-88-15
1949-78	Croinor Pershing Mines; Perron Gold Mines Ltd; Camflo Mattagami Mines Ltd; Anaconda American Brass Co; F. Corcoran; Les Mines Abigold Inc	• Diamond drilling • Trenching • Sampling • 9,979 t bulk sample	• 9,979 t @ 3.7 g/t Au (1,187 oz) (bulk sample) with recovery of 95%	GM 11082, 13997, 17268, 23398, 24487, 28494, 28975, 30000, 30594, 31018, 31862, 32279
1979-82	Onaping Resources Ltd; Spooner Mines and Oils Ltd; Dominion Explorers Ltd	• **17,505 m** surface diamond drilling • Rehabilitation of the shaft • Sinking of 395-m raise	• Work interrupted in 1982 due to weak gold price	Hill, Goettler, Delaporte Ltd, 1982
1983-86	Sullivan Mines Inc; Dominion Explorers Ltd	• **5,230 m** of drilling (underground) • **10,215 m** of drilling (surface) • Dewatering of shaft, mapping and sampling • Bulk sampling	• **1,700 t @ 1.47 g/t Au (93 oz)** (bulk sample) with a recovery of 86% • Resource estimate (Sullivan): **385,978 t @ 5.5 g/t Au (68,252 oz)** • Resource estimate (Dominion): **840,658 t @ 6.0 g/t Au (162,167 oz)**	P. Duhaime, 1986; J. Depatie, 1983; G. Laforet, 1983

Year	Company	Work	Results	References
1987-89	Cambior Inc; Dominion Explorers Ltd	• **10,475 m** of drilling (142 holes) • Trenching, mapping and channelling • Metallurgical tests	• Extended depth of mineralized zone • Resource and reserve estimates: Open pit mineable reserves: **55,820 t @ 5.6 g/t Au (10,050 oz)** Underground reserves: **148,206 t @ 7.4 g/t Au (35,260 oz)** Total resources: **863,568 t @ 8.0 g/t Au (222,115 oz)**	In-house documents from Cambior
1996-97	Goldust Ltd	• Open-pit mining (ore to waste ratio = 1:8.5)	• Amount extracted: 51,010 t @ 3.40 g/t Au (5,536 oz)[1]	Internal reports; GM 55945
1998-99	Exploration Malartic-Sud Inc	• Line cutting: 34 km • Ground geophysics: Mag/EM, 200-m spacing (east part of the deposit) • 2,150 m of drilling (9 holes) • Evaluation report	• Geophysical maps	Internal report; GM 56512; Gaudreault, 1997 (Géologica)
1999	Exploration Malartic-Sud Inc	• Geophysics: magnetic and HEM/MaxMin survey • Trenching – 16 trenches in total	• Discovery of T13 (trench number) showing: 1.93 g/t Au	GM 58791
2000	Exploration Malartic-Sud Inc	• Trenching between sections 500W and 900W • Resource evaluation by Géostat • 6,000 m of drilling (52 holes)	• Resource and reserve estimate (Géostat - SGI): 2.8 Mt @ 2.94 g/t Au	Internal report; SGI, 2000
2001	Exploration Malartic-Sud Inc	• Trenching • 4,087 m of drilling (58 holes)	• Metallurgical testing • Geological modelling	Internal reports
2002	Exploration Malartic-Sud Inc	• Line cutting (170 km) • Ground geophysics (IP) over 154.7 km • 14,137 m of drilling (121 holes)	• New geological and metallogenic model	Internal reports; GM 59933
2003	Exploration Malartic-Sud Inc	• Trenching (11 sites = 5,720 m^2) • 18,265 m of drilling (162 holes) • Dewatering and surveying of Goldust pit • Metallurgical tests on drill core • New mineral inventory • 20,000 t bulk sample • Line cutting (33 km), IP survey (31 km), Mag survey (195 km) • Geological prospecting • Re-interpretation of metallogenic model and new geological model	• New mineral inventory: 2.5 Mt @ 3.46 g/t Au	Internal reports; Saucier, 2003 (Met-Chem)
2004	Exploration Malartic-Sud Inc	• Geophysical surveys (IP and Mag) over western portion of property• Drilling of Bug Lake and Trench #2 area • Mining of the Centre pit	• Many conductors identified • Bug Lake 2.76 g/t Au over 22.9 m; 2.96 g/t Au over 12 m; 2.86 g/t Au over 8.3 m. • Trench #2: Discovery of new structure in tuffs (3.84 g/t Au over 3 m) • Milled from Centre pit: 30,760 t @ 2.2 g/t Au for 2,044 oz	Internal reports; reports filed with MRNFP; Pelletier & Boudrias, 2005 (NI43-101 Technical Report)
2005	Exploration Malartic-Sud Inc	• Mining of the West pit • New mineral resource estimate (InnovExplo)	• Milled from West pit: 24,363 t @ 5.0 g/t Au for 3,834 oz • New mineral resource estimate: **620,218 t @ 10.37 g/t Au for 206,792 oz**; Measured and Indicated categories	Pelletier & Boudrias, 2005 (NI43-101 Technical Report)

Year	Company	Work	Results	References
2007-2008	First-Gold Inc	• 12 914 m of drilling	• Numerous economic grade intercepts	Internal reports
2009	First-Gold Inc	• New mineral resource estimate and 43-101 technical report (P. O'Dowd) • Scoping study (F. Chabot, Golder Associates)	• New mineral resource estimate: **814,228 t @ 9.11 g/t Au @ 238,414 oz**; Measured and Indicated categories.	O'Dowd, 2009 (NI43-101 Technical Report); Chabot, 2009 (NI43-101 Technical Report)
2010-2011	Blue-Note Mining Inc and X-Ore Resources Inc	• 2,789.12m drilling in 2010 • 7,198m drilling in 2011 • Technical Report and Prefeasibility Study for the Croinor Project (InnovExplo) August 2010 • Update of the Prefeasibility by InnovExplo (Poirier 2011) in June 2011 • 3D-IP geophysical field work	• Drilling results (see Section 10 of this report) • NI43-101 compliant technical report and prefeasibility study using data from O'Dowd (2009). • Targets generated based on results of 3D-IP survey	Poirier et al., 2010 (NI43-101 Technical Report) Poirier et al., 2011 Perkins and Bérubé, 2010

Source: Marchand, 2004; Chénard and Turcotte, 2003 and all references therein.

[1] Conflicting data exist for this mining period. The numbers presented here are from Chénard and Turcotte (2003), which were taken from the Aurbel mill schedule from March 6 1997 to April 29 1997.

Table 6.2 – Past production on the Croinor Project

Production	Tonnes (t)	Grade (g/t)	Recovered ounces
Surface production			
Goldust Ltd (1996-1997)	51,010	3.4	5,536
Bulk sampling (2003-2004)	20,629	3.1	2,056
Centre pit	30,760	2.2	2,044
West pit (2004-2005)	24,363	5.4	3,834
Subtotal	**126,762**	**3.4**	**13,470**
Underground production			
Croinor (1949-78)	9,979	3.7	1,187
Sullivan Mines Inc (1983-86)	1,700	1.5	93
Subtotal	**11,679**	**3.4**	**1,280**
Total	**138,441**	**3.3**	**14,750**

Table 6.3 – Historical resource estimates on the Croinor deposit

YEAR	COMPANY	METHOD	TONNAGE (tm) Diluted	GRADE (g/t)	% Dilution
1948*	James & Buffam	-	107047	7.54	15% to 1.02 g/t
1973*	Middleton BM Abigold	-	399161	9.25	15% to 0 g/t
1974*	Schaaf & A./ Abigold inc.	-	259 554 (ND)	7.88	
1976*	Brown, P.K./ Abigold inc.	-	308 442 (ND)	7.88	
1982*	Page, C.E./ Harbinson G.	-	667518	7.2	29% to 1.02 g/t
1982*	Hill, Goet. & De Laporte	-	577651	6.51	21%
1982*	Geol. Dept. / Durham	-	240 113 (ND)	10.28	
1983*	Laforet / Minexpert	-	842 266	5.58	20% to 0.68 g/t
1984*	Veilleux CA/Sullivan	-	386 591	4.8	20% to 0 g/t
			1 014 035	5.55	
1984*	Page, C.E./ Onaping Cross-sections	-	702 886	6.17	20% to 0 g/t
1984*	Taylor MS/H.G. & D. (H,G & DL)	-	863 567	7.88	0.457 m of the wall
1988*	Bérubé, M./ Cambior inc.	-	479 414	6.51	
2000*	Systèmes Geostat International	Block Model (1/D2)	279 400	2.94	None
2001*	Systèmes Geostat International	Block Model (1/D2)	4 622 804	2.06	None
2003*	Malartic-Sud (Chénard/Turcotte)	Block Model (1/D2)	2 501 405	3.46	None
2003*	Met-Chem (Saucier)	Block Model (1/D2)	2 501 405 (mes+ind)	3.46	None
2005	InnovExplo (Pelletier/Boudrias)	Polygonal/ cross section	620 218 (ND mes+ind)	10.37	None
2009	O' Dowd, P.	Polygonal/ cross section	814 228 (ND mes+ind)	9.11	None

ND = not diluted

* These “resources” and/or “reserves” are historical in nature and should not be relied upon. It is unlikely they conform to current Regulation 43-101 criteria or to CIM Standards and Definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting

The Croinor property is located in the eastern part of the Archean Abitibi Greenstone Belt in the southern Superior Province of the Canadian Shield.

The Abitibi Greenstone Belt is one of the most extensive continuous expanses of low metamorphic grade Archean volcanic and sedimentary rocks on Earth (Card and Poulsen, 1998). It also happens to be one of the richest mining regions in the world and has produced large amounts of gold, copper, zinc, silver and iron from the Timmins, Kirkland Lake, Rouyn-Noranda, Val-d'Or, Matagami and Chibougamau mining districts. For these reasons, the Abitibi Greenstone Belt has been the focus of numerous studies. Most of this work is summarized by Card and Poulsen (1998) and presented on a compilation map of the southern Superior Province produced by the *Ministère de l'Énergie et des Ressources du Québec* and the Ontario Geological Survey in 1983. Several articles have also been published about the Abitibi belt, namely by Dimroth et al. (1982; 1983a,b; 1984a,b; 1985a,b), Hodgson (1983), Gélinas and Ludden (1984), Allard et al. (1985), Jensen (1985), Ludden et al. (1986), Card (1990), and Jackson and Fyon (1991).

The Abitibi Greenstone Belt comprises extensive Neoarchean volcano-sedimentary sequences and a multitude of intrusions ranging from synvolcanic to post-tectonic in age and from ultramafic to felsic in composition. The vast majority of volcanic episodes took place from 2.75 to 2.70 Ga and was closely followed by deformation, regional metamorphism and an episode of plutonism during the period from 2.70 to 2.65 Ga (Card and Poulsen, 1998). Hocq (1990) explains that early ductile to brittle deformation (folding and faulting) subsequently became increasingly brittle, eventually producing a tectonic collage with diamond-shaped, weakly deformed domains separated by strongly deformed zones. Near the end of deformation, clastic sedimentary basins and shoshonitic-alkaline volcanic rocks were emplaced (Mueller and Donaldson, 1992).

In many parts of the Abitibi Greenstone Belt, the succession may be divided into two major cycles typically characterized by a mafic to ultramafic volcanic sequence at the base, overlain by a mafic to felsic, tholeiitic to calc-alkaline sequence at the top (Card and Poulsen, 1998). Locally, accumulations of sedimentary rocks (turbiditic and alluvial and/or fluvial) occur associated with alkaline to shoshonitic volcanic rocks, as well as a few intrusions (Dimroth et al., 1982; Jensen, 1985). According to Dimroth et al. (1985a,b), these sequences may be correlated with three major physiographic settings, namely extensive subaqueous lava flows, subaqueous to subaerial volcanic complexes, and intra-arc basins.

A series of major volcanic episodes took place from 2750 to 2698 Ma; the latter represent the vast majority of volcanic sequences in the Abitibi Greenstone Belt (Card and Poulsen, 1998). One of these sequences, dated from 2710 to 2698 Ma and restricted to the southern part of the Abitibi Greenstone Belt, forms a volcano-plutonic assemblage that is very widespread throughout the belt. This sequence is composed of a basal komatiitic unit, followed by a bimodal assemblage alternating from basaltic to rhyolitic in composition, and capped by a tholeiitic and calc-alkaline assemblage (Card and Poulsen, 1998). This sequence hosts the vast majority of volcanogenic massive sulphide deposits in the southern part of the Abitibi Greenstone Belt. Many older volcanic sequences (2720 to 2713 Ma; 2730 to 2725 Ma) are widely scattered throughout the Abitibi Greenstone Belt (Card and Poulsen, 1998). These sequences, in the south part of the belt, host the Kidd Creek (2712 Ma) and Prosser (2716 Ma) rhyolites, the Deloro (2725, 2727 Ma), Wabewawa-Catherine (2720 Ma), Normetal (2728 Ma) and Hunter Mine (2730 Ma) groups, and the Rand (2713 Ma) and Ghost Range (2713 Ma) mafic to ultramafic complexes. In the north part of the belt, these volcanic sequences include the Joutel (2730 Ma), Dumagami (2723 Ma), Watson Lake (2725 Ma) and Garon Lake (2721 Ma) rhyolites, as well as the Waconichi Formation (2728, 2730 Ma), the Bell River (2725 Ma) and Lac Doré (2724, 2728 Ma) gabbro and anorthosite complexes, and the Cummings mafic to ultramafic complex (2717 Ma).

Metasedimentary units in the Abitibi Greenstone Belt include turbiditic flysch sequences overlain by conglomeratic molasse-type sequences (Mueller and Donaldson, 1992). The Porcupine, Cadillac and Kewagama groups occur in the southern Abitibi, whereas the Taibi, Caopatina and Chicobi Lake groups occur in the northern Abitibi. Mueller and Donaldson (1992) explain that these sedimentary units were deposited from 2730 to 2720 Ma in the northern Abitibi, and from 2700 to 2687 Ma in the south.

The economic potential of the Abitibi Greenstone Belt is well established. The Croinor property area is no exception, and the region’s economic potential is illustrated by several past and present producers. The Val-d’Or mining camp has produced more than 15 Moz of gold from more than twenty-six (26) mines. Most of the production comes from the Sigma-Lamaque hydrothermal system from which 9 Moz has been extracted.

7.2 Local Geological Setting

The Croinor property is underlain by two major lithological packages—the dominantly volcanic Assup Domain in the north and the dominantly sedimentary Garden Island Domain in the south. Lithological units in both domains are generally oriented N295° with steep dips to the north. The Assup Domain is further divided into the mafic volcanic Aurora Group in the south and the mafic to intermediate volcanic Assup Group in the north (dominantly mafic on the property). The Pershing Batholith overlaps the southwest end of the property. The property is transected by the Garden Island Deformation Corridor in a NW-SE direction, which overprints and partially follows the contact between the Assup and Garden Island domains.

The Croinor deposit is located 3 km northeast of the Pershing Batholith and 2 km northeast of the Garden Island Deformation Corridor (Fig 7.1).

7.3 Property

The Croinor deposit is hosted by the synvolcanic Croinor Sill. This dioritic sill is from 60 to 120 metres thick and hosted within volcanic rocks of the Assup Domain (Fig 7.1). The deposit is characterized by gold-rich lenses consisting of quartz-carbonate-tourmaline-pyrite veins, altered pyritic host rock material, and/or tectonic breccia (pyritic host fragments within a quartz-carbonate-tourmaline-pyrite vein).

The mineralized lenses are spatially controlled by reverse-oblique shear zones that crosscut and displace both the lenses and its dioritic host. A hydrothermal alteration halo surrounds these structures. Zoning begins with an epidote-chlorite envelope that gradually changes into a chlorite-carbonate zone closer to the shear. Within the shear structure itself, the host rock has undergone extensive alteration characterized by a sericite-ankerite-pyrite assemblage.

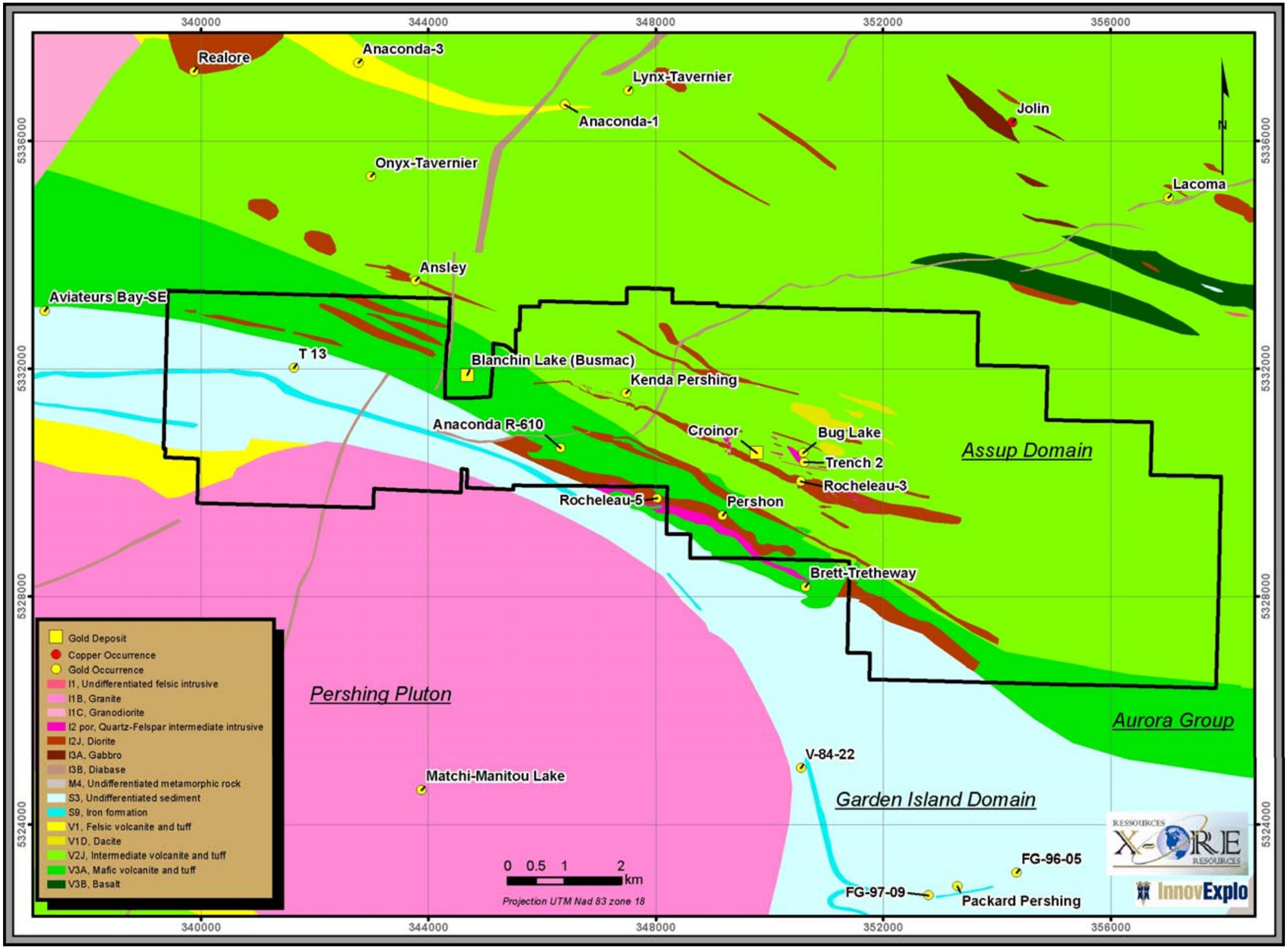


Figure 7.1 – Geology of the Croinor property area

8.0 DEPOSIT TYPES

Lode gold deposits (gold from bedrock sources: Fig. 8.1) occur dominantly in terranes with an abundance of volcanic and clastic sedimentary rocks of a low to medium metamorphic grade (Poulsen, 1996). Greenstone-hosted quartz-carbonate vein deposits are a subtype of lode-gold deposits (Poulsen et al., 2000). They correspond to structurally controlled, complex epigenetic deposits hosted in deformed metamorphosed terranes (Dubé and Gosselin, 2007).

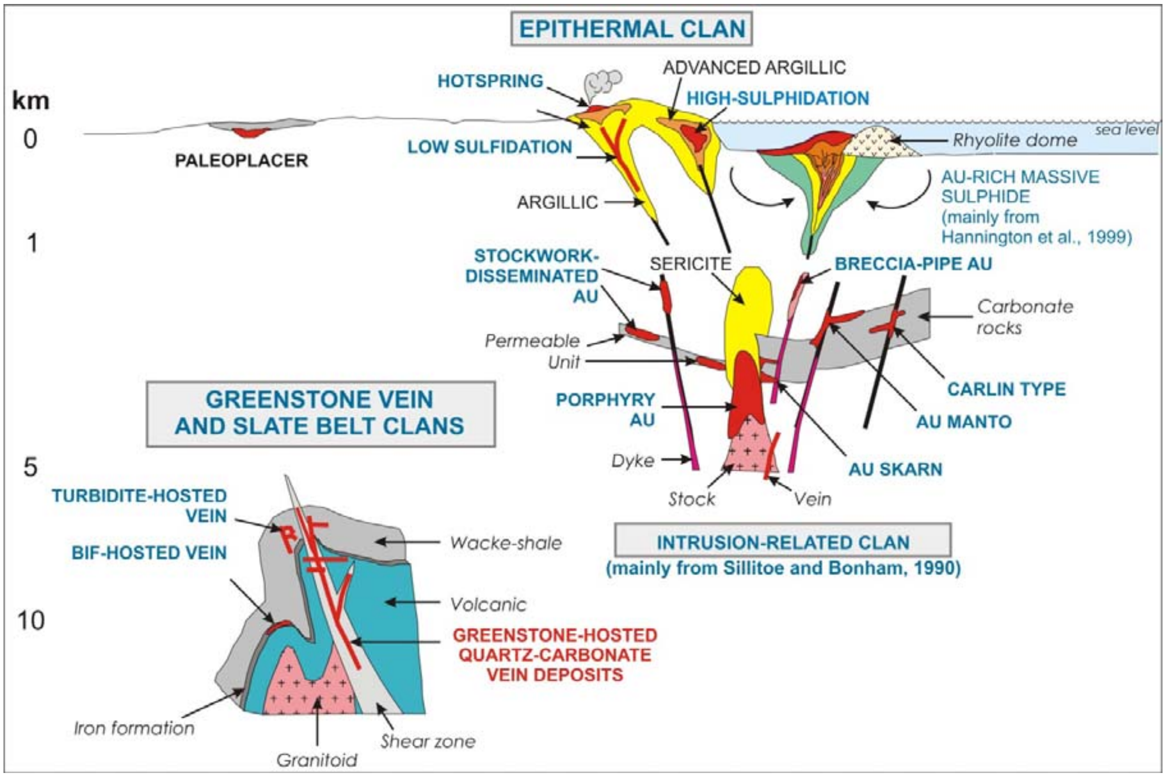


Figure 8.1 – Inferred crustal levels of gold deposition showing the different types of lode gold deposits and the inferred deposit clan (from Dubé et al., 2001; Poulsen et al., 2000).

Greenstone-hosted quartz-carbonate vein deposits consist of simple to complex networks of gold-bearing, laminated quartz-carbonate fault-fill veins in moderately to steeply dipping, compressional brittle-ductile shear zones and faults with locally associated shallow-dipping extensional veins and hydrothermal breccias. They are hosted by greenschist to locally amphibolite facies metamorphic rocks of dominantly mafic composition and formed at intermediate depth in the crust (5-10 km). They are distributed along major compressional to transtensional crustal-scale faults zones (Fig. 8.2) in deformed greenstone terranes of all ages, but are more abundant and significant, in terms of total gold content, in Archean terranes. Greenstone-hosted quartz-carbonate veins are thought to represent a major component of the greenstone deposit clan (Fig. 8.1) (Dubé and Gosselin, 2007). They can coexist regionally with iron formation-hosted vein and disseminated deposits, as well as with turbidite-hosted quartz-carbonate vein deposits.

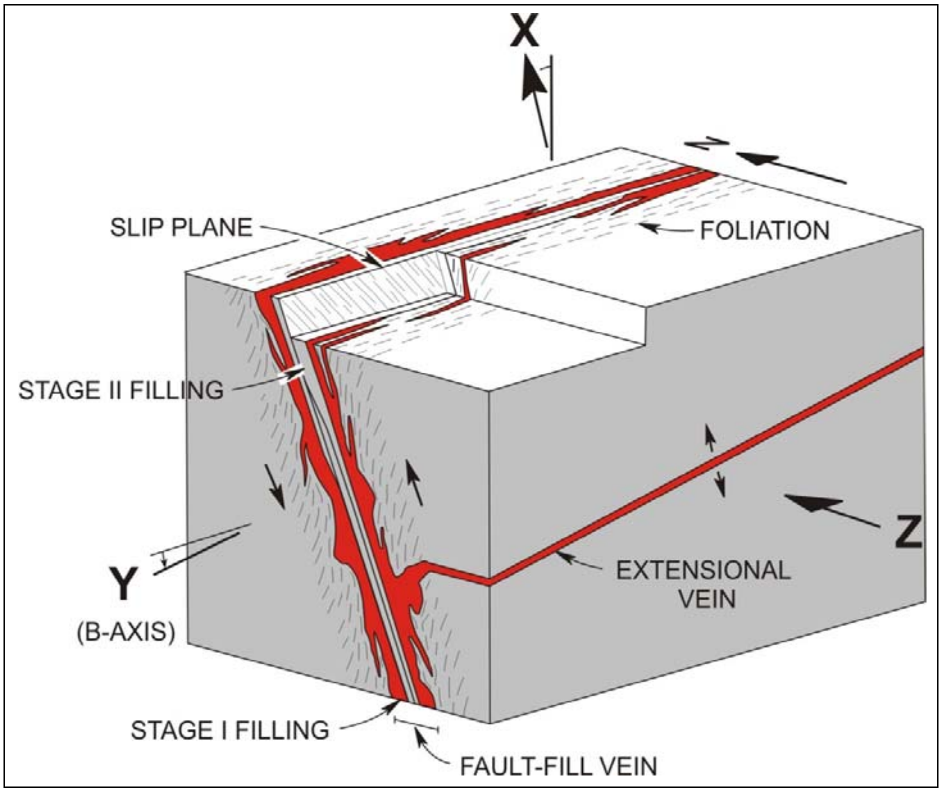


Figure 8.2 – Schematic diagram illustrating the geometric relationships between the deposit-scale strain axes and the vein and shear zone structural elements (from Robert, 1990).

The main gangue minerals are quartz and carbonate with variable amounts of white micas, chlorite, scheelite and tourmaline. The sulphide minerals typically constitute less than 10% of the ore. The main ore minerals are native gold with pyrite, pyrrhotite and chalcopyrite without significant vertical zoning (Dubé and Gosselin, 2007).

8.1 Croinor

The mineralized lenses at Croinor range from 60 to 120 metres long (Chénard and Turcotte, 2003). The lenses can generally be followed from one section to another (10-metre sections) over lateral distances varying from several tens of metres up to 600 metres. To date, about forty (40) gold-rich lenses have been identified. Gold-hosting structures within the dioritic Croinor Sill consist of a series of narrow breaks and reverse shears with dips of 25° to 60° to the north. These structures display en-echelon patterns and are variably oriented between N290°-N330°. They can cause displacements of a few centimetres to tens of metres. Tectonic breccias of pyritic quartz-tourmaline-ankerite material occupy the central portion of these shears (Fig. 8.1).

Two (2) types of veins are observed: 1- shear-parallel veins (shear veins); and 2- subhorizontal tension veins. The veins consist of quartz, tourmaline and carbonates with minor amounts of pyrite, chalcopyrite and native gold.

The tension veins are generally several centimetres thick and dip gently towards the east. Gold is restricted to rocks affected by pyritic metasomatism; that is, it mainly occurs in pyritic vein selvedges, breccia fragments in tectonic breccias, and fragments in brecciated veins. Pyrite is locally associated with minor amounts of chalcopyrite and pyrrhotite. Gold is found as inclusions within pyrite grains or along its boundaries, or within fractures cutting pyrite. Although pyrite is a good indicator of the presence of gold, not all pyritic zones are mineralized with gold, and not all gold occurs with pyrite. Isolated flecks of native gold were also observed in veins and veinlets of smoky quartz associated with shearing, and in the highly altered selvedges of such veins.

Pyrite occurs as two distinct forms that may correspond to distinct crystallization phases. The earliest phase is present as amorphous clusters and threads that frequently show evidence of deformation and a preferential orientation subparallel to the main foliation. The last phase appears as disseminated cubes with individual grains reaching 0.5 cm to locally 1 cm.

Other than pyrite content, the intensity of deformation and the type of alteration constitute the essential criteria for the presence of economic gold grades. Sericite-ankerite alteration combined with tourmaline and silicification appears to be directly associated with the highest grades within brecciated zones containing pyritic quartz-tourmaline-ankerite material.

Quartzo-feldspathic rocks that intrude the Croinor deposit area may have played an important role in gold mineralization. The relatively late emplacement of these intrusives may have accentuated the intensity of deformation and thus increased host rock porosity, in addition to prolonging the mineralization event while contributing to the remobilization of local hydrothermal fluids.

9.0 EXPLORATION

Exploration work performed on the Croinor property up to and including 2010 is summarized in Table 9.1.

Table 9.1 – Summary of exploration work performed on the Croinor property

Date	Work	Results	References
2003	• Trenching (11 sites = 5,720 m^2 • 18,265 m of drilling (162 holes) • Dewatering and surveying of Goldust pit • Metallurgical tests on drill core • New mineral inventory • 20,000 tonne bulk sample • Line cutting (33 km), IP survey (31 km), Mag survey (195 km) • Geological prospecting • Re-interpretation of metallogenic model and new geological model	• New mineral inventory: 2.5 Mt @ 3.46 g/t Au	Internal reports; Met-Chem report
2004	• Geophysical surveys (IP and Mag) over western portion of property • Drilling of Bug Lake and Trench # 2 area (cf. Section 10 Drilling, this report)	• Numerous conductors identified • Bug Lake: 2.76 g/t Au over 22.9 m; 2.96 g/t Au over 12 m; 2.86 g/t Au over 8.3 m. • Trench # 2: Discovery of new structure within tuffs (3.84 g/t Au over 3 m).	Internal reports; reports filed with MRNFP
2010	• Geophysical surveys (3D-IP) covering eastern on-property extension and known deposit area	• Numerous conductors identified	Perkins and Bérubé, 2010

9.1 Recent Exploration Work performed by Blue Note Mining

In 2010, Blue Note Mining commissioned Abitibi Geophysics to carry out a 3D hole-to-hole resistivity/IP survey on the Croinor property. The objective of the study was to assess the potential for gold mineralization at depth and the extent of known mineralization. This information was used to formulate a follow-up program in 2011 on the most promising anomalies.

A total of 26 independent pairs of receiver holes were surveyed over the property. A list of all the holes used in the survey is presented in Table 9.2, and the locations of these holes are shown in Figure 9.1. Overall, the 3D inversion identified a potential structural corridor in the northwest part of the survey area. This corridor is represented by a sharp contact between a resistivity high and a resistivity low, corresponding to a decrease in chargeability. Multiple small IP anomalies were identified adjacent to the structural feature, as well as larger IP anomalies at depth that were intersected by CR-07-332A.

In the southeast part of the survey area, multiple smaller shallow IP anomalies were identified, as well as a large anomaly at depth that was intersected by CN-88-127 and CN-88-128.

Table 9.2 – List of boreholes used for the 3D hole-to-hole survey on the Croinor Project

HOLE-ID	AZIMUTH	DIP	LENGTH	Easting (mE)	Northing (mN)
CN-88-127	200	-70	310.29	350024.36	5330547.59
CN-88-128	200	-70	270.66	349947.28	5330519.89
83-S-261	200	-85	367.89	349814.56	5330598.15
83-S-262	200	-85	371.55	349878.31	5330593.21
83-S-263	200	-83	365.76	349928.90	5330555.61
CR-07-331	205	-78	453	349438.00	5330784.00
CR-07-332-A	25	-62	585	349368.00	5330619.00
CR-07-333	205	-78	516	349624.00	5330729.00
CR-07-334	25	-62	602	349520.33	5330476.77
CR-07-335	205	-72	592	349777.20	5330643.57
CR-07-337	25	-86	399	349685.00	5330627.00
CR-07-338	25	-86	467	349667.00	5330580.00
CR-07-339	108.06	-90	620	349520.00	5330695.00
CR-07-340-A	25	-82	476	349489.00	5330655.00
CR-07-341	25	-87	393	349448.00	5330667.00
CR-07-342	25	-87	384	349304.00	5330730.00
CR-07-345	21	-86	358	349343.00	5330710.00
CR-07-346	65	-87	399	349268.00	5330793.00
CR-07-347	348	-86	399	349213.00	5330790.00
CR-08-349	80	-87	351	349594.08	5330620.64
CR-08-352	0	-87	300	349805.93	5330494.04
CR-08-353	0	-87	251	349249.55	5330744.01
CR-08-360	0	-87	300	349200.01	5330811.01

The information obtained from this study was used to plan the 2011 drilling program. Most of the IP anomalies on the eastern and western extensions of known mineralization were targeted for drill testing in 2011 and are identified as such in the following section. Complete descriptions and images are documented in the report prepared by Abitibi Geophysics.

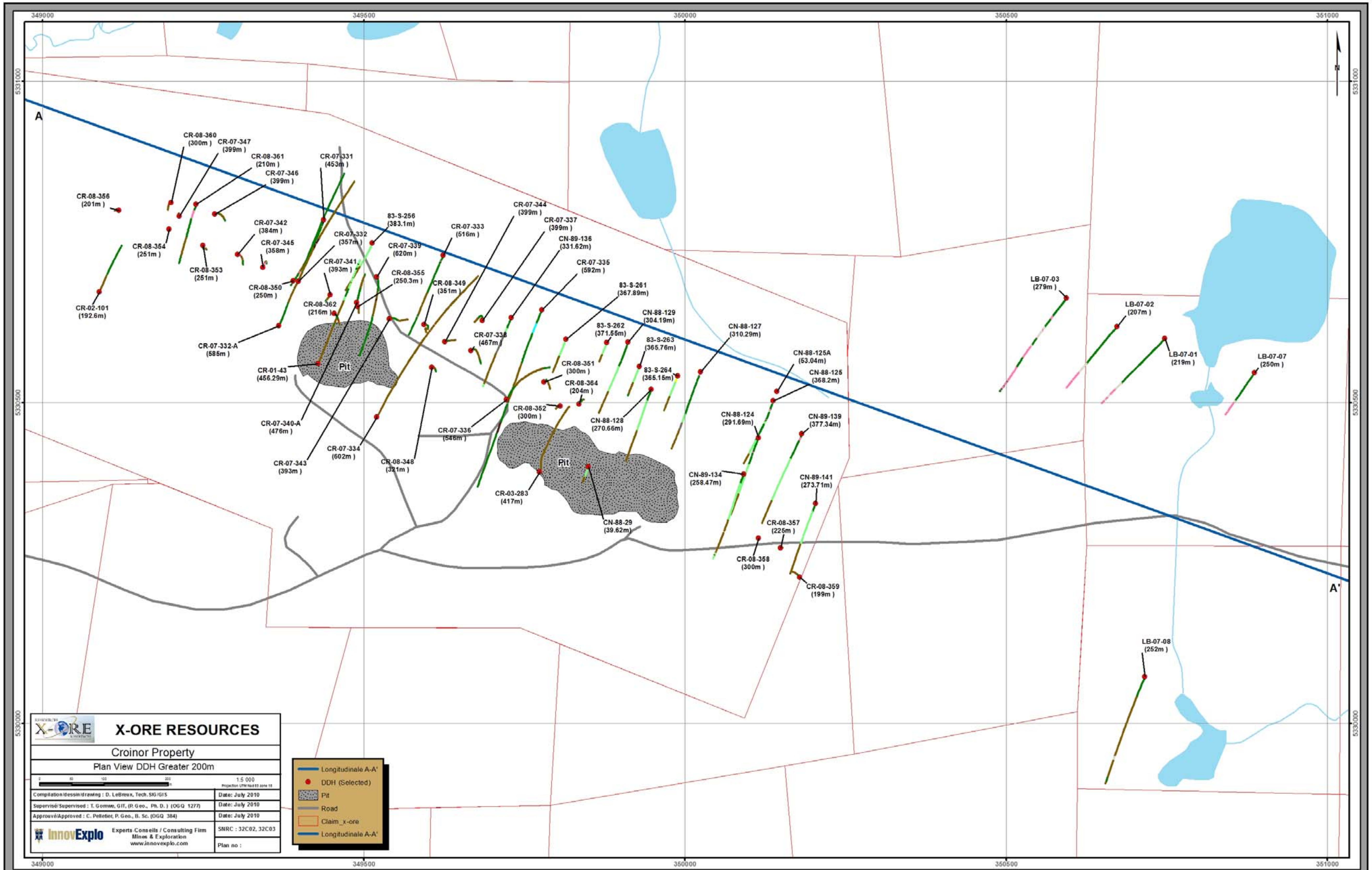


Figure 9.1 – Location of surveyed holes on the Croinor property

10.0 DRILLING

The drilling programs conducted on the Croinor property by Blue Note Mining Inc (also under the name of X-Ore Resources Inc, formerly Malartic-Sud Inc) are described below. The summary of the 1998-2004 drill programs presented in Table 10.1 was compiled using the technical report by Pelletier and Boudrias (2005). The description of the 2007-2008 drilling programs is from the technical report by O'Dowd (2009).

10.1 Drilling during the 1998-2004 Period (Exploration Malartic-Sud)

Table 10.1 – Drilling programs from 1998 to 2004

Year	Description	Comments
1998	• 9 drill holes for **2,150 m**	
2000	• 52 drill holes for **6,000 m**	• Resource update
2001	• 58 drill holes for **4,087 m**	
2002	• 121 drill holes for **14,137 m**	
2003	• 2002-2003: total of 198 drill holes for **24,659 m** in pit area. • 8 geotechnical drill holes (tailings) for **814 m**	• Discovery of Bug Lake: 1.34 g/t Au over 23.5 m, incl. 4.59 g/t Au over 3.5 m and 3.44 g/t Au over 3.0 m
2004	• 31 drill holes and 5 extensions (**4,094 m**) on Bug Lake and the Trench 2 area.	

10.2 Drilling during the 2007-2008 Period (X-Ore Resources/First Gold Exploration)

The total metres for 2007-2008 amount to 12,648 m on Croinor 1 and 3,833 m on Croinor 2. NQ rods were used in 2007, but both NQ and BQ rods were used during the 2008 program. The use of BQ rods in the second half of 2008 was because NQ rods were unavailable and the near-vertical holes (-87°) did not require maximum stabilization equipment. Diamond drilling on the mining lease (BM 862) was mostly aimed at testing lateral and vertical extensions of known mineralization related to the Croinor deposit. Figures 10.1 and 10.2 show the location of drill holes for the 2007-2008 program. Most holes intersected anomalous to significant gold mineralization related to known zones or potential new ones. Tables 10.2 and 10.3 show the best results from the Croinor 1 and Croinor 2 projects, respectively. Some of these intervals are included in the 2009 resource estimate, but other holes had pending results at the time and could not be included. Drill spacing ranged from 40 to 50 m, both laterally and vertically. In 2007, First Gold drilled a few holes that reached 500 to 600 m long, but mineralization was sparse. The scope of the programs in 2007-2008 was eventually limited to drilling to a vertical depth of 400 metres. Given the style of mineralization (as established by InnovExplo's recent interpretation), the current drill spacing below 300 m is entirely inadequate for investigating the gold potential of the area at depth.

Drilling on Croinor 2 was more speculative, targeting small and poorly known showings. These targets were selected based on the general compilation for the Croinor 2 Project. A number of showings, most discovered decades ago, were reviewed and some drill targets defined. The information available for the compilation was generally quite poor and the locations of the showings remained approximate.

Diamond drilling was carried out by Benoit Diamond Drilling of Val-d'Or (now Major Drilling). NQ and BQ sized tubes were used in 2008. Some holes were surveyed by Corriveau J.L. &

Associates Inc (Land-Surveyor) while others were located by company staff using a GPS unit. All casings were left in place and properly identified with aluminum tags on steel posts.

Holes were surveyed every 50 metres down the hole using a Reflex unit (single shot). Permitting and environmental matters were mandated to G.E.S.S.T. of Val-d'Or. Samples were prepared and assayed at Laboratoire Expert of Rouyn-Noranda or Technilab of Sainte-Germaine Boulé. Core logging was performed by geologist Rodolphe Lavoie under the supervision of Pierre O'Dowd. InnovExplo performed all the drafting in Gemcom format.

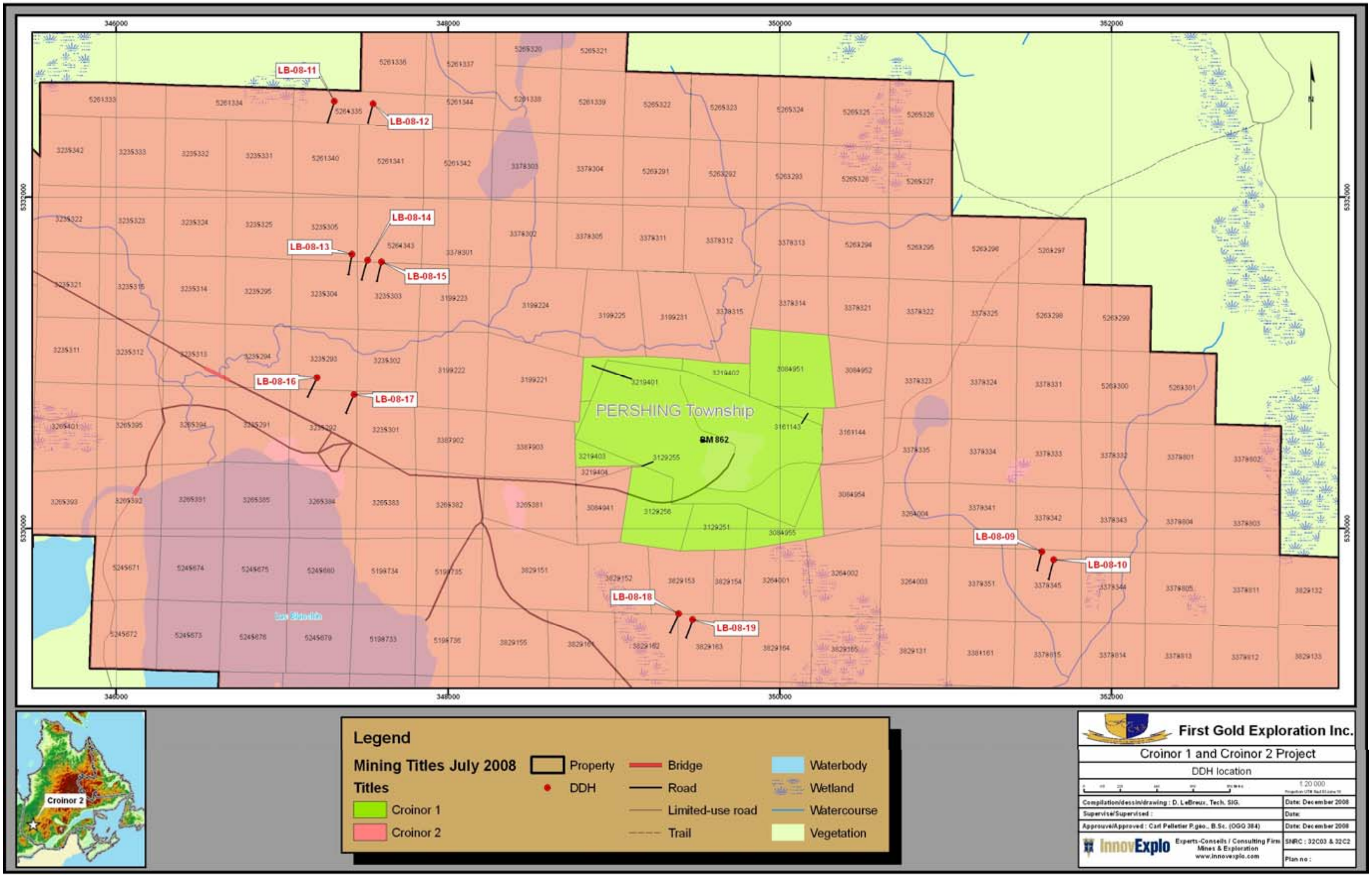


Figure 10.1 – Locations of the 2007-2008 diamond drill holes on the Croinor claim map (Source O'Dowd, 2009)

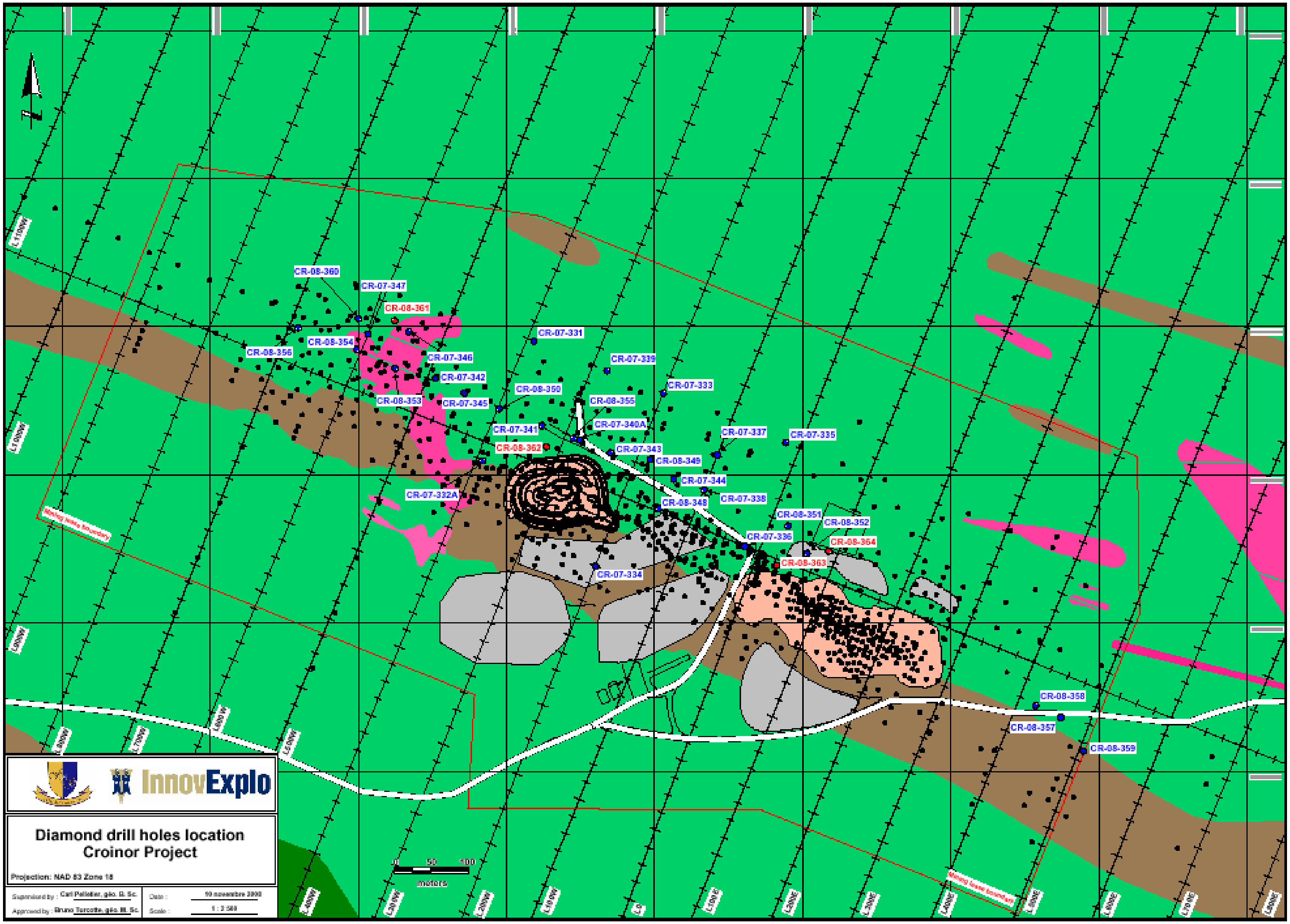


Figure 10.2 – Locations of the 2007-2008 diamond drill holes (Source O'Dowd, 2009)

Table 10.2 – Best intersections: Croinor 1 - 2008

CROINOR 1 - 2008				
DRILL HOLE	**FROM**	**TO**	**WIDTH**	**GRADE g/t**
CR-08-348	122	123	1.00	2.44
CR-08-348	138	142	3.00	4.36
CR-08-348	145	146	1.00	1.57
CR-08-348	**198**	**202**	**4.00**	**4.67**
CR-08-348	221	223	2.00	3.80
CR-08-349	**255**	**260**	**5.00**	**4.74**
CR-08-349	261	262	1.00	1.54
CR-08-350	no value >1g			
CR-08-351	262	263	1.00	1.76
CR-08-352	158	159	1.00	1.30
CR-08-352	162	164	2.00	2.31
CR-08-352	181	182	1.00	2.21
CR-08-353	**154**	**157**	**3.00**	**3.20**
CR-08-353	170	171	1.00	2.21
CR-08-353	191	192	1.00	2.54
CR-08-353	197	198	1.00	1.12
CR-08-353	200	201	1.00	1.60
CR-08-354	155	156	1.00	4.25
CR-08-354	159	162	3.00	2.65
CR-08-354	164	165	1.00	1.77
CR-08-354	**166**	**169**	**3.00**	**4.99**
CR-08-354	**170**	**172**	**2.00**	**5.33**
CR-08-354	181	182	1.00	1.11
CR-08-354	**190**	**192**	**2.00**	**14.26**
CR-08-354	209	210	1.00	2.38
CR-08-354	211	213	2.00	2.26
CR-08-355	no value >1g			
CR-08-356	151.3	154	2.70	1.83
CR-08-356	187	188	1.00	1.90
CR-08-356	196	197	1.00	1.59
CR-08-357	176	181	5.00	2.30
CR-08-358	160	161	1.00	1.62
CR-08-358	172	173	1.00	1.12
CR-08-358	185	187	2.00	2.90
CR-08-359	107	108	1.00	6.22
CR-08-360	213.5	215	1.50	0.93
CR-08-360	**218**	**219.5**	**1.50**	**8.13**
CR-08-360	**251**	**253**	**2.00**	**5.41**
CR-08-360	298	299	1.00	1.03
CR-08-361	179	180	1.00	0.99
CR-08-361	182	186	4.00	1.35
CR-08-362	157.3	157.8	0.50	1.20

CROINOR 1 - 2008				
DRILL HOLE	**FROM**	**TO**	**WIDTH**	**GRADE g/t**
CR-08-362	166.5	168	1.50	1.48
CR-08-362	171	172.5	1.50	1.05
CR-08-362	171	172.5	1.50	1.05
CR-08-362	**179.5**	**181**	**1.50**	**8.51**
CR-08-363	**72**	**78**	**6.00**	**9.07**
CR-08-364	188	189	1.00	1.11
CR-08-364	191	192	1.00	1.01

Table 10.3 – Best Intersections: Croinor 2 - 2008

CROINOR 2 - 2008						
DDH #	**FROM**	**TO**	**LENGTH**	**DIP**	**TRUE WIDTH**	**Au g/t**
LB-08-16	119	120	1.00	55	0.82	1.12
LB-08-19	190	191	1.00	70	0.94	2.48
LB-08-19	191	192	1.00	70	0.94	2.59

10.3 Drilling during the 2010-2011 Period (Blue Note Mining)

The previous study by Poirier et al. (2011) did not include the drilling results from the 2010 and 2011 programs, but they recommended the assay results be included in subsequent updates. In all, 65 holes were drilled for a total of 15,390 m during the 2010 and 2011 drilling programs. This section presents all information on the various drilling programs. Because the cut-off date for the current estimate was mid-November, holes with pending assays at that time (CR-11-414 to CR-11-419) could not be included in the current resource estimate; however, InnovExplo is of the opinion that these results would not have had a significant impact on the results. InnovExplo does not recommend a complete update of the current Mineral Resource Estimate and prefeasibility study to include these assay results. A property map in the pocket (Appendix IV) displays the locations of the 2010-2011 drill holes.

10.3.1 Drilling program – 2010

The initial phase of drilling at Croinor ran from June to the beginning of July, 2010. The phase included holes CRG-10-01 to CRG-10-05 and CR-10-365 to CR-10-371. There were two objectives:

1. Test the ground stability of the crown pillar in the Croinor gold deposit to provide key information for the mine engineering plan for future mining operations. The full report was completed by Golder Associates Ltd. InnovExplo carried out the core logging and selection of mineralized samples for assaying.
2. Test the near-surface mineralization located west of the formerly mined West Pit to assess the potential for open pit mining.

Phase 2 of the Croinor drill program ran from July to August and included holes CR-10-372 to CR-10-378. Phase 2 consisted of follow up drilling to delineate potential resources below and lateral to the current resource, and to further test the mineralized diorite sill at depth.

Eleven (11) diamond drill holes (NQ size) for a total of 720 m were logged and sampled for Phase 1 (Table 10.4). A total of 8 diamond drill holes (NQ size) with a total of 2,069 m were logged and sampled for Phase 2 (Table 10.5). The 2010 program produced 602 samples, including CRM standards, blanks, and field duplicates.

In 2010, seven (7) of the drill holes intersected significant gold grades and widths in close proximity to planned development, demonstrating the potential to expand the reserves. Values included 11.81 g/t Au over 7.5 m, 10.50 g/t Au over 6.7 m, and 29.3 g/t Au over 5.7 m, with values as high as 78.15 g/t Au over 1 m.

Previous drilling west of the West Pit identified gold mineralization, typically within 20 m of surface, which could be easily extracted via shallow open pit. The 2010 campaign generally confirmed the continuity of the mineralization and included higher grade intervals of 7.27 g/t gold over 1.0 m within a 5.1-m interval grading 2.30 g/t gold in CR-10-368 and 7.74 g/t gold over 1.1 m within a 4.1-m interval grading 2.65 g/t gold in CR-10-371.

Table 10.4 – Croinor Phase 1 diamond drill holes executed in July 2010

DDH	YEAR	Easting (UTM83 Z18)	Northing (UTM83 Z18)	Elevation (m)	Azimuth	Dip	Length	Note
CRG-10-01	2010	349386	5330632	3045	180	-70	98.6	Crown Pillar
CRG-10-02	2010	349580	5330529	3049	200	-70	71.7	Crown Pillar
CRG-10-03	2010	349703	5330481	3048	232	-62	71.4	Crown Pillar
CRG-10-04	2010	349817	5330487	3049	200	-57	113.52	Crown Pillar
CRG-10-05	2010	349906	5330316	3049	336	-75	90	Crown Pillar
CR-10-365	2010	349290	5330641	3040	202	-47	10.5	Abandoned hole
CR-10-365A	2010	349290	5330641	3040	202	-47	12	Abandoned hole
CR-10-365B	2010	349290	5330641	3040	202	-47	42	West Pit
CR-10-366	2010	349273	5330629	3040	202	-61	12	Abandoned hole
CR-10-366A	2010	349273	5330629	3040	202	-61	12	Abandoned hole
CR-10-366B	2010	349273	5330629	3040	202	-61	36	West Pit
CR-10-367	2010	349270	5330620	3041	202	-61	30	West Pit
CR-10-368	2010	349293	5330622	3040	202	-47	30	West Pit
CR-10-369	2010	349310	5330610	3041	0	-90	45	West Pit
CR-10-370	2010	349318	5330607	3041	0	-90	45	West Pit
TOTAL Croinor Phase 1							**719.72**	

Table 10.5 – Croinor Phase 2 diamond drill holes executed from July to August 2010

DDH	YEAR	Easting (UTM83 Z18)	Northing (UTM83 Z18)	Elevation (m)	Azimuth	Dip	Length
CR-10-371	2010	349346	5330601	3042	202	-42	30
CR-10-372A	2010	349773	5330550	3045	202	-75	280
CR-10-373	2010	349651	5330541	3045	202	-75	239
CR-10-374	2010	349612	5330604	3045	202	-57	240
CR-10-375	2010	349554	5330730	3045	202	-60	325
CR-10-376	2010	349510	5330781	3045	202	-54	356.4
CR-10-377	2010	349395	5330735	3045	202	-60	295
CR-10-378	2010	349791	5330542	3045	202	-75	304
TOTAL Croinor Phase 2							**2,069.4**

10.3.2 Drilling program – 2011

One of the objectives of the 2011 program was to perform step-out testing of the gold mineralization within and to the east of the planned development, identified by earlier drilling. The program was split into three phases: Phase 1 ran from January to March and included holes CR-11-379 to CR-11-401; Phase 2 ran through the month of May and included holes CR-11-402 to CR-11-408; Phase 3 ran through the month of August and included holes CR-11-409 to CR-11-419.

Other drilling objectives included investigating the targets generated by the previous program (including those identified by the 3D hole-to-hole IP survey to the east of the planned development), and testing the host diorite sill to the west of the reserve area where an IP target had been identified at the edge of the survey and beyond the limit of previous drilling. An IP signal in the east was targeted by holes CR-11-384 to -388. An IP signal in the west was targeted by holes CR-11-395 and CR-11-397 to -401.

One (1) hole was drilled to test mineralized veins down dip within the diorite dyke (CR-11-408). To do this, the hole was drilled through the diorite, nearly parallel to it, to a depth of 750 m.

Twenty-four (24) diamond drill holes (NQ size) for a total of 7,198 m were logged and sampled during Phase 1 (Table 10.6). Seven (7) diamond drill holes (NQ size) for a total of 2,764 m were logged and sampled during Phase 2 (Table 10.7). Eleven (11) holes for a total of 3,084 m were logged and sampled during Phase 3. The 2011 program produced 3,149 samples, including certified reference material standards, blanks, and field duplicates

At the time the resource calculation was initiated, some results from Phase 3 were pending. It was therefore decided to exclude the assay results from six (6) drill holes from the resource calculation database. These holes are CR-11-414 to CR-11-419 inclusive. InnovExplo is of the opinion that the results of the excluded holes from the 2011 drilling program do not have a significant impact on the Mineral Resource Estimate. InnovExplo does not recommend a complete update of the Mineral Resource Estimate and prefeasibility study to include the assay results excluded from the calculation.

Table 10.6 – Croinor Phase 1 diamond drill holes executed from January to March 2011

DDH	YEAR	Easting (UTM83 Z18)	Northing (UTM83 Z18)	Elevation (m)	Azimuth	Dip	Length
CR-11-379	2011	350218	5330370	3045	202	-68	345
CR-11-380	2011	350168	5330323	3047	202	-56	267
CR-11-381	2011	350157	5330165	3049	202	-45	81
CR-11-382	2011	350254	5330327	3046	202	-67	309
CR-11-383	2011	350097	5330227	3049	202	-45	120
CR-11-384	2011	349947	5330582	3042	202	-68	435
CR-11-385	2011	349936	5330541	3043	202	-59	369
CR-11-386A	2011	349913	5330489	3044	202	-58	256
CR-11-387	2011	350054	5330550	3043	202	-62	453
CR-11-388	2011	350037	5330507	3044	202	-61	393
CR-11-389	2011	349524	5330813	3044	203.5	-62.4	405
CR-11-390	2011	349949	5330480	3045	200.7	-61	276
CR-11-391	2011	349423	5330751	3041	202	-70	255
CR-11-392	2011	349591	5330767	3045	202	-65	12
CR-11-392A	2011	349591	5330769	3048	202	-65	396
CR-11-393	2011	349627	5330649	3046	202	-70	309
CR-11-394	2011	350317	5330313	3046	202	-67	309
CR-11-395	2011	349120	5330958	3038	202	-70	327
CR-11-396	2011	350256	5330328	3046	202	-57	285
CR-11-397	2011	348979	5330959	3035	202	-69	295
CR-11-398	2011	349085	5330874	3039	202	-68	240
CR-11-399	2011	348957	5330907	3040	202	-68	212
CR-11-400	2011	349137	5331005	3040	202	-72	423
CR-11-401	2011	348991	5330991	3035	202	-79	426
TOTAL Croinor Phase 1							**7,198**

Table 10.7 – Croinor Phase 2 diamond drill holes executed in May 2011

DDH	YEAR	Easting (UTM83 Z18)	Northing (UTM83 Z18)	Elevation (m)	Azimuth	Dip	Length
CR-11-402	2011	349199	5330999	3045	202	-65	399
CR-11-403	2011	349234	5330968	3045	202	-70	414
CR-11-404	2011	349205	5330906	3045	202	-65	345
CR-11-405	2011	349165	5330914	3045	202	-65	300
CR-11-406	2011	349025	5330889	3045	202	-65	235
CR-11-407	2011	349058	5330970	3045	202	-65	321
CR-11-408	2011	349029	5330727	3045	22	-60	751
TOTAL Croinor Phase 2							**2,764**

Table 10.8 – Croinor Phase 3 diamond drill holes executed in August 2011

DDH	YEAR	Easting (UTM83 Z18)	Northing (UTM83 Z18)	Elevation (m)	Azimuth	Dip	Length
CR-11-409	2011	349393	5330683	3042	202	-66	255
CR-11-410	2011	349421	5330668	3043	202	-60	204
CR-11-411	2011	349407	5330686	3043	202	-56	207
CR-11-412	2011	349413	5330727	3041	202	-67	285
CR-11-413	2011	349460	5330728	3043	202	-65	309
CR-11-414	2011	349525	5330793	3046	202	-56	372
CR-11-415	2011	349531	5330696	3045	202	-72	318
CR-11-416	2011	349513	5330707	3044	202	-67	315
CR-11-417	2011	349568	5330761	3048	202	-59	360
CR-11-418	2011	349490	5330650	3044	202	-56	225
CR-11-419	2011	349547	5330629	3045	202	-55	234
TOTAL Croinor Phase 3							**3,084**

All the 2010 holes investigating mineralization west of the West Pit and at depth encountered significant gold grades and widths, as presented in tables 10.9a and 10.9b.

Table 10.9a – Significant intercepts from the 2010 drilling program west of the West Pit

Hole no.	Target Area	From	To (m)	Length	True width	Au g/t
CR-10-365B	W-West Pit	24	24.9	0.9	0.87	1.85
CR-10-366B	W-West Pit	20.8	21.8	1.0	0.98	2.53
	W-West Pit	27	27.6	0.6	0.6	1.52
CR-10-367	W-West Pit	14.8	16	1.2	1.18	2.61
	W-West Pit	16	16.5	0.5	0.49	0.47
CR-10-368	W-West Pit	13.7	14.7	1.0	0.99	7.27
	W-West Pit	17.4	18.8	1.4	1.04	2.90
	W-West Pit	22.9	24	1.1	1.05	0.48
CR-10-369	W-West Pit	15	16	1.0	0.76	3.31
	W-West Pit	21	21.6	0.6	0.46	0.83
CR-10-370	W-West Pit	14.2	17	2.8	2.15	1.55
	W-West Pit	17	18	1.0	0.76	0.48
CR-10-371	W-West Pit	22.9	24	1.1	1.09	7.74
	W-West Pit	27	28.1	1.1	1.1	0.36

Table 10.9b – Significant intercepts from the 2010 drilling program of holes targeting mineralization at depth

Hole No.	Section	Dip	From (m)	To (m)	Length (m)	True Width (m)	g/t Au
CR-10-372A	10E	-75	148.4	151	2.6	2.3	2.98
			150.2	151	0.8	0.7	7.16
CR-10-373	100W	-64	63.8	70.6	7.5	6.4	11.81
			66.8	67.8	1.0	0.9	32.10
			101.5	102.5	1.0	0.9	8.72
			128.9	130.4	1.5	1.3	6.27
			131.9	132.8	0.9	0.75	1.35
			133.35	134.2	0.85	0.7	1.71
CR-10-374	160W	-57	111.7	112	0.3	0.3	2.37
			123.5	124.5	1.0	0.8	12.65
			136	137	1.0	0.8	2.9
			140	141	1.0	0.8	1.47
			176.9	178	1.1	0.9	1.79
			188	189	1.0	0.8	6.71
			192.5	193.5	1.0	0.8	4.86
CR-10-375	260W	-60	224.1	226.5	2.4	2.2	19.94
			232.5	233.3	0.8	0.7	1.57
CR-10-376	320W	-54	226.1	227	0.9	0.9	10.46
			231.6	232.3	0.7	0.7	36.5
			239.7	240.7	1.0	1.0	21.45
			241.7	242.7	1.0	1.0	23.15
			243.7	244.7	1.0	1.0	78.15
			244.7	245.4	0.7	0.7	39.3
			280.6	281.1	0.5	0.5	10.95
CR-10-377	410W	-60	183	184	1.0	1.0	1.75
CR-10-378	30E	-75	145.5	146	0.5	0.5	2.58
			152	153.6	1.6	1.4	1.32

The holes 2011 drilled on Section 750W, 50 m west of planned development in the current ore reserves, included the following mineralized intervals: 21.70 g/t Au over 1.0 m and 28.15 g/t Au over 1.0 m in CR-11-395; 9.62 g/t Au over 2.5 m, including 17.83 g/t Au over 0.8 m in CR-11-398; and 44.04 g/t Au over 0.5 m in CR-11-400. Step-out drilling on Section 880W intersected 4.04 g/t Au over 1.3 m, including 6.88 g/t Au over 0.5 m; results from this section indicate continuity of gold mineralization to the west (Table 10.10).

Assays received for drill holes between sections 200E and 390W, which tested the extensions of ore lenses proximal to the current ore reserves, returned up to 10.11 g/t gold over 0.9 m in CR-11-390 and 6.35 g/t gold over 1.2 m in CR-11-393 (Table 10.10).

Six of the seven holes from Phase 2 were drilled between sections 650W and 810W to test the continuity of significant intersections from Phase 1, particularly those in CR-11-395 on Section 750W. Several intersections at vertical depths between 200 and 285 metres are interpreted as the western extensions of mineralized zones in the current ore reserve (Table 10.11).

Diamond drill holes on Croinor are typically drilled from north to south and oriented to cut the mineralized zones at a high to perpendicular angle. Hole CR-11-408, however, was drilled from south to north to follow the diorite sill at depth to search for new mineralized zones. The hole was drilled to 751 m (660 m vertical). Numerous mineralized zones were encountered throughout the hole from a vertical depth of 51 m through to 654 m. Results from hole CR-11-408 clearly demonstrate that the type and density of Croinor mineralization extends to 654 m and is still open in all directions (Table 10.12).

The drill program was designed to extend the limits of the mineralized lenses in the area of the current ore reserves in order to provide the additional definition required for mine planning. Significant mineralization was found in every hole of the program, ranging from 2.01 g/t Au to 50.76 g/t Au. The best results among the first five holes (CR-11-409 to -413; Table 10.13) were two main mineralized zones from CR-11-413: 15.11 g/t Au over 4.1 m (3.9 m true width) from 186.8 to 190.9 m, and 17.63 g/t Au over 4.3 m (4.2 m true width) from 203.7 to 208.0 m. The results from holes CR-11-416 to -419 (Table 10.13) include the best overall result of the program, intersected in CR-11-416: a true width of 5.19 m grading 5.28 g/t Au from 210.4 to 215.7 m, including 9.28 g/t Au over 2.16 m.

Table 10.10 – Significant intercepts from the 2011 drilling program phase 1

Hole no.	Section	Dip	From (m)	To (m)	Length* (m)	Au g/t
CR-11-379	490E	-68	225.6	226.1	0.5	3.03
			226.8	227.7	0.9	7.03
			229.4	230.0	0.6	3.30
			268.3	268.8	0.5	3.49
CR-11-380	460E	-56	194.1	194.7	0.6	0.57
CR-11-381	510E	-45	61.4	61.9	0.5	5.69
			61.9	62.6	0.7	4.33
CR-11-382	540E	-67	199	199.6	0.6	11.75
CR-11-383	430E	-45	85.9	86.4	0.5	0.14
CR-11-384	160E	-68	237.6	238.5	0.9	3.0
CR-11-385	160E	-59	304	305.3	1.3	1.62
CR-11-386A	160E	-58	110.2	110.7	0.5	0.46
CR-11-387	270E	-62	248.6	249.6	1.0	1.81
CR-11-388	270E	-61	198.9	199.5	0.6	2.13
			199.5	200.2	0.7	8.23
CR-11-389	320W	-62	279.5	280.5	1.0	2.01
CR-11-390	200E	-61	153.8	154.7	0.9	10.11
			175.5	176.6	1.1	3.69
			227.9	228.4	0.5	1.07
CR-11-391	390W	-70	136	136.6	0.6	1.73
			205.3	205.8	0.5	1.38
CR-11-392A	240W	-65	297	299	2.0	4.24
			319	320.3	1.3	1.19
CR-11-393	160W	-70	180.2	181.1	0.9	1.38
			181.1	181.6	0.5	5.73
			245.8	247	1.2	6.35
			247	248.1	1.1	0.69
			248.1	249	0.9	4.0
CR-11-394	509E	-67	205.1	205.9	0.8	4.24
CR-11-395	750W	-70	259.7	260.7	1.0	21.7
			285	285.5	0.5	6.17
			310	311	1.0	28.15
CR-11-396	540E	-57	188.4	189.5	1.1	0.22
CR-11-397	750W	-69	232.6	233.6	1.0	28.15
CR-11-398	750W	-68	158.4	159.2	0.8	1.45
			159.2	159.7	0.5	15.03
			159.7	160.4	0.7	2.73
			160.4	160.9	0.5	0.73
			160.9	161.7	0.8	17.83
CR-11-399	880W	-68	168.4	169.2	0.8	0.79
CR-11-400	880W	-72	263.5	264	0.5	44.04
			314.8	315.5	0.7	1.94
			339.2	340.3	1.1	2.79
			359.8	360.9	1.1	3.87
CR-11-401	880W	-79	303.6	304.1	0.5	6.88
			304.1	304.9	0.8	2.27
			321.7	322.7	1.0	1.89
			335.3	335.8	0.5	1.85
			340.3	342.9	2.6	1.58

Table 10.11 – Significant intercepts from the 2011 drilling program western extention

Hole no.	Section	Dip	From (m)	To	Length*	Au
CR-11-402	690W	65	333.2	333.7	0.5	5.26
			346.3	347.3	1.0	0.54
			352.9	353.8	0.9	2.64
CR-11-403	650W	-70	318.4	318.9	0.5	1.78
			320.9	322.9	2.0	6.66
			321.9	322.9	1.0	11.94
			359.7	360.7	1.0	5.74
			359.7	360.2	0.5	11.2
			360.7	361.2	0.5	1.48
			387.5	388.0	0.5	1.9
			389.5	390.7	1.2	6.66
			390.0	390.7	0.7	11.4
CR-11-404	650W	-65	234.3	239.3	5.0	3.74
			234.3	235.0	0.7	17.15
			240.8	241.3	0.5	1.04
			242.8	243.3	0.5	2.59
			262.5	264.5	2.0	3.92
			262.5	263.0	0.5	6.77
			254.0	264.5	0.5	8.04
CR-11-405	690W	-65	235.6	236.6	1.0	1.14
			236.6	238.2	1.6	11.15
			236.6	237.6	1.0	14.59
			241.5	243.1	1.6	3.18
			241.5	242.0	0.5	6.85
			251.9	254.5	2.6	4.88
			252.4	252.9	0.5	21.05
CR-11-406	810W	-65	146.6	147.2	0.6	0.67
			150.8	151.6	0.8	0.08
			183.5	184.1	0.6	3.19
CR-11-407	810W	-65	252.0	252.6	0.6	7.2
			265.2	266.1	0.9	1.89

Table 10.12 – Significant intercepts from the 2011 drilling program for Drill hole parallel to diorite

Hole no.	Section	Dip	From (m)	To	Length*	Au
CR-11-408	750W	-60	58	59	1.0	1.52
			90	96.6	6.6	0.89
			90	90.6	0.6	4.5
			108	109.5	1.5	7.57
			108	109	1.0	9.7
			121.4	121.9	0.5	2.62
			141.5	142.1	0.6	1.43
			147.9	161.7	13.8	1.01
			148.4	149.4	1.0	3.54
			158.7	159.4	0.7	2.24
			213.8	215.0	1.2	5.64
			228.0	230.2	2.2	1.38
			229.4	230.2	0.8	2.54
			290.7	291.4	0.7	1.4
			345.8	350.1	4.3	2.65
			345.8	347.1	1.3	6.21
			346.4	347.1	0.7	11.03
			355.4	355.9	0.5	1.27
			357.4	366.0	8.6	2.48
			357.4	359.6	2.2	5.55
			362.2	363.2	1.0	5.64
			365.1	366.0	0.9	2.81
			372.4	375.8	3.4	1.29
			372.4	373.3	0.9	2.64
			404.9	405.4	0.5	1.4
			552.2	554.4	2.2	2.57
			553.8	554.4	0.6	7.48
			683.2	683.7	0.5	1.95
			692.2	692.7	0.5	2.45
			707.3	713.0	5.7	0.6
			710.2	710.8	0.6	2.52
			732.8	733.5	0.7	0.5
			741.1	742.3	1.2	0.5
			743.6	744.1	0.5	0.84

* Based on the current interpretation, holes CR-11-402 to CR-11-407 were drilled approximately perpendicular to the mineralized zones, and core length intervals in these holes are considered to be approximate true widths. True widths of the mineralized zones in CR-11-408 could not be determined due to insufficient information.

Table 10.13 – Significant intercepts from the 2011 drilling program concentrated on known ore reserves

Hole no.	Section	Dip	From (m)	To	Length	True Width	Au
CR-11-409	390W	-66	155.5	157.0	1.5	1.41	6.63
			162.2	162.9	0.7	0.68	3.78
			166.4	170.1	3.7	3.58	10.92
			168.8	170.1	1.3	1.26	19.24
			169.6	170.1	0.5	0.48	20.7
			176.0	177.0	1.0	1.00	5.64
CR-11-410	360W	-60	143.1	145.3	2.2	2.13	2.87
			147.6	148.2	0.6	0.58	2.28
			186.2	186.8	0.6	0.58	3.22
CR-11-411	380W	-56	159.8	160.3	0.5	0.50	2.01
			171.9	172.5	0.6	0.59	4.56
CR-11-412	390W	-67	147.2	148.0	0.8	0.75	15.08
CR-11-413	350W	-65	186.8	190.9	4.1	3.91	15.11
			186.8	187.8	1.0	0.95	27.61
			187.3	187.8	0.5	0.48	38.92
			190.4	190.9	0.5	0.48	50.76
			198.1	198.8	0.7	0.67	4.98
			203.7	209.7	6.0	5.87	13.12
			203.7	208.0	4.3	4.20	17.63
			206.1	207.3	1.2	1.17	32.38
			242.7	243.3	0.6	0.58	4.97
CR-11-414	31-W	-56	204	241.6	1.6	1.08	4.41
CR-11-415	270-W	-72	207.3	207.8	0.5	0.47	10.68
			211.5	212.1	0.6	0.58	3.41
			213.3	214.9	1.6	1.54	2.8
			214.4	214.9	0.5	0.48	6.19
			216.5	217	0.5	0.48	1.1
CR-11-416	290-W	-67	210.4	215.7	5.3	5.19	5.28
			213.5	215.7	2.2	2.16	9.28
			215.2	215.7	0.5	0.49	20.41
CR-11-417	260-W	-59	231.3	232.3	1.0	1.0	4.09
			249.7	250.2	0.5	0.5	1.48
			255	260	5.0	5.0	2.19
			259	260	1.0	1.0	6.39
			283.1	283.6	0.5	0.5	7.3
			287.1	287.7	0.6	0.6	1.12
CR-11-418	290-W	-56	137.3	138	0.7	0.7	3.85
			162	163.9	1.9	1.9	4.28
CR-11-419	240-W	-55	102	102.5	0.5	0.5	1.92
			105	105.8	0.8	0.8	4.08
			129.6	130.1	0.5	0.5	4.46
			134.4	135	0.6	0.6	1.75
			180.3	181.7	1.4	1.4	4.27
			181.2	181.7	0.5	0.5	9.06

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This section presents the sampling method and approach adopted during previous work programs by Exploration Malartic-Sud Inc from 1998 to 2005 (Pelletier and Boudrias, 2005), X-Ore Resources Inc and First Gold Exploration Inc from 2007 to 2008 (O'Dowd, 2009), and Blue Note Mining Inc in 2010 and 2011.

11.1 Exploration Malartic-Sud (1998-2005)

11.1.1 Drill core sampling

Samples were taken from the core after it was measured and described (logged). Samples were typically from alteration zones, quartz veins, shear zones and pyritic intervals.

According to Malartic-Sud, there was no written protocol for the sampling procedure but they did sample according to prevailing industry standards, using a maximum sample length of 1.50 m and a minimum of 0.5 m for any BQ or NQ core. The maximum length was often shortened to 1.00 m in mineralized zones for NQ core, and the minimum length occasionally reduced to less than 50 cm (i.e., 25–30 cm) if warranted (K. Marchand, written communication).

InnovExplo verified the sample lengths in Malartic-Sud's computerized database. Drill core sample lengths varied from 0.3 m to 2 m, with an average of about 1.3 m.

Samples were marked on the core by the geologist, and the core was split in half by a technician using a hydraulic splitter, returning half the core into the box as reference material. The other half was placed in a thick plastic sample bag with a numbered tag, and a tag bearing the same number was placed in the core box with the reference half-core. The splitter and sampling box were cleaned between each sample preparation to prevent contamination. Samples were then prepared for shipment along with a shipping list documenting the samples (Chénard and Turcotte, 2003). Malartic-Sud's technician delivered the samples to the Chimitec-ALS Chemex laboratory in Val-d'Or.

Strongly mineralized samples (those with quartz veining) were assayed using a coarse gold separation method to counteract the free gold effect. The remainder were assayed using the fire assay method with atomic absorption finish on 30-g fractions. Results grading over 1 g/t Au were systematically re-assayed using fire assay with gravimetric finish.

11.1.2 Blast hole sampling

InnovExplo is of the opinion that the sampling procedure adopted by Malartic-Sud at the beginning of open pit operations in 2003 was inadequate. The procedure, which lasted until InnovExplo established a new protocol in September 2004, involved systematic sampling every 2.5 m without taking into account the starting elevation. This caused a non-uniform horizontal distribution of the grades (upper part of Fig. 11.1). Consequently, the grade established for a given bench was not necessarily representative of the material ultimately extracted.

When InnovExplo came into the project in September 2004, the protocol was modified so that each bench was sampled in a truly representative manner (lower part of Fig. 11.1) according to the following instructions:

1. The first sample should end at the 1st bench elevation encountered. If the length is less than 1 m, the sample is added to the 2nd bench sample;
2. No samples are to be taken from the sub-drilling (that is, from the underlying bench);
3. Sample tickets must be placed in the bags;
4. Sample books must be legible;
5. Samples must be collected from the pile of cuttings from the drilling waste using a sampling shovel.
6. Sampling is performed on a systematic grid of 2.5 m by 2.5 m.

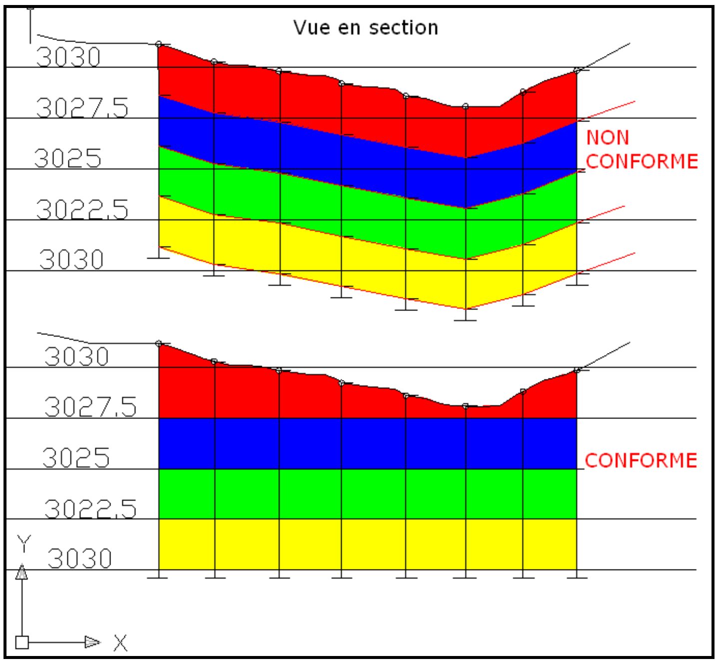


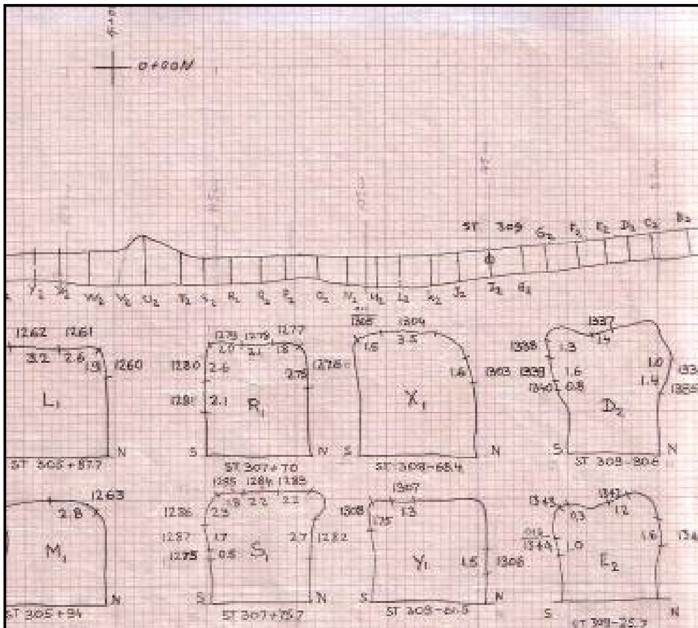
Figure 11.1 – Blast hole sampling. Top: non-uniform horizontal grade distribution caused by the pre-September 2004 sampling protocol; Bottom: InnovExplo's 2004 sampling protocol provides representative grades for a given bench.

11.1.3 Channel sampling on outcrops

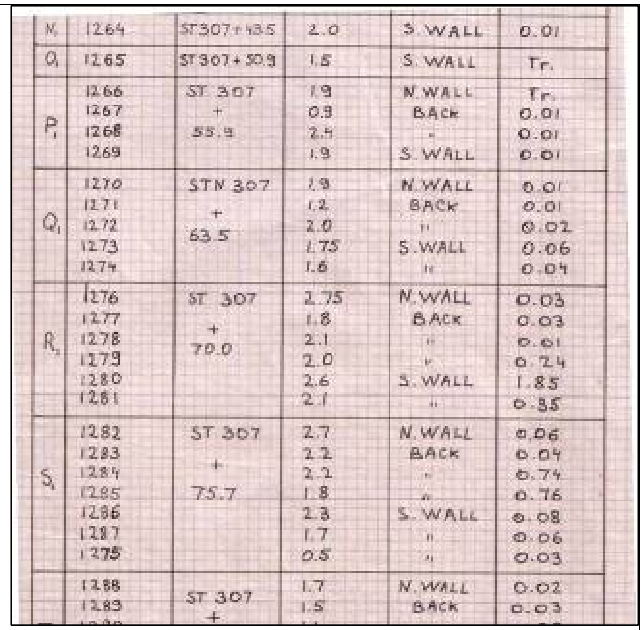
Channel samples were taken with a diamond saw. The channels were cut perpendicularly to the mineralized zone with the following dimensions: 1 m long, 5 to 10 cm wide, and 2.5 to 3 cm deep.

11.1.4 Chip sampling

Croinor Pershing Mines sampled underground drifts in the 1940s by chip sampling the walls and backs of the drifts (Fig. 11.2). Chip samples were systematically taken after each blasting sequence as drifting progressed (Saucier, 2003). The drift sampling was completed by channelling every two (2) metres, on average, between samples. The average length of individual samples was 0.5 m. The walls and back were sampled when sulphide mineralization or quartz veins were present.



Drifts and sections



N_1	1264	ST307+43.5	2.0	S. WALL	0.01
O_1	1265	ST307+50.9	1.5	S. WALL	Tr.
P_1	1266	ST 307	1.9	N. WALL	Tr.
	1267	+	0.9	BACK	0.01
	1268	55.9	2.4	"	0.01
	1269		1.9	S. WALL	0.01
Q_1	1270	STN 307	1.9	N. WALL	0.01
	1271	+	1.2	BACK	0.01
	1272	63.5	2.0	"	0.02
	1273		1.75	S. WALL	0.06
	1274		1.6	"	0.04
R_1	1276	ST 307	2.75	N. WALL	0.03
	1277	+	1.8	BACK	0.03
	1278	70.0	2.1	"	0.01
	1279		2.0	"	0.24
	1280		2.6	S. WALL	1.85
	1281		2.1	"	0.35
S_1	1282	ST 307	2.7	N. WALL	0.06
	1283	+	2.2	BACK	0.04
	1284		2.2	"	0.74
	1285	75.7	1.8	"	0.76
	1286		2.3	S. WALL	0.08
	1287		1.7	"	0.06
	1275		0.5	"	0.03
	1288	ST 307	1.7	N. WALL	0.02
	1289	+	1.5	BACK	0.03

Channel sampling results

Figure 11.2 – Underground channelling, west drift, level 375. (From Malartic-Sud archives)

11.2 First Gold Exploration and X-Ore Resources (2007-2008)

Core boxes were collected at the drill site every morning by the project geologist or the assistant. Core boxes were opened at the company core shed and eventually labelled according to hole number and depth of the interval in the box (e.g.: CR-08-348 141 to 145.5).

The core was then reviewed and described by the geologist, who would delineate intervals to be sampled with red marks. Sample information was listed in the sample book and in the geologist's log (date, interval sampled). The geologist determined the size of the interval based on geological parameters such as geological contact, alteration, mineralization, etc. Samples rarely exceeded 1.5 m and were usually a minimum of 30 cm. Most of the 2007 and 2008 samplings were done using 1.0-metre intervals. The procedure was identical for BQ or NQ size core.

Selected intervals for assaying were split in two using a manual or hydraulic core splitter. One half of the interval was placed with a numbered tag in a plastic bag provided by the laboratory. The other half was put back in its original location in the core box with a second tag (same number) to identify the interval for future reference. The sample bag was sealed and readied for shipping to the laboratory.

The core splitter was thoroughly cleaned between every sample to avoid contamination. Once all intervals had been collected in a box, it was piled outside the company core shed and strapped when piles reach 1.5 m high. Samples were brought or shipped to the laboratory at fairly regular intervals determined by volume (every week or every few days).

Only company employees were allowed to handle the samples before they reached the laboratory of the shipping company.

Appendix III provides a description (in French only) of the laboratory sample preparation and analytical procedures in 2007 and 2008. First Gold used two accredited independent laboratories: Laboratoire Expert in Rouyn-Noranda and Technilab in Sainte-Germaine Boulé.

11.3 Blue Note Mining (2010-2011)

InnovExplo was responsible for establishing the sample preparation, analysis and security protocols for the drilling conducted by Blue Note Mining in 2010-2011. InnovExplo's sampling protocol specifies that samples should be half-split core measuring 0.5 to 1.5 m long. The length is based on geological criteria. If samples require re-analysis for verification purposes, a quarter-split of the remaining sample may be taken for that purpose. Every zone containing visible pyrite and/or quartz vein mineralization is to be considered potentially mineralized and is systematically sampled.

A core shack facility, equipped with a rock saw in a partitioned housing area, was established at the InnovExplo office in Val-d'Or for the duration of all drilling programs. Diamond drill hole core at Croinor is regularly intact with little possibility of loss due to wash out and is considered to be of good quality. The core is rarely ground, and where this occurs, it is only over short distances (less than 0.8 m). Overall, the drill core recovered from a mineralized zone can be considered representative of that zone.

According to InnovExplo's protocol, drill core is boxed, covered and sealed at the drill rig and moved to the logging and sample preparation facility by the drillers or InnovExplo personnel. After being examined and logged (described), intervals to be sampled are delineated. The core of the selected section is first cut in half using a typical table-feed circular rock saw, with one half placed in a plastic sample bag with a numbered tag for eventual shipment to the laboratory. The second half of the core is put back in the core box, and a tag bearing the same number is placed at the end of the sawed core half. All drill core since 2010 is stored and categorized for future reference purposes at the Blue Note Mining storage facility in Val-d'Or (200-1745 Chemin Sullivan, Val-d'Or, Québec).

Assays were performed at the independent and accredited laboratories of ALS-Chemex in Val-d'Or, Québec, or Timmins, Ontario for phases 1 and 2 in 2010 and phases 1 to 2 in 2011. Phase 3 assays of 2011 were performed at the independent and accredited Techni-Lab laboratories in Sainte-Germaine Boulé, Québec. There is no indication of anything in the drilling, core handling, or sampling procedures that could have had a negative impact on the reliability of the reported assay results.

Once the samples arrive at the ALS-Chemex laboratories, they are dried then crushed (jaw crusher) to 90% passing 10 mesh (2 mm). Samples are then riffle-split (Jones riffle splitters) to reduce the sample size for pulverization to a maximum of 1 kg. The 1-kg subsamples are then pulverized (ring and puck) to 90% passing 200 mesh (75µm). A 50-g split is taken from each pulp for fusion. Analytical protocols require all samples be finished using acid digestion-AAS finish (Au-AA26). All results with initial AAS results greater than 3 g/t Au were re-assayed from the same pulp using gravimetric finish (Au-GRA22).

In addition to regular sampling and assaying, additional quality control protocols (initiated externally by InnovExplo) require the preparation of various duplicate samples to estimate the precision (i.e., reproducibility) and accuracy (i.e., correctness) of the reported values. Each batch of 25 samples included a certified standard, a blank, and a field duplicate. Field duplicates were selected randomly and each batch of 25 samples should contain a duplicate from a previous batch of 25. The data was crosschecked once the batch results were complete. Internally, ALS Chemex also conducts quality control protocols by systematically conducting a duplicate for every 25th sample within the batch sent to the lab.This sample is not a regular field duplicate, certified standard or blank.

The laboratory delivered results in electronic format through the ALS Chemex Webtrieve^{TM} system via the Internet, as well as by e-mail sent to various recipients at InnovExplo. Assay results were reported in grams per tonne (g/t) and transferred directly into the centralized assay database (available in two formats: GeoticLog and Gemcom).

Details of the analytical procedure at Techni-Lab are contained in Appendix III (French version only). The procedures are very similar to those of ALS-Chemex. The only difference is that during the fusion process, the batches were separated into two sub-batches at the laboratory during the fusion process, one containing 19 samples plus standards, and the second containing the other 6 samples along with samples from the next batch from the same project. Techni-Lab inserts into each batch 1 blank and 2 standards, and processes 1 duplicate taken at random from the samples within the batch. The laboratory delivers results via e-mail sent to various recipients from InnovExplo and Blue-Note Mining. Assays were reported in grams per tonne (g/t) and transferred directly into the centralized database. Techni-Lab has a policy that it does not routinely re-analyze standards using Au-GRA22 if the values are above 3 g/t Au. InnovExplo had to request that the standards be re-analyzed by this method. There are therefore some standards in the database that were not checked using Au-GRA22.

12.0 DATA VERIFICATION

A QA/QC program was implemented at the beginning of the drilling program in 2010 and was followed throughout the 2010-2011 drilling programs. In addition to the various and systematic checks made by the laboratories (use of standards and duplicates), seven (7) different gold standards and one (1) blank were used. In this section, only assay results from holes included in the Mineral Resource Estimate are considered; information for holes CR-11-414 to CR-11-419 inclusive is therefore excluded.

One standard, one blank and one duplicate field sample were randomly introduced among every 25 samples on average (total of 3,361 samples, including 280 blanks and gold standards). In all, three types of duplicates were used in this program: field, coarse and pulp.

All samples returning values above 3 g/t Au are systematically re-assayed from the same pulp using gravimetric finish (Au-GRA22). The only exception was for standards with >3 g/t Au, which are not re-assayed according to TechniLab's in-house policy. Figure 12.1 shows the graph of Au-GRA22 values versus the original values. The correlation coefficient for these results is 0.98. A total of 227 control samples were assayed for gold, which includes the 130 laboratory duplicates but excludes any re-assays of high-grade samples. Metallic sieve assaying was used on samples exhibiting coarse gold.

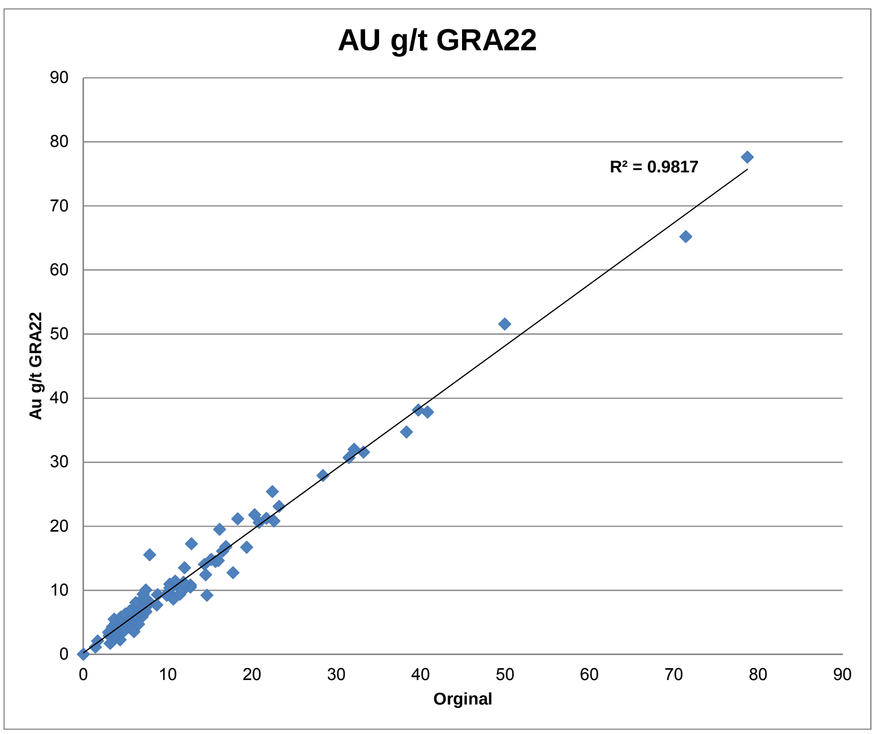


Figure 12.1 – Comparison of Au-GRA 22 re-assays to original values for samples from the 2010-2011 programs

12.1 Duplicates

Field Duplicate: A field duplicate is prepared for one (1) sample selected at random from each field sub-batch (with some bias to ensure that results are included from all grade ranges), and included as a regular sample, blind to the laboratory. The samples to be analyzed are provided from half of the half-split core; that is, from a quarter-split of the original whole core. InnovExplo inserted a total of 131 field duplicates among the samples from the 2010 and 2011 programs.

The results for field duplicates can be used to determine total precision (i.e., reproducibility) of the sample analysis process, from sampling through to sample preparation. When used in conjunction with other sample preparation duplicates, the incremental loss of precision can be determined for each of the various stages of the sampling, preparation and assay process. For the field duplicate increment, this can indicate whether loss of precision can be attributed to the initial sample size.

As seen on Figure 12.2 A and B, the correlation between the two populations is good (80 to 82%). The field duplicates contain two samples that may be the result of a nugget effect and are shown in the first graph (Fig. 12.2 A) in relation to the rest of the samples. No re-analysis was performed since inherent sample heterogeneity or the nugget effect may strongly affect these gold values.

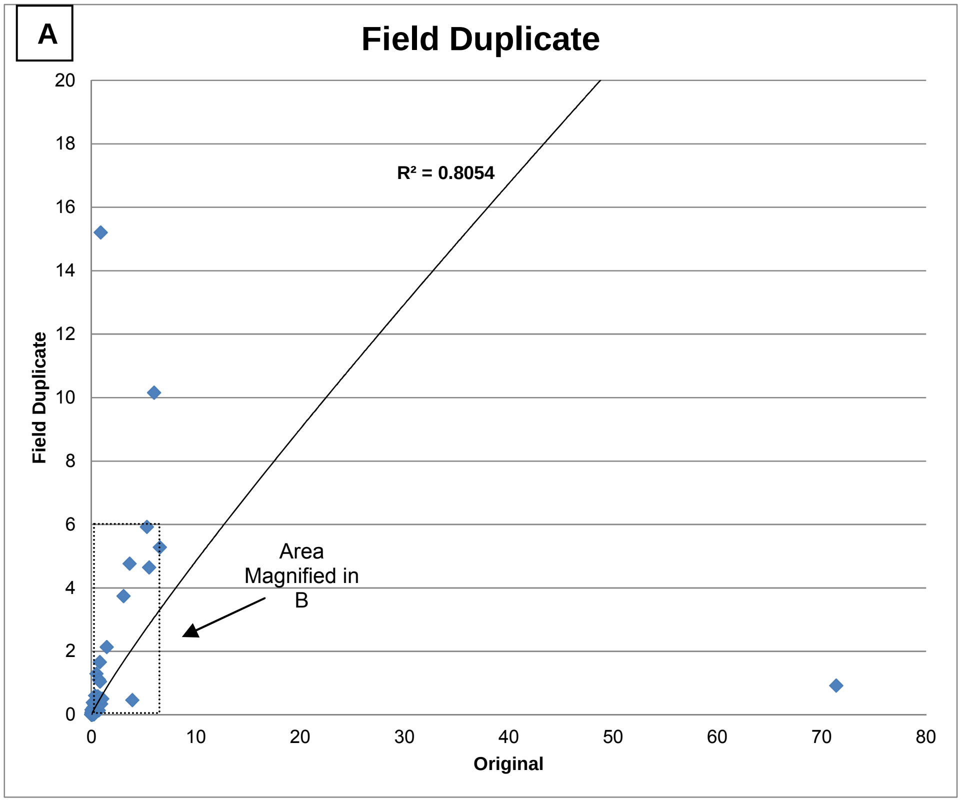
Coarse Crush Duplicate: ALS-Chemex was instructed to prepare a coarse crush duplicate for the 25th sample in each batch. Coarse crush duplicates follow the same sample preparation and assay procedures as regular samples. Techni-Lab did not prepare any coarse duplicates.

The coarse duplicate sample (1,000g) is taken after the primary crushing stage, before proceeding with the regular samples. At 0.98, correlation tables for these duplicates are shown in Figure 12.3. The correlation between the two populations is good, suggesting good reproducibility in the compared samples.

By measuring the precision of the coarse crush duplicates, the incremental loss of precision can be determined for the coarse crush stage of the process, thereby indicating whether a sub-sample size of 1,000 g taken after primary crushing is adequate to ensure a representative sub-split

Pulp duplicate: Both laboratories were instructed to assay a pulp duplicate prepared from one (1) sample selected at random from each sub-batch of samples. Techni-Lab selected at random up to two duplicates within a batch of 25 samples.

By measuring the precision of the pulp duplicates, the incremental loss of precision can be determined for the pulp pulverizing state of the process, thus indicating whether a pulp size of 50g taken after pulverization is adequate to ensure representative fusing and analysis. The pulp duplicates (Fig. 12.4) show a linear correlation of 0.97.



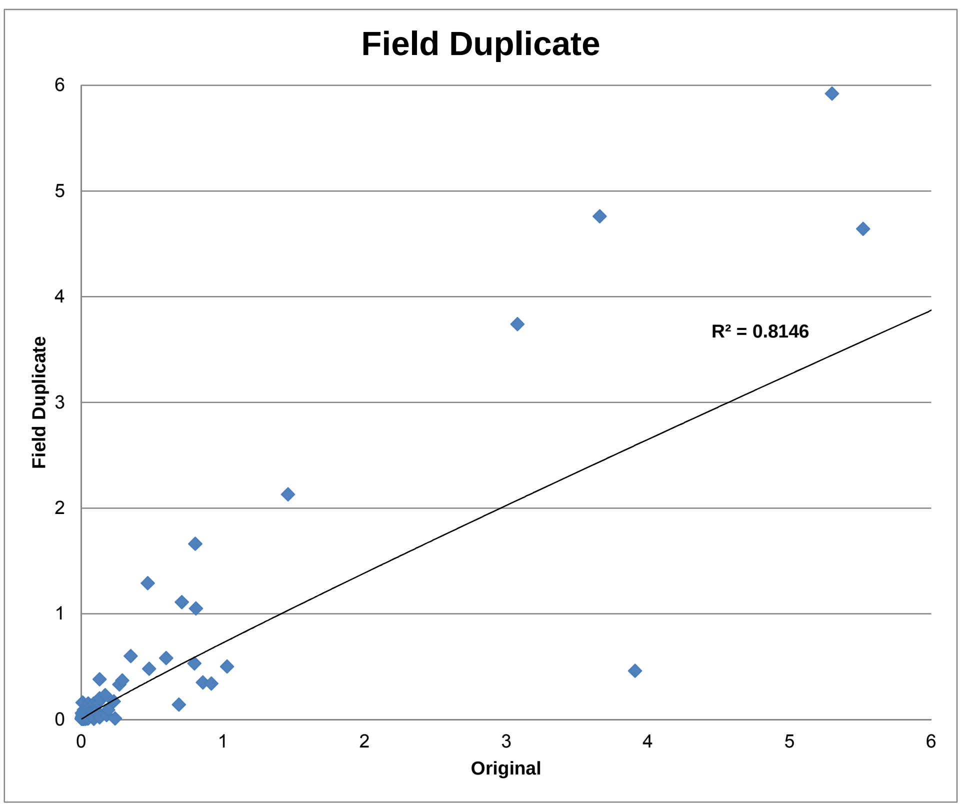


Figure 12.2 –Field duplicates from 2010-2011

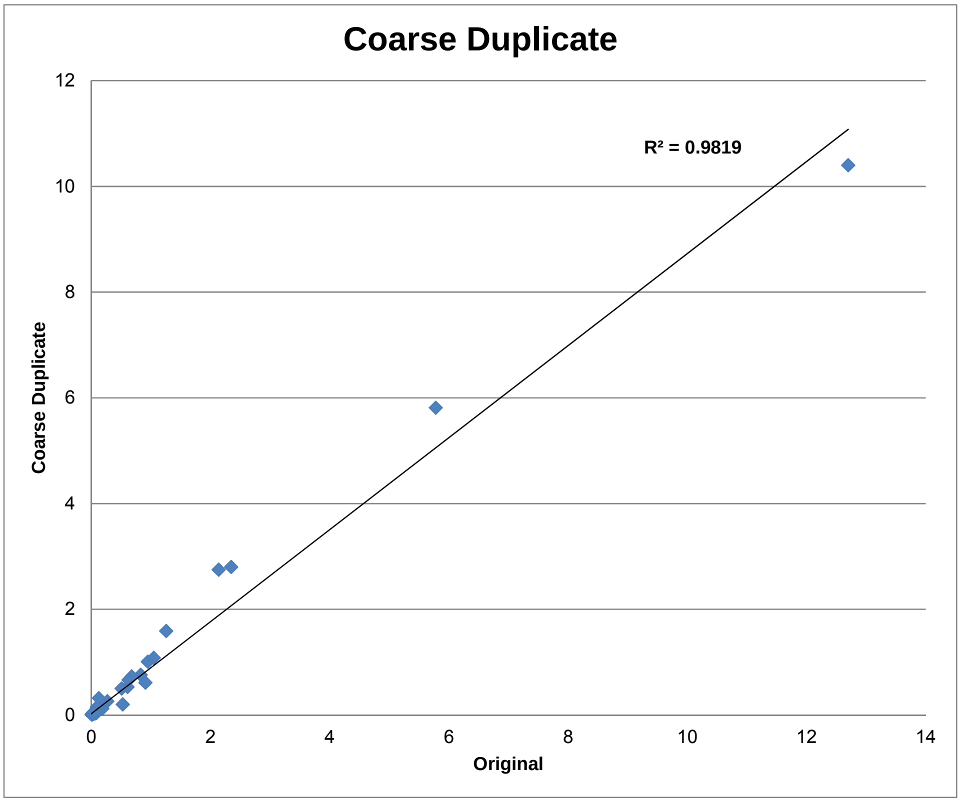


Figure 12.3 –Coarse duplicates from 2010-2011

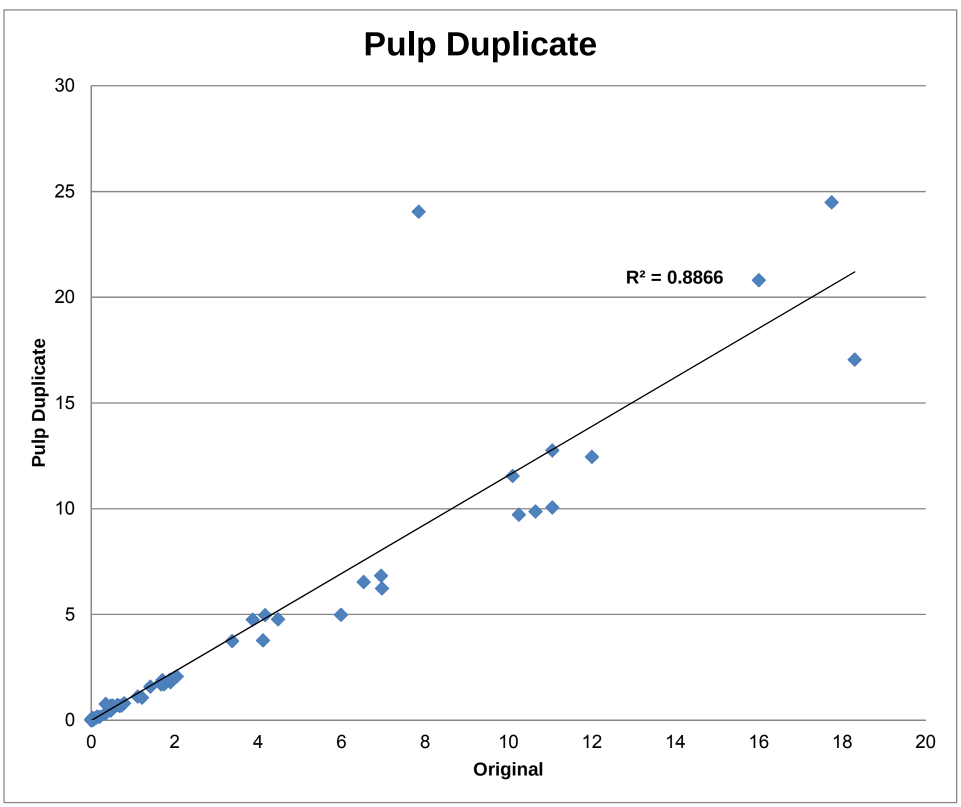


Figure 12.4 – Pulp duplicates from 2010-2011

12.2 Blanks

A total of 141 blank samples were inserted with the regular samples in the 2010 to 2011 programs (Fig 12.5). Both labs had the same detection limit of 0.01 g/t Au. To express "<0.01 g/t Au" numerically, a value of 0.005 g/t Au (half of the detection limit) was assigned, regardless of the laboratory. InnovExplo's recommended quality control protocol stipulates that if any blank yields a gold value above 0.1 g/t Au, the entire batch should be re-assayed..

The average of all blanks is 0.007 g/t Au. This average is well within acceptable ranges given the high-grade nature of the mineralized intervals on the project. There are three outliers, two with values of 0.03 g/t Au and one with 0.07 g/t Au. The value of 0.07 g/t follows an interval of high-grade samples and likely reflects some level of contamination in the equipment used at the lab. The contamination level was considered to be acceptable since the high-grade samples in this batch were re-analyzed using the gravimetric finish and the results were comparable.

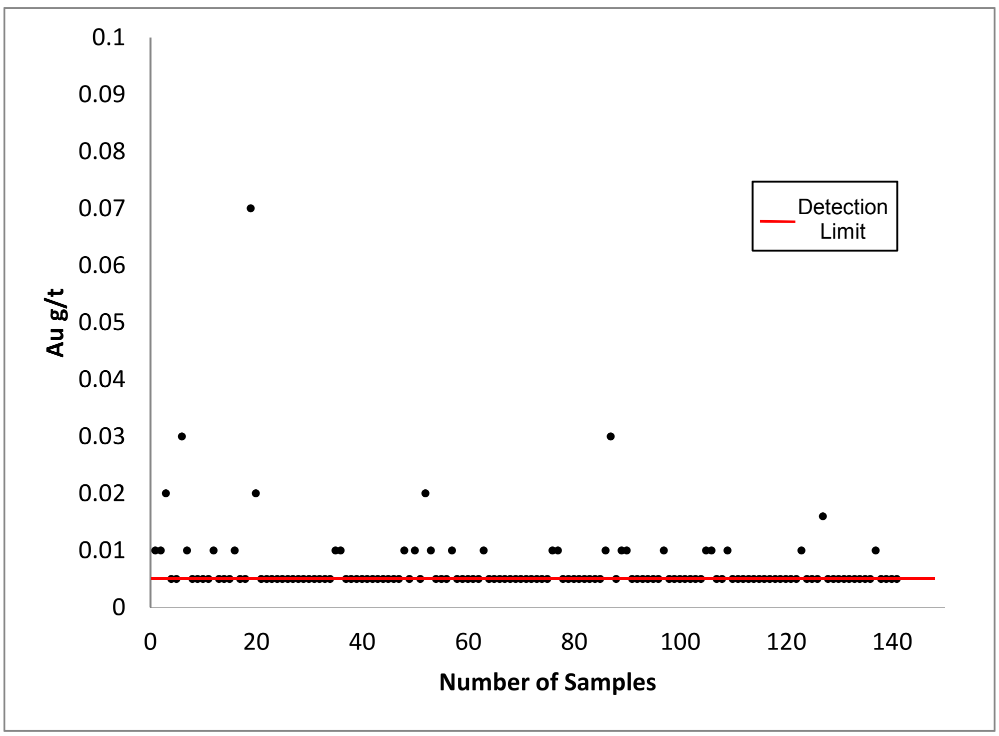


Figure 12.5 – Blank samples from 2010-2011

(Note: Au analyses below the detection limit are shown as 0.005 g/t, representing half the detection limit)

12.3 Gold Standards

For the 2010 and 2011 drill programs, a total of 7 standards were used as control samples (Figs. 12.6a to 12.6k).

The first standard has a certified grade of 0.976 g/t Au. The average of the 35 results is 0.95 g/t Au. There is one value that is low (0.63 g/t), but it does fall within acceptable limits. Removal of this outlier brings the average up to 0.96 g/t Au, and hence acceptable. The standard deviation is 4.9%, including the outlier.

The second standard has a certified grade of 1.344 g/t Au. Only 2 samples of this standard were analyzed, and the average is 1.32 g/t Au. The standard deviation is 0.7%. The low number of results should be taken into account.

The third standard has a certified value of 5.909 g/t Au. The average of 14 results is 5.73 g/t Au. The standard deviation is 8.5%; the low number of results is a contributing factor to this poor value (see above). The variation of the values does fall within acceptable limits. The average from the gravimetric analyses for 7 results is 5.92 g/t Au, with a standard deviation of 4.3%.

The fourth standard has a certified value of 8.685 g/t Au. The average of 30 results is 8.55 g/t Au. The standard deviation is 4.1%. There is one value with a low value of 7.38 g/t Au, but it does fall within acceptable limits. Removal of this sample gives an average of 8.59 g/t Au, with a standard deviation of 3.4%. One result was <0.01 g/t Au, which is well outside acceptable limits. The batch of samples within this batch was reanalyzed and the first results deleted from the database.

The fifth standard has a certified value of 8.573 g/t Au. The average of the 4 results is 8.01 g/t Au with a standard deviation of 1.9%. The gravimetric analyses average is higher, at 8.53 g/t Au, with a standard deviation of 1.7%.

The sixth standard has a certified grade of 18.14 g/t Au. The average of the 41 results is 18.02 g/t Au. There are two outliers at 20.3 g/t and 14.45 g/t Au, which fall within acceptable limits. The standard deviation is 4.2%. Gravimetric analyses were carried out on 35 samples and the average is 17.89 g/t Au. The standard deviation is 3.3%. There is one outlier with a value of 14.35 g/t Au. This outlier falls within acceptable limits.

The seventh standard has a certified grade of 30.04 g/t Au. The average for the 7 results is 29.38 g/t Au. The standard deviation is 8.0%. There is one value at 24.68 g/t, and removal of this value gives an average of 30.05 g/t Au, a difference of 2% and a standard deviation of 3.4%. The results using the gravimetric finish show an average of 28.36% with a standard deviation of 7.0%. A low value of 25.4 g/t Au influences this value. When this low value is excluded, the average is 29.10 g/t Au with a relative standard deviation of 4.3%. The low number of results is partly responsible for these low values, however the values do fall within acceptable limits.

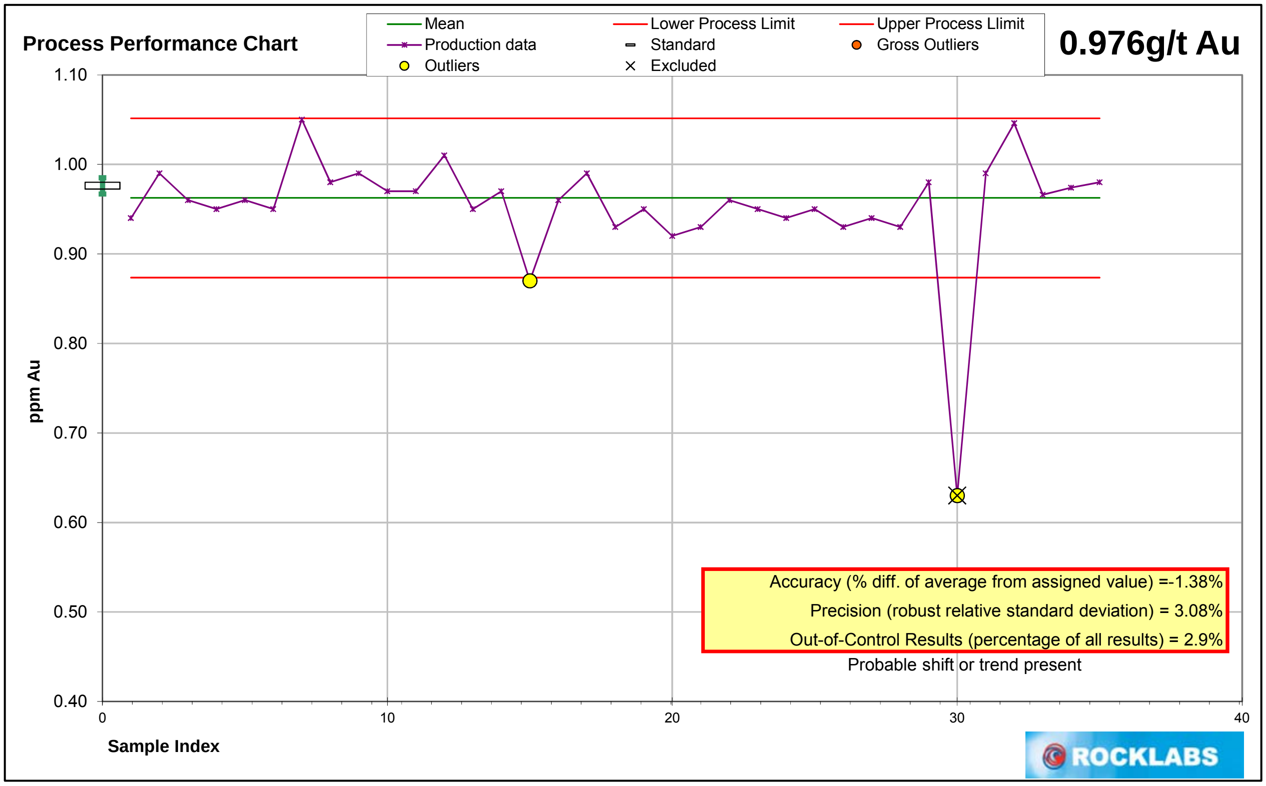


Figure 12.6a – Gold standard from 2010-2011 (0.976 g/t Au)

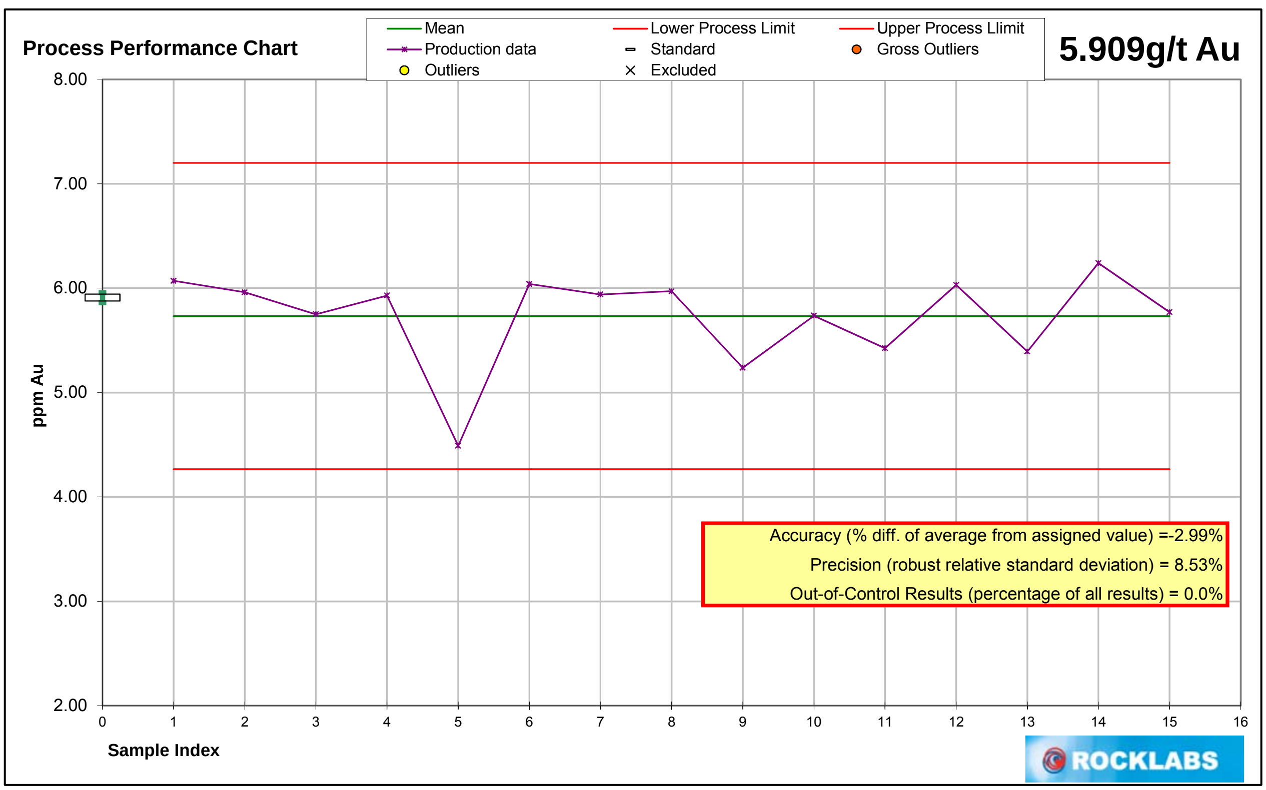


Figure 12.6b – Gold standard from 2010-2011 (5.909 g/t Au)

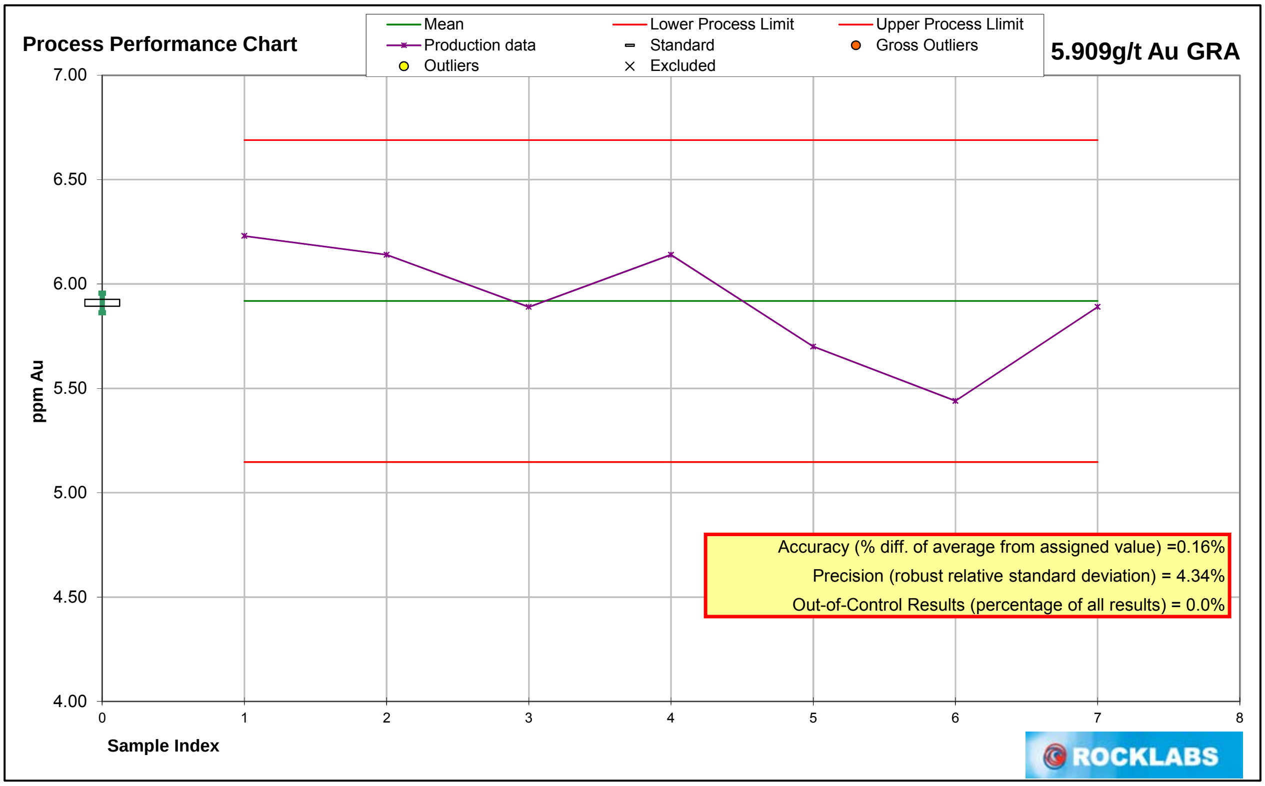


Figure 12.6c – Gold standard from 2010-2011 (5.909 g/t Au using gravimetric finish)

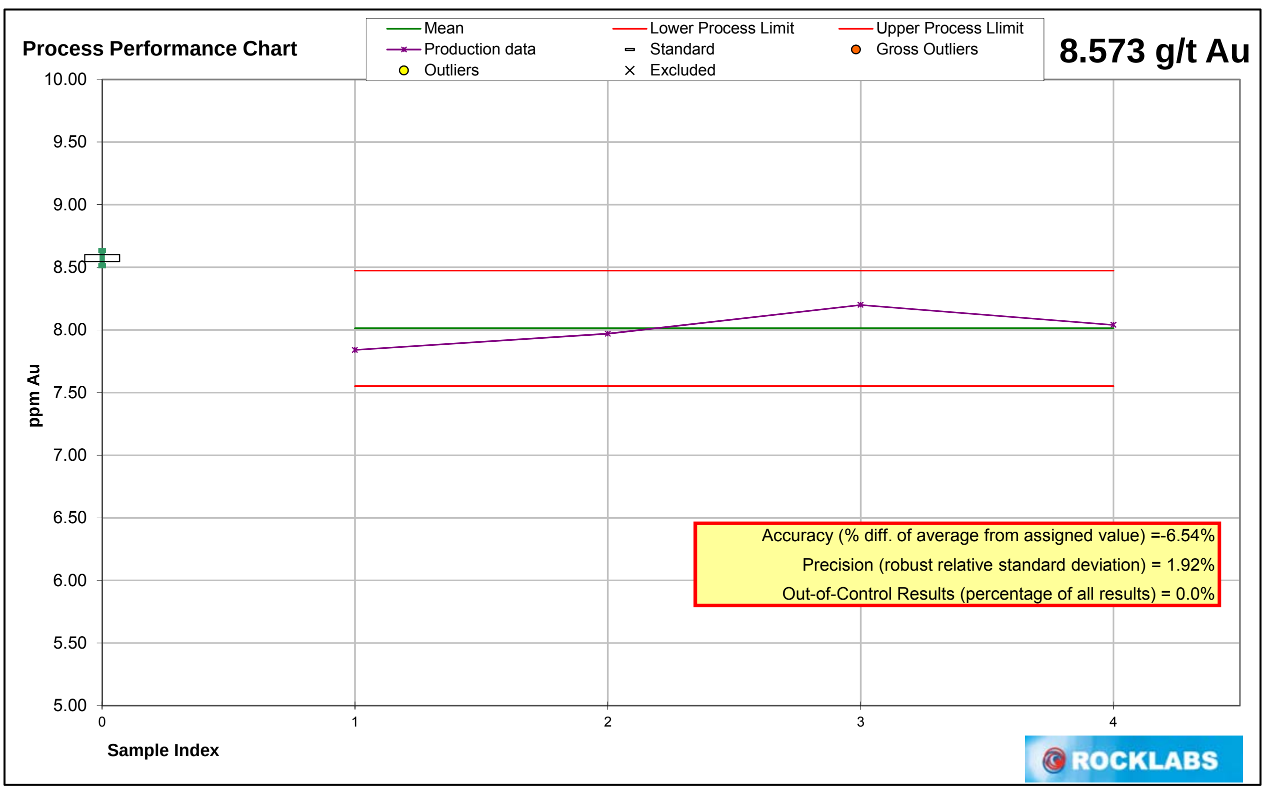


Figure 12.6d – Gold standard from 2010-2011 (8.573 g/t Au)

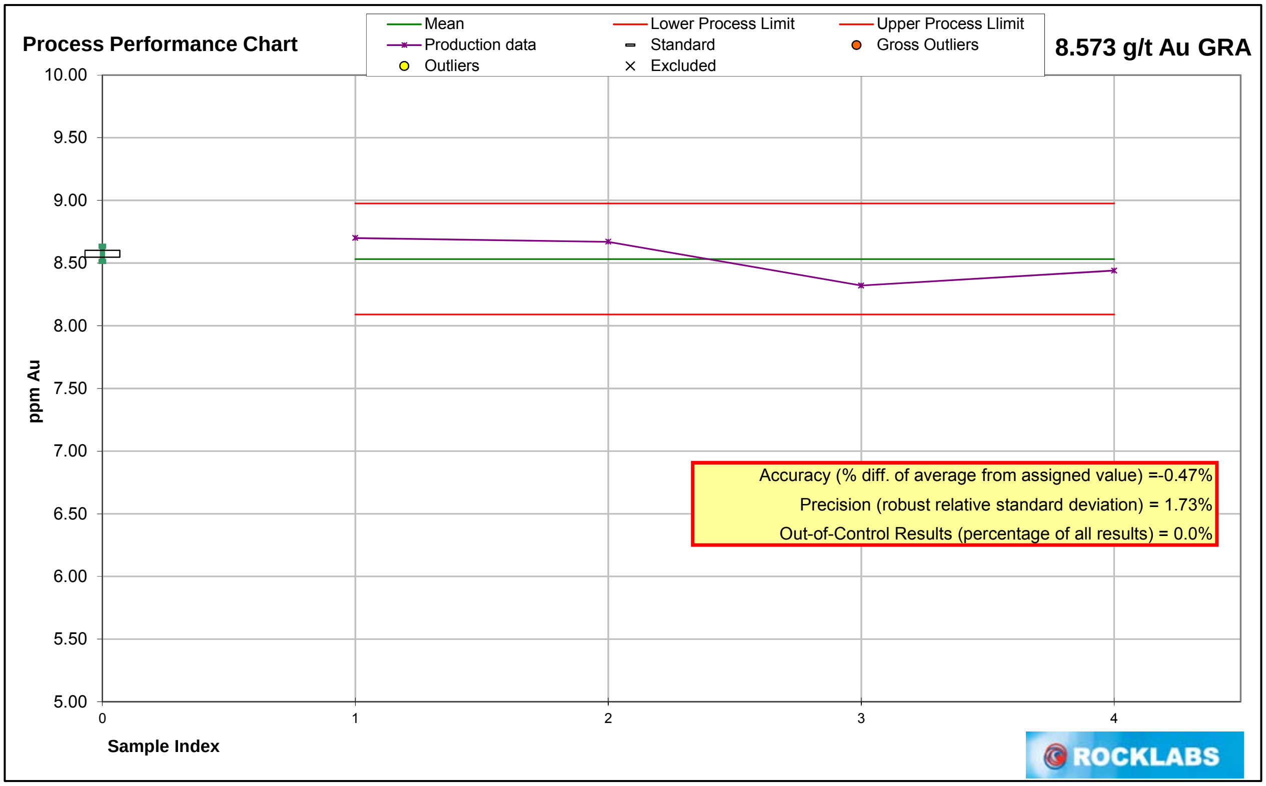


Figure 12.6e – Gold standard from 2010-2011 (8.573 g/t Au using gravimetric finish)

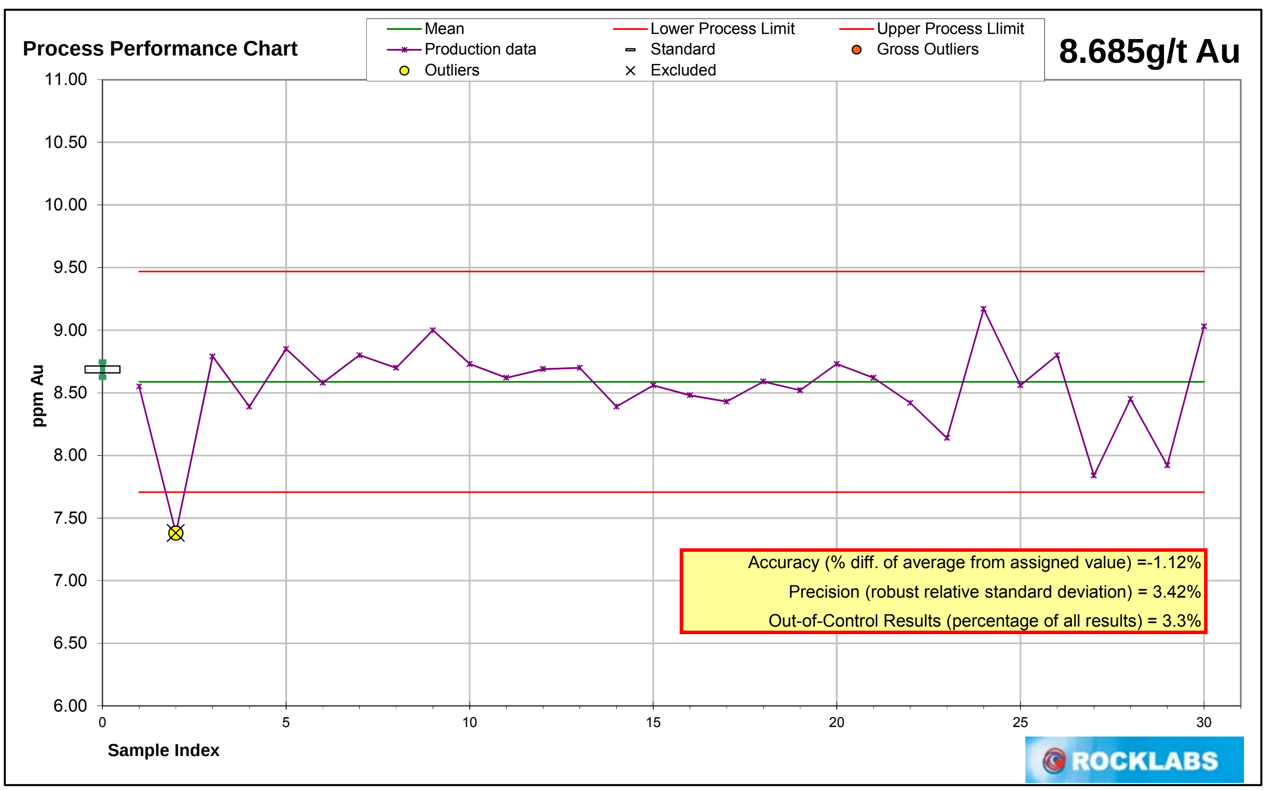


Figure 12.6f – Gold standard from 2010-2011 (8.685 g/t Au)

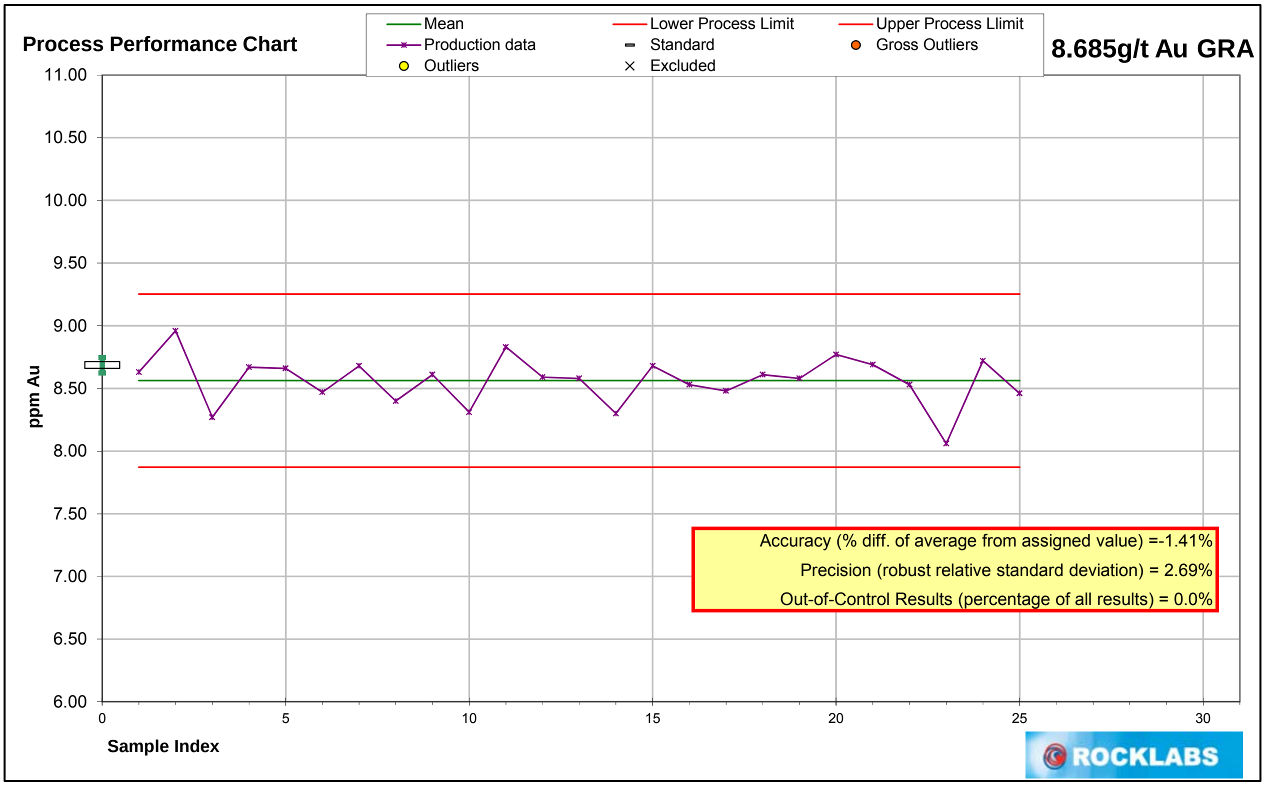


Figure 12.6g – Gold standard from 2010-2011 (8.685 g/t Au using gravimetric finish)

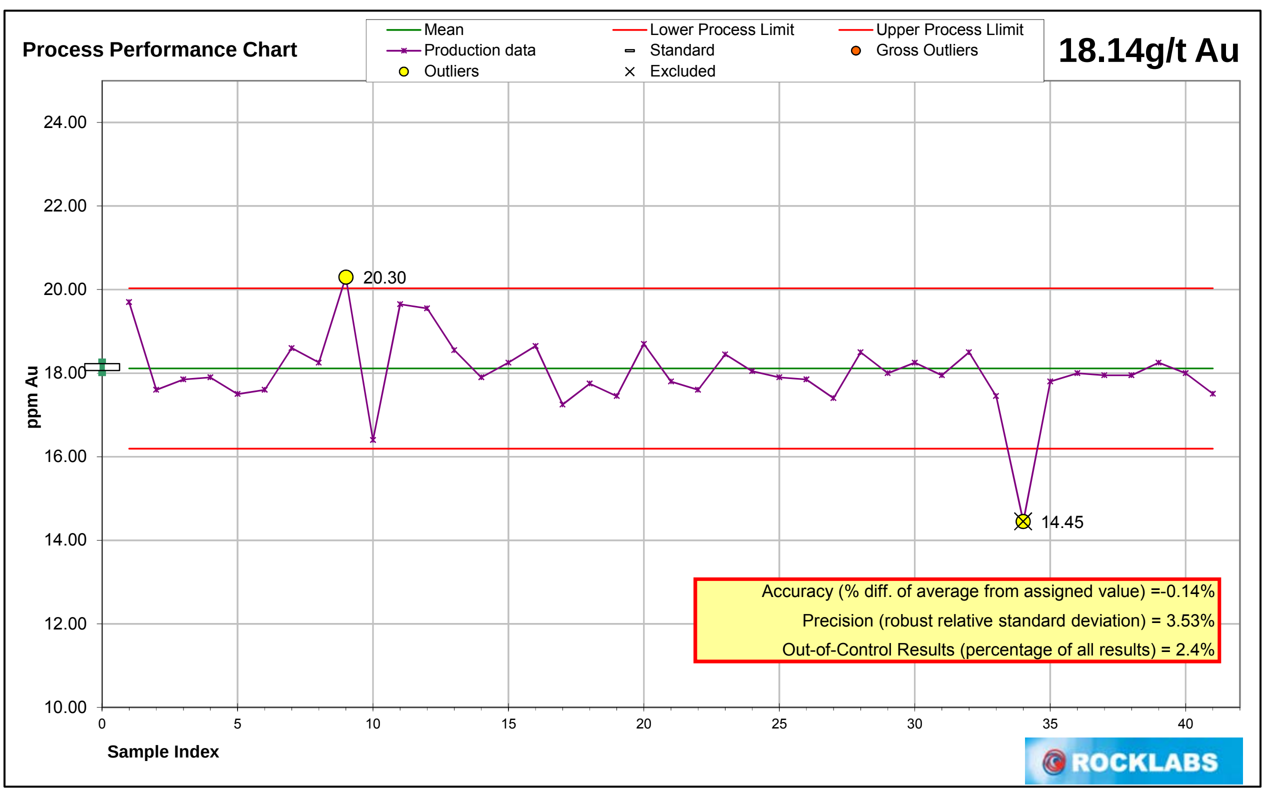


Figure 12.6h – Gold standard from 2010-2011 (18.14 g/t Au)

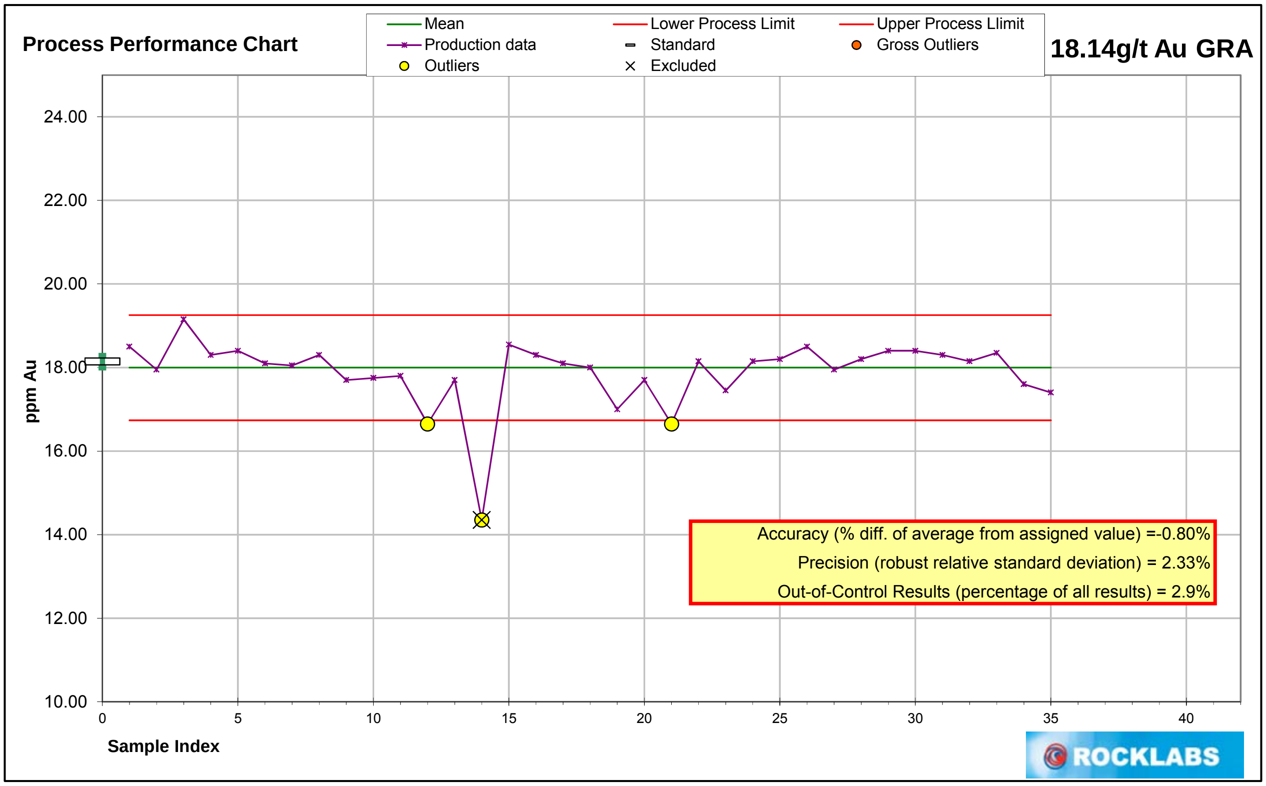


Figure 12.6i – Gold standard from 2010-2011 (18.14 g/t Au using gravimetric finish)

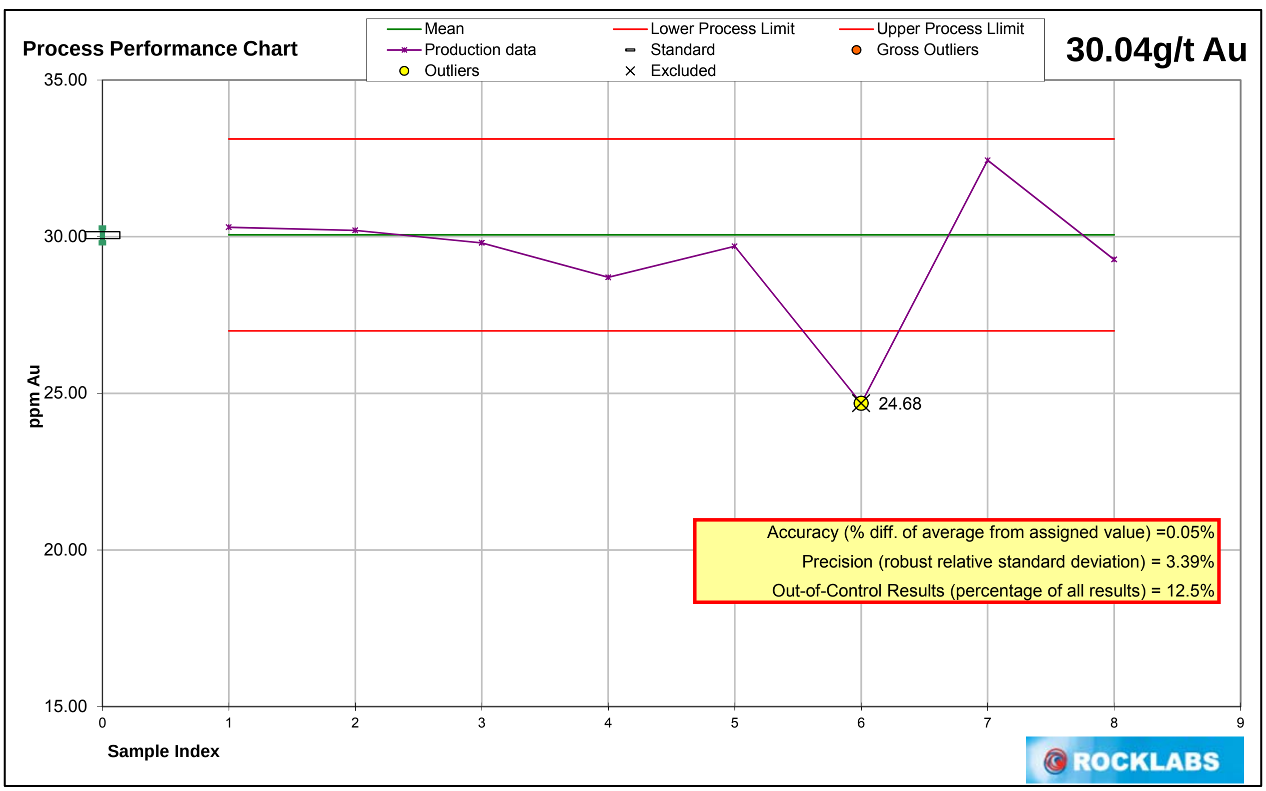


Figure 12.6j – Gold standard from 2010-2011 (30.04 g/t Au)

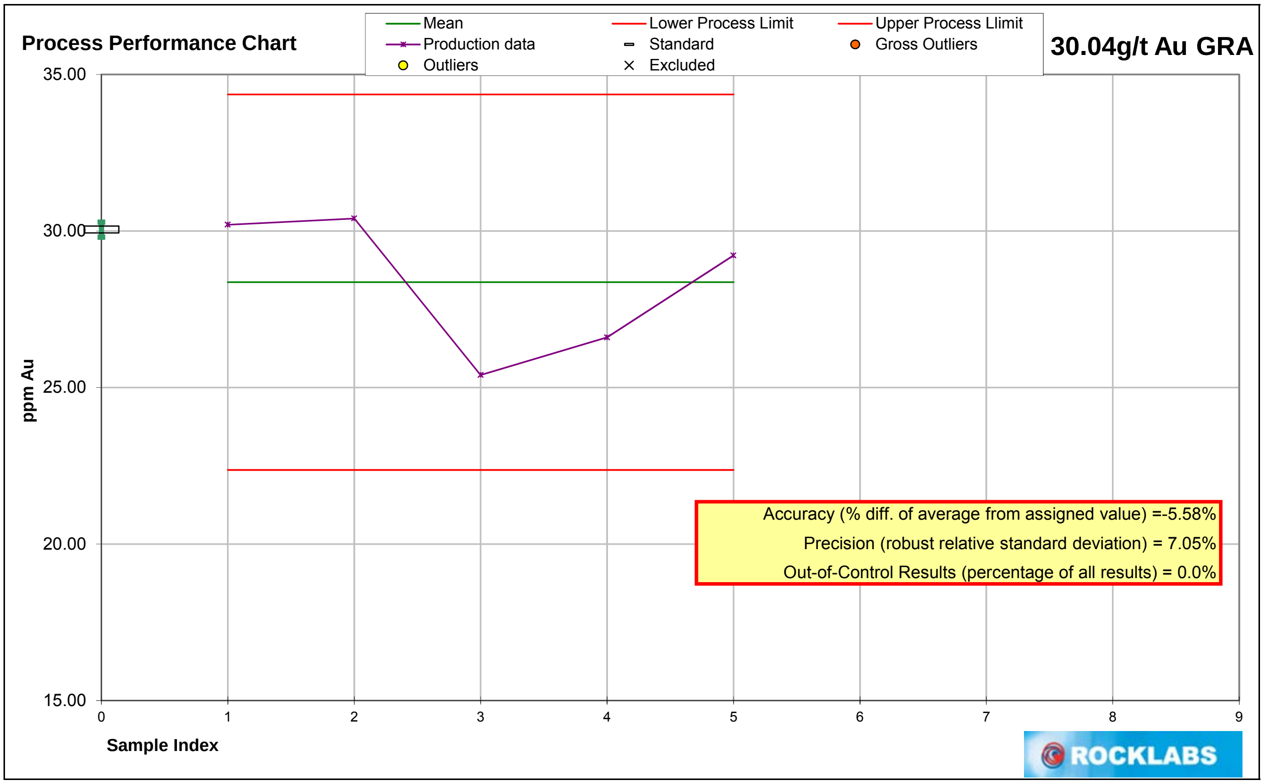


Figure 12.6k – Gold standard from 2010-2011 (30.04 g/t Au using gravimetric finish)

12.4 Data verification

When InnovExplo began drilling in 2010, the firm's exploration team made several verifications to familiarize themselves with the property's gold mineralization. Cross sections prepared by InnovExplo for O'Dowd in 2009 were reviewed and proposals by Francois Chabot of Golder Associates were plotted on these sections.

The exploration team adhered to the following procedure:

- Verify the locations of old casings using handheld GPS units in the field, especially holes in close proximity to proposed new drill sites (Fig. 12.7);
- Verify the presence of casing in the field at all proposed drill sites, for the IP borehole survey;
- Review old logs and cross sections;
- Visit mineralized areas in the field to check the type of mineralization and their historical descriptions (ore and waste piles, open pit walls, the Bug Lake showing, the Trench 2 area, and others);
- Spot all drill holes in the field and have a technician or geologist verify the alignment of all drills before drilling starts. Figure 12.8 shows a typical drill set-up after alignment verification;
- Have a geologist on site to terminate all drill holes in the early stages of the drilling program to become familiar with the geology of the area;

At the end of each drilling phase from 2010 to 2011, the completed holes were surveyed by Corriveau J.L. & Associates Inc (Land-Surveyor). The surveyed locations were updated in the central database. All holes drilled in 2010 and 2011 were identified by metal casing caps and red metal flags (an example is shown in Figure 12.9).

The Gemcom database was verified for consistency between the Geotic logs and the information contained in the database. Some errors were identified and corrected accordingly. InnovExplo is of the opinion that the data contained in the Croinor database is of good quality and can be relied upon for the current Mineral Resource Estimate.

Figure 12.7 – Technician verifying the location of a historical hole by handheld GPS before spotting the new location for a planned drill hole (2010).

Figure 12.8 – Drill set-up behind a rock pile at the small West Pit, summer 2010. Drilling is towards the south. The sloop is on the right in red, with the tractor on the left.

Figure 12.9 – Example of metal casing cap with red metal flag attached for hole CR-11-414, showing engraved drill hole identification number on metal cap.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section presents the results of historical milling operations using the information available in reports.

InnovExplo did not perform metallurgical tests or any other type of mineral processing testing. The reader is directed to the reports by Chénard and Turcotte (2003) and St-Jean (2001, 2003) for a more complete review of previous milling tests and milling operations.

13.1 Earlier Milling Operations

13.1.1 Cambior's 1988 metallurgical tests

Cambior requested that Lakefield Research Laboratory perform metallurgical tests (Salter and Furey, 1988; Chénard and Turcotte, 2003). Flotation and direct cyanidation tests were performed on a 76.8-kg sample composed of mineralized intersections from sixteen (16) drill holes. The direct cyanidation tests reported recovery rates close to 98%, whereas flotation tests gave recovery rates of better than 96%. During these tests, the density of the treated mineralized material was 2.82 g/cm^3. Note that the calculated grades obtained during Lakefield's tests are comparable to Cambior's original core assays, but the values for both were higher than Lakefield's assays on the same core samples.

13.1.2 Malartic-Sud's 2001 and 2003 metallurgical tests

Malartic-Sud performed two (2) metallurgical tests (Table 13.1). The mandates for these tests were assigned to Laboratoire LTM Inc (St-Jean, 2001; 2003). The first test was to verify the behaviour of the Croinor mineralized material in the Chimo mill. The Knelson tests gave a recovery of 47.3%, flotation tests yielded rates from 83 to 98%, and cyanidation tests gave recovery rates of 96.3% with grinding. Note that the assay grades for the core samples were higher than the grades calculated for the feed during the tests.

Malartic-Sud's second test was performed on two (2) samples from two different zones. Flotation tests gave recovery rates of 94.9% and 97%, and cyanidation tests yielded corresponding recovery rates of 95.1% and 99.1%. Note that the grades obtained during these tests were higher than the grades given for the assayed core. The first sample graded 4.45 g/t Au when assayed from core, whereas the average calculated grade of the feed during the cyanidation tests for the same sample was 6.35 g/t Au. The second sample yielded 6.68 g/t Au when assayed, whereas the average calculated grade of the feed during the cyanidation and flotation tests was 7.45 g/t Au.

The results obtained over these two periods show that there is no difference in the processing characteristics of mineralized material with different grades.

Table 13.1 – Results from metallurgical testing

Company	Year	Type of tests	Pulverization	Recov. (%)	Calculated grade (g/t)	Measured grade (g/t)	Grade difference (g/t)	Grade difference (%)
Cambior	1988	Cyanide	76.8% -200 MESH	97.8	5.99	5.88	-0.11	1.8
		Cyanide	82.8% -200 MESH	97.9	5.99	5.58	-0.41	6.8
		Cyanide	88.8% -200 MESH	97.8	5.99	5.87	-0.12	2.0
		Cyanide	80.0% -200 MESH	96.7	5.99	5.98	-0.01	0.2
		Flotation	70.0 % -200 MESH	96.4	5.99	6.01	0.02	0.3
		Flotation	80.0 % -200 MESH	96.3	5.99	5.46	-0.53	8.8
		Flotation	90.0 % -200 MESH	96.6	5.99	6.41	0.42	6.6
Malartic-Sud	2001	Knelson	100% -10 MESH	47.3	3.24	2.06	-1.18	36.4
		Flotation	94.1% -200 MESH	83.3	3.24	2.24	-1.00	30.9
		Flotation	94.1% -200 MESH	69.9	3.24	2.97	-0.27	8.3
		Flotation	94.1% -200 MESH	97.7	3.24	2.94	-0.30	9.3
		Cyanide	94.1% -200 MESH	96.3	3.24	2.01	-1.23	38.0
		Cyanide	97.3% -200 MESH	97.3	3.24	2.01	-1.23	38.0
Malartic-Sud	2003	Flotation	77.8% -200 MESH	97.0	4.45	4.12	-0.33	7.4
		Cyanide	68.0% -200 MESH	94.3	4.45	5.04	0.59	11.7
		Cyanide	79.9% -200 MESH	95.8	4.45	6.42	1.97	30.7
		Cyanide	82.6% -200 MESH	93.8	4.45	5.97	1.52	25.5
		Cyanide	84.9% -200 MESH	94.2	4.45	9.00	4.55	50.6
		Cyanide	87.6% -200 MESH	95.7	4.45	6.74	2.29	34.0
		Cyanide	94.8% -200 MESH	93.9	4.45	6.44	1.99	30.9
		Cyanide	97.4% -200 MESH	96.1	4.45	4.87	0.42	8.6
		Flotation	80.5% -200 MESH	94.9	6.68	7.33	0.65	8.9
		Cyanide	93.0% -200 MESH	99.1	6.68	7.56	0.88	11.6

Source: Chénard and Turcotte (2003)

13.1.3 Bulk sampling

Before Exploration Malartic-Sud began working on the Croinor Project, three (3) bulk samples had been extracted from the property as per the summary in Table 13.2.

Table 13.2 – Bulk samples taken from 1972 to 1997

Year	Location	Mill	Tonnage (t)	Grade (g/t Au)	Recovery Au (%)
1972	Underground	Goldfields	9 979	3.70	95.0
1983	Underground	Belmoral	1 700	1.47	86.0
1996-1997	Goldust pit	Aur-Bel/Chimo	51 010	3.40	97.0

Since 2003, Malartic-Sud has milled Croinor ore five times: one period for the bulk sample in 2003, and four other periods for the Centre and West open pit operations. All the ore was processed through the Camflo mill in Malartic, owned by Richmont Mines Inc.

13.2 Milling and Metallurgy

The following sections report the milling and metallurgy results for the bulk sample of mineralized material that was processed at the Camflo mill from February 3 to February 25, 2004, as described by Mr Oscar Lafrance, a consultant hired by Malartic-Sud to supervise the milling operations. The text is from the report by Moryoussef (2004). InnovExplo did not verify the data but assumes they are valid and pertinent. No report exists for the milling of mineralized material from the Centre open pit, but the process was exactly the same and also supervised by Lafrance. The Camflo mill customizes the milling of ore from many different sources, and a circuit inventory is therefore done before and after each program to obtain a representative metallurgical balance. Table 13.3 presents the summary of Malartic-Sud's milling performed at the Camflo mill.

13.2.1 Milling method

The Camflo mill uses a direct cyanidation circuit for the mineralized material. All the mineralized material is crushed and grinded until 70 to 85% of the material pass through a 200 mesh sieve. It should be noted that grinding could be adjusted to be finer or coarser if needed. All crushed mineralized material in a cyanide and lime solution is agitated in retention tanks in presence of cyanide which, combined to oxygen, dissolves the gold. The solution is then recovered through decant and filtration then, taking out the oxygen, by creating a vacuum, and also by adding zinc powder to the solution, gold is precipitated and then recovered in press filters. The precipitate is then recovered and mixed to a flux to cast an unpure [sic] gold bar which is carried to the Royal Canadian Mint or any other refiner to be purified and marketed. The balance of the mineralized material is taken to the tailing pond, constituting the final waste. (From O. Lafrance, 2004, as reported in Moryoussef, 2004)

13.2.2 Summary of the mineralized material metallurgy

The Croinor mineralized material reacts to the direct cyanidation process practically perfectly. It is difficult to calculate the bond index. In comparison, I would say that the bond index is between 13 and 16 KWh. The tonnage processed to the mills was always thin and the feeding to the cyclone very unstable, this being due to the blocking of feeding chutes by frozen mineralized material. A coarse grinding was obtained, varying between 74% and 79% passing a 200 mesh sieve. The filtration was easy with average liquid waste of 0.0003 ounces per short ton, which is excellent. The average solid rejects were of 0.002 ounces per short ton. This being the assaying detection limit, thus the recovery can be considered almost perfect. Cyanide consumption was of 0.2 kg/short ton, which is very low. Putting apart the physical mineralized material handling due to freezing, all other aspects seem perfect. (From O. Lafrance, 2004, as reported in Moryoussef, 2004)

In summary, the various mineralized material milling steps performed on the Croinor mineralized material at the Camflo mill confirm the excellent response of the mineralized material to direct cyanidation and a satisfactory recovery rate. The tonnage milled to date (75,752 tonnes), coming from various veins and with different grades, is considered to be representative enough of Croinor mineralized material that no problem is expected for any future operation on the same type of mineralized material.

Table 13.3 – Milling results for the Croinor ore at the Camflo mill

Date	Tonnes (t)	Grade (g/t)	Total Ounces	Recovery	Recovered Ounces
February 3-25, 2004	20,629	3.10	2,033	97.4%	1,981
August 6-13, 2004	7,883	1.80	456	95.3%	435
August 18-31, 2004	9,750	1.97	619	95.5%	591
October 7-18, 2004	13,127	2.50	1,055	96.6%	1,019
July 2005	24,363	5.38	3,920	97.9%	3,834
Total	**75,752**	**3.33**	**8,081**	**97.0%**	**7,860**

14.0 MINERAL RESSOURCE ESTIMATE

The Mineral Resource Estimate herein was performed by Karine Brousseau, Eng. and Tafadzwa Gomwe, PhD, GIT, under the supervision of Carl Pelletier, BSc, PGeo, using all available results. One of the objectives of InnovExplo's work was to prepare a Regulation 43-101 compliant Mineral Resource Estimate for the Croinor deposit using 3D block modelling instead of a polygonal method like the previous estimates. The result is a Mineral Resource Estimate with Measured, Indicated, and Inferred Resources for the Croinor deposit. The effective date of this Mineral Resource Estimate is November 4, 2011.

14.1 Methodology

The Mineral Resource Estimate detailed in this report was made using 3D block modelling and the inverse distance power six (1/D6) interpolation method for a 1,570-m strike-length corridor of the Croinor property, from 910mW to 640mE (local grid, oriented on a 291.9167° Azimuth) and down to a vertical depth of 545 metres below surface.

14.1.1 Drill hole and channel sample database

InnovExplo compiled and merged all existing drill hole databases for the Croinor property. The 2010 and 2011 surface drill holes were added to the database, up to and including CR-11-413, which were complete in terms of assay results at the time of the current Mineral Resource Estimate. The current Mineral Resource Estimate considered 1,219 surface and underground diamond drill holes with conventional analytical gold assay results and coded lithologies. The 1,219 holes cover an east-west distance of 1,530 m of the Croinor deposit (local grid).

All header (i.e., coordinate), survey (i.e., down-hole) and lithological data, as well as conventional assay grades, were integrated into a Gemcom database. In addition to the basic tables of raw data, the Gemcom database contains several tables with various drill hole and wireframe solid intersection composite calculations required for statistical evaluation and resource block modelling. The database contains a total of 27,655 assays taken from the 122,339 metres of core drilled in the 1,219 drill holes.

A channel sample database was integrated into the GEMS project. This database was compiled by InnovExplo in 2005 (Pelletier and Boudrias, 2005) and incorporated chip samples from development headings collected between 1983 and 1986. The header table includes the channel sample number, the collar location and the length of each channel sample. The conventional assay grades were compiled. The database contains a total of 4,309 assays taken from 1,927 channel samples.

14.1.2 Interpretation of mineralized zones

In order to facilitate resource modeling of the Croinor deposit, InnovExplo constructed a mineralized-zone solid model delimiting the geologically defined extent of the mineralized zones included in a 1,560-metre strike-length corridor measuring 230 metres wide and extending up to 500 metres below surface.

InnovExplo performed the geological interpretation and correlated the lenses on vertical sections spaced 10 metres apart. The mineralized-zone model was constructed to outline zones of continuous mineralization along an average trend of 290° Az and an average dip of -35°. A minimum width of 1.8 metres (true width) was respected for the interpretation model. The model contains a total of fifty-four (54) mineralized-zone solids (lenses). The names used for the 2011 mineralized-zone solids are based on the nomenclature of Pelletier and Boudrias (2005).

The wireframe solids of the mineralized-zone model were created by digitizing the data and performing an interpretation on sections spaced 10 metres apart, and then using tie-lines between the 3D rings to complete the wireframes for each solid.

Figure 14.1 presents a 3D isometric view of the mineralized-zone model, the existing open pits, and the drift network of the Croinor deposit.

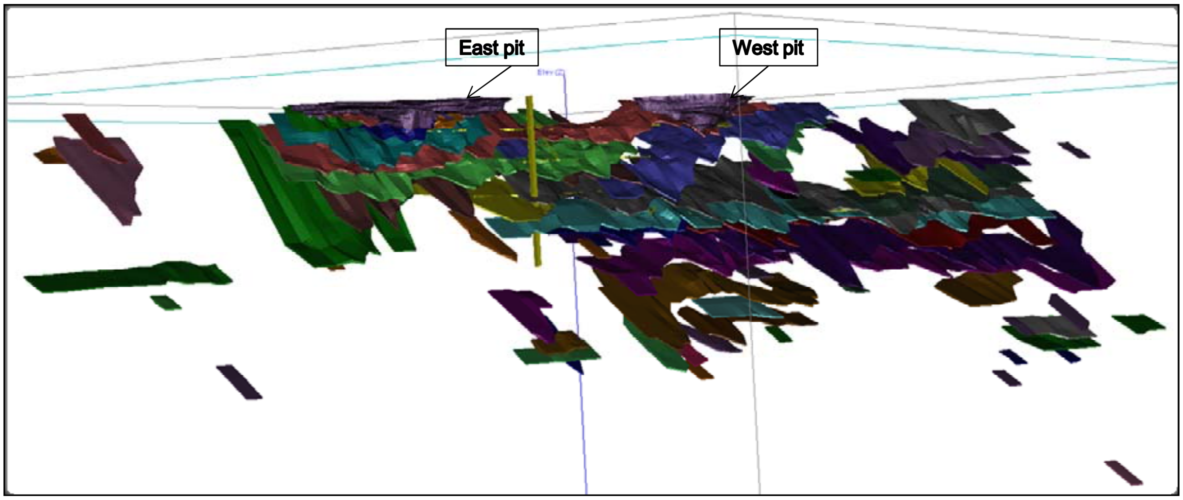


Figure 14.1 – 3D isometric view of the Croinor deposit looking southwest

14.1.3 Statistics on raw assays

Drill hole assay intervals intersecting interpreted mineralized zones were automatically coded in the database. This database was used to analyze sample lengths and generate statistics, composites and variography.

Basic univariate statistics on datasets grouped by zone were performed using point area files containing raw analytical assay data for a total of 8,680 diamond drill hole (DDH) samples. Only one assay population was plotted on a log-normal histogram for the DDH intersecting each zone. Table 14.1 presents the summary statistics for the DDH population.

The capping level was based on a study of the effect of high-grade values on the mean and standard deviations, as well as on normal histograms and probability plots. Capping the high assays represents a total loss of 4.98% of gold considering the volume of each lens and the percent loss of gold per lens. The high-grade capping value of 70 g/t Au for the DDH population is similar to the one used by O'Dowd in 2009 (65 g/t Au; O'Dowd, 2009).

Basic univariate statistics on channel sample datasets were also performed using point area files containing raw analytical assay data for a total of 4,144 chip samples. Only one assay population was plotted on a log-normal histogram for the channel samples intersecting each zone. A high-grade capping value of 55 g/t Au was applied to the channel sample population, based on a study of the effect of high-grade values on the mean and standard deviations, as well as on normal histograms and probability plots. Capping the high assays represents a total loss of 2.67% of gold for the channel sample population. Table 14.2 presents the summary statistics for the channel sample population.

Table 14.1 – Summary statistics for the raw assays for the DDH population by zone

Zone	Blockcode	Number of samples	Max	Uncut Mean (g/t)	High Grade Capping	Cut Mean	# Samples cut	% Samples capped	% loss metal content	Volume (m3)
ZONE1	101	685	161.82	3.58	70	3.45	1	0.15	3.74	60 975
ZONE2	102	1034	344.63	4.48	70	4.10	4	0.39	8.47	136 218
ZONE3	103	309	74.74	4.78	70	4.77	1	0.32	0.32	19 413
ZONE4	104	606	57.50	2.25	70	2.25	0	0.00	0.00	200 540
ZONE5	105	400	72.36	3.89	70	3.89	1	0.25	0.15	187 019
ZONE6	106	688	256.13	3.60	70	3.26	3	0.44	9.66	212 437
ZONE7	107	138	64.86	4.79	70	4.79	0	0.00	0.00	9 814
ZONE8	108	112	48.82	2.23	70	2.23	0	0.00	0.00	77 215
ZONE9	109	331	54.51	1.58	70	1.58	0	0.00	0.00	93 904
ZONE10	110	185	136.45	3.48	70	3.12	1	0.54	10.31	53 169
ZONE11	111	28	14.05	2.39	70	2.39	0	0.00	0.00	24 536
ZONE12	112	39	5.40	0.41	70	0.41	0	0.00	0.00	4 296
ZONE13	113	350	95.99	3.85	70	3.77	2	0.57	2.05	136 169
ZONE14	114	217	99.08	5.03	70	4.77	4	1.84	5.13	151 934
ZONE15	115	117	30.62	3.37	70	3.37	0	0.00	0.00	102 123
ZONE16	116	1141	306.62	3.97	70	3.53	10	0.88	11.25	139 939
ZONE17	117	51	8.50	1.33	70	1.33	0	0.00	0.00	4 828
ZONE18	118	548	169.30	3.23	70	2.97	2	0.36	8.01	85 737
ZONE19	119	7	7.03	1.97	70	1.97	0	0.00	0.00	1 997
ZONE20	120	132	82.97	4.22	70	4.12	1	0.76	2.33	41 790
ZONE21	121	12	11.45	1.93	70	1.93	0	0.00	0.00	5 570
ZONE22	122	5	6.97	2.06	70	2.06	0	0.00	0.00	1 055
ZONE23	123	12	6.86	0.85	70	0.85	0	0.00	0.00	5 024
ZONE24	124	389	53.98	2.53	70	2.53	0	0.00	0.00	39 609
ZONE25	125	40	20.03	1.29	70	1.29	0	0.00	0.00	23 028
ZONE26	126	118	53.14	2.15	70	2.15	0	0.00	0.00	36 028
ZONE27	127	6	5.69	2.19	70	2.19	0	0.00	0.00	3 101
ZONE28	128	210	35.43	3.02	70	3.02	0	0.00	0.00	74 822
ZONE29	129	30	4.46	0.77	70	0.77	0	0.00	0.00	12 507
ZONE30	130	201	148.04	2.56	70	2.18	1	0.50	15.15	53 496
ZONE31	131	95	74.05	6.00	70	5.93	2	2.11	1.26	10 596
ZONE32	132	12	25.93	4.38	70	4.38	0	0.00	0.00	2 596
ZONE33	133	69	66.13	2.63	70	2.63	0	0.00	0.00	27 827
ZONE34	134	11	11.94	2.19	70	2.19	0	0.00	0.00	6 752
ZONE35	135	33	38.39	2.00	70	2.00	0	0.00	0.00	30 936
ZONE36	136	11	11.03	1.91	70	1.91	0	0.00	0.00	9 406
ZONE37	137	12	8.07	1.19	70	1.19	0	0.00	0.00	2 319
ZONE38	138	4	16.05	4.03	70	4.03	0	0.00	0.00	1 678
ZONE39	139	4	3.28	1.04	70	1.04	0	0.00	0.00	1 488
ZONE40	140	78	13.16	1.62	70	1.62	0	0.00	0.00	5 338
ZONE41	141	25	28.15	3.36	70	3.36	0	0.00	0.00	25 372
ZONE42	142	42	4.97	0.83	70	0.83	0	0.00	0.00	16 410
ZONE44	144	9	35.72	6.18	70	6.18	0	0.00	0.00	1 780
ZONE45	145	9	6.51	1.28	70	1.28	0	0.00	0.00	5 137
ZONE48	148	24	14.75	1.24	70	1.24	0	0.00	0.00	7 168
ZONE50	150	9	24.48	5.59	70	5.59	0	0.00	0.00	9 348
ZONE55	155	7	22.97	5.14	70	5.14	0	0.00	0.00	8 229
ZONE70	170	13	112.97	10.57	70	7.27	1	7.69	31.26	7 055
ZONE75	175	16	50.05	6.99	70	6.99	0	0.00	0.00	18 441
ZONE80	180	5	11.54	4.77	70	4.77	0	0.00	0.00	3 936
ZONE85	185	5	5.89	3.00	70	3.00	0	0.00	0.00	1 812
ZONE90	190	18	11.31	3.12	70	3.12	0	0.00	0.00	11 337
ZONE95	195	21	9.96	2.70	70	2.70	0	0.00	0.00	20 467
ZONE100	200	7	7.48	1.46	70	1.46	0	0.00	0.00	2 121

Table 14.2 – Summary assay statistics for the raw assays for the channel sample population by zone

Zone	Blockcode	Number of samples	Max	Uncut Mean (g/t)	High Grade Capping	Cut Mean	# Samples cut	% Samples capped	% loss metal content	Volume (m3)
ZONE1	101	462	86.39	3.71	55	3.57	3	0.65	3.97	60 975.00
ZONE2	102	846	102.85	5.83	55	5.62	9	1.06	3.55	136 218.00
ZONE3	103	4	2.39	1.28	55	1.28	0	0.00	0.00	19 413.00
ZONE4	104	549	68.22	2.72	55	2.69	2	0.36	1.45	200 540.00
ZONE5	105	171	17.14	1.31	55	1.31	0	0.00	0.00	187 019.00
ZONE6	106	512	101.48	3.47	55	3.38	1	0.20	2.61	212 437.00
ZONE9	109	42	25.37	2.35	55	2.35	0	0.00	0.00	93 904.00
ZONE10	110	384	23.99	0.98	55	0.98	0	0.00	0.00	53 169.00
ZONE13	113	142	10.28	1.03	55	1.03	0	0.00	0.00	136 169.00
ZONE16	116	1021	308.56	4.22	55	3.62	7	0.69	14.41	139 939.00
ZONE28	128	7	4.11	1.71	55	1.71	0	0.00	0.00	74 822.00
ZONE35	135	4	3.77	1.54	55	1.54	0	0.00	0.00	30 936.00

14.1.4 Compositing

In order to minimize any bias introduced by the variable sample lengths, the capped gold assays were composited to 1.0-metre (1.0m) equal lengths within all DDH intervals defining each of the mineralized zones. The same composite length was used for the DDH and channel sample assays. A total of 11,504 DDH and 3,485 channel sample equal-length composites were created in the zones. Each mineralized-zone solid (lens) was estimated separately using its own set of composites. When all composites generated within the assayed interval were taken into account, 4.31% of the DDH population and 6.54% of the channel sample population have lengths less than 0.15 m. All composited assays less than 15 cm long for the DDH population and the channel sample population were removed from each population before performing the block model interpolation and variography. No grades were assigned to missing sample intervals.

Tables 14.3 and 14.4 present the distribution statistics by zones for the 1.0m composites generated for the DDH and the channel sample populations respectively. For the DDH population, twenty-four (24) of the zones present a moderate Coefficient of Variation, indicating that high-grade values contribute moderately to the mean grades. For the other thirty (30) zones, high grades are more erratic, with a Coefficient of Variation ranging from 2.0 to 6.24.

Table 14.3 – Summary statistics for the 1.0-metre DDH composites by zone

Zone	BlockCode	Number of 1m Composites	Min	Max	Mean	Std Deviation	Coeff. Of Variation
ZONE1	101	788	0.00	48.97	2.67	4.49	1.68
ZONE2	102	1274	0.00	70.00	3.23	6.99	2.16
ZONE3	103	430	0.00	55.93	3.07	6.48	2.11
ZONE4	104	813	0.00	40.99	1.31	3.20	2.44
ZONE5	105	441	0.00	69.18	2.94	6.62	2.25
ZONE6	106	854	0.00	69.93	2.08	5.28	2.54
ZONE7	107	233	0.00	31.26	2.71	5.08	1.87
ZONE8	108	135	0.00	48.82	1.69	4.69	2.77
ZONE9	109	564	0.00	36.92	0.88	2.56	2.91
ZONE10	110	242	0.00	56.88	2.01	5.65	2.81
ZONE11	111	29	0.00	8.82	1.98	2.51	1.27
ZONE12	112	43	0.00	3.28	0.29	0.58	2.03
ZONE13	113	460	0.00	64.70	2.26	6.18	2.73
ZONE14	114	230	0.00	69.99	3.36	7.41	2.21
ZONE15	115	127	0.00	30.62	3.16	4.46	1.41
ZONE16	116	1515	0.00	70.00	2.22	6.45	2.90
ZONE17	117	72	0.00	6.72	0.83	1.45	1.76
ZONE18	118	643	0.00	69.91	2.02	6.36	3.15
ZONE19	119	5	0.81	5.29	2.41	1.68	0.70
ZONE20	120	147	0.00	69.32	3.18	7.87	2.47
ZONE21	121	17	0.00	3.69	0.42	1.15	2.71
ZONE22	122	10	0.00	6.97	1.21	2.16	1.79
ZONE23	123	15	0.00	6.86	0.64	1.71	2.67
ZONE24	124	444	0.00	49.93	1.97	4.30	2.18
ZONE25	125	34	0.00	9.91	1.14	2.37	2.08
ZONE26	126	126	0.00	35.72	1.32	3.68	2.78
ZONE27	127	5	0.00	3.60	1.50	1.38	0.92
ZONE28	128	247	0.00	29.51	2.02	3.71	1.84
ZONE29	129	44	0.00	4.46	0.55	1.11	2.03
ZONE30	130	245	0.00	29.55	1.19	3.23	2.72
ZONE31	131	150	0.00	57.78	3.07	7.87	2.56
ZONE32	132	12	0.01	25.90	4.45	6.78	1.52
ZONE33	133	83	0.00	48.16	1.74	5.49	3.16
ZONE34	134	11	0.06	11.94	2.32	3.58	1.54
ZONE35	135	103	0.00	35.00	0.55	3.45	6.24
ZONE36	136	10	0.00	4.77	1.37	1.63	1.19
ZONE37	137	20	0.00	5.69	0.63	1.37	2.17
ZONE38	138	7	0.00	9.25	1.33	3.23	2.42
ZONE39	139	6	0.00	3.28	0.69	1.19	1.72
ZONE40	140	114	0.00	13.16	1.16	2.23	1.92
ZONE41	141	26	0.00	28.15	3.51	6.06	1.72
ZONE42	142	80	0.00	4.02	0.41	0.83	2.03
ZONE44	144	10	0.00	17.71	3.65	5.75	1.57
ZONE45	145	16	0.00	6.51	0.76	1.67	2.21
ZONE48	148	23	0.00	14.72	1.24	3.11	2.52
ZONE50	150	7	1.59	24.41	5.96	7.62	1.28
ZONE55	155	11	0.00	22.92	3.96	6.41	1.62
ZONE70	170	11	0.77	58.32	7.56	16.18	2.14
ZONE75	175	15	0.85	50.05	7.21	12.04	1.67
ZONE80	180	5	0.00	11.38	4.92	4.27	0.87
ZONE85	185	8	0.00	5.89	1.88	2.20	1.17
ZONE90	190	16	0.00	7.16	2.63	2.40	0.91
ZONE95	195	27	0.00	9.95	1.87	2.54	1.36
ZONE100	200	5	0.16	4.54	1.35	1.63	1.21

Table 14.4 – Summary statistics for the 1.0-metre channel sample composites by zone

Zone	BlockCode	Number of 1m composites	Min	Max	Mean	Std Deviation	Coeff. Of variation
ZONE1	101	375	0.00	46.17	3.50	5.93	1.69
ZONE2	102	691	0.00	55.00	5.59	8.60	1.54
ZONE3	103	3	0.68	1.90	1.20	n.d.	n.d.
ZONE4	104	462	0.00	45.97	2.40	4.67	1.94
ZONE5	105	123	0.00	14.47	1.31	2.11	1.61
ZONE6	106	368	0.00	30.54	3.23	5.09	1.58
ZONE9	109	43	0.00	25.37	2.35	4.66	1.98
ZONE10	110	309	0.00	20.70	0.90	1.72	1.91
ZONE13	113	105	0.00	7.56	0.95	1.37	1.45
ZONE16	116	764	0.00	55.00	3.43	6.02	1.75
ZONE28	128	9	0.00	3.42	1.41	1.36	0.97
ZONE35	135	5	0.14	3.77	1.26	1.38	1.09

14.1.5 Variography

Because the down-hole oriented samples are the most closely spaced samples in the database, the nugget value (i.e., the variation in grade for the same sample point) is best estimated using a linear variogram. In addition, because the holes are oriented to cut across the width of the mineralization, the linear variogram provides an indication of the range of influence across the width of mineralization and the tertiary axis of the search/interpolation ellipse. Down-hole linear variography was completed on the 1.0m equal-length composites of the capped gold assay data for each of the mineralized zones to establish the best estimate of the nugget value and the range of grade continuity across the width of each zone.

The 3D directional-specific variography was completed using the 1.0m equal-length composites of the capped gold assay data for the DDH population of all mineralized zones. The investigation involved 10° incremental searches in the horizontal plane, followed by 5° to 10° incremental searches in the vertical planes indicated by the preferred azimuths, as well as planes normal to the preferred azimuth.

The results of the linear and 3D variographic investigations for the mineralized zone composites are consistent with the univariate statistics and generally correlate with geological features of the deposit. Some changes were introduced to the best-fit model to better reflect the geological model.

The 3D variography combined with the modified best-fit model produce the Inferred category search ellipse. The influence of the search ellipse for the Indicated category has been fixed at two-thirds (2/3) of the influence of the search ellipse for the Inferred category. No 3D variographic study was done for the channel sample population. Instead, a sphere of 9 metres radius was used for the channel samples to interpolate the grade of the Measured category; the size of the sphere was based on the dimensions of the mine drift of the Croinor deposit. Table 14.5 summarizes the search and interpolation ellipse orientations and ranges derived from the variography and the modified best-fit model for the 1.0m DDH composites.

Table 14.5 – Croinor ranges derived from variography and the modified best-fit model for DDH samples

	Rotation			Range (m)		
Category	Z	Y	Z	X	Y	Z
Inferred	90	35	90	35	25	15
Indicated	90	35	90	20	15	9

14.1.6 Bulk density

Since no new density determinations are available for the Croinor deposit, InnovExplo used the calculated values presented in the NI43-101 Technical Report of 2005 (Pelletier and Boudrias, 2005). Accordingly, a mean value of 2.80 was used in the mineralized zones and waste. Bulk densities were used to calculate tonnages from the volume estimates in the resource-grade block model.

14.1.7 Block model geometry

A block model was established over a 1,570-metre segment of Croinor deposit to a depth of 545 metres below surface. The limits of the block model are as follows:

- From 930mW to 640mE (314 columns x 5 m each)
- From 112mN to 692mN (232 rows x 2.5 m each)
- From level 3070 to level 2500 (228 levels x 2.5 m each)

The block model has parallel orientation, with the Y-axis oriented along a north local grid azimuth of 21.9167°. The individual block cells have dimensions of 5 m long (X-axis) by 2.5 m wide (Y) by 2.5 m vertical (Z). The block dimensions reflect the dimension of the mineralized zones. In order to avoid overlaps in rock code identification, three folders were generated within the same block model. Table 14.6 shows the Croinor block model with its interpolated zones and their associated folders.

Table 14.6 – Croinor block model and associated mineralized zones

Folder	Rock code	Block code
Group A	Zone 3	103
	Zone 6	106
	Zone 9	109
	Zone 11	111
	Zone 14	114
	Zone 16	116
	Zone 21	121
	Zone 22	122
	Zone 24	124
	Zone 26	126
	Zone 28	128
	Zone 29	129
	Zone 32	132
	Zone 34	134
	Zone 41	141
	Zone 42	142
	Zone 70	170
	Zone 75	175
	Zone 85	185
	Zone 90	190
	Zone 95	195
	Zone 100	200
Group B	Zone 1	101
	Zone 2	102
	Zone 4	104
	Zone 5	105
	Zone 7	107
	Zone 8	108
	Zone 10	110
	Zone 15	115
	Zone 18	118
	Zone 19	119
	Zone 25	125
	Zone 30	130
	Zone 33	133
	Zone 36	136
	Zone 38	138
	Zone 44	144
	Zone 45	145
	Zone 48	148
	Zone 80	180
Group C	Zone 12	112
	Zone 13	113
	Zone 17	117
	Zone 20	120
	Zone 23	123
	Zone 27	127
	Zone 31	131
	Zone 35	135
	Zone 37	137
	Zone 39	139
	Zone 40	140
	Zone 50	150
	Zone 55	155

14.1.8 Mineralized-zone block model

All blocks with more than 0.01% falling within the selected mineralized-zone solids were assigned the corresponding mineralized-zone rock code. A percent block model was generated reflecting the proportion of each block inside the mineralized solids. The percent block model was used in the resource estimation process. All remaining blocks were assigned code "999" for waste rock, "8" for underground development, "7" for open pit, "6" for overburden, or "5" for air.

14.1.9 Grade block model

As per the geostatistical results detailed in this report, a grade model was interpolated using the 1.0m composite derivates from the conventional assay grade data capped to produce the best possible grade estimate for the defined resources in the Croinor deposit. A set of nine (9) interpolation profiles was established for grade estimation. The interpolation profiles were customized to estimate grades separately within the Group A, Group B and Group C folders for the DDH population and the channel sample population. The method retained for the final resource estimation was inverse distance power six (1/D6) with capping of high-grade values.

A point area workspace providing the X, Y, Z and gold data points was used for block interpolation in the grade model. The composite points in each of the point area files were assigned rock codes and block codes corresponding to the mineralized zone in which they occur. The interpolation profiles specify a single target and sample rock code for each mineralized-zone solid, thus establishing hard boundaries between the mineralized zones and preventing block grades from being estimated using sample points with different integer block codes than the block being estimated. The search/interpolation ellipse orientations and ranges defined in the nine (9) interpolation profiles used for grade estimation correspond to those developed in Section 14.1.5 and Table 14.5 of this report (Variography).

Other specifications for the control grade estimation are as follows:

For all three folders:

- inverse distance power six (1/D6) interpolation method for data points;
- minimum of one (1) and maximum of three (3) DDH sample points in the search ellipse for interpolation of the indicated and inferred categories and minimum of two (2) and maximum of twelve (12) channel sample points for the interpolation of the measured category;
- no limit of samples from any one DDH or channel sample;
- high-grade values capped.

These interpolation parameters were specifically selected for the purpose of comparing the 3D block model method herein with the polygonal cross-sectional method used by O'Dowd in 2009 (O'Dowd, 2009).

14.1.10 Resource category block model

The Measured, Indicated and Inferred categories were identified by the interpolation process based on the search ellipse criteria and specific interpolation parameters. In order to generate the final resource grades for the block model, the Measured category interpolation profile for Group A, B and C folders were executed first. In the second step, the Indicated interpolation profiles were executed for all three folders, but excluded all previously identified Measured blocks. The third step calculated the Inferred interpolation profiles excluding the already identified Measured and Indicated blocks. Subsequently, isolated blocks or a series of blocks showing no spatial continuity in terms of grade and/or density of information were reclassified as Inferred. In the same way, isolated blocks or a series of blocks showing good spatial continuity in terms of grade and/or density of information were reclassified as Indicated. Only blocks having an assigned rock code were interpolated for grade and resource categories. Figure 14.2 displays an isometric 3D view of an example of resource categories for the Group A folder.

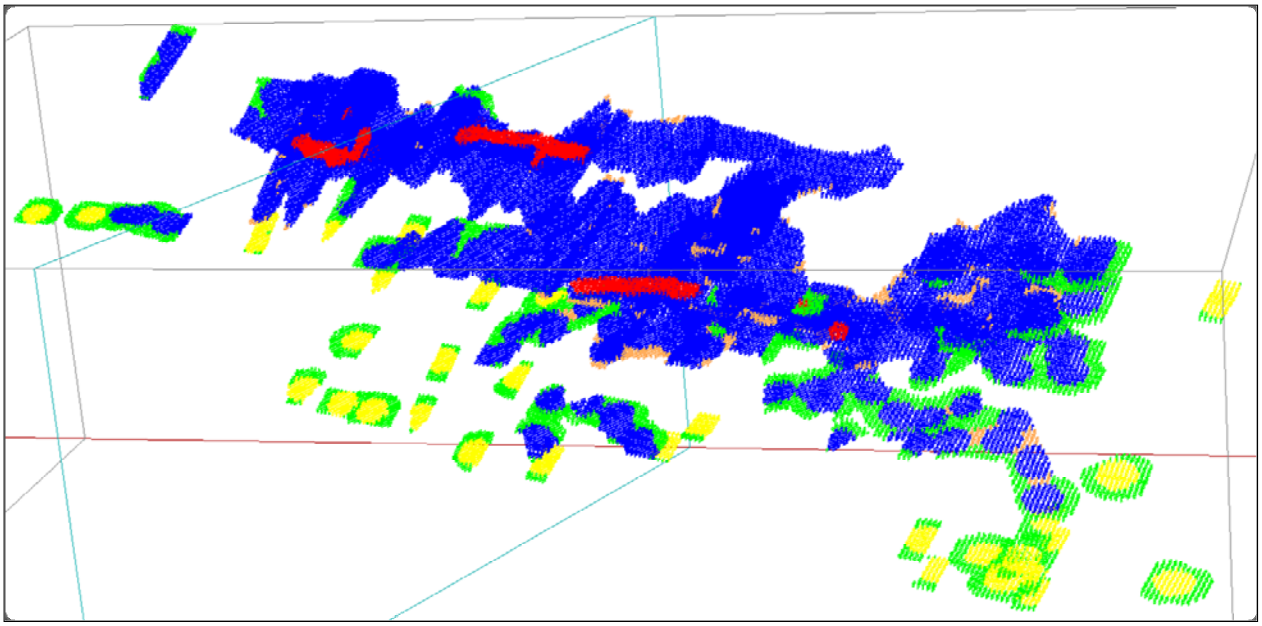


Figure 14.2 – Group A resource categories
(Red= Measured; Blue= Indicated; Yellow= Indicated reclassified as Inferred; Green= Inferred; Orange=Inferred reclassified as Indicated).

14.2 Mineral Resource Classification, Category or Definition

The resource classification definitions used for this report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their document "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines".

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

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

Inferred Mineral Resource: that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

14.3 Global Resource Estimation

InnovExplo is of the opinion that the current Mineral Resource Estimate can be classified as Measured, Indicated and Inferred resources based on the density of the processed data, the search ellipse criteria, and the specific interpolation parameters. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves. The resources were estimated using different gold cut-off grades and a minimum width of 1.8 metres (true width). The selected cut-off of 4 g/t was used to determine the mineral potential of the deposit for the underground mining option. Tables 14.7 and 14.8 display the results of the In Situ Mineral Resource Estimate for the Croinor deposit (54 lenses). The detailed results are presented by zone in Appendix II. InnovExplo is of the opinion that the current Mineral Resource Estimate can be used for a pre-feasibility study.

Figures 14.3 to 14.6 display the grade model for the different resource categories on cross-sections.

Table 14.7 – Results of the In Situ Mineral Resource Estimate (Measured, Indicated, and Measured+Indicated Resources) at different cut-off grades

Measured Resources				Indicated Resources				Measured + Indicated Resources			
Cut-off	Tonnes	Au g/t	Oz Au	Cut-off	Tonnes	Au g/t	Oz Au	Cut-off	Tonnes	Au g/t	Oz Au
> 6 g/t	46 800	10.98	16 500	> 6 g/t	355 800	12.13	138 800	> 6 g/t	402 600	12.00	155 300
> 5 g/t	59 400	9.81	18 700	> 5 g/t	447 300	10.78	155 000	> 5 g/t	506 700	10.66	173 700
> 4 g/t	80 500	8.41	21 800	> 4 g/t	599 600	9.18	176 900	> 4 g/t	680 100	9.08	198 700
> 3 g/t	112 400	7.00	25 300	> 3 g/t	848 300	7.51	204 700	> 3 g/t	960 700	7.45	230 000
> 2 g/t	165 200	5.55	29 500	> 2 g/t	1 192 200	6.05	231 700	> 2 g/t	1 357 400	5.99	261 200
> 1 g/t	257 600	4.08	33 800	> 1 g/t	1 889 600	4.35	264 100	> 1 g/t	2 147 200	4.31	297 900
> 0.6 g/t	297 700	3.64	34 800	> 0.6 g/t	2 283 700	3.73	274 100	> 0.6 g/t	2 581 400	3.72	308 900
> 0.3 g/t	335 200	3.28	35 300	> 0.3 g/t	2 685 400	3.24	279 800	> 0.3 g/t	3 020 600	3.25	315 100

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by Regulation 43-101, are Carl Pelletier, B.Sc., Geo. and Karine Brousseau, Eng. (InnovExplo Inc), and the effective date of the estimate is November 4, 2011.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Results are presented undiluted and in situ. The estimate includes 54 gold-bearing lenses.
- Resources were compiled using cut-off grades of 0.3 g/t, 0.6 g/t, 1.0 g/t, 2.0 g/t, 3.0 g/t, 4.0 g/t, 5.0 g/t and 6.0 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 2.8 g/cm^3 was used in zones and waste.
- A minimum true thickness of 1.8 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High-grade capping was done on raw data and established at 70.0 g/t Au for diamond drill hole assays and 55.0 g/t Au for underground chip sample assays.
- Compositing was not done over entire drill hole lengths. Instead, compositing was done on drill hole and chip sample sections that fall within the mineralized zone envelopes (composite = 1 metre).
- Resources were evaluated from drill hole and underground chip samples using an I/D6 interpolation method in a block model.
- The Measured, Indicated and Inferred categories were defined using different search ellipsoid parameters.
- The Inferred reclassified category is the result of isolated blocks or series of blocks showing no spatial continuity in terms of grade and/or density of information that were reclassified from Indicated to Inferred.
- The Indicated reclassified category is the result of blocks or series of blocks showing good spatial continuity in terms of grade and/or density of information that were reclassified from Inferred to Indicated.
- Ounce (troy) = Metric Tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation 43-101.
- Other than the drill hole CR-11-414 to CR-11-419, that are not included in the current resource estimate because they had pending assays at that time the cut-off date (mid-November) of the Mineral Resource Estimate,InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource Estimate.

Table 14.8 – Results of the In Situ Mineral Resource Estimate (Inferred Resources) at different cut-off grades

Inferred Resources			
Cut-off	Tonnes	Au g/t	Oz Au
> 6 g/t	84 500	12.02	32 700
> 5 g/t	102 400	10.90	35 900
> 4 g/t	160 100	8.56	44 100
> 3 g/t	227 800	7.03	51 500
> 2 g/t	358 300	5.36	61 800
> 1 g/t	564 600	3.94	71 500
> 0.6 g/t	664 700	3.47	74 200
> 0.3 g/t	726 300	3.21	75 100

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by Regulation 43-101, are Carl Pelletier, B.Sc., Geo. and Karine Brousseau, Eng. (InnovExplo Inc), and the effective date of the estimate is November 4, 2011.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Results are presented undiluted and in situ. The estimate includes 54 gold-bearing lenses.
- Resources were compiled using cut-off grades of 0.3 g/t, 0.6 g/t, 1.0 g/t, 2.0 g/t, 3.0 g/t, 4.0 g/t, 5.0 g/t and 6.0 g/t Au.
- Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate, mining cost).
- A fixed density of 2.8 g/cm^3 was used in zones and waste.
- A minimum true thickness of 1.8 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High-grade capping was done on raw data and established at 70.0 g/t Au for diamond drill hole assays and 55.0 g/t Au for underground chip sample assays.
- Compositing was not done over entire drill hole lengths. Instead, compositing was done on drill hole and chip sample sections that fall within the mineralized zone envelopes (composite = 1 metre).
- Resources were evaluated from drill hole and underground chip samples using an I/D6 interpolation method in a block model.
- The Measured, Indicated and Inferred categories were defined using different search ellipsoid parameters.
- The Inferred reclassified category is the result of isolated blocks or series of blocks showing no spatial continuity in terms of grade and/or density of information that were reclassified from Indicated to Inferred.
- The Indicated reclassified category is the result of blocks or series of blocks showing good spatial continuity in terms of grade and/or density of information that were reclassified from Inferred to Indicated.
- Ounce (troy) = Metric Tons x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource Estimate.

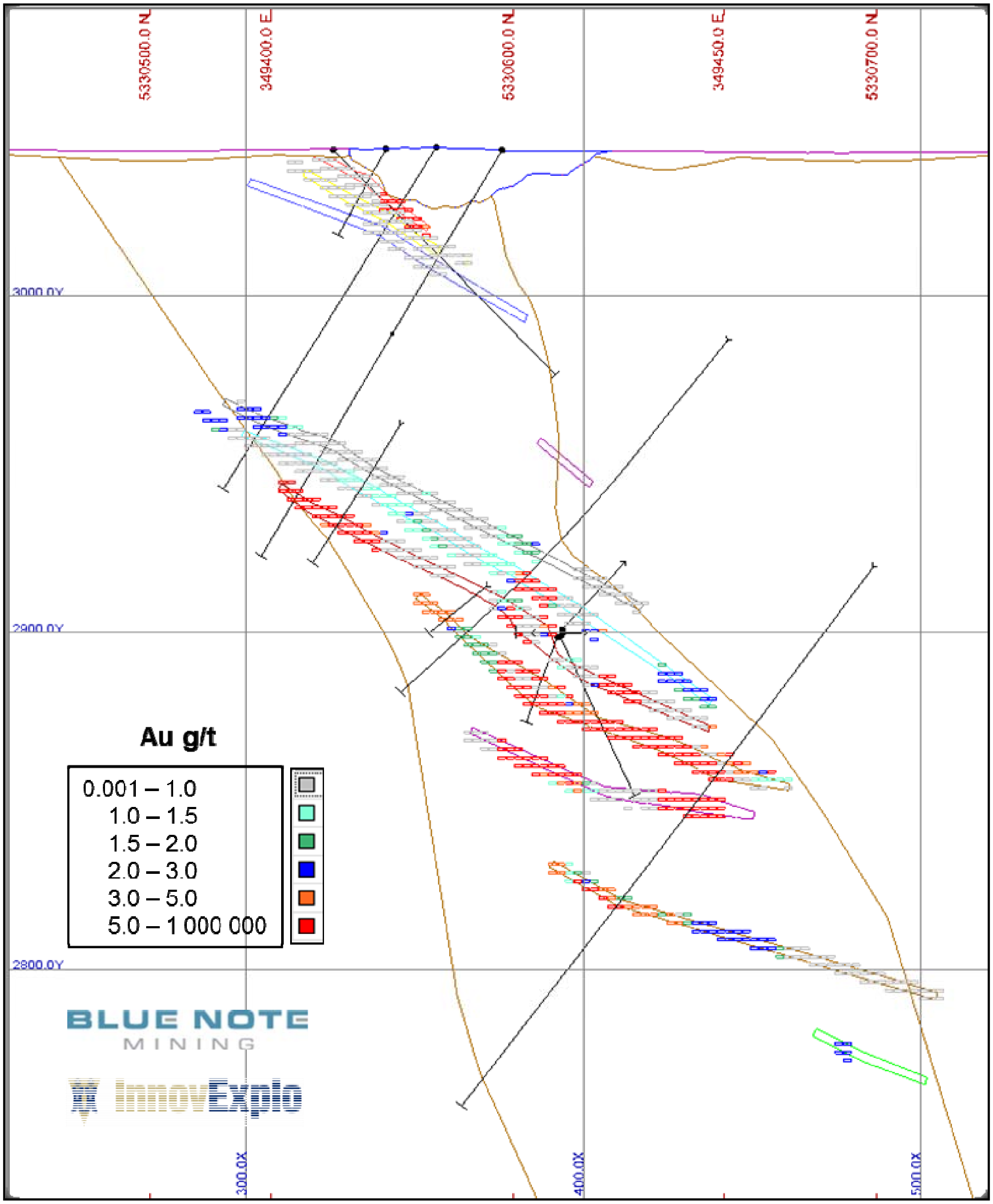


Figure 14.3 – Measured, Indicated and Inferred Resources in the Croinor deposit (Section 330W)

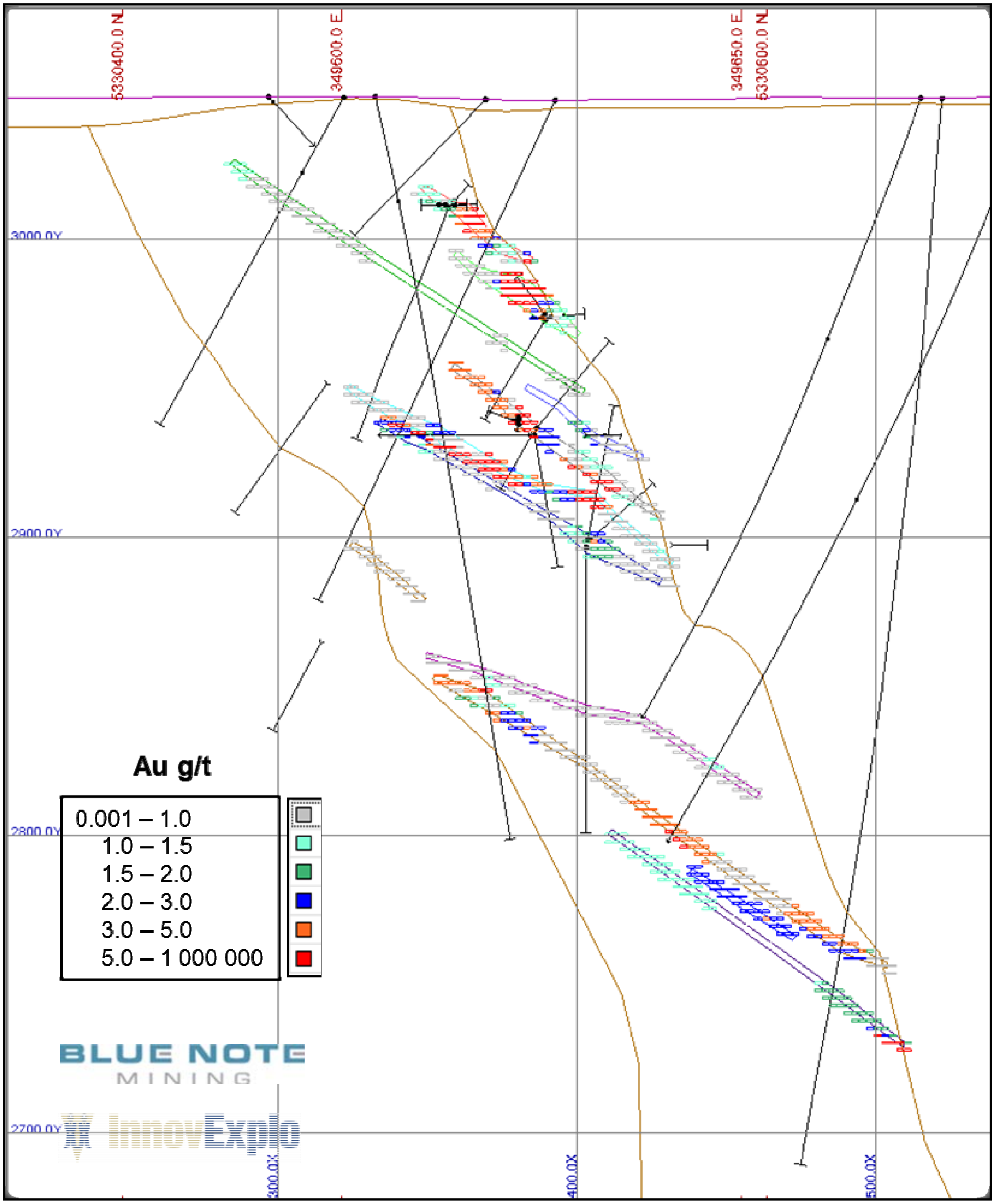


Figure 14.4 – Measured, Indicated and Inferred Resources in the Croinor deposit (Section 120W)

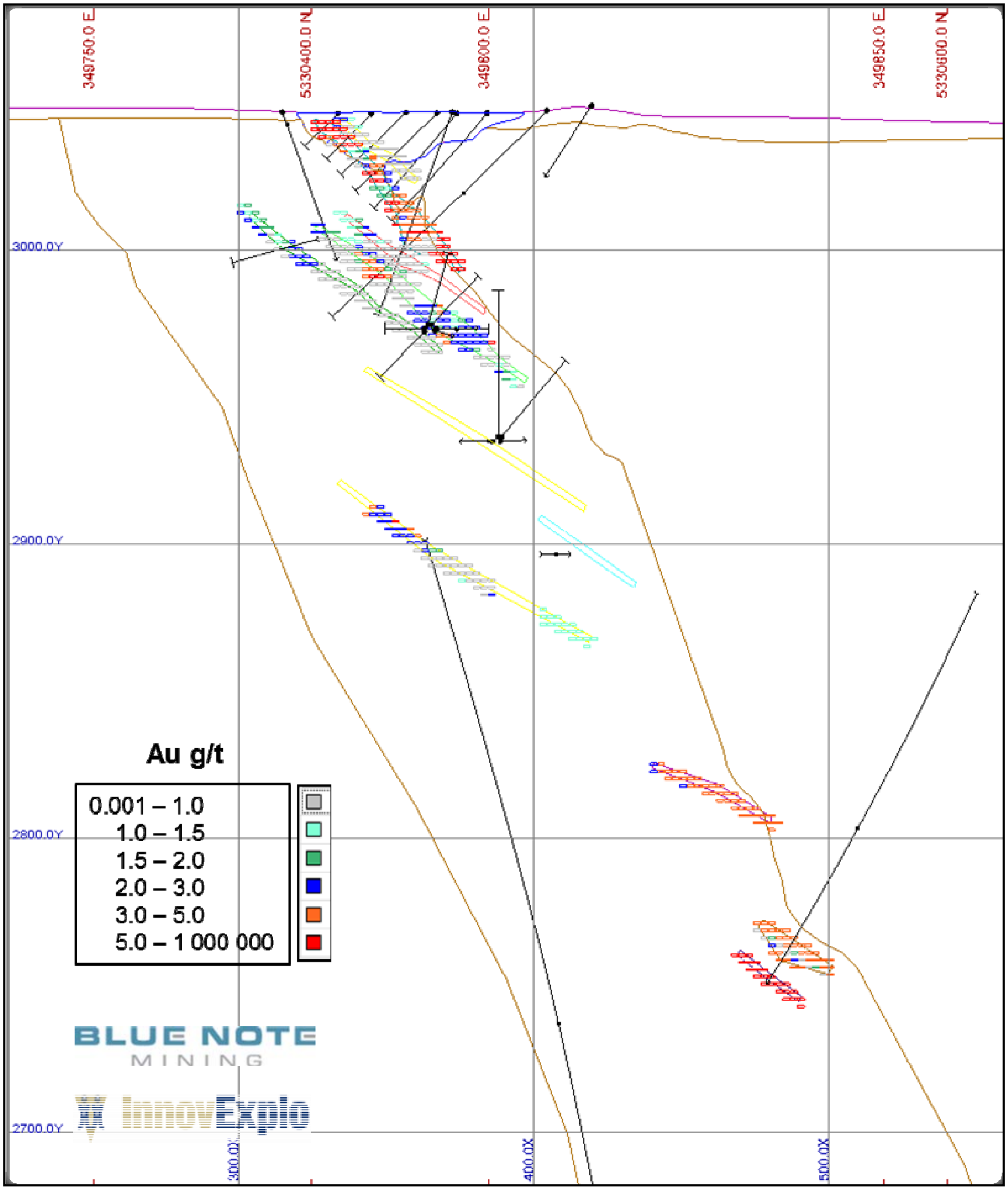


Figure 14.5 – Measured, Indicated and Inferred Resources in the Croinor deposit (Section 70E)

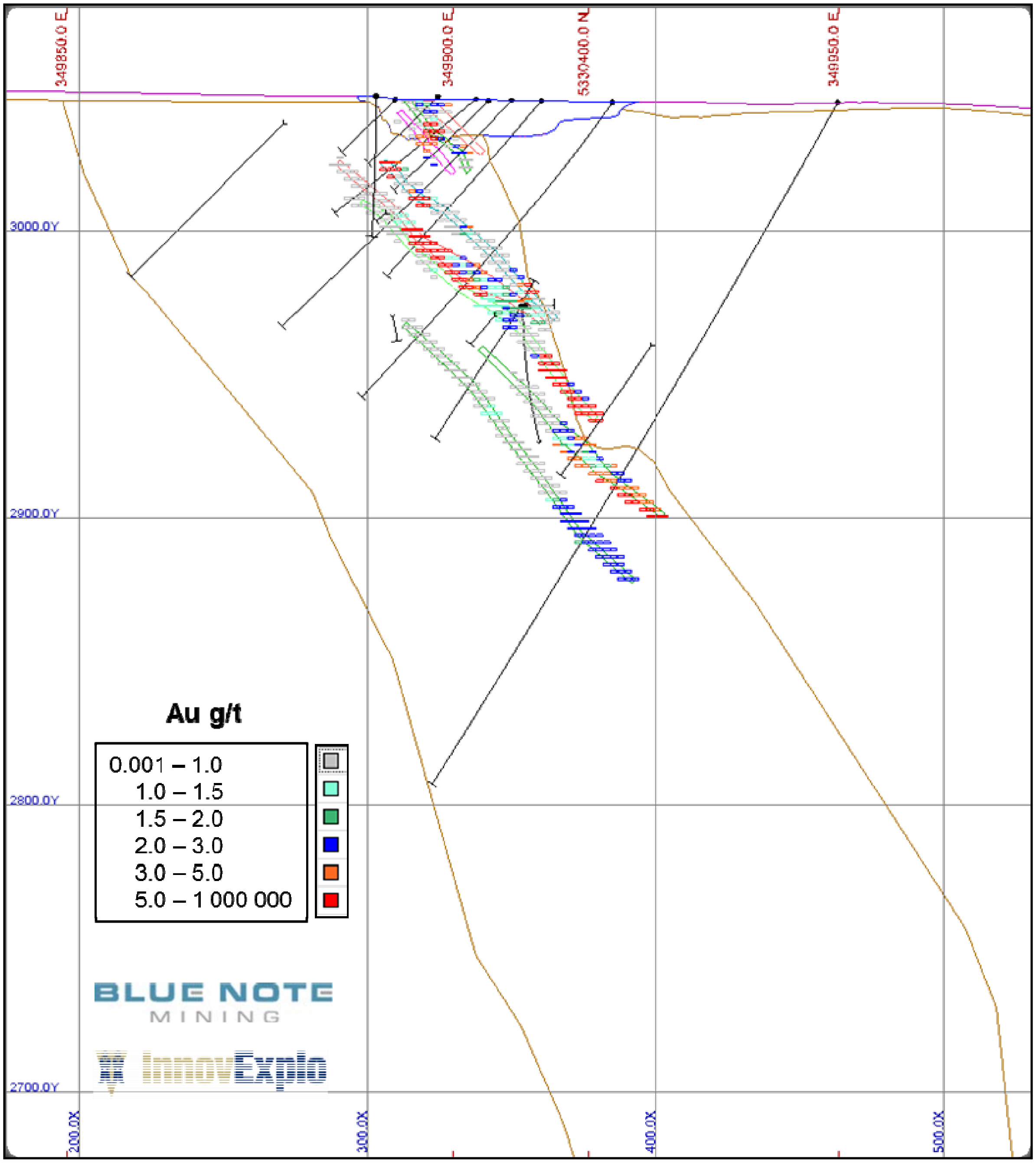


Figure 14.6 – Measured, Indicated and Inferred Resources in the Croinor deposit (Section 200E)

14.4 Discussion

The interpolation method selected for the Mineral Resource Estimate herein was inverse distance power six (1/D6) in order to facilitate the comparison with the previous estimate of O'Dowd (2009), which used the polygonal cross-sectional method. Although the choice of a higher power value makes the two methods more similar, it does not make them equivalent.

Table 14.9 compares the two mineral resource estimates using a cut-off grade of 5 g/t Au.

Table 14.9 – Comparison of the 2011 and 2009 mineral resource estimates for the Croinor deposit

	Measured + Indicated Resources			Inferred Resources		
Cut-off > 5 g/t Au	Tonnes	Au g/t	Oz Au	Tonnes	Au g/t	Oz Au
2011	506 700	10.66	173 700	102 400	10.90	35 900
2009	814 200	9.11	238 400	n.a	n.a	n.a
Difference	-307 500	1.55	-64 700	102 400	10.90	35 900

After the resource estimation by O'Dowd in 2009, Blue Note Mining drilled 65 drill holes for a total of 15,390 m. In order to quantify the effect of these holes on the mineral resources, InnovExplo performed a second calculation of the block model without the 2010-2011 drill holes. A total of 47 drill holes and 2,444 assays were removed from the database and the block model was recalculated using the same 3D solids and the same parameters.

The changes in resource estimation methodology and geological interpretation for the current 2011 Mineral Resource Estimate resulted in a decrease of 307,500 tonnes in the Measured+Indicated category compared to the 2009 estimate. On the other hand, the addition of data and results from the 2010 and 2011 diamond drilling programs increased Indicated resources by 16,209 tonnes, and Inferred resources by 37,990 tonnes.

This exercise demonstrates that the 2010-2011 drilling programs had a positive impact on the mineral resources, and that the decrease in Mineral Resources is mainly caused by the use of a different estimation method.

Inferred Resources have been calculated for the current Mineral Resource Estimate, but were not calculated in the previous estimate (O'Dowd, 2009). This was made possible by the additional drilling in 2010 and 2011 along the lateral extensions and at depth, demonstrating the continuity of mineralized zones outside the area of known resources. This provided sufficient information to establish the geological and grade continuities of these zones. Inferred resources in the Croinor deposit are all in the immediate vicinity of the Indicated resources. The majority form a fringe around the Indicated resources, up to a maximum distance of 70 m, in which there are not enough drill hole intersects to categorize the resources as Indicated. It should be relatively easy to convert all or parts of these Inferred resources to Indicated by adding definition drilling.

15.0 MINERAL RESERVE ESTIMATE

Mineral reserves were classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Reserves for the current Project incorporate appropriate mining dilution and mining recovery allowances for the selected mining method.

In order to determine the resources to be converted to reserves, we used the Mineable Shape Optimizer (MSO), a Datamine software application. According to specified stope parameters, MSO generates individual stope shapes from the block model.

Two mining methods appear to be most convenient for the Croinor deposit: long-hole retreat and room and pillar. In order to select the most appropriate mining method, two MSO runs were completed on the block model using the following parameters for both methods. A small block size was selected in order to better reflect the narrow vein nature of the deposit.

Long-hole mining method:

- Cut-off grade value: 3.7 g/t;
- Minimum mining width of 1.8 m (stope thickness);
- Sublevel interval of 13 m (vertical height);
- Spacing interval of 5 m (stope length);
- Mining dilution of 0.4 m on the hanging wall and 0.2 m on the footwall;
- Minimum slope wall angle of 45 degrees.

Room and pillar mining method:

- Cut-off grade value: 5.4 g/t;
- Minimum mining width of 1.8 m (stope thickness);
- Maximum mining width of 3 m (stope thickness);
- Spacing interval of 5 m x 5 m (stope size along strike);
- Maximum slope wall angle of 45 degrees.

Table 15.1 presents the estimated Proven and Probable reserves, which total 120,883 oz after applying the appropriate mining recovery and dilution factor for the selected method.

Table 15.1 – Mineral Reserve Estimate

Category	tonnes	g/t	ounces
Proven	68,849	6.23	13,789
Probable	498,023	6.69	107,094
Total reserve	566,872	6.64	120,883

The current study reports a lower tonnage and grade, resulting in a lower number of ounces, compared to the 2010 prefeasibility study. The lower tonnage is due to changes in the resource estimation methodology, the geological interpretation, and the resource category criteria. The lower grade is due to an increase in the application of the long-hole mining method. In the current mine plan, 75% of the resources would be mined using the long-hole method as opposed to 20% in the 2010 prefeasibility study. The long-hole method was applied as much as possible due to the lower mining cost and higher productivity.

15.1 Ore Recovery and Dilution

The ore recovery and dilution factor applied in the mining plan and reserve calculations were based on a rock geomechanics study and on common factors applied to the selected method.

In the long-hole method, as a first step, the small stope units generated by the MSO were grouped into larger stopes. Pillar locations were determined according to guidelines provided by the geomechanical evaluation. A 95% recovery factor was then applied to the remaining tonnage. A dilution of 0.6 metre was initially included among the MSO parameters. Once compiled, the resulting overall stope dilution was 24%. In order to remain conservative, a 6% dilution factor was added to bring the overall dilution factor to 30% for the long-hole stopes. The resulting average mining width is 4.0 metres, including 1.2 metres of dilution. The dilution grade was set at 0.0 g/t Au.

The room and pillar stopes were evaluated using a recovery factor of 85%. In cases where the stope dimension was smaller or equal to the stable size determined by the geomechanical study, a 100% recovery factor was applied. A dilution factor of 5% was applied.

15.2 Cut-off Grade

The estimated cut-off grade was calculated using a metal price of $1,205.52 at an exchange rate of 1.07. This metal price reflected the 3-year average metal prices as of October 31 2011, the date the stope shapes were generated. The remaining parameters used in the cut-off grade estimation are presented in the following table.

Table 15.2 – Cut-off grade parameters

	Long-hole	Room and pillar
Operating Cost	$150.00/t	$203.00/t
Mint cost	$5.00 /oz	$ 5.00/oz
Mill recovery	97.5%	97.5%
Mining dilution	Included in MSO parameters	5.0%

InnovExplo designed the underground mine plan based on the current Mineral Resource (refer to section 14). The proposed approach uses sublevel long-hole retreat and room and pillar, with 75% of the ounces from long-hole and 25% from room and pillar. On the basis of the information presented in sections 16 to 22 of this report, InnovExplo was able to demonstrate the economic viability of the proposed extraction and processing of the Measured and Indicated resources found within the mine plan.

InnovExplo is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that would materially affect the Mineral Reserve Estimate. InnovExplo considers the present prefeasibility study to be reliable and thorough, based on quality data, reasonable hypotheses and parameters compliant with Regulation 43-101 and CIM standards with regard to Mineral Reserve and Resource estimates.

Overall, InnovExplo considers that its basic engineering project meets the requirements of a preliminary feasibility study.

16.0 MINING METHODS

The proposed mining plan of the Croinor Project was prepared using the Measured and Indicated resources presented in section 14. These resources were converted to Proven and Probable reserves based on the parameters described in Section 15. It involves the underground mining of narrow subvertical veins. A large portion of the identified resources present a dip running below 45 degrees. This dip is unfavorable to long-hole mining since the broken ore does not flow easily on the footwall. It is also unfavorable for room and pillar as the dip makes it hard for workers to travel in the stope with the equipment and the material. However, in recent years, the introduction of electronic detonators has yielded better results in controlling the blasting outcome, leading to better mining recovery of material in the long-hole stope at a footwall angle of 45 degrees or less.

The mining plan for the Croinor Project comprises a combination of conventional and mechanized mining. The approach in this study has been to force the application of long-hole mining by adding dilution to ensure a minimum footwall angle of 45 degrees. When this approach was not convenient, room and pillar mining was selected. The use of MSO software made these stope analyses possible by calculating optimized stope shapes according to specified mining parameters.

The ore will be transported to surface using a combination of 3.5-yd and 6-yd scooptrams and 30-ton trucks. Waste material will either be brought to surface or used to backfill mined out stopes when possible.

The deposit will be accessed via a ramp. The existing ramp will be restored to level 125 and a new section will be excavated to access all resources. The production drifts will be accessed via crosscuts connecting to the ramp. A small portion of the resources will be mined with captive methods, however the haulage will always be mechanized.

16.1 Geotechnical Evaluation

This rock mechanics evaluation was prepared by Jane Alcott, MASc, P.Eng. (Ontario), eng. (Québec), an engineer with InnovExplo.

The following geomechanical assessment and design recommendations are based entirely on the review of documents provided by Golder Associates Limited (Golder, 2004; Golder, 2009; Golder, 2011). The majority of the information was taken from the crown pillar stability assessment (Golder, 2011). The data in this report pertains to a depth range of 0-100 m below surface. The values reported by Golder (2004; 2009; 2011) have not been verified.

16.1.1 Structural data

Structural data (Golder, 2011) have been compiled from open pit mapping data (Golder, 2004) and oriented core data (Golder, 2011). These are summarized in Table 16.1.

Table 16.1 – Major and minor structural sets at the Croinor mine (Golder, 2011)

Set	Dip / Dip Direction	Description
FO1	67°/010°	**Major** foliation set
J1	43°/014°	**Major** joint set
J2	72°/314°	Major joint set based on open pit mapping, minor based on oriented core data
J2a	40°/339°	**Major** joint set
J3	74°/044°	Joint based on open pit mapping
J3a	46°/049°	Minor joint set
J4	09°/121°	Minor set based on oriented core data
J5	47°/253°, 54°/098°	Minor sets based on open pit mapping

16.1.2 Geomechanical classification

The geomechanical domains and their descriptions (Golder, 2011) are summarized in Table 16.2.

Table 16.2 – Croinor mine geotechnical domains and descriptions (Golder, 2011)

Geotechnical Domains	Description
Tuff-Weathered	V1Tu, V2Tu, V3Tu, V2-V3Tu units between depths of 0 to 10 m below the bedrock surface; increased fracturing and weathering
IJ2-Weathered	All diorite units between depths of 0 to 10 m below the bedrock surface; increased fracturing and weathering
Tuff	V1Tu, V2Tu, V3Tu, V2-V3Tu units below 10 m depth
IJ2	All diorite units below 10 m depth
IJ2-Vein	Quartz veins within diorite units
Fracture Zone	Discrete fracture zones located below 10 m depth

The fracture zone domain identified in Golder (2011) does not correspond to the stoping areas or to the adjacent hanging and foot walls. Accordingly, the fracture zones have not been considered for the stope dimensioning analysis.

The typical and lower bound values of Q' and RMR1976, as determined by Golder (2011), are summarized in Table 16.3.

Table 16.3 – Croinor mine Q' and RMR1976 values (Golder, 2011)

Geotechnical Domains	Q'		RMR1976	
	Typical	Lower Bound	Typical	Lower Bound
Tuff-Weathered	4	4	54	46
IJ2-Weathered	5	5	55	54
Tuff	20	10	66	60
IJ2	16	9	67	63
IJ2-Vein	20	12	68	58
Fracture Zone	5	5	52	52

The unconfined compressive strength (UCS) values as determined by Golder (2011) are summarized in Table 16.4. For this report, the intact UCS (σ_{ci}H-B) values have been used.

Table 16.4 – Croinor mine unconfined compressive values (Golder, 2011)

Geotechnical Domains	UCS (MPa)†	UCSeq (MPa)*	σ_{ci}H-B (MPa)**
Tuff-Weathered		70	70
IJ2-Weathered		70	70
Tuff	68 ± 35	65	84
IJ2	86 ± 32	74	75
IJ2-Vein	72 ± 11	80	72

† *Laboratory testing (Queen's University, 2010 in Golder, 2011)*
** Equivalent UCS values estimated from Point Load Testing (Golder, 2011).*
*** Intact UCS determined from RocLab data analysis (Golder, 2011).*

16.1.3 Stope dimensions

Golder (2011) recommended the crown pillar thicknesses presented in Table 16.5. Based on these thicknesses, it is assumed that no stoping will be carried out within the weathered geotechnical domains (i.e., 0-10 m below the bedrock surface).

Table 16.5 – Recommended crown pillar thicknesses for the Croinor mine (Golder, 2011)

Stope Dip	Method	Crown Pillar Thickness (m)
< 40° - 45°	Room and Pillar	20 m
45° - 55°	Long-hole	20 m
≥ 55°	Long-hole	20-25 m

This study considers that all excavations are located in the IJ2 (diorite units), IJ2-Vein and Tuff units.

16.1.4 Long-hole stopes

Veins dipping at 45° to 60° are assumed to be mined by long-hole methods. The long-hole stopes will be mined from 3-metre-high sublevels, spaced at 15-metre intervals, along dip. It is assumed that stopes will not be backfilled and that pillars will be left between panels and mining horizons.

Long-hole stope dimensions were analyzed using the Mathews-Potvin stability graph method. The analysis assumes:

- Stopes are located in the IJ2, IJ2-Vein or Tuff geotechnical domains;
- Stopes dip at 45° to 60°;
- Lower bound Q' values = 9 (IJ2) – 10 (Tuff) – 12 (IJ2-Vein); and
- HW parallel joint set (45° – J1, 60° – FO1).

The hanging wall (HW) stability graph parameters for the Croinor mine are summarized in Table 16.6.

Table 16.6 – Mathews-Potvin stability graph parameters for the Croinor mine

Stope Dip	Method	Crown Pillar Thickness (m)
< 40° - 45°	Room and pillar	20 m
45° - 55°	Long-hole	20 m
≥ 55°	Long-hole	20 – 25 m

The stope/vein dip has a significant impact on HW stability. As the dip becomes steeper, the stable panel dimensions increase (i.e., larger hydraulic radius, HR). The following figures and tables present the stability graph analyses for the 45° (Fig. 16.1 & Table 16.7) and 60° scenarios (Fig. 16.2 & Table 16.8).

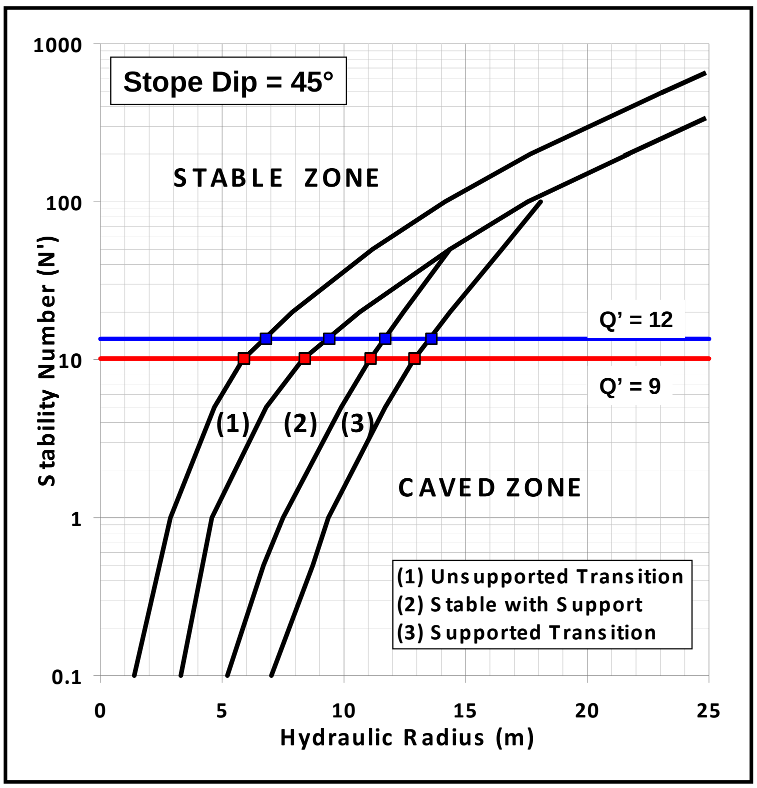


Figure 16.1 – Mathews-Potvin stability graph (45° scenario)

Table 16.7 – Interpretation of the Mathews-Potvin stability graph (45° scenario)

	Vein Dip = 45° HW	
Hydraulic Radius (m)	**Q' = 9**	**Q' = 12**
Stable	< 5.9	<6.8
Unsupported Transition	5.9 – 8.4	6.8 – 9.4
Stable with Support	8.4 – 11.1	9.4 – 11.7
Supported Transition	11.1 – 12.9	11.7 – 13.6

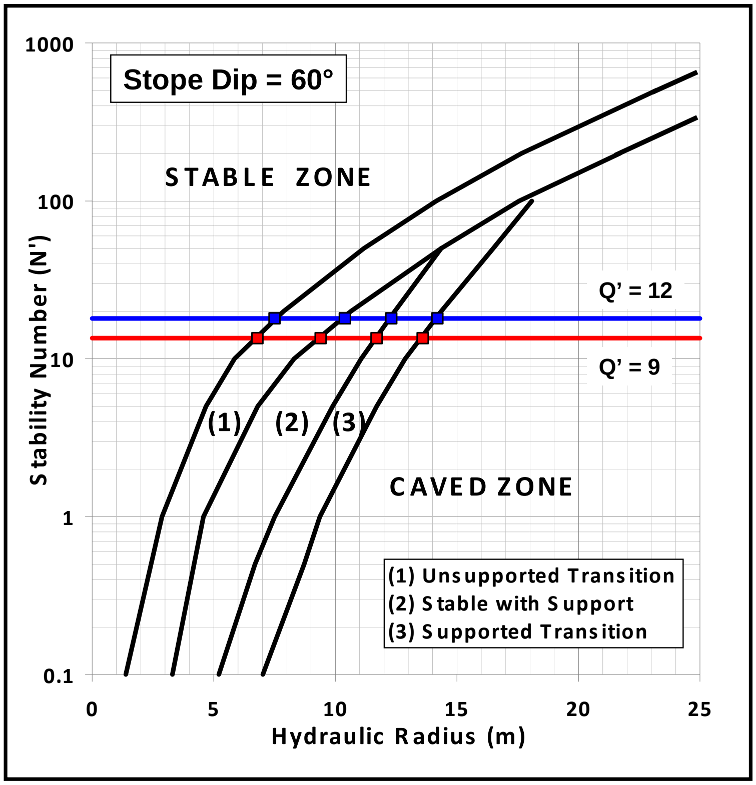


Figure 16.2 – Mathews-Potvin stability graph (60° scenario)

Table 16.8 – Interpretation of the Mathews-Potvin stability graph (60° scenario)

	Vein Dip = 60° HW	
Hydraulic Radius (m)	**Q' = 9**	**Q' = 12**
Stable	<6.8	<7.5
Unsupported Transition	6.8 – 9.4	7.5 – 10.4
Stable with Support	9.4 – 11.7	10.4 – 12.3
Supported Transition	11.7 – 13.6	12.3 – 14.2

The long-hole stope panel dimensions are based on the worst-case scenario (lower bound value: Q' = 9 and stable-without-support criterion). The design criteria yield hydraulic radii of 5.9 m for 45° stopes and 6.8 m for 60° stopes. The sublevel interval was assumed to be fixed at 15 m (along dip). Accordingly, the panel length was adjusted in order to satisfy the design criteria (Table 16.9).

Table 16.9 – Panel length assessment

	Panel Height	Hydraulic Radius (m)				
		L = 25 m	L = 20 m	L = 17 m	L = 16 m	L = 15 m
Configuration #1	36 m	7.4	6.4	5.8	5.5	5.2
Configuration #2	42 m	7.8	6.7	6.1	5.8	5.5

The recommended panel configuration and corresponding hydraulic radius values are presented in the Table 16.10 and Figure 16.3.

Table 16.10 – Proposed long-hole stope configurations

	HW Panel Dimensions	
	45° scenario Design HR_{max} = 5.9	60° scenario Design HR_{max} = 6.8
Configuration #1 • 15 m long downholes drilled from #1 & #2 Subs • 5 m wide pillar between panels	36 m (H) x 16 m (L) (HR = 5.5 m)	36 m (H) x 16 m (L) (HR = 6.4 m)
Configuration #2 • 15 m long downholes drilled from #1 & #2 Subs • 7 m long upholes drilled from #2 Sub • 5 m wide pillar between panels	42 m (H) x 16 m (L) (HR = 5.8 m)	42 m (H) x 16 m (L) (HR = 6.7 m)

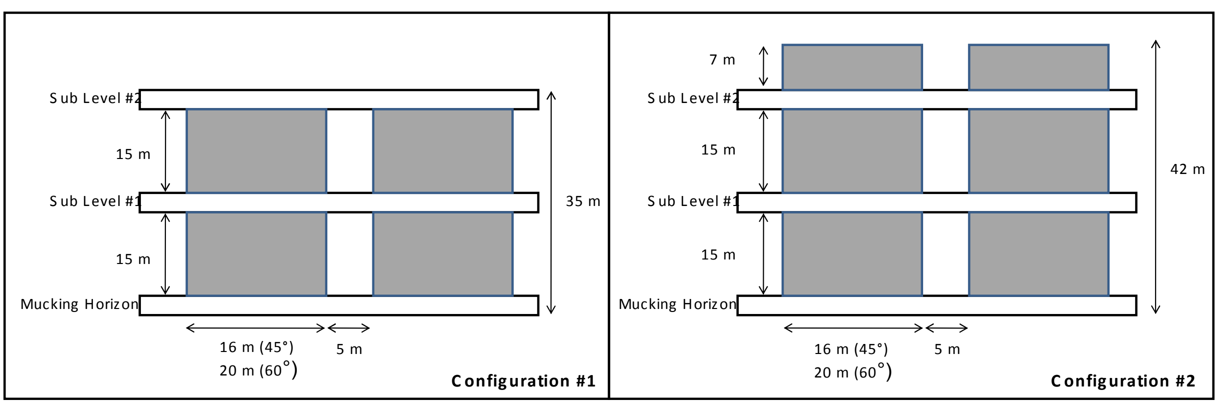


Figure 16.3 – Proposed long-hole stope configurations (longsection view). All height measurements are stated along dip.

The following elements are not considered in the Mathews Potvin stability graph analysis:

1. Variations in geometry due to changes in vein orientation and/or dip (i.e., undulations). Stopes are assumed to have a regular rectangular geometry and any variations in geometry will negatively impact stability.
2. Undercutting of stope hanging walls and footwalls due to oversized development. Undercutting may create an additional release surface for slabbing. Cabling may be required for wall control in these situations.

This study has not considered the dimensioning and stability of the pillars between long-hole panels.

16.1.5 Room and pillar stopes

The proposed room and pillar stope configuration is based on typical industry practices for currently operating mines in deposits with similar vein geometry.

The typical mining height will vary from 1.8 m (minimum) to 3.0 m (maximum). The 3.0 m maximum reflects safety and operational constraints. For mineralized zones in excess of 3.0 m, either selective mining or backfilling and multiple cut methods should be considered. The proposed room and pillar dimensions are summarized in Table 16.11 and Figure 16.4.

Table 16.11 – Proposed room and pillar stope configuration

Rooms	6.5 m wide
Pillars	3 m wide x 5 m long x 1.8 – 3.0 m high
Maximum Span	8.85 m (intersection)
Mining Height	1.8 – 3.0 m

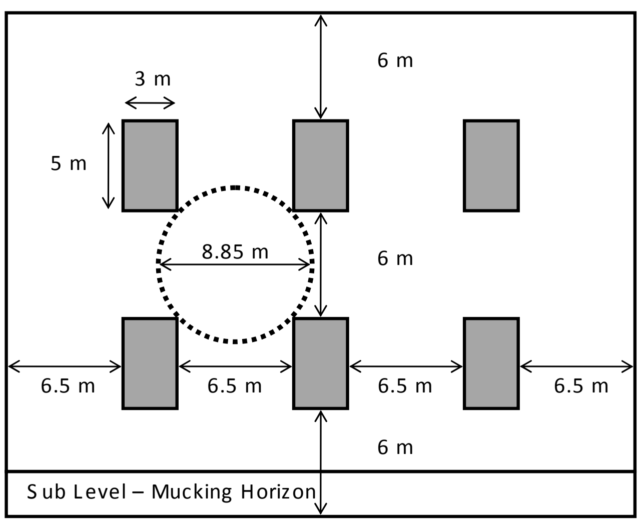


Figure 16.4 – Proposed room and pillar stope configuration (plan view)

The stability of the excavations was analyzed using the Underground Entry-Type Excavation Span Design method (Lang, 1994; Wang et al., 2000). This method empirically determines the stable span for an excavation supported by primary support. The stability of a given excavation span is assessed based on rock mass classification (RMR), to which corrections are applied for the presence of unfavourable structures (flat-lying joints and dykes) and the induced stress conditions (Table 16.12).

Table 16.12 – Entry-Type Excavation Span Design correction factors (Lang, 1994).

Condition	Correction Factor
Flat lying joints (<30º)	- 10%
Dyke or fault (0-60º)	- 10%
Low burst potential (σ_1/σ_c = 0.3 – 0.4)	- 10%
Moderate burst potential (σ_1/σ_c = 0.4 – 0.5)	- 15%
High burst potential (σ_1/σ_c > 0.5)	- 20%

Primary support is defined as 6 ft bolts on a 4 ft x 4 ft pattern. The method does not consider discrete wedges or excavations that fall in the *potentially unstable* or *unstable* categories but which may become stable with the addition of secondary support (e.g., cablebolts, shotcrete, etc). Stability refers to short-term stability, which is defined as a period of less than three months. A "stable" excavation is one where the primary support is sufficient to maintain the excavation; "potentially unstable" refers to an excavation where additional or secondary is required; and "unstable" refers to an excavation where failure has occurred.

The proposed room and pillar stope configuration has a maximum span of 8.85 m (~9 m). Figure 16.5 shows that the lower bound RMR values (i.e., worst case scenario) correspond to maximum spans of 8 to 10.5 m. Therefore, secondary ground support is recommended for the maximum span intersections. In the intersection, 10 ft (3 m) resin-grouted cables (i.e., Standlok cables) will be installed on a 6 m x 6 m diamond around a central cable (Fig. 16.6).

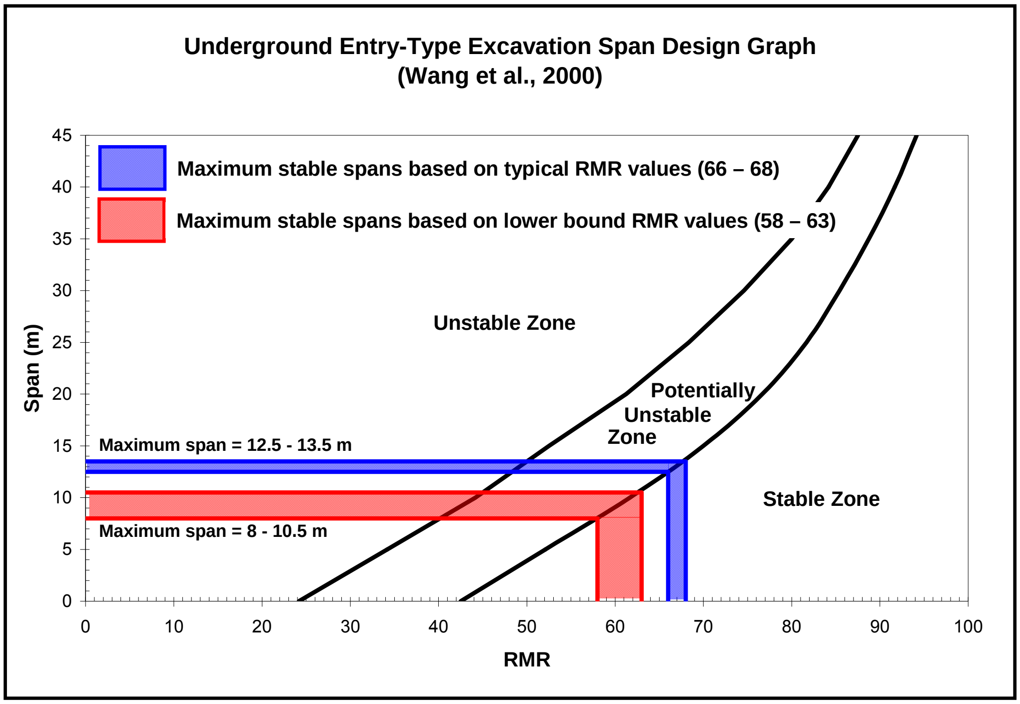


Figure 16.5 – Underground Entry-Type Excavation Span Design analysis

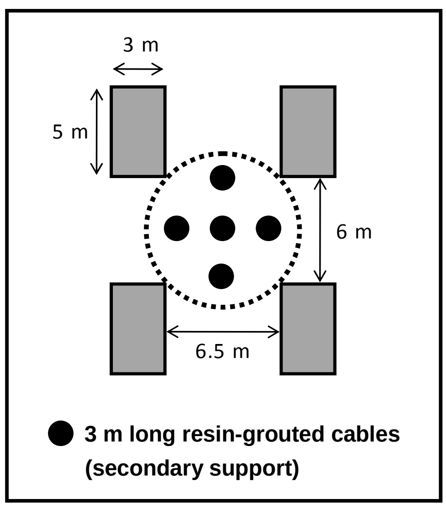


Figure 16.6 – Proposed secondary support pattern for room and pillar stope intersections (plan view)

16.1.6 Typical ground support patterns

These preliminary ground support recommendations are based on standard industry practices. More detailed recommendations will require additional information regarding joint spacing and continuity.

Based on Farmer and Shelton (1983), the following proposed bolt lengths for the back are based on the excavation span (bolt length = 0.3 span; Table 16.13).

Table 16.13 – Bolt length as a function of span

Bolt length*	Maximum Span
4 ft (1.2 m)	13.1 ft (4 m)
5 ft (1.5 m)	16.5 ft (5 m)
6 ft (1.8 m)	19.7 ft (6 m)
7 ft (2.1 m)	23.0 ft (7 m)
8 ft (2.4 m)	26.2 ft (8 m)

* Bolt length indicates the length installed within the rock and excludes any threads or bar outside of the drill hole.

The standard support (Fig. 16.7) consists of:

- Back: rock bolts (length based on excavation span) on a 1.2 m x 1.2 m (4' x 4') pattern with screen as required (based on excavation height);
- Wall: rows (number of rows based on excavation height) of rock bolts (length = 1.2-1.5 m) on a 1.2 m x 1.2 m (4' x 4') pattern.

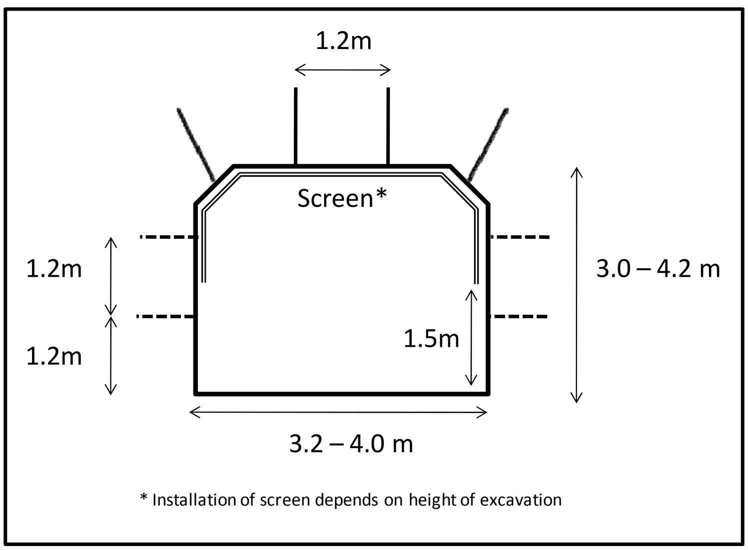


Figure 16.7 – Standard support pattern.

Screening of the back to 1.5 m above base-of-rail (BOR) is recommended for all excavations 3.5 m or higher. The screen is intended as a safety measure where back height will make routine inspections and scaling more difficult. Once additional structural and rock quality information are available, it will be possible to optimize the ground support standards (e.g., possibly eliminate wall bolts for excavations less than 3 m high).

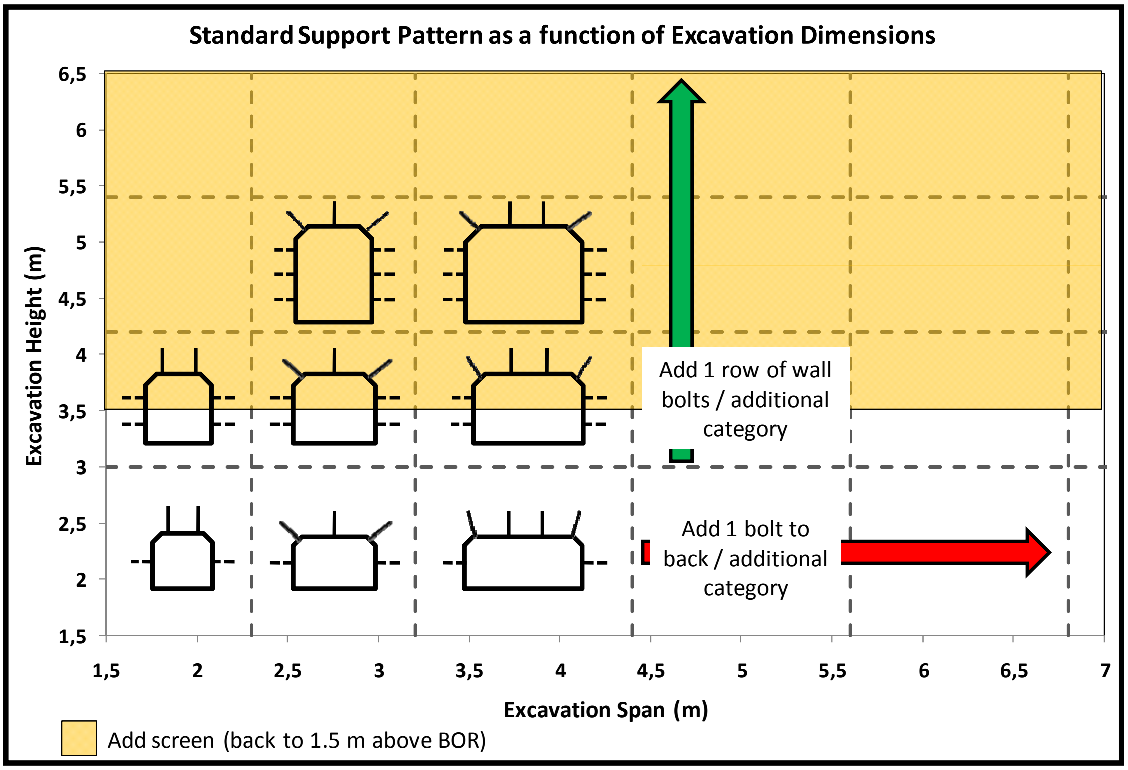


Figure 16.8-Ground support guidelines as a function of excavation span and height
(Note: The transitions between bolting patterns are suggested guidelines and will vary slightly depending on the degree of arching of the excavation back.)

16.1.7 Stress

The in-situ stresses employed by Golder (2011) for the Croinor mine have been adopted for this analysis. At the maximum mining depth of 305 m and with a rock mass strength of 72-84 MPa, induced stresses are likely to cause some spalling around excavations. The ratio of induced stress to strength for development excavations at 305 m of depth is anticipated to vary from 0.47 to 0.55. These values exceed the threshold of the induced stress to strength ratio corresponding to the onset of stress-driven cracking (0.33-0.4; Martin, 1993). The stress analysis and assumptions are summarized in Table 16.14.

Table 16.14 – Induced stress analysis

Parameter	Depth = 305 m	Source
Minor principal stress (α_3)	9.6 MPa	Golder (2011)
Major principal stress (α_1)	25 MPa	Golder (2011)
k_o (α_1/ α_3)	2.6	Golder (2011)
Maximum induced stress (α_1)	40 MPa	Square (4 m x 4 m) excavation assumed
Rock mass strength (α_c)	72 - 84 MPa	Golder (2011)
Induced stress: strength (α_1/ α_c)	0.47 – 0.55	

16.1.8 Recommendation for Future Work

Values taken from the crown pillar study (Golder, 2011) have been assumed to be valid for the surface to a depth of 100 m. In general, one would expect rock mass conditions to improve with depth. It is recommended that assumptions (rock mass strength, classification and structure) be verified once access to underground becomes available. Detailed analysis (3D numerical modeling) of the proposed stope configuration is recommended in order to proceed to a feasibility study. Such an analysis would examine the stability of the pillars between the long-hole panels and within room and pillar stopes, as well as examining sequencing and stope interactions.

16.2 Mining Method

As mentioned earlier, two mining methods are proposed to accommodate the geometry of the mineralization: sublevel long-hole retreat and conventional room and pillar.

16.2.1 Long-hole method

The long-hole stopes will be mined from 3-m-high sublevels at 13-m vertical intervals. For stope heights exceeding 13 m, the maximum stope panel was defined as 15 m long. It is assumed that stopes will not be backfilled and 5-m pillars are left between panels and mining horizons. For stope heights of 13 m and less, longer stope panels were considered on a case by case basis.

The method consists of drilling and blasting 54-mm-diameter holes in a pattern parallel to the walls. Holes are drilled upward or downward depending on the context.

The development sequence consists of accessing the mineralized zone and excavating a level cut in the mineralized zone. The mining sequence will require the excavation of a raise opening, which is either developed as a conventional raise or as a drop raise when a top access is available. Once development is completed, the mineralized zone is surveyed with precision for the preparation of the drilling and blasting pattern.

Blasting will be done using electronic detonators to allow for a larger blast, thereby reducing the number of blasts per stope. This method has been proven to reduce dilution and be more productive by eliminating the number of moves for the workers. Rock fragmentation is also greatly improved with electronic detonators and this factor is of great importance to improve recovery in long-hole mining at lower dips. The broken material is mucked out using a remote system.

16.2.2 Room and pillar

In some areas, room and pillar was an economically better choice than long-hole. The proposed room and pillar stope configuration is based on typical industry practices for currently operating mines in deposits with similar vein geometry. The typical mining height will vary from 1.8 m (minimum) to 3.0 m (maximum).

The room and pillar mining method entails the excavation of a series of "rooms" following the vein, leaving "pillars" or columns of rock in place to help support the mine roof. In conventional room and pillar mining, drilling is achieved using hand-held drill equipment and holes are loaded with explosives. Bolts are then installed in the mine roof to ensure the roof is properly supported. The broken rock is scraped to either a raise or a drawpoint where it is taken with a scooptram to be hauled to surface.

16.3 Dewatering

Prior to any underground rehabilitation or development work, the existing mine infrastructure will have to be dewatered. The total estimated volume of water present in the mine and the two open pits is 504,080 m^3.

- East pit: 270,700 m^3
- West pit: 211,900 m^3
- Underground opening: 21,482 m^3

Water will be pumped from the East and West pits. Initial dewatering is expected to be carried out at a rate of 5,500 m^3/day, resulting in an approximate dewatering period of 92 days. Mine water will be pumped and processed through Envirotubes to clarify the water and collect the mine sludge. Discharge from the Envirotubes will be directed to the existing settling pond (60 m x 60 m). The pond effluent will be monitored and managed in accordance with MMER and Directive 019 requirements.

16.4 Underground Mine Design

16.4.1 Primary development

The mine design incorporated in this study takes advantage of existing infrastructure when possible and appropriate. Underground access is planned using a single ramp developed from the existing ramp at level 125. The existing ramp will be extended to a final depth of 260 metres. The ramp will connect to the existing levels and to the new sublevels.

The sublevels are developed at 13-m vertical intervals. The mine plan requires that portions of the existing levels be enlarged. Each level or sublevel is accessed using 4m x 4m crosscuts from the main ramp or secondary ramp. The opening dimensions are planned as follows:

- Ramp: 4m x 4.5m
- Crosscut: 4m x 4m
- Production drift: 3.7m x 3m
- Captive sublevel: 2.4m x 3m

The ore and waste will be hauled by LHDs from the production area to either a remuck bay or to loading points that will be excavated close to the ramp. The material will then be loaded in 30-tonne trucks to be hauled directly to surface.

The shaft will be rehabilitated to level 500 and will serve as the main emergency egress as well as for ventilation purposes. Short ventilation raises will be required as development progresses to accommodate the various production areas.

16.4.2 Secondary development

Two permanent refuge stations are planned, one on the 250 level and a second one on level 625. These refuge stations will be 4 m x 12 m x 3 m. There will be two main sump stations to collect water inflow. The first one is planned on level 125 and the second one on level 625. The water will be pumped in stages to the surface.

16.4.3 Stope development

In general, the long-hole stope level or sublevel will be excavated directly in the mineralized zones. In some cases, to allow for a better mine sequencing, bypass drifts and drawpoints have been considered to preserve access to future resources. The long-hole stopes will be mined by retreating and the broken material will be collected in the lower level or sublevel. The room and pillar stopes are accessed with crosscuts that normally serve as a draw point for the ore. When the stopes are located at an elevation higher than the level, short raises will be developed and separated into two compartments by a timber wall, one side to serve as a manway and the other to be used as a chute.

16.5 Mine Sequence

Mine development will be accelerated in the first two years of the project to provide a degree of flexibility in terms of access, which should facilitate scheduling during the production period. The development sequence will ensure that many stopes are available for mining at a number of different locations at any given time. However, some of the stopes can only be mined at the end of the mine life since they are located directly over or under the level, therefore preventing any further access on that level when mined.

16.6 Mining Rate

The expected production rate will start at an average of 475 t/day in the six months following preproduction, and will slowly ramp up to an average of 675 t/day in Year 4. The overall project mine life is expected to be approximately 5 years, including an 18-month pre-production period.

In the opinion of author Sylvie Poirier, Eng., the mine plan should be achievable given the flexibility and number of available working places. Table 16.15 summarizes the yearly tonnage distribution according to the mine plan.

Table 16.15 – Mine plan tonnage distribution

	Year 1	Year 2		Year 3	Year 4	Year 5	Total
	Pre-prod	Pre-Prod	Prod	Month 25-36	Month 36-48	Month 49-58	
Long hole (t)	0	19 110	53 455	81 966	154 299	112 746	**421 576**
Grade (g/t)	0.00	6.05	7.70	5.17	6.18	5.48	**5.98**
Development(t)	4 914	7 623	6 552	20 686	12 726	0	**52 501**
Grade (g/t)	5.63	6.40	5.73	5.42	5.75	0.00	**5.70**
Room and pillar (t)	0	4 554	3 522	33 319	34 584	16 374	**92 352**
Grade (g/t)	0.00	9.06	6.66	14	8.66	7.41	**10.19**
Mill tonnage(mt)	4 914	31 287	63 529	135 971	201 609	129 120	**566 429**
Grade (g/t)	5.63	6.58	7.44	7.29	6.58	5.73	**6.64**

16.7 Mine Plan Schedule Criteria

Contractors will be used for all mine development, mine production and ore haulage activities. A small staff will be hired to provide technical and administrative support and direction to the contractors.

The design criteria used to develop the mine plan are as follows:

- Overall drift development or drift enlargement: 7 m/day;
- Sublevel development: 3.8 m/day;
- Raises: 1.8 m/shift.

16.8 Development and Production Schedule

InnovExplo developed a preliminary development and production schedule based on the existing underground development and mineral resources discussed in Section 14. Development and production activities are based on a schedule of two 10-hour shifts, 12 shifts a week, for an average of 25 days per month (300 days/year). The underground mine design provides for a 5-year mine plan producing 566,429 tonnes of ore grading 6.64 g/t. Using a mill recovery of 97.5%, this translates to 117,956 oz of gold produced during this period.

According to the mine plan, 84% of the tonnage will be from long-hole and sub level development ant that and 16% of the tonnage will be mined by room and pillar.

The mining plan includes all development required to access and mine the mineralized zones. Estimated development quantities are presented in Table 16.16 and the production schedule is presented in Table 16.17. Figure 16.9 gives a general overview of the total development and Figure 16.10 illustrates the minable zones.

The reserves included in the mining plan were obtained by applying the mining recovery and dilution factor presented in Section 15.0.

Table 16.16 – Croinor mine development quantities

	Year 1	Year 2		Year 3	Year 4	Year 5	Total
	Pre-prod	Pre-Prod	Prod	Month 25-36	Month 36-48	Month 49-58	
Development CAPEX							
Development ramp 4mx4.5m (m)	723	36	736	345	0		**1 840**
Secondary ramp 4mx4m (m)	0	0	192	463	100		**755**
Development drift 4mx4m (m)	368	635	281	1 227	425		**2 936**
Development drift 3mx3.7m (m)	0	0	0	196	0		**196**
Drift enlargement 4mx4m (m)	80	0	0	0	0		**80**
Drift enlargement 3mx3.7m (m)							**0**
Raise(m)				288			**288**
Development OPEX							
Development drift 3mx3.7m (m)	181	1 110	1 124	3 192	2 133		**7 740**
Sub level drift 3mx2.4m (m)	433	0	0	255	773		**1 461**
Drift enlargement 3mx3.7m (m)	65	390	149	472	64		**1 140**
Raise (m)	26	116	202	123	267	136	**870**

Table 16.17 – Croinor mine production rates

	Year 1	Year 2		Year 3	Year 4	Year 5	Total
	Pre-prod	Pre-Prod	Prod	Month 25-36	Month 36-48	Month 49-58	
Tonnage summary							
Ore (t)	4 914	31 287	63 529	135 971	201 609	129 120	566 429
Waste (t)	52 993	55 966	70 961	161 896	91 996	1 234	435 046
Total (t)	**57 907**	**87 253**	**134 489**	**297 866**	**293 605**	**130 353**	**1 001 475**
Total ore (t/day)		209	424	453	672	574	
Total ore and waste (t/day)		**582**	**897**	**993**	**979**	**579**	**802**

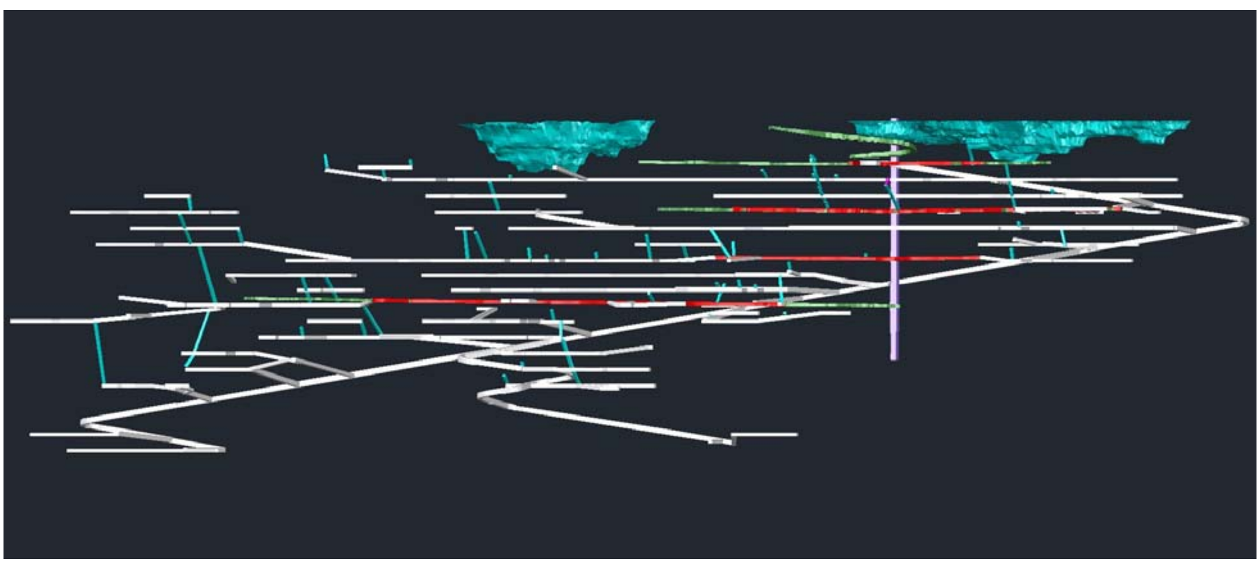


Figure 16.9 – Croinor mine development

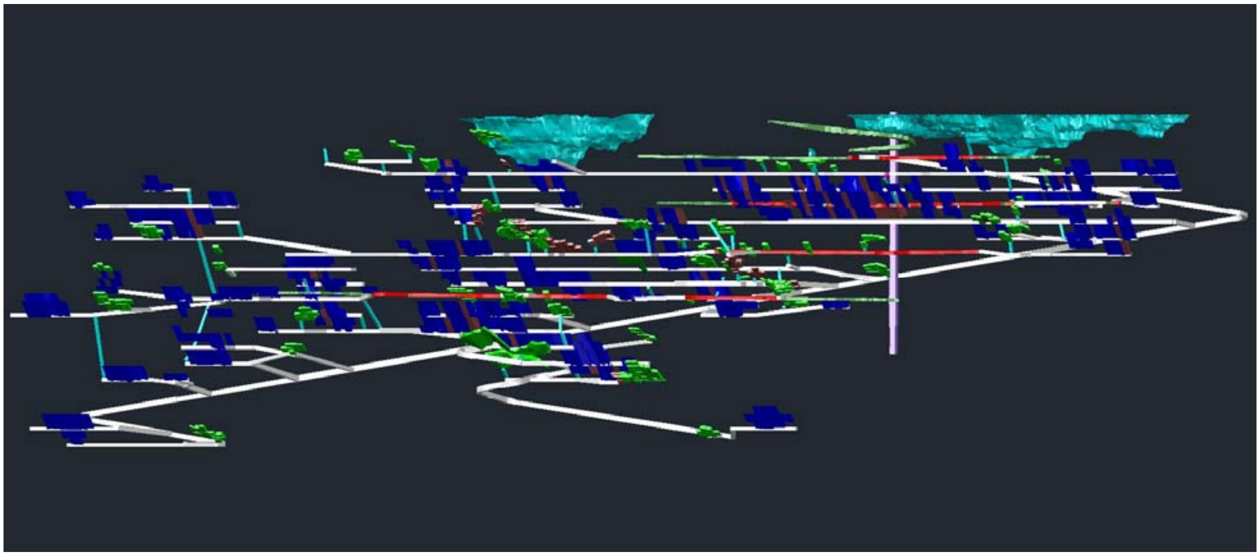


Figure 16.10 – Croinor mine development and reserves

16.9 Equipment Selection and Requirements

This study is based on new equipment that would be acquired by Blue Note Mining through rental agreements. A contractor will provide the Jack leg and slusher required for room and pillar mining as well as the stopper drills required for ground support work. The list of Blue Note Mining equipment is presented in Table 16.18.

Table 16.18 – Equipment list

Equipment	Quantity
Mine truck (30t)	2
Scooptram 6 yd	1
Scooptram 3.5 yd	2
Jumbo 2B	1
Jumbo 1B	1
Scissor lift	2
Tractor (mech and electric)	2
Tractor (development)	2
Tractor (spare)	1
Gear truck	1
Mine mule	4
Electric slusher	Provided by the contractor
Jackleg and stopper drill	Provided by the contractor
Long-hole drill	Provided by the contractor
Surface loader	1
Grader	1
E-series cargo	2
Pick-up	1

16.10 Manpower Requirement

Blue Note Mining will hire its own staff for the project's administrative, technical and surface services. Some positions will be partially staffed at the beginning of pre-production, and will progressively reach the final fully-staffed scenario presented in Table 16.19. The Blue Note Mining staff will be reduced in the last year of production.

Table 16.19 – Blue Note Mining staff list

Description	Quantity
Administration	
Manager	1
Accounting	1
Purchase Agent	1
Clerk	1
Executive secretary	1
Technical Services	
Chief geologist	1
Geologist	1
Chief engineer	1
Engineer	1
Geological technician	3
Mine technician	3
Surface Services	
Dry man	2
Service man	4
Gate keeper	4

The contractor will provide all manpower related to supervision, maintenance and production.

The manpower resource on each working shift used to achieve the mine schedule includes:

- 3 two-man crews for room and pillar;
- 2 long-hole driller;
- 2 long-hole blaster;
- 1 truck driver;
- 3 LHD operators;
- 2 development crews:
    - One crew with a two-boom jumbo for the ramp and level development, and one team with a one-boom jumbo;
    - Each crew consists of 1 jumbo operator and 3 workers for ground support and services.

16.11 Mining Services

16.11.1 Ventilation

The existing shaft will be rehabilitated and used for mine ventilation as well as an emergency escapeway. For the current study, InnovExplo performed a preliminary simulation using Ventsim software to establish a main ventilation network from which secondary fans will bring air to the working area. The required ventilation was established at 95 cubic metres per second (200,000 cfm) and will be provided by a 224 kW (300 hp) main air fan. Fresh air will be heated by two 3224-kW (11-MBtu/hr) capacity propane burner systems and will exhaust via the ramp.

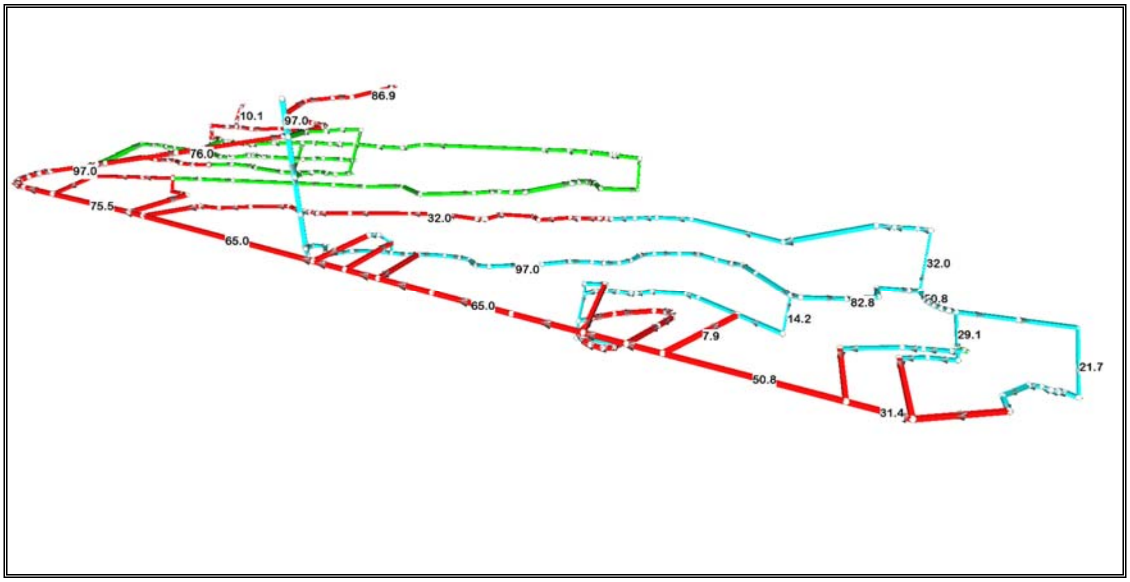


Figure 16.11 – Main ventilation circuit schematic

16.11.2 Dewatering

An innovative dewatering system is proposed for the Croinor Project. It consists of four main pumping stations: one on 125, one on 625, and two on the lowest levels of the mine. Ongoing dewatering during mining operations is estimated at 963 m^3/day based on previous operating data.

The level 125 station includes a modular pumping station at which two pumps capable of pumping suspended abrasive solids will be assembled in series (one operating and one as back-up). The electric motor will be 100 hp and the V-belts will be adjusted for a flow of 2180 m^3/day @ 45 m. The feed tank capacity will be approximately 14,400 liters. At surface, environmentally safe polymers will be added to the water, which will then be clarified in Envirotubes. This type of installation captures 99% of solids, and the clear filtrate can be collected and used for production water or discarded into the environment. The solids remain in the Envirotube bag. When full, the Envirotube container and contents can be disposed of in a landfill, or the solids removed and used as a surface cover when appropriate. This type of system allows for smaller sumps and eliminates underground mud management.

The level 625 station is also a modular type pumping station at which two sets of pumps will be assembled in series (one pair operating and one pair as back-up), for a total of four pumps. The system is capable of pumping dirty water to level 125. The electric motor will be 100 hp and the V-belts will be adjusted for a flow of 2180 m^3/day @ 170 m. The feed tank capacity will be approximately 14,400 liters.

The two smaller pumping modular installations will be set up at the lower locations in the mine to bring water to the level 625 station.

16.11.3 Compressed air

Two 41.8 m^3/min (1,476 cfm), self-enclosed electric compressors will be installed at surface. A network of pipelines will be installed down the shaft and along the ramp and drifts throughout the mine. Compressed air will be provided to various handheld drills and production long-hole rigs, and will also provide emergency air supply to the refuge station. A parallel network complete with pressure-reducing valves will supply water to the underground operations.

16.11.4 Underground power distribution

One (1) 4.16 kV substation at 600/347 V, 750 KVA, will be installed near the shaft. The installation will include two soft-start starters (300 hp) for the main fan and two starters for the pumps. This installation will supply all underground mobile substations.

Underground, a main substation (1000 KVA at 4160 V) is planned on level 375 and another on level 625 to provide power to the underground loads, such as pumps, fans, lunchroom, etc.

For the entire project, two 500 KVA, 4.16 KV at 600-347 V mobile substations are to be installed at different levels to follow production requirements. These substations will be skid-mounted and junction boxes will be available at each level to feed a fixed substation or the mining mobile substation, thereby ensuring all the flexibility required by the mining operations.

17.0 RECOVERY METHODS

GENIVAR was commissioned by InnovExplo and Blue Note Mining to review custom milling options in the context of a prefeasibility study for the Croinor gold project. The review provided a custom milling recommendation based on information provided by the client.

GENIVAR did not perform sampling, assaying or metallurgical work of any kind, but considered the provided data to be from reliable sources.

17.1 Review of Metallurgical Test Results

The first reported metallurgical study on Croinor ore was performed by Lakefield Research in 1988 for Cambior. Results were promising for direct leaching, providing recoveries in the 96.7-97.9% range, with no clear correlation to grind size. Flotation processing yielded slightly lower recoveries, averaging 96.4%, and was also insensitive to grind size in the range tested. It must be emphasized that further gold unit losses for downstream processing (leaching or smelting) would have to be added to this, making the flotation-based route less appealing than direct cyanidation.

In early 2000, metallurgical testwork results (St-Jean, 2001; 2003) again suggested that the ore responds better to direct leaching than sequential flotation-leaching, with recoveries in excess of 94%. The only sequential flotation-leaching test yielded very poor results: an overall recovery of 87.8 % and significantly higher sodium cyanide consumption (twice as much).

The most recent bulk sample milling programs at the Camflo mill, which processed ore from the East and West pits, confirmed that the Croinor ore is readily amenable to conventional processing (Pelletier and Boudrias, 2005), achieving recoveries ranging from 95.3% to 97.9%, with a weighted average of 97%. These recoveries were achieved on ore with significantly lower gold content than what will be processed from the underground mine. According to the expected grade at Croinor, GENIVAR believes a recovery of 97.5% is attainable, and this value was used in the present study.

The available metallurgical results are detailed in tables 13.2 and 13.3. It should be noted that gold recovery usually correlates directly with head grade. Caution is thus advised when comparing the results of the different test programs.

17.2 Analysis and Recommendations

The planned production rate at the Croinor deposit ranges from 475 to 675 tpd over a 5-year period. Only four gold concentrators within a 120 km radius have the capacity to process the Croinor ore: Beacon Gold Mill, Aurbel Gold Mill, Sigma-Lamaque Complex and Camflo Mill. Table 17.1 summarizes their main features.

Table 17.1 – Potential plants for custom milling

Mill	Company	Process	Capacity	Distance
Beacon Gold Mill	Northern Star Mining	leaching/Merrill-Crowe	900 tpd	60 km
Aurbel Gold Mill	Alexis Minerals	flotation & leaching	1,200 tpd	69 km
Sigma-Lamaque Complex	Century Mining	gravity concentration & leaching	2,500 tpd	73 km
Camflo Mill	Richmond Mines	leaching/Merrill-Crowe	1,200 tpd	105 km

Typical flow sheets for direct leaching (Merrill-Crowe) and sequential flotation-leaching (Merrill-Crowe) are depicted in figures 17.1 and 17.2 respectively.

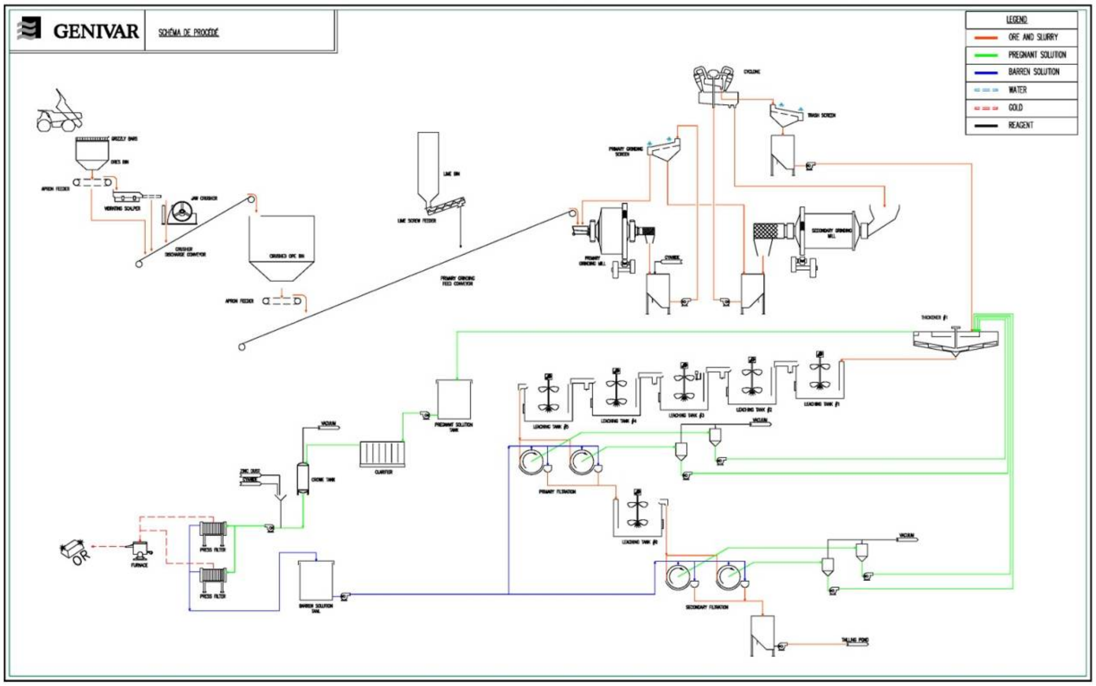


Figure 17.1 – Typical direct leaching (Merrill-Crowe) flow sheet

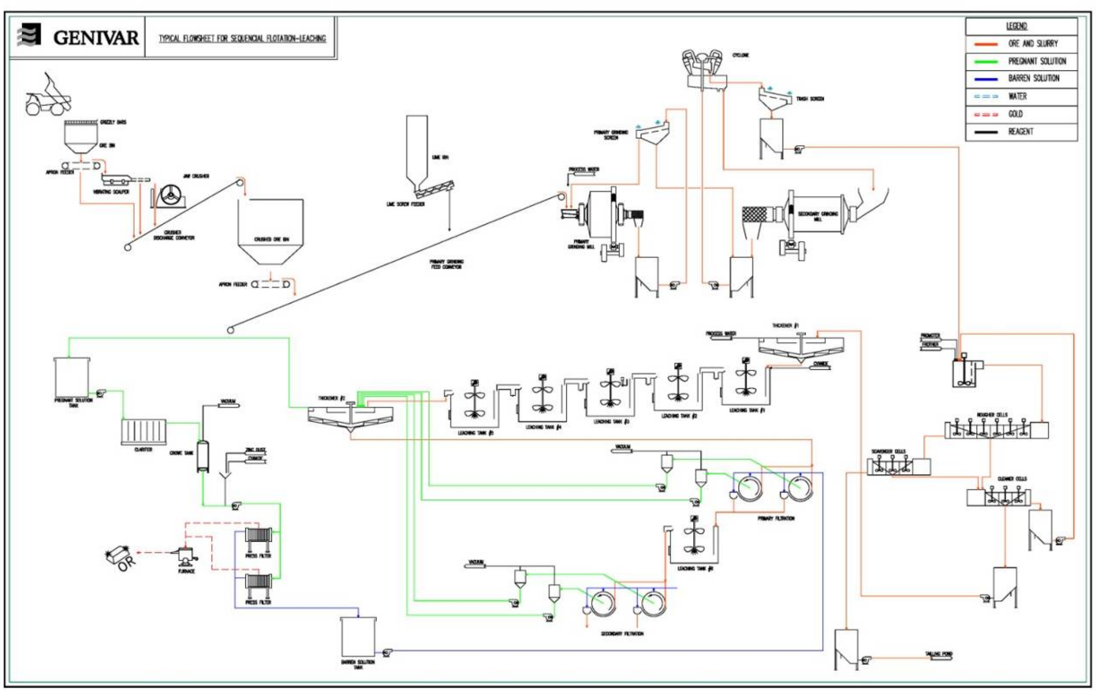


Figure 17.2 – Typical sequential flotation-leaching (Merrill-Crowe) flow sheet

Sigma-Lamaque Complex could technically be an interesting choice since some test results suggested gravity concentration could be beneficial for ore originating from the A5 Zone (St-Jean, 2003). However at 2,500 tpd, the plant capacity would require several weeks of stockpiling for every program. The CIP (carbon-in-pulp) gold recovery process would present another hurdle: the management of loaded carbon inventory in the circuit. In fact, reconciliation of gold recovery in a carbon-based process can be problematic in a custom milling situation involving independent feed sources.

The Aurbel Gold Mill would have been an attractive option from a location standpoint. However, the metallurgical results currently available for the flotation-based flowsheet raise some concerns about potential gold unit losses and higher sodium cyanide consumption.

Beacon Gold Mill has been refurbished and is currently under care and maintenance. The interest of Northern Star Mining for custom milling is unknown, but its proximity, gold recovery process, and capacity (900 tpd) make it an attractive candidate to treat the Croinor ore.

Based on the metallurgical results and above considerations, the Camflo mill may be the best option. In addition to having successfully processed the Croinor ore in the past, thereby mitigating any technical risk, the plant is likely to have availability for new custom feed, and has been in the custom milling business for years. This plan assumes that the ore will be processed at Camflo for the purpose of estimating milling and transportation costs although a final evaluation and decision will be made when the project commences.

It must be emphasized that in addition to conventional milling, the Croinor ore could be a good candidate for pre-sorting at the mine site. Currently available/proven technologies could help significantly reduce the amount of material requiring processing at the custom-milling facilities, thereby reducing transportation and treatment charges. Off-the-shelf processing equipment can be installed with a small footprint and low capital investment. GENIVAR highly recommends considering ore sorting at the mine site prior to conventional milling. In addition, there may be an opportunity to install a gravity separator at the chosen custom mill to reduce operating costs and potentially improve gold recovery.

18.0 PROJECT INFRASTRUCTURE

18.1 Plant and site layout

Figure 18.1 presents the general surface layout, including the proposed location of the required infrastructure for the Croinor mine.

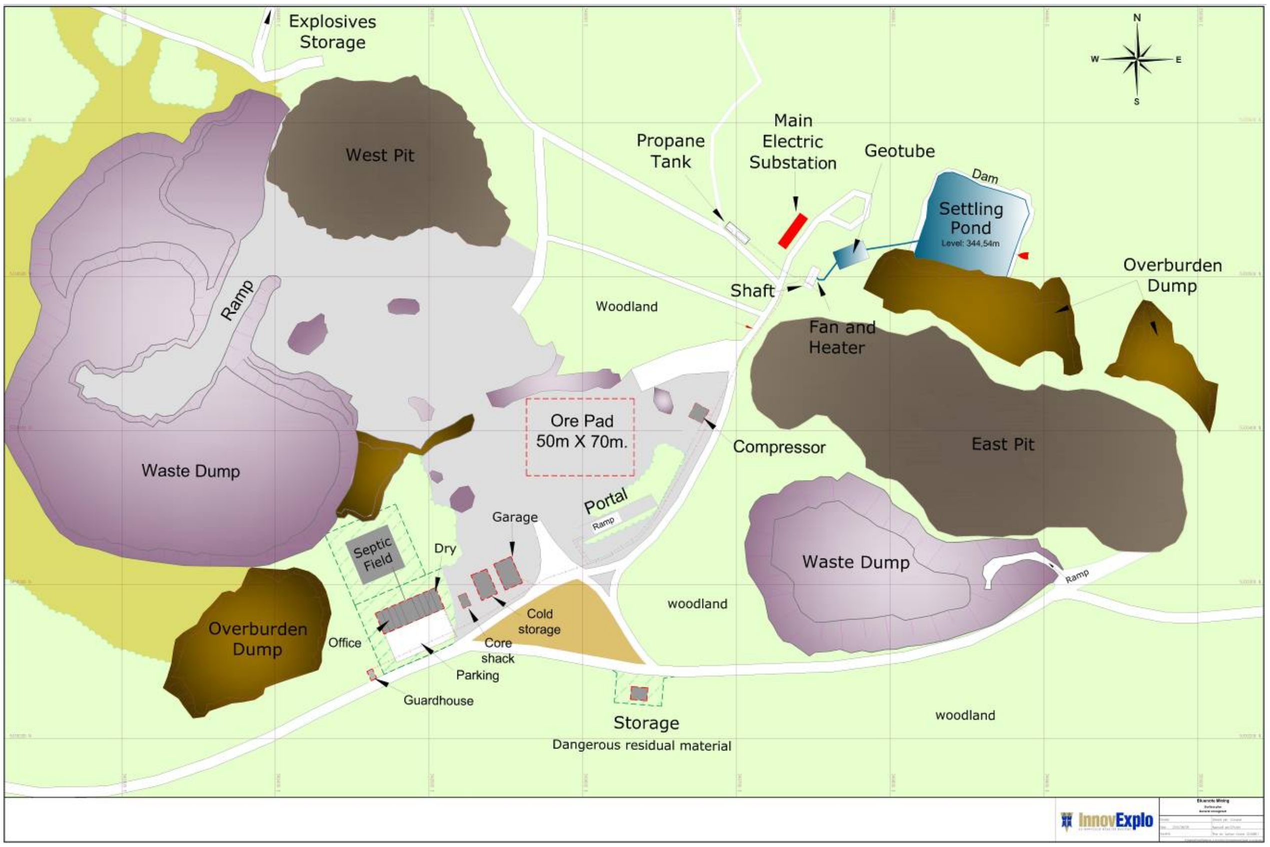


Figure 18.1 – Surface plan, general layout

18.2 Electrical Power Supply and Distribution

The Croinor property is currently not serviced by an electric power line. The closest power line is located 26.5 km west of the Croinor site, at the former Chimo mine.

The power demand for the mine site is estimated at 1,925 KW. The Croinor mine project requirements include the following electrical installation:

- New 25 kV three-phase overhead power line between the former Chimo mine and the Croinor mine site;
- New 25 kV three-phase overhead power line on site to supply the surface installations, such as the dry, offices, garage, etc. and the new 25 kV/4.16 kV substation;
- Underground main distribution (details are given in Section 16.12.4)

18.3 25 kV Three-Phase Overhead Power line (Off-Site)

To supply electric power to the site, a new 26.5 km 25 kV three-phase overhead power line is planned. In this study, this new power line is assumed to be held and owned by Hydro-Québec. Further discussions between the client and Hydro-Québec will be required to define the project.

18.4 Croinor Project 25 kV Surface Distribution

A 25 kV voltage level was selected to optimize voltage regulation and use lower-cost standard equipment. As previously stated, the underground main distribution feeders will be operating at 4.16 kV to ensure proper cable sizing and adequate voltage regulation.

The site surface buildings will be supplied by one 25 kV three-phase overhead ACSR line mounted on wood poles. Since all service buildings require less than 500 kVA, all buildings will be supplied by transformer banks on wooden poles with AC lightning arresters, fuse disconnect switches, and grounding.

The site exterior lighting will also be supplied by one 25 kV line through a single-phase transformer. Dusk-to-dawn lights will be mounted on poles in order to maintain minimum lighting in the areas covering the camp site (offices, dry, warehouse, garage and ramp access). In addition, 100W floodlights will be mounted on 50-foot wooden poles near the ramp, ore and waste piles.

18.5 Croinor Project 25 kV Main Substation

This substation will be installed in a mobile container. It will be located near the existing shaft to limit the length of power cables feeding the largest loads. The 3 MVA transformers are able to support the full underground mine load, compressors, and the main ventilation fan loads.

18.6 Site Access

The Croinor site is accessible by way of a 37-km gravel road branching off Route 117 (Trans-Canada Highway), approximately 40 km east of Val-d'Or. Existing roads to the site will be upgraded to support vehicle travel, including the offsite transportation of ore for processing. A 13-km segment of the gravel road requires major maintenance. Another access to the site is possible via Senneterre using a gravel road.

18.7 Camp

No permanent or temporary camp is included. Given the proximity to Senneterre and Val-d'Or, transportation will be organized by Blue Note Mining for its staff and by the contractor for its workers.

18.8 Mine Site Entrance/Guardhouse

All visitors, contractors, delivery personnel and mine personnel will go through a main entrance at the west side of the site. A 3 m x 8 m guardhouse will be erected at this point and anyone entering the site will do so only after being authorized by security personnel. A fenced-in car park will be located next to the gate and will have electric outlets to plug in the vehicles during cold weather.

18.9 Office Building and Dry Complex

A 67-m^2 temporary mobile unit will provide the office, lunch room and infirmary from the start of the project until month 7 when the larger modular installation will be set up. The more permanent installation will serve as offices for administration, engineering, geology, and contractors. It also comprises a dry installation for 85 men and 12 women, a lunchroom, the office for the rescue team, and an infirmary.

18.10 Service Buildings

The existing garage, consisting of a single arch-type building (70x70m) on a concrete slab, is in good condition and will be restored and used as cold storage. A steel-structure garage building will be built adjacent to the existing one. The addition includes a 12 m x 15 m section comprising a crane for equipment maintenance. There is also a 6 m x 15 m section for the warehouse and finally a 7 m x 7 m space for the mechanical and electrical room. A 53.5-m^2 trailer-type temporary building will be installed near the garage to be used as a core shack.

18.11 Site Roads

A site road providing access to various parts of the property already exists and will require restoration work to be fully operational

18.12 Compressor Building

Two 41.8 m^3/min (1476 cfm) self-enclosed compressors will be located mid-distance between the portal and the main electrical substation.

18.13 Fuel Storage

Diesel fuel for the mine equipment and vehicles will be stored in an above-ground tank. A 5100-litre diesel tank will be installed according to prevailing environmental laws and regulations. There is no gas station since only the surface pick-up trucks are gas-operated and they will be fuelled off-site.

18.14 Site Fencing

Fencing will only be provided for the main substation, the propane tanks, and for a short distance on each side of the main entrance gate.

18.15 Water Systems

Fresh water for the mine site will consist of pumped water from a well that will be drilled and tested for its water quality. The water quality must meet the requirements of Québec’s drinking water regulations. Regular monitoring will be carried out as required by such regulations.

If the well water does not meet the provincial norms and its treatment is not possible, bottled water will be available for human consumption.

18.16 Communication System

The Croinor site will be connected to the public telephone service using a wireless telephone network.

A preliminary evaluation of the topography suggests the need for two 70-foot communication towers equipped with microwave radio transmitters to be installed on a concrete slab.

Internet connection will be provided using a satellite system providing a 10 Mbs transfer rate at a reasonable cost of installation. Internet could be included in the point-to-point telephone network. Further discussions with the telephone provider (Télébec) or other third party providers should be held to evaluate all installation and operation options before completing the final design.

The surface radio system consists primarily of channels with local short-distance coverage or extended coverage. At this stage, one radio signal repeater is planned and will be installed inside the dry building where emergency power will be available.
The following channels are planned for the site:

- Security/Emergency;
- Surface Operations;
- General and Maintenance (mechanical/electrical/housekeeping/etc.);
- Underground Operations (underground link with surface).

IP technology is recommended for site communication systems for both voice and data communications. This technology is widespread, versatile and reliable. It allows similar equipment to be used for both purposes, thereby reducing the different types of equipment to install and maintain on such a large site.

IP technology permits easy integration of the following services:

- Telephony;
- Computer data (Internet/Intranet).

A voice-over IP (VoIP) telephone system with 6 lines/24 extensions will be installed. High-range IP phones are planned for the offices while basic IP phones will be sufficient for other services.

18.17 Sewage

It is planned to install a treatment system for the sewage water coming from the buildings. The system is not designed yet and some soil testing is required before proceeding with the design. The system will most likely be composed of a septic tank and bed.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Markets for doré are readily available and the doré bars produced from Croinor ore will be sold on the Spot Market. Gold markets are mature, the demand is presently high, and the gold price has risen significantly during recent years. The Bloomberg forecast for gold as at December 19, 2011 is presented in the following table, and the 3-year trailing average as at December 31 was $1,256/oz at an exchange rate of 1.05.

Table 19.1 – Bloomberg base case consensus forecasts as at December 19, 2011

	2012	2013	2014	2015
Gold price ($US/oz)	1,834	1,893	1,572	1,506
Exchange rate ($C/$US)	1.03	1.01	1.04	1.01

19.2 Contracts

No contracts have yet been assigned for the Croinor Project. Blue Note Mining plans on establishing an agreement with a third party for the following items:

- Doré transportation and refining;
- Mine dewatering;
- Mining and development work;
- Custom milling;
- Ore transportation from the mine site to the mill;
- Rental of an explosives storage building and procurement of explosives;
- Propane tank farm rental and procurement.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL/COMMUNITY IMPACT

Golder has been mandated by Blue Note Mining to prepare this section of the current report.

20.1 Hydrology

Surface water from the Croinor property is part of the Bell River watershed. The water flows towards an unnamed stream that is a tributary of Lake Blanchin. The outlet of Lake Blanchin flows into the Marquis River to Guégen Lake, then to Simon Lake, Tiblemont Lake, and Bell River. The mine effluent will be directed through an existing drainage ditch to the unnamed stream.

20.2 Hydrogeology

Since it is not expected that the Croinor deposit will generate ore and waste rock that will be acid-generating or that will leach metals under neutral pH or acid rain conditions, a formal hydrogeological study was not requested as part of the permitting process.

Based on some information gathered from a geotechnical program carried out by Golder in 2003, the soils are mainly composed of sand and silt on the Croinor site. The overburden is thin and water saturated soils were encountered in 3 of the 13 trenches completed.

The likelihood of having a class I or II aquifer on the Croinor site is low.

20.3 Waste Rock and Mineralized Rock Characterization

In order to characterize the ore and waste rock, selected composite samples were collected from drill core samples. The core represented typical ore and geological units that will be extracted during the mine operation.

Ten (10) ore samples, 8 waste rock samples and 3 mud samples (from the settling pond) were collected and tested using the modified Sobek ABA method (acid-base-accounting) in order to determine the acid generating potential.

Ten (10) ore samples, the eight (8) waste rock samples and one (1) composite sample from the pond mud were assayed for trace elements in order to estimate whether their metal content was above Criterion A of the *Soil Protection and Contaminated Site Rehabilitation Policy* (SPCSRP). If metal values are found to be above Criterion A, leach tests are required to evaluate whether metal leaching is possible.

For the leach tests, the 10 ore samples were combined into 5 composite samples. The 8 waste rock samples were combined into 4 composite samples. Leach tests on these 9 composite samples were carried out using the TCLP, SPLP and CTEU-9 protocols (which represent leaching under acetic acid, acid rain conditions and neutral pH respectively).

Based on the results of these tests, the mud of the settling pond is a low-risk non-acid-generating residue. The ore and the waste are not acid-generating and they do not leach metals under neutral pH and acid rain conditions, which are the most likely conditions at Croinor.

20.4 Environmental Management

20.4.1 Mine Water Management – Initial Development

In order to initiate underground development, the open pits and existing underground openings have to be dewatered (the mine workings are connected to the main pit). Initial dewatering is expected to be carried out at a rate of 5,500 m^3/day for an approximate period of 92 days. Ongoing dewatering during mine operation is estimated at 963 m^3/day based on previous operating data.

Based on data from previous dewatering during the 2003-2005 period, the effluent quality always met release limit thresholds and metal values were low. Mine water will be pumped and processed through Envirotubes to collect mine sludge.

Discharge from the Envirotubes will be directed to the existing settling pond (60 x 60 m). The pond effluent will be monitored according to MMER and *Directive 019* requirements.

20.4.2 Mine Water Management – Operation

During operation, the mine water will be collected and treated on surface using Envirotubes. The Envirotubes allow suspended solids and possible contaminants to be removed. The clear filtrate will be collected and used for production water or discarded into the settling pond. The retention time in the pond will provide further polishing of the water. Excess water will be allowed to flow out of the settling pond, and monitoring of this effluent will be carried out according to prevailing regulations.

20.4.3 Runoff Management

The site is not prone to runoff due to the sandy and silty nature of the soils. There is no system to collect runoff due to the high infiltration rates in the soils, waste piles and backfilled areas of the site.

20.5 Emergency Response Plan

To ensure the health and safety of workers and protection of the environment, an emergency response plan will be developed for the duration of mining operations. This plan will include appropriate emergency measures for different potential situations in order to ensure a rapid and effective response.

20.6 Monitoring

It is planned to contract out the environmental monitoring and control activities since the scope of the project does not support the need for a full-time coordinator. The sampling of the settling pond effluent will be carried out once a week. The effluent will be assayed for pH, total suspended solids (TSS) and metals (As, Cu, Fe, Ni, Pb, Zn, Ra) according to *Directive 019* and MMER requirements. Assaying of an extended list of parameters is required once every quarter and once a year.

The contractor will submit monthly and annual reports to the provincial and federal agencies.

20.7 Air Quality

The Québec regulation on air quality requires that no dust clouds be visible more than 2 metres from the source. On surface, the handling and breaking of ore could be a source of dust. If dust clouds become visible during ore handling and breaking, water will be sprayed on the ore to prevent dust from spreading.

Trucking on the mine site is another source of dust. Dust control agents will be used if needed.

20.8 Waste Rock Acid-Based Accounting

Waste rock permanently brought to surface either for disposal on surface or for use as a construction material must be tested for acid generation potential.

20.9 Effluent Quality

The pond effluent will be monitored according to the MMER and *Directive 019* requirements.

20.10 Closure Consideration

One of the site liabilities is its reclamation in accordance with the Québec Mining Act. Waste piles will be reclaimed by re-establishing a soil cover and vegetation. The backfilled area of the site will be reclaimed the same way. Buildings will be dismantled and foundations backfilled and vegetated. Underground openings will be blocked or capped. Equipment will be removed from the mine and the site.

The second liability is to carry out, at final mine closure, an Environmental Site Assessment (ESA) as required by the Environmental Quality Act. If ground contamination is discovered during the ESA, an environmental rehabilitation plan must be submitted to the Ministry for approval and the site must be decontaminated.

There are no other existing environmental liabilities.

In order to fulfil requirements of the Mining Act, waste rock piles, areas backfilled with waste rock, and cement foundations must be reclaimed by re-establishing vegetation. The settling pond needs to be emptied and levelled. A breach will be opened in the dike to accommodate free water flow. The pond will be seeded.

Self-sustaining vegetation must be in place 5 years following final site reclamation.

Agronomical and geotechnical inspections will be carried out to ensure the site's integrity.

The method selected for re-establishing vegetation over waste rock piles and areas backfilled with waste rock is to place a 15-cm layer of silt and/or organic soil over the rock and then proceed with fertilizing and seeding of grass. A layer of silt and/or organic soil will help retain water in the surface layer. A borrow pit for this material needs to be found near the mine site.

Fertilizing and seeding will be done directly in the settling pond and on the piles of overburden.

Mine workings (ramp and shaft) will be blocked and capped.

A protection abutment will be maintained around the open pits, which will eventually be flooded after mine closure.

20.11 Permitting Requirements

The last period of mining at the Croinor site ended in May 2005. In order to facilitate the resumption of mining operations, Golder undertook a review of environmental permitting and licensing requirements. Refer to Section 4.4 for a description of the environmental obligations and to Table 4.5 for a summary of the permit status.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Expenditure

Most of the capital cost was estimated using quotes from equipment suppliers and contractors. In some cases, comparable installations at other projects were used. The capital cost estimate is accurate within ±20%.

The pre-production costs are estimated at $20.68 million, including $2.4 million of capitalized operating costs net of production revenue received during the pre-production period. Sustaining capital is estimated at $17.9 million, excluding $0.66 million for final closure costs. The cost breakdown is presented in Table 21.1.

Table 21.1 – Capital expenditure breakdown

Description	Pre-production	Sustaining	Total cost
Capitalized operating cost	$16,363,677		$16,363,677
Capitalized revenue	($13,950,520)		($13,950,520)
Dewatering and rehabilitation	$1,249,609		$1,249,609
Development	$4,537,911	$10,760,313	$15,298,224
Ventilation equipment	$340,075		$340,075
Mine dewatering	$442,718	$56,614	$499,331
Surface installation and equipment	$2,081,591	$670,628	$2,752,219
Electrical distribution	$6,029,064	$1,232,000	$7,261,064
Mobile equipment	$2,955,638	$4,903,494	$7,859,132
Environment	$371,596	$258,162	$629,758
Contractor demobilization	$255,642		$255,642
Total capital expenditures	**$20,677,000**	**$17,881,210**	**$38,558,210**

21.1.1 Capital operating cost and revenue

During the 18-month pre-production period, the Croinor mine is expected to produce a total of 7,316 ounces of gold, providing revenue of $13.95 million. The estimated operating cost for the pre-production period is $16.36 million. The pre-production revenue and pre-production operating costs were capitalized.

21.1.2 Dewatering and rehabilitation

The capital cost related to initial dewatering and rehabilitation amounts to $1.25 million. The dewatering item includes the equipment and manpower required for mine dewatering, such as pumps and all needed equipment for the Envirotubes. The rehabilitation work comprises shaft rehabilitation, portal preparation work, and ramp rehabilitation to level 125. A 25% contingency is included in the dewatering and rehabilitation estimate.

21.1.3 Development

A considerable amount of development and rehabilitation work is planned for the pre-production year to provide a degree of flexibility in terms of access, which should facilitate scheduling during the production period.

The estimated capital development cost is $4.54 million for the pre-production period and the estimated sustaining capital is $10.76 million. It includes the entire development of the ramp, the excavation required to access the production drift, and the excavation of all required ventilation raises. Primary development, such as the electrical station, refuge station, and sump, are part of the development capital cost and were included by adding 15% to all development required in the mine plan. No further contingency was added to the development cost.

21.1.4 Ventilation equipment

The costs for the ventilation system were estimated from budget quotes provided by suppliers. Details of the ventilation costs are presented in Table 21.2. The propane tank installation is not capitalized because the plan is to rent these units.

Table 21.2 – Ventilation equipment and installation capital cost

Description	Quantity	Cost
Main fan (300 hp) and accessories	1	$43,000
Burner and accessories	1	$112,000
Underground fan	8	$58,400
Contingency (25%)		$53,350
Compressor and fan installation		
Concrete slab		$9,575
Compressor installation		$6,500
Hardware and piping installation		$12,250
Ventilation system installation		$45,000
	Total	**$340,075**

21.1.5 Mine dewatering

The costs for the mine dewatering equipment were estimated using budget quotes provided by suppliers. The capital costs associated with mine dewatering equipment total $499,331 and are detailed in Table 21.3.

Table 21.3 – Mine Dewatering Capital Cost

Description	Quantity	Cost
Level 125 pumping station	1	$52,491
Level 170 pumping station	1	$241,392
Level 250 pumping station	1	$90,582
Fresh water supply system	1	$15,000
Contingency (25%)		$99,866
	Total:	**$499,331**

21.1.6 Surface installation and equipment

Table 21.4 presents the cost of surface installations and systems based on budget quotes provided by suppliers and existing comparable installations at other projects.

Table 21.4 – Surface installation and equipment capital cost

	Unit cost		Total
Transition trailer			
Mobilization	$1,950	*lump sum*	$1,950
Rental	$820	*$/month*	$5,740
Office and dry building			
Mobilization	$86,470	*lump sum*	$86,470
Rental	$14,810	*$/month*	$666,450
Final payment	$24,810	*lump sum*	$24,810
Core shack rental			
Mobilization	$9,900	*lump sum*	$9,900
Rental	$1,185	*$/month*	$53,325
Final payment	$2,385	*lump sum*	$2,385
Security building			
Mobilization	$5,615	*lump sum*	$5,615
Rental	$750	*$/month*	$33,750
Final payment	$2,385	*lump sum*	$2,385
Garage			
Mobilization	$548,150	*lump sum*	$548,150
Ventilation building	$36,000	*lump sum*	$36,000
Explosives magazine			
Mobilization	$5,780	*lump sum*	$5,780
Rental	$300	*$/month*	$18,000
Compressors (2)	$170,640	*$ (for both)*	$170,640
Parking lot pad preparation	$20,000	*$/unit*	$20,000
Propane farm pad	$5,000	*lump sum*	$5,000
Surveying equipment	$80,000	*$/unit*	$80,000
Material for office	$75,000	*$/unit*	$75,000
Road improvement	$519,228	*lump sum*	$519,228
Contingency (15%)			$381,640
Subtotal			$2,752,219

21.1.7 Electrical distribution

The capital cost estimation related to the electrical infrastructure for the Croinor property and mine was provided by Meglab Inc and is broken down in tables 21.5 to 21.8.

Table 21.5 – Pre-production electrical distribution capital cost

Electrical Distribution Pre-production Budget	Budgetary cost
Main substation installed near the shaft: NEMA box 3R, 3 MVA, 25 KV at 4,160/2400 volts, M.A.L.T., measuring, cables, neutral resistance	$325,000
Electric distribution cabinet 4.16 KV, main switch gear and six 400A feeder breaker cell, volts and amps indicators, ground fault protection, 120 VDC supply on protection	$287,000
Substation 4.16KV at 600/347 volts, 750 KVA for mechanical shop, office, dry, compressors and parking supply	$76,000
Substation 4.16KV at 600/347 volts, 750 KVA and distribution panel at the shaft with 2 soft start starter 300HP for the main ventilation fan, two starter for the pumps 60HP, two PTO 150 Amps, lighting and switches	$145,000
A portable electrical substation 500 KVA, 4.16 KV at 600-347 volts to follow jumbo, pump and fan required for the development	$54,000
Mobil distribution cabinet with multiple switch	$26,000
Electrical cables, connectors and hardware for distribution	$300,000
Electrical cables, connectors and hardware for underground distribution	$125,000
Engineering	$40,000
Manpower (4 electricians for a year)	$560,000
Communication system (two-channel radios for surface and underground)	$60,000
Mercaptan gas system	$8,000
Air heating remote control and protection system	$26,000
Total pre-production budget	**$2,032,000**

Table 21.6 – Year 1 and Year 2 electrical distribution cost

Electrical Distribution Year 1 and Year 2 Budget	Budgetary cost
Main substation 1000 KVA at 4,160 volts installed on level 3 to supply levels 2, 3 and 4, multiple outlet panel to supply mobile equipment, pumps, drills and fan	$121,000
Hardware to control and visualize ventilation equipment, water level, gas and air volume by Leaky Feeder. HMI located on surface to read data collected from a station underground.	$55,000
Gas probe and flow meter	$12,000
Smart Blast blasting system	$40,000
Hardware for 4 electrical panels 600-volt stations on levels 2 and 3	$104,000
Electric cables at 5 KV and 600 volts, connector and hardware	$200,000
Engineering	$35,000
Manpower (3 electricians, 1 technician, and 1 programmer)	$560,000
Equipment for the lunchroom	$60,000
Total pre-production budget	**$1,187,000**

Table 21.7 – Year 3 and Year 4 electrical distribution cost

Electrical Distribution Year 3 and Year 4 Budget	Budgetary cost
Main substation 1000 KVA at 4,160 volts installed on level 5 to supply levels 4, 5 and 6, multiple outlet panel to supply mobile equipment, pumps, drills and fan	$121,000
Gas probe and flow meter	$12,000
Extension of Leaky Feeder and control system for the ventilation fan and to carry data to surface	$55,000
Hardware for 4 electrical panels at 600-volt stations on levels 2 and 3.	$104,000
5 KV and 600 volt electrical cables, connector and hardware	$200,000
Engineering	$20,000
Manpower (3 electricians, technician and programmer)	$480,000
Total pre-production budget	**$992,000**

Table 21.8 – Electrical distribution capital cost

Electrical distribution	Unit cost	
Pre-production	$2,032,000	lump
Year 1 and 2	$627,000	lump
Manpower Year 1 and 2	$560,000	$
Year 3 and Year 4	$992,000	lump
Off-site power line	$2,461,771	$
Surface communication system	$95,939	$/unit
Contingency (on power line only)	$492,354	lump
Total	$7,261,064	

21.1.8 Mobile equipment

The study considers all the production equipment to be purchased by Blue Note Mining. Most of the costs listed in Table 21.9 were estimated using quotes from suppliers with a 20% down payment upon receipt of the equipment and the remaining amount financed by the supplier at 6.5%.

Table 21.9 – Building and infrastructure mobilization and installation capitalized cost

Equipment	Cost
Mine truck (30t)	$906,031
Mine truck (30t)	$922,343
Scooptram 6yd	$913,312
Scooptram 3.5yd	$793,591
Scooptram 3.5yd	$793,591
Jumbo 2B	$953,644
Jumbo 1B	$650,697
Scissor lift	$122,400
Scissor lift	$122,400

Equipment	Cost
Tractor (mechanical and electric)	$268,800
Tractor (development)	$187,200
Tractor (spare)	$93,600
Gear truck	$50,000
Mine mule	$152,000
Electric slusher	Provided by the contractor
Jackleg and stopper drill	
Long-hole drill	
Surface loader	$344,051
Grader	$431,100
E-series cargo	$103,373
Pick-up	$51,000
Total:	**$7,859,132**

21.1.9 Environment

Environmental capital cost estimates were provided by Golder and are summarized in Table 21.10.

Table 21.10 – Environmental capitalized cost

Description	Cost
Final effluent station and ditch	$76,780
Characterization of waste for road upgrading	$11,000
Fees for using water (dewatering)	$1,260
Drinking water well	$34,050
Turbidity curtain	$8,800
Sewage system	$84,750
CofA for gravel pit	$5,500
Miscellaneous permits	$22,000
Plans for effluent station and ditch	$13,200
Environmental operating manual	$22,000
Environmental effects monitoring study	$43,787
Contingency (15%)	$48,469
Subtotal:	**$371,596**
Financial guarantee to MRNF ($105,178 already in guarantee)	$99,223
Environmental effects monitoring study	$158,939
Subtotal:	**$258,162**
Total:	**$629,758**

21.1.10 Sustaining capital

The sustaining capital estimate for the operating life of the project includes ongoing development but does not include stope development costs that are part of the operating cost estimate. Also included in the sustaining capital estimate are ongoing costs related to electrical equipment the environmental control and environmental studies, as well as the cost of contractor demobilization.

21.1.11 Project closure costs

As part of their mandate, Golder estimated the project closure costs (Table 21.11) to be $655,689

Table 21.11 – Closure cost estimation

Description	Yearly cost
Environmental site assessment (ESA)	$44,000
Environmental effects monitoring study	$143,000
Garage dismantling, site cleanup etc.	$55,000
Reclamation of the mine site	$328,164
Contingency (15%)	$85,525
Total closure cost	655,689
Financial guarantee reimbursement	$204,401
Amount including financial guarantee:	$451,288

21.1.12 Equipment residual value

A residual value of $5.8 million has been considered, mostly for mobile, building, electrical, ventilation and air heating equipment.

21.2 Operating cost

Operating costs are estimated in 2011 Canadian dollars with no allowance for escalation. The total life-of-mine operating cost and average unit operating costs are summarized in Table 21.12. The overall unit operating cost is $164 per tonne of milled ore.

InnovExplo estimated mine operating costs using budget quotes from contractors and suppliers.

Table 21.12 – Summary of total life-of-mine operating costs

Description	Total cost	Unit cost	
Definition drilling	$2,332,270	4.40 $/t	20.44 US$/oz
Stope development	$16,039,477	30.25 $/t	140.55 US$/oz
Mining	$19,372,502	36.54 $/t	169.76 US$/oz
Blue Note Mining staff	$10,847,925	20.46 $/t	95.06 US$/oz
Contractor (indirect cost)	$10,885,100	20.53 $/t	95.38 US$/oz
Surface services	$189,508	0.36 $/t	1.66 US$/oz
Energy cost	$4,482,168	8.45 $/t	39.28 US$/oz
Milling and transportation	$22,055,439	41.60 $/t	193.27 US$/oz
Environment	$779,739	1.47 $/t	6.83 US$/oz
Total:	**$86,984,128**	**164 $/t**	**762 US$/oz**

21.2.1 Definition drilling

InnovExplo estimated that 10,000 metres of definition drilling will be required per year until early in Year 4 when most of the definition drilling will have been completed. This estimate is based on similar mine operating practices. According to the mine-life conceptual mining plan, access for drill set-up should generally be relatively simple. A cost of $90/m was used, and the resulting total estimate for definition drilling is $2.33M or $4.40/tonne milled.

21.2.2 Stope development

The unit cost for stope preparation stands at $30.25 per tonne milled (based on tonnage milled assigned to production), and consists of all of the 3 m x 3.7 m drift excavation or enlargement work, and 100% of the raises and sublevels for long-hole mining. A 15% contingency was added to the development lengths for drifts and raises and is included in the development sequence.

The development unit cost was estimated using quotes from contractors for direct and support labour, as well as for the supply of steel, bits and small tools. The consumable material will be managed and provided by Blue Note Mining. The unit costs were therefore estimated based on the quantities needed according to the dimensions of each type of excavation. The consumable cost includes explosives, ground control, ventilation water and compressed air pipe. A contingency of 20% was added to accounts for the various losses.

21.2.3 Mining

Mining costs are estimated to be $36.54/tonne milled using a ratioconsidering that 84% of the tonnage will be from long-hole and sub level development ant that and 16% of the tonnage will be mined by room and pillar.

21.2.4 Blue Note Mining staff and employees

Staff and employee salaries and associated expenses include those for the administrative, technical and surface services. Most salaries were evaluated using the 75th percentile of the salary range for each job classification according to data in Mining Cost Services published by Cost Mine, a division of InfoMine USA Inc. To account for benefits, 38% was added; depending on the job, bonuses were also included. The estimate of a department's general operating cost was based on a 15% factor derived from a comparable mine operating budget. The yearly cost is $3.5M, averaging $20.46 per tonne milled. The annual cost of the Blue Note Mining staff assigned to the Croinor Project is detailed in Table 21.13.

Table 21.13 – Blue Note Mining staff salary

Description	Quantity	Annual salary
Administration		
Manager	1	$269,100
Accounting	1	$121,440
Purchase agent	1	$115,920
Clerk	1	$60,278
Executive secretary	1	$69,552
Materials and other		$522,712
Subtotal:		$1,159,003
Technical Services		
Chief geologist	1	$174,570
Geologist	1	$129,030
Chief engineer	1	$174,570
Engineer	1	$136,620
Geological technician	3	$341,136
Mine technician	3	$341,136
Materials and other		$194,559
Subtotal:		$1,491,621
Surface Services		
Dry man	2	$120,557
Service man	4	$252,595
Gate keeper	4	$172,224
Materials and other		$302,223
Subtotal:		$847,599
	Total:	**$3,498,223**

21.2.5 Contractor indirect costs

All the salaries for the manpower required to operate the mine, including supervision, maintenance and underground workers, are included in contractor indirect costs.

21.2.6 Surface services

The cost for surface services includes the cost related to the supplies required to operate the treatment station for the Envirotubes. It is estimated that 26 bags will be required each year. The estimated yearly cost is thus $43,316. A 25% contingency has been added to cover the costs of the required chemicals. The average life-of-mine cost is $0.36 per tonne milled.

21.2.7 Energy cost

The energy cost includes all electrical consumption, the propane needed to heat the underground air, and the rental of a propane tank. The estimated average annual electrical consumption is 12,415,240 kWh, representing an annual cost of $739,685. The estimated annual propane consumption is 1,151,000 litres per year, amounting to $529,460 per year at a price of $0.46/litre (budget quotation from Superior Propane). As shown in Table 21.14, the estimated total annual energy cost is $1.32M, or an average of $8.45 per tonne milled.

Table 21.14 – Yearly energy cost

Description	Yearly cost
Electricity	$739,685
Propane	$529,460
Propane tank rental	$46,200
Total:	$1,315,345

21.2.8 Milling and transportation

Ore from Croinor will be processed at a mill in the Val-d'Or area with excess capacity for the duration of the Croinor operation. Contact has been made with potential custom milling partners and tentative commitments have been arranged for processing the ore. For the study, it is assumed that the ore will be trucked to a custom mill located approximately 100 km from the Croinor Project. The unit cost for truck loading, transportation and milling is estimated at $41.60/tonne and is based on budgetary quotes.

21.2.9 Environment

Golder estimated the yearly environmental costs to be $231,406 (Table 21.15) for an average of $1.47 per tonne milled.

Table 21.15 – Yearly environment cost

Description	Yearly cost
Assays	$38,338
Personnel	$120,000
Miscellaneous	$22,000
Fees for using water (dewatering)	$879
Fees for using water (well)	$6
Maintenance	$20,000
Contingency (15%)	$30,183
Total:	**$231,406**

21.2.10 Capitalized Opex

The operating costs incurred during the pre-production period ($16,363,677) were capitalized.

22.0 ECONOMIC ANALYSIS

An after-tax model was developed for the Croinor Project. All costs are in 2011 Canadian dollars with no allowance for inflation or escalation.

The Croinor Project is subject to the following taxes: Québec mining rights; Federal and provincial taxes.

The income tax rate is 26.9% (2012 federal and Québec tax rate) and the mining tax rate is 16% (2012); this latter rate will be sanctioned as proposed in the May 2011 bill.

It is assumed that Blue Note Mining Inc ("Blue Note Mining"), a federal corporation, and X-Ore Resources Inc ("X-Ore"), a Québec corporation, will proceed with a vertical amalgamation. X-Ore and Blue Note Mining are allowed to do so, but only following the continuance of X-Ore under the Canada Business Corporations Act, or of Blue Note Mining under the Business Corporations Act (Québec). This vertical amalgamation will allow the available non-capital losses of Blue Note Mining and X-Ore to be used by the resulting corporation to offset any future income derived from its mining activities.

The Croinor property is subjected to the Corcoran-Agar Royalty which consist of 15% applied on net profit from commercial production of which $15,000 is payable in September of each year as an advanced payment on royalties.

Another royalty is affecting the Croinor property, the Canadian Spooner Royalty. Considering that any expense (of whatever nature) incurred on the property since 1983 would have to be recouped by the operator, it is highly unlikely that any amount would have to be paid with respect to the agreement regarding this royalty.

The economic valuation of the project was performed using the Internal Rate of Return (IRR) and Net Present Value (NPV) methods. The IRR on an investment is defined as the rate of interest earned on the unrecovered balance of an investment. The NPV method converts all cash flows for investments and revenues occurring throughout the planning horizon of a project to an equivalent single sum at present time at a specific discount rate. The discount rate used in the analysis is 7%. According to the NPV method, a positive NPV represents a profitable investment where the initial investment plus any financing interests are recovered.

The following parameters were considered in the financial analysis:

- An average gold price of US$1495/oz and an exchange rate of 1.03 CAD/1USD, which correspond to the Bloomberg consensus estimate of December 19, 2011. It also considers that preproduction would be initiated in April 2012. Table 22.1 provides details of the Bloomberg base case consensus forecasts.

Table 22.1 – Bloomberg base case consensus forecasts as of December 19, 2011

	2012	2013	2014	2015
Gold price ($US/oz)	1,834	1,893	1,572	1,506
Exchange rate (CAD/USD)	1.03	1.01	1.04	1.01

- Reserves parameters as described in Section 15.
- Gold recovery of 97.5%. This value was based on recovery obtained at the time the mine was operating.

- Royal Mint fees of $5/oz.
- Estimated average annual tonnage of 94,820 to 201,610 tonnes.
- Estimated average annual output of 21,260 to 41,580 ounces of gold.
- Royalty payment evaluated as follows: a royalty of 15% applied on profit over the carried expenses, which amounts $13,153,158.
- Future annual cash flow estimates based on grade, gold recoveries and cost estimates as previously discussed in this report.
- 36,201 tonnes of ore (to be processed during the pre-production period) deemed as capital production and not included in production nor the revenue derived from it.

The resulting main parameters and cash flow analysis for the entire project are presented in Table 22.2, and Table 22.3 provides the details of the cash flow analysis.

Table 22.2 – Cash flow analysis summary

Parameters	Results
Proven & Probable mineral reserves	566,429 t @ 6.64 g/t
Total contained gold reserve	120,833 oz
Mine life (including 18-month pre-production)	5 years
Daily mine production	425 tpd ramping up to 675 tpd in Year 4
Gold recovery	97.5%
Annual gold production	21.259 to 41,578 oz
LOM recovered gold	117,956 oz
Average cash operating cost	$164/tonne
Average cash operating cost	US$762/oz
Capital cost (including $17.9M sustaining/working capital)	$38.6 million
Total cost per ounce	US$1032/oz
Total gross revenue	$182 million
Total operating cost	$86.9 million
Total project cost	**$125.5 million**
Total operating cash flow (before tax & royalties)	**$47.2 million**
Estimated mining and income taxes	**$12.5 million**
Net cash flow	**$31 million**
Pre-tax NPV (7% discount)	**$30.6 million**
Pre-tax IRR	**57%**
After-tax NPV (7% discount)	**$21 million**
After-tax IRR	**44%**
Payback period	**38 months**
Pre-production period (including 36,201t of production)	**18 months**

Table 22.3 – Economic analysis for the Croinor Project (figures in Canadian dollars)

Croinor- Prefeasibility Economic Analysis	Pre-production (18 month)		Production (starts at month 19th)			
Figures in Canadian dollars Bloomberg forecast as of December 19 2011	Year 1 (12 month pre-prod)	Year 2 (6 month pre-prod -6 month prod)	Year 3	Year 4	Year 5	Total
PRODUCTION						
Long Hole		72 565	81 966	154 299	112 746	421 576
Grade (g/t)		7.26	5.17	6.18	5.48	5.98
Development	4 914	14 175	20 686	12 726	0	52 501
Grade (g/t)	5.63	6.09	5.42	5.75	0.00	5.70
room and pillar		8 076	33 319	34 584	16 374	92 352
Grade (g/t)		8.01	13.66	8.66	7.41	10.19
Total (tonne mined)	4 914	94 816	135 971	201 609	129 120	566 429
Grade (g/t)	5.63	7.15	7.29	6.58	5.73	6.64
Total (tonne milled)	4 914	94 816	135 971	201 609	129 120	566 429
Grade (g/t)	5.63	7.15	7.29	6.58	5.73	6.64
Recovery (%)	97.50%	97.50%	97.50%	97.50%	97.50%	97.50%
Gold Produced (oz)	867	21 259	31 064	41 578	23 188	117 956
Tonne milled assigned to capital	4 914	31 287				36 201
Gold Produced assigned to capital (oz)	867	6 449				7 316
Tonne milled assigned to production		**63 529**	**135 971**	**201 609**	**129 120**	**530 228**
Grade (g/t)		**7.44**	**7.29**	**6.58**	**5.73**	6.66
Gold Produced (oz)		**14 810**	**31 064**	**41 578**	**23 188**	**110 640**
Grade (g/t)						
Gold Price ($US/oz)	$1 849	$1 813	$1 556	$1 443	$1 256	$1 495
Exchange rate ($CAN/$US)	1.02	1.02	1.03	1.02	1.05	1.03
Gold Price ($CAN/oz)	$1 888	$1 843	$1 606	$1 474	$1 324	$1 542
Gross Revenue	**$1 657 786**	**$38 359 362**	**$49 890 131**	**$61 277 643**	**$30 695 122**	**$181 880 042**
Mint (cost 5,00$ per oz)	$4 335	$106 293	$155 319	$207 891	$115 941	$589 779
Capitalized revenue	$1 653 450	$12 297 070				**-$13 950 520**
Net Revenue	**$0**	**$25 955 999**	**$49 734 812**	**$61 069 752**	**$30 579 181**	**$167 339 743**
OPERATING EXPENDITURES						
Definition drilling	900 000	900 000	$900 000	$900 000	$82 270	$3 682 270
Stope development	1 399 164	6 421 327	$8 275 225	$4 634 309	$11 474	$20 741 500
Mining	45 700	2 929 195	$5 313 308	$7 477 822	$4 656 478	$20 422 504
Blue Note Staff	1 423 715	3 038 658	$3 498 223	$3 498 223	$2 332 149	$13 790 968
Contractor Indirect	2 020 800	3 086 200	$3 691 200	$3 196 200	$2 427 100	$14 421 500
Surface services	4 512	54 145	$54 145	$54 145	$54 145	$221 092
Energy cost	361 565	1 072 266	$1 315 345	$1 315 345	$1 315 345	$5 379 867
Milling and transportation	204 404	3 943 962	$5 655 854	$8 386 153	$5 370 883	$23 561 255
Environment	231 406	231 406	$231 406	$231 406	$201 223	$1 126 849
Capitalized operating cost	-6 591 266	-9 772 411				-$16 363 677
Total operating costs	**$0**	**$11 904 750**	**$28 934 707**	**$29 693 604**	**$16 451 067**	**$86 984 128**
Op. cost/tonne $CAN		$187	$213	$147	$127	$164
Op. cost/oz $CAN			$931	$714	$709	$786
Op. cost/tonne $US			$206.10	$144.25	$120.88	$159
Op. cost/oz $US			$902	$699	$673	$762
Operating Cash Flow		**$14 051 249**	**$20 800 104**	**$31 376 148**	**$14 128 114**	**$80 355 615**
CAPITAL EXPENDITURES						
Capitalized operating cost	$6 591 266	$9 772 411	$0	$0	$0	$16 363 677
Capitalized revenue	-$1 653 450	-$12 297 070	$0	$0	$0	-$13 950 520
dewatering and rehab	1 249 609	$0	$0	$0	$0	$1 249 609
Contractor mobilization	255 642	$0	$0	$0	$0	$255 642
Development	2 827 105	$4 815 786	$6 317 717	$1 337 616	$0	$15 298 224
Ventilation equipment	321 825	$18 250	$0	$0	$0	$340 075
Mine dewatering	442 718	$0	$56 614	$0	$0	$499 331
Surface installation and equipment	1 963 980	$235 221	$235 221	$234 031	$83 766	$2 752 219
Electrical distribution	5 789 064	$480 000	$496 000	$496 000	$0	$7 261 064
Mobile equipment	1 760 833	$2 038 580	$1 687 550	$1 581 319	$790 850	$7 859 132
Environment	371 596	$0	$99 223	$158 939	$0	$629 758
Total capital expenditures	**$19 920 188**	**$5 063 178**	**$8 892 325**	**$3 807 904**	**$874 616**	**$38 558 210**
Total Cost cost/oz $CAN						$1 064
Total Cost cost/oz $US						$1 032
Salvage (portion of electrical equipment)					$5 815 903	$5 815 903
Financial guarantee reimbursement					$204 401	$204 401
Closure Costs		$0	$0	$0	$655 689	$655 689
Net Cash flow before NPI	**-$19 920 188**	**$8 988 071**	**$11 907 780**	**$27 568 244**	**$18 618 113**	**$47 162 020**
Administrative Cost (5%)	**$996 009**	**$848 396**	**$1 891 352**	**$1 675 075**	**$866 284**	**$6 277 117**
NPI (15%) (Carried expenses: $13,153,158)	**$15 000**	**$15 000**	**$15 000**	**$971 987**	**$2 662 774**	**$3 679 762**
Net Cash flow After NPI	**-$19 935 188**	**$8 973 071**	**$11 892 780**	**$26 596 256**	**$15 955 339**	**$43 482 259**
Estimated Mining and Income taxes		**$1 240 514**	**$1 600 722**	**$6 543 598**	**$3 080 320**	**$12 465 154**
Cash Surplus After Taxes	**-$19 935 188**	**$7 732 557**	**$10 292 058**	**$20 052 658**	**$12 875 019**	**$31 017 105**
Cumulative Cash flow	**-$19 935 188**	**-$12 202 630**	**-$1 910 572**	**$18 142 086**	**$31 017 105**	

Pre-tax NPV (7%)	**30 580 555 $**
Pre-tax IRR	**57%**
After-tax NPV (7%)	**21 002 071 $**
After-tax IRR	**44%**

22.1 Sensitivity

The parameters in the sensitivity analysis were chosen based on their potential impact on the outcome of the economic evaluation. The parameters were divided into two different groups: production parameters and economic parameters.

The following production parameters were analyzed:

- Grade (g/t)
- Gold price (US$/oz)

The following economic parameters were analyzed:

- Total net revenue (REVENUE)
- Operating expenditure (OPEX)
- Capital expenditure (CAPEX)

Sensitivity calculations were performed on the project's after-tax NPV, IRR and total cash flow by applying a range of variation (±25%) to the parameter values. Results are presented in tables 22.4, 22.5 and 22.6. The effects on NPV, total cash flow and IRR are shown graphically in figures 22.1, 22.2 and 22.3.

As illustrated in the figures, the Croinor Project is highly sensitive to changes in grade, gold price and revenue. It is moderately sensitive to changes in OPEX and CAPEX.

22.2 Discount Rate Sensitivity

The cash flow has been discounted to 7% and is considered reasonable; however, depending on the financing option, further analysis should be undertaken to establish the true discount rate based on the real cost of capital. Table 22.4 presents sensitivity to discount rate.

Table 22.4 – Sensitivity analysis for various discount rates

Discount rate	**0%**	**2%**	**4%**	**6%**	**7%**	**8%**	**10%**
	$31	$27.8M	$24.9M	$22.2M	$21M	$19.8M	$17.7M
Change (%)	48%	32%	19%	6%	Base case	-6%	-16%

Table 22.5 – Sensitivity analysis on NPV at 7% discounted rate

	-25%	-20%	-15%	-10%	-5%	Base Case Scenario	+5%	+10%	+15%	20%	25%
PRODUCTION PARAMETERS											
Grade	1.80	5.70	9.60	13.48	17.25	21.00	24.75	28.50	32.25	36.00	39.75
Change (%)	-91%	-73%	-54%	-36%	-18%		18%	36%	54%	71%	89%
Gold Price	1.74	5.65	9.56	13.46	17.23	21.00	24.77	28.53	32.29	36.05	39.81
Change (%)	-92%	-73%	-54%	-36%	-18%		18%	36%	54%	72%	90%
ECONOMIC PARAMETERS											
REVENUE	1.80	5.70	9.60	13.48	17.25	21.00	24.75	28.50	32.25	36.00	39.75
Change (%)	-91%	-73%	-54%	-36%	-18%		18%	36%	54%	71%	89%
OPEX	31.24	29.17	27.12	25.07	23.03	21.00	18.97	16.87	14.65	12.44	10.23
Change (%)	49%	39%	29%	19%	10%		-10%	-20%	-30%	-41%	-51%
CAPEX	28.29	26.83	25.37	23.92	22.46	21.00	19.55	18.09	16.63	15.18	13.72
Change (%)	35%	28%	21%	14%	7%		-7%	-14%	-21%	-28%	-35%

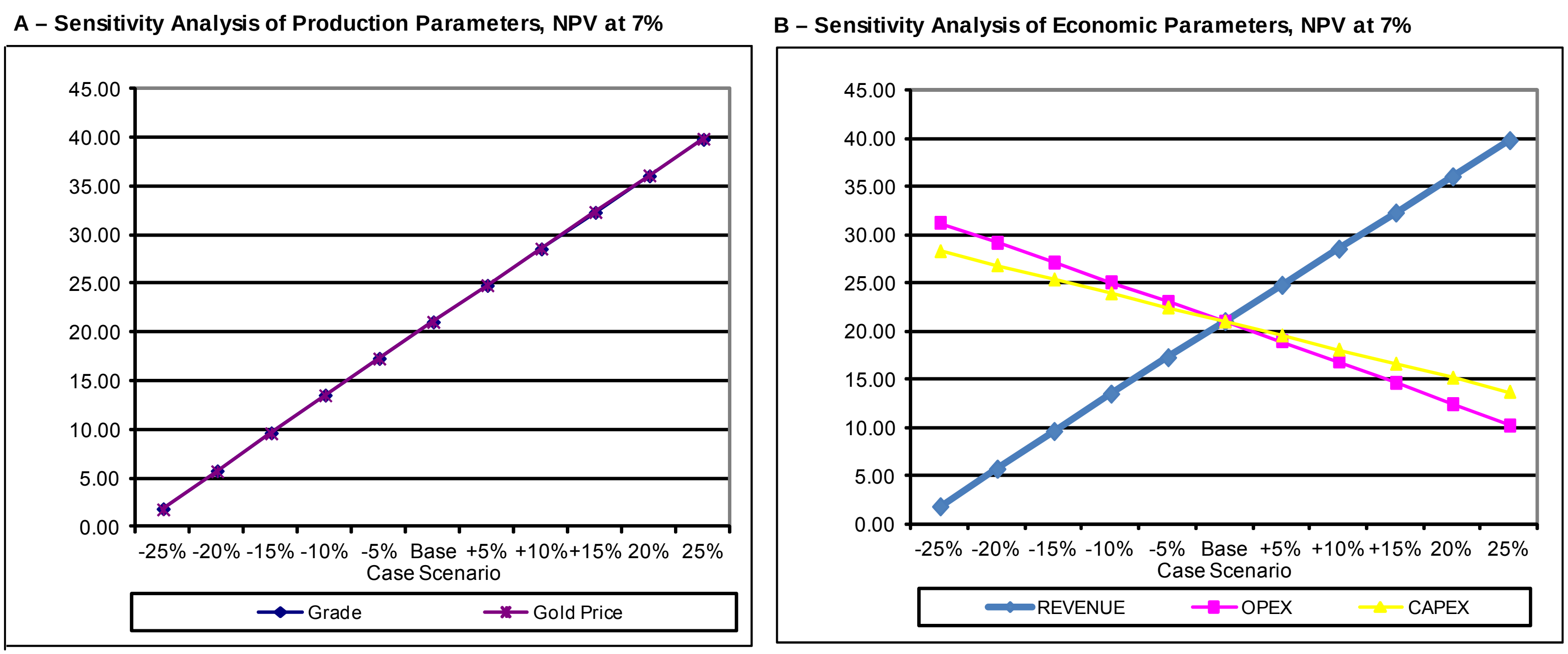
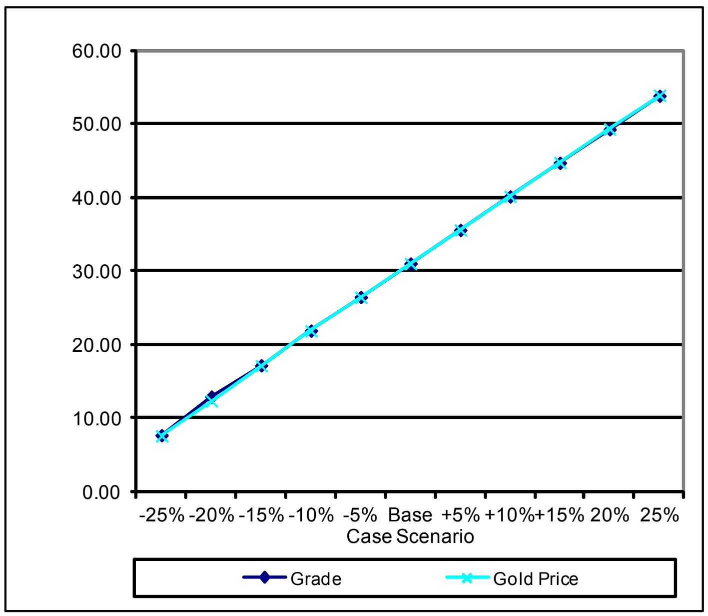


Figure 22.1 – Sensitivity diagram on NPV at a discount rate of 7%

Table 22.6 – Sensitivity analysis on total cash flow

	-25%	-20%	-15%	-10%	-5%	Base Case Scenario	+5%	+10%	+15%	20%	25%
PRODUCTION PARAMETERS											
Grade	7.67	12.91	17.16	21.88	26.44	30.99	35.55	40.11	44.67	49.23	53.78
Change (%)	-75%	-58%	-45%	-29%	-15%		15%	29%	44%	59%	74%
Gold Price	7.59	12.35	17.11	21.85	26.42	30.99	35.57	40.14	44.71	49.29	53.86
Change (%)	-76%	-60%	-45%	-29%	-15%		15%	30%	44%	59%	74%
ECONOMIC PARAMETERS											
REVENUE	7.67	12.91	17.16	21.88	26.44	30.99	35.55	40.11	44.67	49.23	53.78
Change (%)	-75%	-58%	-45%	-29%	-15%		15%	29%	44%	59%	74%
OPEX	43.06	40.62	38.20	35.78	33.38	30.99	28.62	26.16	23.55	20.95	18.34
Change (%)	39%	31%	23%	15%	8%		-8%	-16%	-24%	-32%	-41%
CAPEX	39.14	37.51	35.88	34.25	32.62	30.99	29.37	27.74	26.11	24.48	22.85
Change (%)	26%	21%	16%	11%	5%		-5%	-10%	-16%	-21%	-26%

A – Sensitivity Analysis of Production Parameters, Total Cash Flow



B – Sensitivity Analysis of Economic Parameters, Total Cash Flow

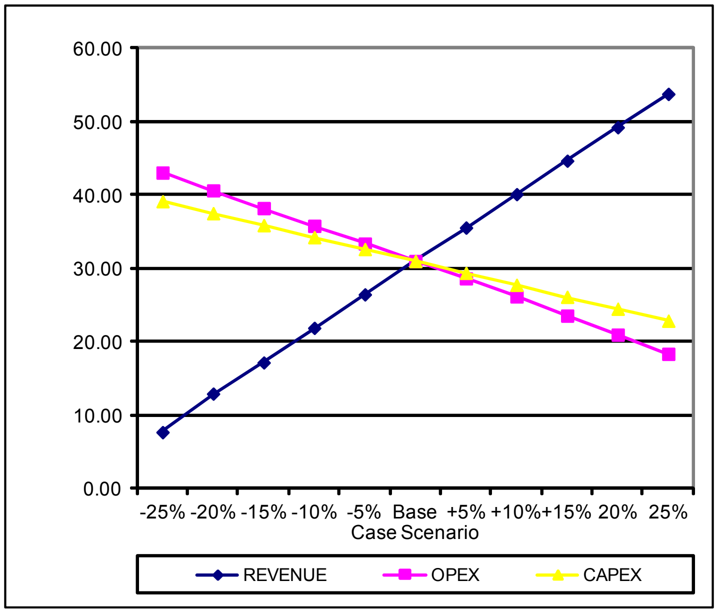
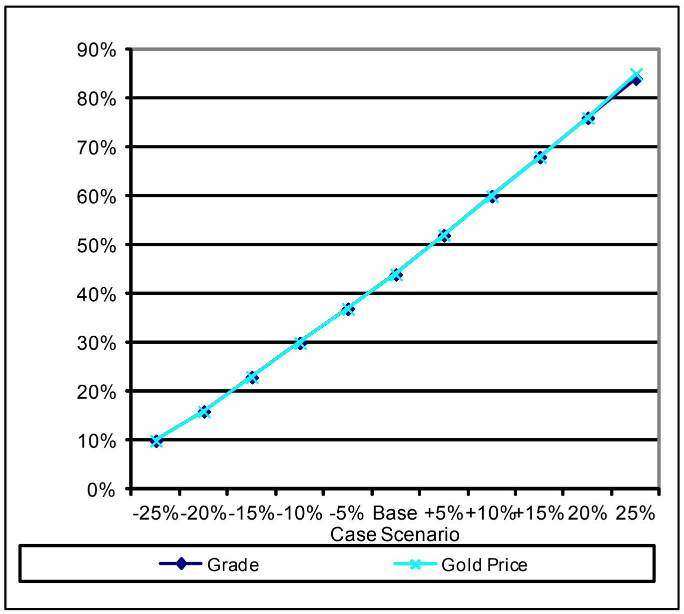


Figure 22.2 – Sensitivity diagram on total cash flow

Table 22.7 – Sensitivity analysis of IRR

	-25%	-20%	-15%	-10%	-5%	Base Case Scenario	+5%	+10%	+15%	20%	25%
PRODUCTION PARAMETERS											
Grade	10%	16%	23%	30%	37%	44%	52%	60%	68%	76%	84%
Change (%)	*-77%*	*-64%*	*-48%*	*-32%*	*-16%*		*18%*	*36%*	*55%*	*73%*	*91%*
Gold Price	10%	16%	23%	30%	37%	44%	52%	60%	68%	76%	85%
Change (%)	*-77%*	*-64%*	*-48%*	*-32%*	*-16%*		*18%*	*36%*	*55%*	*73%*	*93%*
ECONOMIC PARAMETERS											
REVENUE	10%	16%	23%	30%	37%	44%	52%	60%	68%	76%	84%
Change (%)	*-77%*	*-64%*	*-48%*	*-32%*	*-16%*		*18%*	*36%*	*55%*	*73%*	*91%*
OPEX	71%	65%	60%	54%	49%	44%	40%	35%	31%	27%	23%
Change (%)	*61%*	*48%*	*36%*	*23%*	*11%*		*-9%*	*-20%*	*-30%*	*-39%*	*-48%*
CAPEX	71%	64%	59%	54%	49%	44%	41%	37%	34%	30%	27%
Change (%)	*61%*	*45%*	*34%*	*23%*	*11%*		*-7%*	*-16%*	*-23%*	*-32%*	*-39%*

A – Sensitivity Analysis of Production Parameters, IRR



B – Sensitivity Analysis of Economic Parameters, IRR

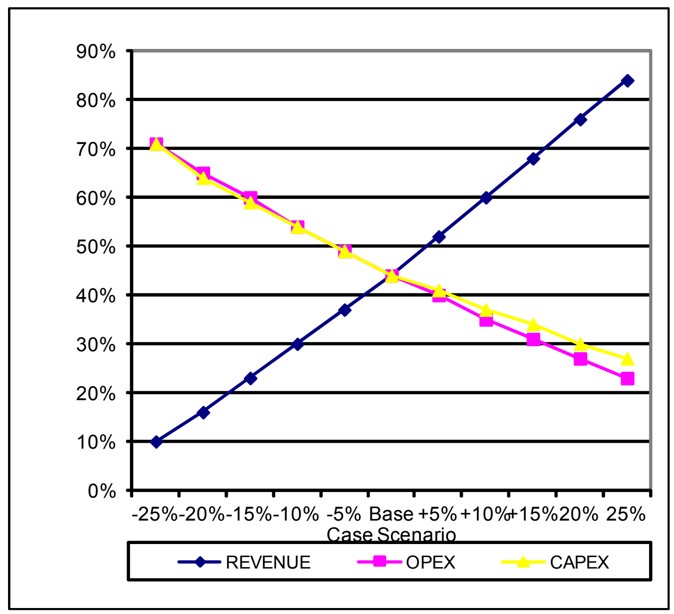


Figure 22.3 – Sensitivity diagram for total IRR

23.0 ADJACENT PROPERTIES

Two of the properties surrounding the Croinor property are owned by Blue Note Mining: the Pershing property to the south and the Bel-Rive property to the North. Between the Croinor and Bel-Rive properties, there is a block of claims owned by Yves Lemieux. To the west, the property is limited by a block of claims owned by René Rousseau. Three other properties are in contact with the Croinor property: a property owned by Brionor Resources to the north, one owned by Wen Fan to the east, and one owned by Canadian Mining House to the south (Fig. 23.1).

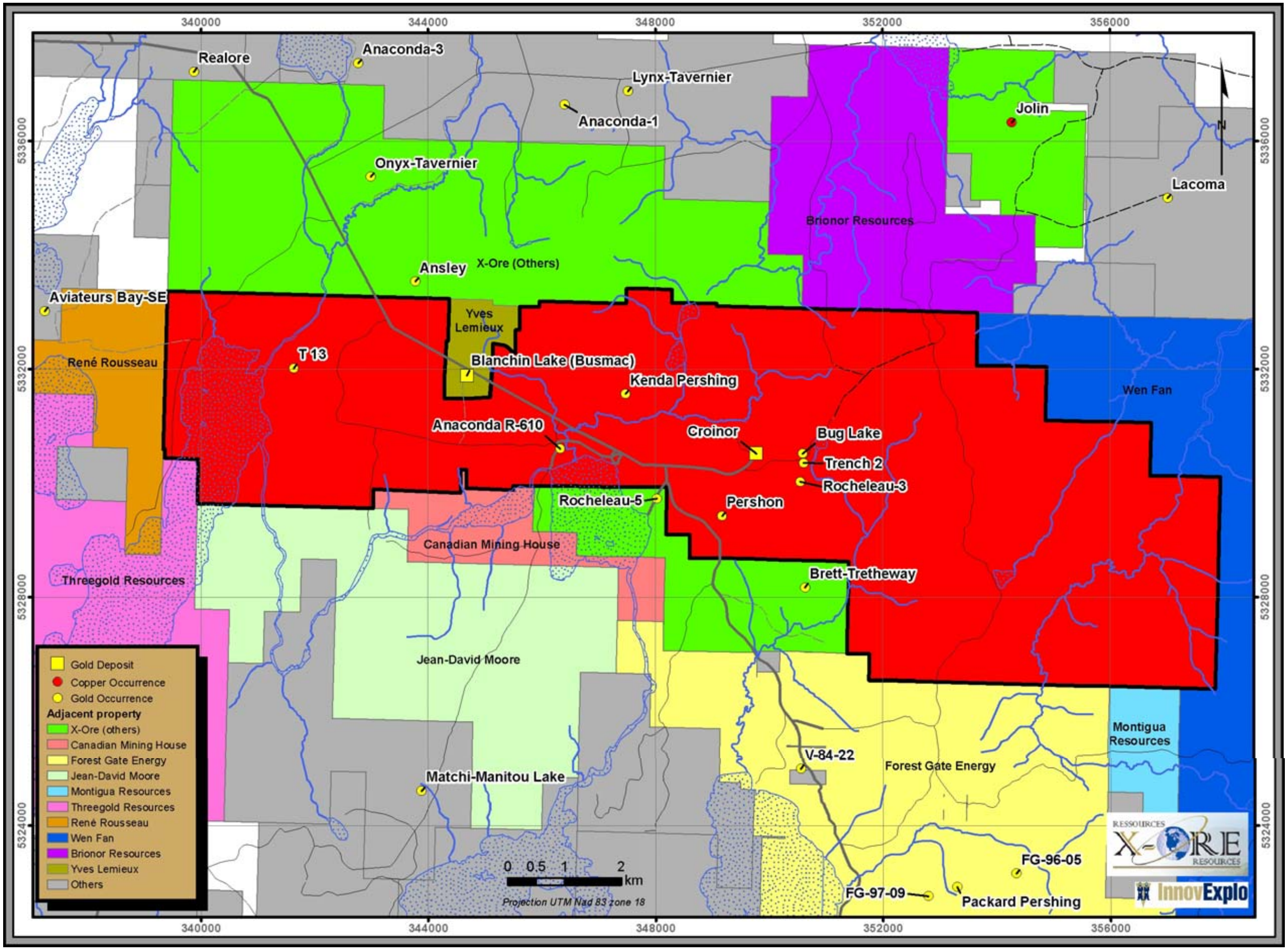


Figure 23.1 – Properties and mineral occurrences in the vicinity of the Croinor property

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional information to be included in this report.

25.0 INTERPRETATION AND CONCLUSIONS

The first objective of the 2010-2011 drilling programs was to improve the level of understanding of mineralization in areas with densely spaced historical surface drilling by drilling new holes in the vicinity of the known reserves and at depth. The second objective was to test the continuity of the mineralization from east to west and at depth. This work resulted in additional information being *added* to the 2009 resource calculation by O'Dowd. This report presented the results achieved.

The 2010 and 2011 drill program brought to light the following interpretations and conclusions with respect to the mineral deposit:

- There is an east to west continuity of high-grade mineralization within zones, which can be traced from section to section. Some areas may not be high-grade, but they do show anomalous positive values that connect with the high-grade results;
- Hole CR-11-408 shows that the mineralised zones and the diorite host rocks are still open at depth. This also suggests a further east to west extension at depth for the zones intercepted along the hole;
- The east and west limits of the host rock diorite are still open. Drill holes show the continuity of the high-grade results, indicating that the mineralized zones are also open to the east and west;
- The drill spacing in the areas containing Indicated Resources is not consistent and additional drilling in these areas could increase the Indicated Resources;
- The drill spacing in the areas containing Inferred Resources is not tight enough to increase confidence in the results and convert some of the resources to the Indicated category, or to increase the Inferred Resources overall; additional drilling in these areas could convert Inferred Resources into Indicated Resources or increase the Inferred Resources;
- The 3D IP targets, when drilled, yielded some elevated gold results within the expected range of influence.

The results from the 2010 and 2011 drilling programs presented in the current report were used to update the Mineral Resource Estimate.

The estimated proven and probable reserves resulting from the prefeasibility study, as summarized in the following table, represent 120,883 oz of gold after applying the appropriate mining recovery and dilution factor according to the selected method.

Mineral Reserve Estimate (Table 15.1)

Category	Tonnes	Grade (g/t)	Ounces
Proven	68,849	6.23	13,789
Probable	498,023	6.69	107,094
Total Reserves	566,872	6.63	120,883

The current study reports a lower tonnage and grade, resulting in less ounces compared to the 2010 prefeasibility study. The lower tonnage is due to changes in the resource estimation methodology, the geological interpretation, and the resource category criteria. The lower grade is due to an increase in the application of the long-hole mining method. In the current mine plan, 75% of the resources would be mined using the long-hole method as opposed to 20% in the 2010 prefeasibility study. This method was applied as much as possible due to the lower mining cost and higher productivity.

Development and production activities are based on a schedule of two 10-hour shifts, 12 shifts a week. To minimize capital requirements, contractors will be used for all mine development, mine production and ore haulage activities. A small in-house staff workforce will be hired to provide technical support and direction to the contractors.

The expected production rate starts at an average of 475 tpd in the six months following pre-production, and will slowly ramp up to an average of 675 tpd in Year 4. The overall project mine life is expected to be approximately 5 years, including an 18-month pre-production period.

After mining and milling recoveries, InnovExplo's prefeasibility study estimates a production total of 117,956 ounces of gold over a 5-year period. This represents 566,429 tonnes of ore at a diluted grade of 6.64 g/t. A summary of the yearly mine plan is presented in the following table.

Mining plan tonnage distribution (Table 16.15)

	Year 1	Year 2		Year 3	Year 4	Year 5	Total
	Pre-prod	Pre-Prod	Prod	Month 25-36	Month 36-48	Month 49-58	
Long hole (t)	0	19 110	53 455	81 966	154 299	112 746	**421 576**
Grade (g/t)	0.00	6.05	7.70	5.17	6.18	5.48	**5.98**
Development(t)	4 914	7 623	6 552	20 686	12 726	0	**52 501**
Grade (g/t)	5.63	6.40	5.73	5.42	5.75	0.00	**5.70**
Room and pillar (t)	0	4 554	3 522	33 319	34 584	16 374	**92 352**
Grade (g/t)	0.00	9.06	6.66	14	8.66	7.41	**10.19**
Mill tonnage(mt)	4 914	31 287	63 529	135 971	201 609	129 120	**566 429**
Grade (g/t)	5.63	6.58	7.44	7.29	6.58	5.73	**6.64**

The Croinor mine is currently flooded to the portal elevation. As an initial step, an estimated 504,080 m^3 of mine water would be pumped from the existing pit and mine infrastructure. Initial dewatering is expected to be carried out at a rate of 5,500 m^3/day for an approximate period of 92 days. Mine water will be pumped and processed through Envirotubes to collect mine sludge. The existing 200-metre deep shaft will be reconditioned up to level 500 and will be used as a ventilation raise and emergency escapeway. Ore and waste haulage to surface will be via ramp.

Genivar was commissioned by InnovExplo and Blue Note Mining to review custom milling options in the context of a prefeasibility study for the Croinor gold project. Only four gold concentrators located within a 120-km radius could potentially process the Croinor ore: Beacon Gold, Aurbel Gold, Sigma-Lamaque Complex and Camflo. This study assumes that the ore will be processed at the Camflo mill. In addition to having successfully processed Croinor ore in the past (hence mitigating the technical risk), the plant has been in the custom milling business for years and is likely to have availability for new custom feed that fits the life-of-mine requirements for Croinor. Ore previously mined from the Croinor open pit operations was processed at the Camflo mill, and based on the results of those runs, a gold recovery of 97.5% has been used in the present prefeasibility study.

InnovExplo prepared a preliminary design for the proposed project infrastructure. Most of the capital cost was estimated using quotes from equipment suppliers and contractors. In some cases, comparable installations at other projects were used. The capital cost estimate is accurate within ±20%.

The pre-production costs are estimated at $20.68 million, including $2.4 million of capitalized operating costs net of production revenue received during the pre-production period. Sustaining capital is estimated at $17.88 million, excluding $0.66 million for final closure costs. The cost breakdown is presented in the following table.

Capital expenditure breakdown (Table 21.1)

Description	Pre-production	Sustaining	Total cost
Capitalized operating cost	$16,363,677		$16,363,677
Capitalized revenue	($13,950,520)		($13,950,520)
Dewatering and rehabilitation	$1,249,609		$1,249,609
Development	$4,537,911	$10,760,313	$15,298,224
Ventilation equipment	$340,075		$340,075
Mine dewatering	$442,718	$56,614	$499,331
Surface installation and equipment	$2,081,591	$670,628	$2,752,219
Electrical distribution	$6,029,064	$1,232,000	$7,261,064
Mobile equipment	$2,955,638	$4,903,494	$7,859,132
Environment	$371,596	$258,162	$629,758
Contractor demobilization	$255,642		$255,642
Total capital expenditures	**$20,677,000**	**$17,881,210**	**$38,558,210**

Operating costs are estimated in 2011 Canadian dollars with no allowance for escalation. The total life-of-mine operating cost and average unit operating costs are summarized in the following table. The overall unit operating cost is $164/tonne of milled ore.

InnovExplo estimated mine operating costs using data from budget quotes from contractors and suppliers.

Summary of Total Life-of-Mine Operating Costs (Table 21.12)

Description	Total cost	Unit cost	
Definition drilling	$2,332,270	4.40 $/t	20.44 US$/oz
Stope development	$16,039,477	30.25 $/t	140.55 US$/oz
Mining	$19,372,502	36.54 $/t	169.76 US$/oz
Blue Note staff	$10,847,925	20.46 $/t	95.06 US$/oz
Contractor (indirect cost)	$10,885,100	20.53 $/t	95.38 US$/oz
Surface services	$189,508	0.36 $/t	1.66 US$/oz
Energy cost	$4,482,168	8.45 $/t	39.28 US$/oz
Milling and transportation	$22,055,439	41.60 $/t	193.27 US$/oz
Environment	$779,739	1.47 $/t	6.83 US$/oz
Total:	**$$86,984,128**	**164 $/t**	**762 US$/oz**

To supply electric power to the site, a new 26 km 25 kV three-phase overhead power line is planned. In this study, this new power line is assumed to be held and owned by Hydro-Québec. Further discussions between the client and Hydro-Québec will be required to define the project.

An after-tax model was developed for the Croinor project. All costs are in 2011 Canadian dollars with no allowance for inflation or escalation.

The economic valuation of the project was performed using the Internal Rate of Return (IRR) and Net Present Value (NPV) methods. The discount rate used in the analysis is 7%. The following parameters were considered in the financial analysis:

- An average gold price of US$1495/oz and an exchange rate of 1.03CAD/1USD, which correspond to the Bloomberg consensus estimate of December 19, 2011. It also considers that pre-production would be initiated in April 2012. Table 22.1 gives the details of the Bloomberg base case consensus forecasts.

Bloomberg base case consensus forecasts as of December 19, 2011 (Table 22.1)

	2012	2013	2014	2015
Gold price ($US/oz)	1,834	1,893	1,572	1,506
Exchange rate ($C/$US)	1.03	1.01	1.04	1.01

- Reserve parameters are as described in section 15.
- Gold recovery of 97.5%. This value was based on the recovery obtained at the time the mine was operating.
- Royal Mint fees of $5/oz.
- Estimated average annual tonnage of 94,820 to 201,610 tonnes.
- Estimated average annual output of 21,260 to 41,580 ounces of gold.
- A royalty payment was considered and evaluated as follows: a royalty of 15% was applied on profit over the carried expenses, which amounts to $13,153,158
- Future annual cash flow estimates based on grade, gold recoveries and cost estimates are as discussed in this report.
- A total of 36,201 tonnes of ore, which will be processed during the pre-production period, is deemed to be capital production and is not included in production, nor is the revenue derived from it.

The resulting main parameters and results of the cash flow analysis for the entire project are presented in the following table.

Cash flow analysis summary (Table 22.2)

Parameters	Results
Proven & Probable mineral reserves	566,429 t at 6.64 g/t
Total contained gold reserve	120,883 oz
Mine life (including 18-month pre-production)	5 years
Daily mine production	425 tpd ramping up to 675 tpd in Year 4
Gold recovery	97.5%
Annual gold production	21.259 to 41,578 oz
LOM recovered gold	117,956 oz
Average cash operating cost	$164/tonne
Average cash operating cost	US$762/oz
Capital cost (including $17.9M sustaining/working capital)	$38.6 million
Total cost per ounce	US$1,032/oz
Total gross revenue	$ 182 million
Total operating cost	$87.0 million
Total project cost	**$ 125.6 million**
Total operating cash flow (before tax & royalties)	**$47.2 million**
Estimated mining and income taxes	**$12.5 million**

Parameters	Results
Net cash flow	**$31 million**
Pre-tax NPV (7% discount)	**$30.6 million**
Pre-tax IRR	**57%**
After-tax NPV (7% discount)	**$21 million**
After-tax IRR	**44%**
Payback period	**38 months**
Pre-production period (including 36,201t of production)	**18 months**

Sensitivity analyses were performed on parameters selected for their potential impact on the outcome of the economic evaluation. The following production parameters were analyzed:

- Grade (g/t)
- Gold price (US$/oz)
- Total net revenue (REVENUE)
- Operating expenditure (OPEX)
- Capital expenditure (CAPEX)

Sensitivity calculations were performed on the project's NPV, IRR and total cash flow by applying a range of variation (±25%) to the parameter values.

The Croinor Project is highly sensitive to changes in grade, gold price and revenue. It is moderately sensitive to changes in OPEX and to changes in CAPEX.

Other than the drill hole CR-11-414 to CR-11-419, that are not included in the current resource estimate because they had pending assays at that time the cut-off date (mid-November) of the Mineral Resource Estimate, InnovExplo is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that would materially affect the Mineral Reserve Estimate. InnovExplo considers the present prefeasibility study to be reliable and thorough, based on quality data, reasonable hypotheses and parameters compliant with Regulation 43-101 and CIM standards with regard to Mineral Reserve and Resource estimates.

26.0 RECOMMENDATIONS

The results from this Mineral Reserve Estimate and Prefeasibility Study demonstrate that the Croinor Project is technically and economically viable and InnovExplo recommends that Blue Note Mining continues to advance the project towards production.

InnovExplo proposes an exploration program that includes follow-up drilling. The drilling program will augment the information on the Indicated Resources before mining begins. The program should also include the Inferred zone in an effort to convert those resources to Indicated. Included in the program should be a follow-up on the extensions to the east and west and at depth to increase the extent of the Inferred Resources. A simplified sketch (Fig. 26.1) shows the current understanding and location of the Indicated and Inferred zones to be targeted by drilling. The sketch also shows significant intercepts located beyond the eastern and western limits as defined by the 2011 drilling program.

The following recommendations pertain to the Mineral Resources and Mineral Reserves. InnovExplo provides the following recommendations:

- Continue infill and down-plunge exploration drilling aimed at expanding the current resources.
- Update the Mineral Resource Estimate with further drilling.
- Pursue discussions with Hydro-Québec to finalize a contract and define the scope of work to advance the design of the electric power line extension to the feasibility study stage and initiate related permitting requests.
- Continue to work on general permitting for the project.
- Complete additional engineering work to optimize the mine design and production schedule;
- Evaluate the possibility of applying ore sorting technology at the Croinor site.
- Prepare contract documents for the activities to be contracted for the work.
- Pursue negotiations to finalize agreements for custom milling and ore transportation.

To advance the project, InnovExplo estimates an overall budget of approximately $2,185,000 as presented in Table 26.1. A brief breakdown of the approximate cost is as follows:

Table 26.1 – Proposed Work Program and Budget

Item	Cost
Exploration and definition drilling program	
Definition drilling : (approx. 8,000 m at $110/m)	$880,000
Exploration drilling: (approx. 10,000 m at $110/m)	$1,100,000
Property-scale exploration, to locate diorite and verify surface outcrops	$100,000
Subtotal exploration:	**$2,080,000**
Preparation to advance project towards production	
Electric power line extension feasibility study	$20,000
General permitting for the project	$25,000
Complete additional engineering work to optimize the mine design and production schedule	$30,000
Evaluate the possibility of applying ore sorting technology at the Croinor site	$15,000
Prepare contract documents for the activities to be contracted for the work	$15,000
Subtotal preparation work:	**$105,000**
Grand Total:	**$2,185,000**

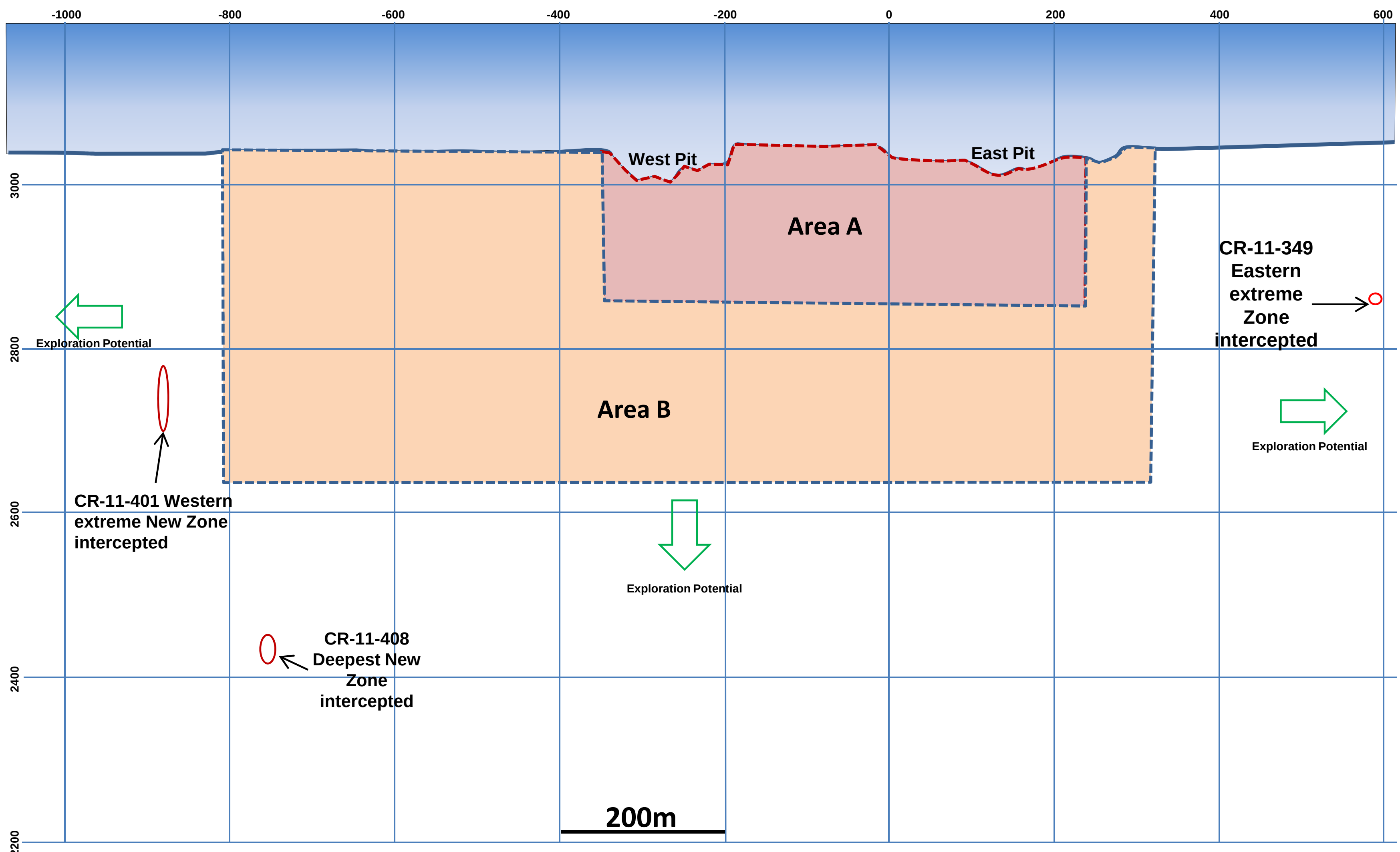


Figure 26.1 – Simplified longitudinal looking north. Area A represents the influence of the Measured and Indicated resources and Area B represents the influence of the Inferred Resources. The intercept areas for the best holes show the potential expansion of the current known resources.

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APPENDIX I

UNITS, CONVERSION FACTORS, ABBREVIATIONS

Units

Units in this report are metric unless otherwise specified. Precious metal content is reported in gram of metal per metric ton (g/t Au or Ag) except otherwise stated. Tonnage figures are dry metric tons unless otherwise stated. The ounces are in Troy ounces.

Abbreviation used

°C	degrees Celsius	oz	troy ounces
ha	hectares	avdp	avoirdupois pound
g	grams	st	short ton
kg	kilograms	oz/t	ounces per short ton
mm	millimetres	t	metric ton (tonne)
cm	centimetres	Mt	millions of metric tonnes
m	metres	g/t	grams per metric ton
km	kilometres	tpd	metric tons per day
masl	metres above sea level	ppb	parts per billion
’ or ft	feet	ppm	parts per million
cfm	cubic feet per minute	cps	counts per second
m^3/min	cubic metres per minute	hp	horsepower
Mbs	megabytes per second	Btu	British thermal units
$ or C$ or CAD	Canadian dollars	kV/kVA	kilovolts/kilovolt-amps
US$ or USD	American dollars	MPa	mega pascals

Conversion factors for measurements

Imperial Unit	Multiplied by	Metric Unit
1 inch	25.4	mm
1 foot	0.305	m
1 acre	0.405	ha
1 ounce (troy)	31.103	g
1 pound (avdp)	0.454	kg
1 ton (short)	0.907	t
1 ounce (troy) / t (short)	34.286	g/t

APPENDIX II

CROINOR MINERAL RESOURCE ESTIMATE DETAILED BY ZONE

Table II.1 – Measured and Indicated Resources by zone at a cut-off grade of 4.0 g/t Au

Measured			
Zone	Tonnes	Au g/t	Oz Au
Zone 1	8 193	5.57	1 468
Zone 2	31 413	10.16	10 258
Zone 3	0	0.00	0
Zone 4	7 469	6.52	1 565
Zone 5	504	4.74	77
Zone 6	10 554	7.55	2 560
Zone 7	0	0.00	0
Zone 8	0	0.00	0
Zone 9	1 066	8.30	285
Zone 10	735	6.16	146
Zone 11	0	0.00	0
Zone 12	0	0.00	0
Zone 13	210	5.00	34
Zone 14	0	0.00	0
Zone 15	0	0.00	0
Zone 16	20 375	8.19	5 367
Zone 17	0	0.00	0
Zone 18	0	0.00	0
Zone 19	0	0.00	0
Zone 20	0	0.00	0
Zone 21	0	0.00	0
Zone 22	0	0.00	0
Zone 23	0	0.00	0
Zone 24	0	0.00	0
Zone 25	0	0.00	0
Zone 26	0	0.00	0
Zone 27	0	0.00	0
Zone 28	0	0.00	0
Zone 29	0	0.00	0
Zone 30	0	0.00	0
Zone 31	0	0.00	0
Zone 32	0	0.00	0
Zone 33	0	0.00	0
Zone 34	0	0.00	0
Zone 35	0	0.00	0
Zone 36	0	0.00	0
Zone 37	0	0.00	0
Zone 38	0	0.00	0
Zone 39	0	0.00	0
Zone 40	0	0.00	0
Zone 41	0	0.00	0
Zone 42	0	0.00	0
Zone 44	0	0.00	0
Zone 45	0	0.00	0
Zone 48	0	0.00	0
Zone 50	0	0.00	0
Zone 55	0	0.00	0
Zone 70	0	0.00	0
Zone 75	0	0.00	0
Zone 80	0	0.00	0
Zone 85	0	0.00	0
Zone 90	0	0.00	0
Zone 95	0	0.00	0
Zone 100	0	0.00	0
TOTAL	**80 517**	**8.41**	**21 759**

**** Not rounded***

Indicated			
Zone	Tonnes	Au g/t	Oz Au
Zone 1	14 208	7.45	3 402
Zone 2	58 953	10.30	19 519
Zone 3	1 586	6.82	348
Zone 4	39 264	7.71	9 738
Zone 5	84 729	9.30	25 344
Zone 6	68 528	9.57	21 093
Zone 7	901	5.94	172
Zone 8	8 304	6.47	1 727
Zone 9	13 087	7.15	3 009
Zone 10	17 341	10.12	5 641
Zone 11	2 229	4.30	308
Zone 12	0	0.00	0
Zone 13	50 704	9.26	15 094
Zone 14	67 750	11.05	24 067
Zone 15	30 590	6.53	6 424
Zone 16	23 884	9.07	6 967
Zone 17	106	4.74	16
Zone 18	33 167	10.11	10 779
Zone 19	0	0.00	0
Zone 20	21 561	12.06	8 358
Zone 21	0	0.00	0
Zone 22	95	5.72	17
Zone 23	132	4.61	20
Zone 24	12 847	7.79	3 216
Zone 25	705	5.44	123
Zone 26	1 342	9.21	397
Zone 27	0	0.00	0
Zone 28	24 815	6.80	5 425
Zone 29	290	4.26	40
Zone 30	11 131	6.35	2 271
Zone 31	1 751	11.94	672
Zone 32	0	0.00	0
Zone 33	2 716	9.59	837
Zone 34	0	0.00	0
Zone 35	1 625	16.30	852
Zone 36	0	0.00	0
Zone 37	53	4.46	8
Zone 38	0	0.00	0
Zone 39	0	0.00	0
Zone 40	0	0.00	0
Zone 41	0	0.00	0
Zone 42	0	0.00	0
Zone 44	0	0.00	0
Zone 45	133	4.45	19
Zone 48	2 209	5.45	387
Zone 50	0	0.00	0
Zone 55	0	0.00	0
Zone 70	0	0.00	0
Zone 75	0	0.00	0
Zone 80	0	0.00	0
Zone 85	0	0.00	0
Zone 90	0	0.00	0
Zone 95	2 831	6.31	574
Zone 100	0	0.00	0
TOTAL	**599 565**	9.18	**176 866**

**** Not rounded***

Table II.2 – Inferred Resources by zone at a cut-off grade of 4.0 g/t Au

Inferred			
Zone	Tonnes	Au g/t	Oz Au
Zone 1	0	0.00	0
Zone 2	0	0.00	0
Zone 3	0	0.00	0
Zone 4	109	4.44	16
Zone 5	15 235	6.80	3 331
Zone 6	7 287	7.12	1 668
Zone 7	0	0.00	0
Zone 8	1 282	6.23	257
Zone 9	3 657	5.36	630
Zone 10	0	0.00	0
Zone 11	12 455	4.39	1 759
Zone 12	0	0.00	0
Zone 13	3 209	8.00	825
Zone 14	8 223	8.33	2 203
Zone 15	27 312	5.00	4 390
Zone 16	0	0.00	0
Zone 17	0	0.00	0
Zone 18	0	0.00	0
Zone 19	0	0.00	0
Zone 20	360	5.63	65
Zone 21	0	0.00	0
Zone 22	0	0.00	0
Zone 23	0	0.00	0
Zone 24	0	0.00	0
Zone 25	58	5.64	11
Zone 26	0	0.00	0
Zone 27	0	0.00	0
Zone 28	550	5.40	96
Zone 29	0	0.00	0
Zone 30	1	4.75	0
Zone 31	0	0.00	0
Zone 32	1 661	11.23	600
Zone 33	0	0.00	0
Zone 34	3 200	6.57	676
Zone 35	0	0.00	0
Zone 36	0	0.00	0
Zone 37	0	0.00	0
Zone 38	1 273	5.14	210
Zone 39	0	0.00	0
Zone 40	0	0.00	0
Zone 41	10 793	11.60	4 025
Zone 42	0	0.00	0
Zone 44	2 351	7.00	529
Zone 45	0	0.00	0
Zone 48	184	5.39	32
Zone 50	15 537	8.85	4 422
Zone 55	5 604	10.73	1 934
Zone 70	5 639	22.20	4 026
Zone 75	21 366	15.06	10 345
Zone 80	3 993	5.83	748
Zone 85	2 455	4.91	388
Zone 90	5 903	4.26	809
Zone 95	380	5.70	70
Zone 100	62	4.54	9
TOTAL	**160 140**	8.56	**44 071**

Not rounded

APPENDIX III
DESCRIPTION (IN FRENCH ONLY) OF THE LABORATORY SAMPLE PREPARATION AND ANALYTICAL PROCEDURES IN 2007 AND 2008

1- Réception des échantillons

a) Les échantillons reçus sont classés et comparés à la liste/demande d'analyse fournie par le client. Ce dernier est contacté afin de clarifier toute disparité. Les échantillons sont, par la suite, placés dans le séchoir et séchés à 100-110°C aussi longtemps que nécessaire (typiquement, 3 à 4 heures).

b) Par la suite, les échantillons sont concassés (Rhino Jaw crusher, T.M. Engineering) selon la granulométrie désirée.

c) L'échantillon concassé est versé dans un séparateur (riffle splitter), autant de fois que nécessaire, jusqu'à l'obtention d'un sous échantillon d'environ 200 à 300g.

d) Le sous échantillon est pulvérisé (Ring pulverizer, T.M. Engineering) jusqu'à l'obtention d'une pulpe de la granulométrie souhaitée et transféré à la section pyro analyse.

2- Pyro analyse et analyse du produit

a) Le sous échantillon est homogénéisé puis pesé au poids désiré (30 ou 50g). Cet aliquot est transféré dans un creuset, auquel est ajouté une quantité appropriée de mélange fondant, ainsi que tout additif jugé nécessaire par le personnel qualifié. L'échantillon est placé dans la fournaise de fusion (1090°C), fusionné et versé dans un moule conique. Lorsqu'il a suffisamment refroidi, la scorie est séparée du bouton de plomb et celui-ci est coupellé. Le produit final, une bille d'argent contenant les métaux précieux, est par la suite dissoute et analysée par absorption atomique (méthode TMT-G5B) ou traitée à l'acide nitrique, détrempée et pesée (méthode TMT-G5C).

b) La limite de détection de la méthode gravimétrique, 0.15 g/t, a été déterminée à l'aide d'une étude statistique des résultats de l'analyse de 10 réplicas d'un échantillon. La teneur moyenne en or de cet échantillon était de 0.80 ± 0.05g/t, la limite de détection étant définie comme étant le triple de l'écart-type, soit 0.15 g/t. Une approche similaire est utilisée pour la méthode d'absorption atomique avec une limite de détection de 0.005 g/t (5 ppb).

c) L'analyse des métaux de base (méthode TMT-G5A) est précédée d'une digestion à l'eau régale (*aqua regia*). L'analyse, en elle-même, est effectuée par absorption atomique à la flamme. La limite de détection des métaux de base est de 1 ppm et de 0.2 ppm pour l'argent.

3- Analyse fractionnée (metallic sieve)

a) Le «rejet» (portion restante concassée) est passé au séparateur, de manière à obtenir un sous échantillon représentatif (typiquement 250 à 350 g). Le sous échantillon est par la suite pulvérisé *de façon incomplète*, c'est-à-dire de façon à avoir une fraction de 15 à 30 % du sous échantillon ne passant pas dans un tamis de 140 mesh.

b) Le sous échantillon partiellement pulvérisé est pesé puis transféré dans un assemblage tamis/plat de réception. Le tout est placé sur un vibreur (Ro-Tap) pour une durée suffisante à la séparation des fractions.

c) Les fractions (fines et grossières) sont pesées, emballées séparément et transférées à la section pyro analyse. La fraction grossière est analysée au complet et la fraction fine voit 2 aliquots traités pour analyse par absorption atomique ou gravimétrie.

4- Densité

a) L'échantillon est pesé à son arrivé.

b) La balance est, par la suite, tarée face à un assemblage comprenant :

- Un plat à échantillon, placé sur le plateau de la balance
- Un récipient métallique «**B**» dans lequel plusieurs orifices ont été percés, afin de permettre à l'eau d'entrer et sortir librement
- Un câble, reliant le récipient à un anneau sous la balance (l'anneau fait parti du mécanisme de la balance)
- Un récipient métallique de plus petite taille «**A**», pouvant être complètement recouvert d'eau lorsque placé dans le récipient suspendu sous la balance et servant à contenir l'échantillon à mesurer
- Un récipient de plastique, de grande taille «C», situé sous la balance et rempli d'eau

c) La température de l'eau est mesurée, afin de pouvoir effectuer une correction dans les calculs, si nécessaire.

d) L'échantillon est placé dans le récipient «A»; de l'eau est ajoutée jusqu'à ce que l'échantillon soit recouvert complètement.

e) Le récipient «A» est transféré dans le récipient «B» et le tout est descendu lentement dans le récipient «C», de manière à ce que ce dernier soit complètement recouvert d'eau, tout comme le récipient «A».

f) Le poids indiqué sur la balance est noté et l'échantillon est par la suite envoyé au séchoir pour poursuivre le processus normal de préparation.

5- Contrôle de la qualité

a) La granulométrie des échantillons est vérifiée à intervalle régulier après le concassage et la pulvérisation (typiquement à chaque 20 échantillons).

b) Chaque série de 24 échantillons (ou moins) inclus un blanc de méthode, un duplica et 2 matériaux de référence certifiés.

c) La réanalyse d'un échantillon est effectuée lors du dépassement de la valeur limite dictée par le client ou lorsque l'analyste ou son superviseur le juge nécessaire.

d) Concasseurs et pulvériseurs sont nettoyés à l'aide de pièces de verre propres (concasseur) ou de verre pilé commercial (pulvériseur).

APPENDIX IV
PROPERTY MAP


DDH collars not surveyed
DDH collars 2010-2011
Road
Croinor Road
Watercourse
Waterbody
Wetland
Property
Open Pit
I2 por, Quartz-Felspar intermediate intrusive
I2J, Diorite
V2J, Intermediate volcanite and tuff
V3A, Mafic volcanite and tuff
V3B, Basalt
PRELIMINARY
X-ORE RESOURCES
Croinor Property
Location of 2010-2011 Diamond Drill Holes
0 12.5 25 50 Meters
1:1 500
Projection UTM Nad 83 zone 18
Compilation/dessin/drawing : L.Charbonneau, Tech. DAO/CAD
Date: May 31st, 2011
Supervisé/Supervised : T. Gomwe, GIT,(P. Geo.,Ph.D.) (OGQ 1277)
Date: May 31st, 2011
Approuvé/Approved : C. Pelletier, P. Geo., B. Sc. (OGQ 384)
Date: May 31st, 2011
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350600
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