Éléonore Gold Project
Quebec, Canada
NI 43-101 Technical Report

compliant with Regulation 43-101
and Form 43-101F1

Project Location

427470 E, 5839829 N (UTM Zone 18N (NAD83))

Effective Date: January 26, 2014
Signature Date: March 27, 2014

Qualified Persons:

Christine Beausoleil, P.Geo

Deny Fleury, P.Eng.

Andy Fortin, P.Eng.

Tony Brisson, P.Geo

Luc Joncas, P.Eng.

SIGNATURE PAGE

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Effective date: January 26, 2014
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(Original signed and sealed)
Christine Beausoleil, P.Geo
Goldcorp Inc.

Signed at Rouyn-Noranda on March 27, 2014

(Original signed and sealed)
Deny Fleury, Eng.
Goldcorp Inc.

Signed at Rouyn-Noranda on March 27, 2014

(Original signed and sealed)
Andy Fortin, Eng.
Goldcorp Inc.

Signed at Rouyn-Noranda on March 27, 2014

(Original signed and sealed)
Tony Brisson, P.Geo
Goldcorp Inc.

Signed at Rouyn-Noranda on March 27, 2014

(Original signed and sealed)
Luc Joncas, P.Eng
Goldcorp Inc.

Signed at Rouyn-Noranda on March 27, 2014

CERTIFICATE OF QUALIFIED PERSON

I, Christine Beausoleil, P.Geo., as an author of this report entitled "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report" with an effective date of January 26, 2014, prepared for Goldcorp Inc. (the "Issuer") do hereby certify that:

1. I am Chief Geologist (interim) Éléonore Project, Goldcorp Inc., 853 boul. Rideau, Rouyn-Noranda, Quebec, Canada, J9Y 0G3.
2. This certificate applies to the technical report entitled "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report" with an effective date of January 26, 2014 (the "Technical Report").
3. I am a graduate of Université du Québec à Montréal with a BSc. in Geology in 1997. I am a geologist and have been practicing my profession continuously since 1997. I am a Professional Geoscientist registered with the Ordre des Géologues du Québec (OGQ #656) and of the Association of Professional Engineers and Geoscientists of the province of British Columbia (APEGBC #36156).
4. I am familiar with National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and by reason of education, experience and professional registration I fulfill the requirements of a "qualified person" as defined in NI 43-101.
5. I have been working on the Éléonore Project full time since February 20th, 2012.
6. I am responsible for Section 1.11, 14 (all) and 25.5 of the Technical Report.
7. I am not an independent qualified person as described in section 1.5 of NI 43-101, as I am an employee of the Issuer.
8. I have no prior involvement with the property.
9. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th day of March, 2014

(Original signed and sealed)
Christine Beausoleil, P.Geo.,
Chief Geologist (interim), Éléonore Project
Goldcorp Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Denis Fleury, Eng., as an author of this report entitled "Éléonore Quebec, Canada NI 43-101 Technical Report" with an effective date of January 26, 2014, prepared for Goldcorp Inc. (the "Issuer") do hereby certify that:

1. I am Chief Engineer, Éléonore Project, Goldcorp Inc., 853 boul. Rideau, Rouyn- Noranda, Quebec, Canada J9Y 0G3.
2. This certificate applies to the technical report entitled "Éléonore, Quebec, Canada NI 43-101 Technical Report" with an effective date of January 26, 2014 (the "Technical Report").
3. I graduated with a B.Sc. degree in Mine Engineering from Ecole Polytechnique, Montreal, in 1985. I am a member in full standing of the Ordre des Ingenieurs du Quebec, under identification number 40177, ICM (100951) and AusIMM (211395). I have worked as a junior engineer from May 1986 to 1988. Following this experience, I obtained my professional registration, and I have worked as a Mining Engineer and Chief Engineer for a total of 26 years, with Remnor Mines - Noranda, Silidor Mine - Hemlo Gold, Louvicourt Mine - Aur Resources, George Fisher Mine, Xstrata Zinc, Argyle Diamonds Mine - Rio Tinto, Cannington Mine – BHPBilliton.
4. I am familiar with National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and Australian JORC Code. I also did the Reserves Evaluation for the Louvicourt Mine and was a CP for BHPBilliton and by reason of education, experience and professional registration I fulfill the requirements of a "qualified person" as defined in NI 43-101.
5. I work at the Éléonore Gold Project since June 2013.
6. I am responsible for sections 1.12, 1.13, 1.14, 15 (all sections), 16 (all sections), 18.1, 24.1, 25.6, 25.7 of the Technical Report.
7. I am not an independent qualified person as described in section 1.5 of NI
8. I have no prior involvement with the property.
9. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th day of March, 2014

(Original signed and sealed)
Denis Fleury, Eng.
Chief Engineer, Éléonore Project
Goldcorp Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Andy Fortin, Eng., as an author of this report entitled "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report" dated January 26, 2014 prepared for Goldcorp Inc. (the "Issuer") do hereby certify that:

1. I am Manager, Process and Surface Operations, Éléonore Project, Goldcorp Inc., at 333, 3 ième Rue, Bureau 2, Chibougamau, Québec , Canada, G8P 1N4.
2. This certificate applies to the technical report "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report" dated January 26, 2014 (the "Technical Report").
3. I am a member of Ordre des Ingénieurs du Québec. I graduated with a Bachelor of Science degree in Metallurgy from Laval University in 1995. I have a background of 19 years, in gold mineral processing, through 3 gold mine projects and two process plant expansions. I occupied several positions as Chief Metallurgist, Operation General Foreman, Project Manager and Process Plant Manager. I developed an expertise in process plant start-up and gold mineral processing. I have worked as a senior manager level for a total of 12 years.
4. I am familiar with National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and by reason of education, experience and professional registration I fulfill the requirements of a "qualified person" as defined in NI 43-101.
5. I have been working on the Éléonore Project full time since January, 2011.
6. I am responsible for Sections 1.10, 1.15, 1.16, 13.0, 17.0, 18.2, 18.3, 18.4, 18.5, 25.4, 25.8, and 25.9 'of the Technical Report.
7. I am not an independent qualified person as described in section 1.4 of NI 43-101, as I am an employee of the Issuer.
8. I was first involved in the metallurgical testing and gold recovery methods in January 2011.
9. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th day of March, 2014

(Original signed and sealed)
Andy Fortin, Eng.
Manager, Process and Surface Operations, Éléonore Project
Goldcorp Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Tony Brisson, P.Geo., as an author of this report entitled "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report" with an effective date of January 26, 2014, prepared for Goldcorp Inc. (the "Issuer") do hereby certify that:

1. I am Exploration Manager, Éléonore Project, Goldcorp Inc., 853 boul. Rideau, Rouyn-Noranda, Quebec, Canada, J9Y 0G3.
2. This certificate applies to the technical report entitled "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report" with an effective date of January 26, 2014 (the "Technical Report").
3. I am a graduate of Université du Québec à Chicoutimi with a BSc. in Geology in 1996. I am a geologist and have been practicing my profession continuously since 1996. I am a Professional Geoscientist registered with the Ordre des Géologues du Québec (OGQ #437).
4. I am familiar with National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and by reason of education, experience and professional registration I fulfill the requirements of a "qualified person" as defined in NI 43-101.
5. I have been working on the Éléonore Project full time since October 15th, 2012.
6. I am responsible for Section 1.3, 1.5, 1.6, 1.7, 1.8, 1.9, 1.23, 4,6,7,8,9,10,11,12,23,24,25,26 and 27 of the Technical Report.
7. I am not an independent qualified person as described in section 1.5 of NI 43-101, as I am an employee of the Issuer.
8. I have no prior involvement with the property.
9. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th day of March, 2014

(Original signed and sealed)
Tony Brisson, P.Geo.,
Exploration manager, Éléonore Project
Goldcorp Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Luc Joncas, P.Eng., as an author of this report entitled "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report" with an effective date of January 26, 2014, prepared for Goldcorp Inc. (the "Issuer") do hereby certify that:

1. I am Mining Manager at Éléonore Project, Goldcorp Inc., 853 boul. Rideau, Rouyn-Noranda, Quebec, Canada, J9Y 0G3.
2. This certificate applies to the technical report entitled "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report" with an effective date of January 26, 2014 (the "Technical Report").
3. I am a graduate of LAVAL University with a mining engineering degree in 1994. I am a mine engineer and have been practicing my profession continuously since 1994. I am a Professional Engineer registered with the Ordre des Ingenieurs du Québec (OIQ #117137).
4. I am familiar with National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and by reason of education, experience and professional registration I fulfill the requirements of a "qualified person" as defined in NI 43-101.
5. I have been working on the Éléonore Project full time since March 1st, 2010.
6. I am responsible for sections: 1, 2, 3, 5, 18.6, 18.7, 19, 20, 21, 22, 25.10, 25.11, 25.12, 25.13 and 25.14 of the Technical Report.
7. I am not an independent qualified person as described in section 1.5 of NI 43-101, as I am an employee of the Issuer.
8. I have no prior involvement with the property.
9. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 27th day of March, 2014

(Original signed and sealed)
Luc Joncas, P.Eng.,
Mine Manager, Éléonore Project
Goldcorp Inc.

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1. SUMMARY

Ms. Christine Beausoleil, P.Geo., Mr. Denis Fleury, P.Eng., Mr. Andy Fortin, P.Eng., Mr. Tony Brisson, P.Geo., and Mr. Luc Joncas, P.Eng. prepared this Technical Report (the "Report") for Goldcorp Inc. ("Goldcorp") on the wholly-owned Éléonore Gold Project (the "Project") located in Quebec, Canada. The Project hosts the Roberto gold deposit, which consists of the Roberto, East Roberto, and Zone du Lac lenses.

This Report supports the disclosure of Mineral Resources and Reserves for the Project, and is based on geological information and the latest geological interpretation related to the block model from October 2013. The contents of the Report are also based on the 2014 Éléonore Project Budget, an internal pre-feasibility study completed by Goldcorp in 2010, and the pre-feasibility update released in 2011. Goldcorp will be using the Report in support of its 2013 Annual Information Form filing.

The operating entity for the Project is a wholly-owned Goldcorp subsidiary, Les Mines Opinaca Ltée. For the purposes of this Report, "Goldcorp" is used to refer interchangeably to the parent and subsidiary companies.

1.1. Location, Climate, and Access

The Éléonore Project is located in the Lake Ell area, in the northeastern part of the Opinaca Reservoir of the James Bay region, in the province of Quebec, Canada. The Project is located approximately 350 km north of the towns of Matagami and Chibougamau, and 825 km north of Montreal.

The Project area is characterized by cold winters and short, cool summers. Precipitation varies throughout the year, reaching an average of 2 metres annually.

A permanent road links the property to the road to Hydro-Québec's Sarcelle hydro-electric facility. Workers are mobilized to the site via a year-round air strip located approximately 1.5 km north of the camp.

1.2. Agreements and Royalties

A royalty payable on production from Éléonore to Virginia Mines Inc. is set at 2.20% on the first 3 Moz of gold, increasing by 0.25% per million ounces thereafter, up to a maximum of 3.50%. The royalty payable in each period is adjusted up or down by an amount ranging between zero and 10%, depending on the gold price in effect during that period. An annual payment is required to the Cree Nation under a confidential Cree Collaboration Agreement.

1.3. Mineral Tenure and Surface Rights

The Éléonore Project comprises 369 contiguous claims totalling 19,274 ha. The claims are 100% owned by Les Mines Opinaca Ltée.

A request for a 284 ha Mining Lease covering the Eleonore deposit was sent to the Ministry of Natural Resources of Quebec (MNR) in 2011 and is currently under final review. In 2013, following the normal procedure, eleven (11) claims affected by the future Mining Lease were temporarily suspended by the MNR. The portion of these claims that extend beyond the Mining Lease boundaries will be re-activated once the Mining Lease will be delivered.

Claims are map-staked, not surveyed on the ground, and are valid for a two-year period and can be renewed every two years. Renewal fees are $127 per claim if renewed 60 days before the expiry date and $254 if renewed within the 60 days of renewal. In order to maintain tenure, exploration work equivalent ranging from $135 to $2,500 per claim is required depending on the number of renewals previously granted. Work expenditures that are in excess of the amount required for the term can be transferred to other contiguous claims that are within 4.5 km from the center of the claim or can be credited for future renewal.

In the opinion of the Qualified Persons ("QPs"), information from legal experts and Goldcorp experts support that the mineral tenure held is valid and sufficient to support the disclosure of Mineral Resources and Mineral Reserves.

Surface rights in the Project area are classified as Category III provincial territories as per the James Bay and Northern Quebec Agreement, which gives certain hunting and fishing rights to the First Nation communities of the region.

In the opinion of the QPs, there are no pre-existing surface rights which are in conflict with the development of the project. In Quebec, a mining lease is required before mining activity commences. The lease for the Project was signed by Minister of the MNR in February 2014.

1.4. Environment, Permitting and Socio-Economics

The Project is not required to be evaluated under the federal environmental process, and has achieved provincial permitting under Chapter II of the Environmental Quality Act (EQA) for a project located north of the 49th parallel. An Environmental Impact and Social Assessment (ESIA) was carried out, subject to consultation by the Cree
Nation, local communities and the general public, and culminated in Goldcorp obtaining the principal Global Certificate of Authorization for the Project on November 11, 2011.

With the Global Certificate of Approval granted, most of the remaining Certificates of Approval to be issued under Chapter I of the EQA for site construction activities have been received from the Ministry of Sustainable Development, Environment and Parks (MDDEP). Other applications for Certificates of Approval are currently being prepared and will be submitted to the appropriate ministries during 2014.

1.5. Geology and Mineralization

The Roberto deposit is a clastic sediment-hosted stockwork-disseminated gold deposit in an orogenic setting, but is an atypical deposit in that it displays only some of the characteristics associated with the greenstone-hosted quartz-carbonate vein deposits as described by the Geological Survey of Canada.

The Project is hosted in Achaean-age rocks of a volcano-sedimentary greenstone belt developed near the contact between the Opinaca and La Grande subprovinces of the Superior Province.

Rock units from the Opinaca Subprovince make up the northeast corner of the Project area. Lithologies are dominated by granite, granodiorites and heterogeneous assemblages of pegmatites, tonalites and granites from the Janin Intrusive Suite intermixed with partially migmatized paragneiss from the Laguiche Complex.

Rock units belonging to the La Grande Subprovince make up most of the Project area west of the contact and host the Roberto deposit. Lithologies are dominated by metasedimentary units of the Low Formation and include conglomerates, greywackes, arenites and cherts. Discordantly overlying the Low Formation are basalts and intermediate to felsic tuff units of the Kasak Formation. These units are folded in a large, NE-trending, 10-km-long open F1 fold. The Ell Lake diorite intrusion, also 10 km long, occupies the centre of the Project, more or less along the centre of the large fold structure.

The Roberto deposit is hosted in polydeformed greywacke units in contact with aluminosilicate-bearing greywacke and thin conglomerate units. The 1.9-km-long crescent shape of the deposit is the result of F2 folding.

To date, mineralization has been intersected to a vertical depth of 1,400 m. Gold-bearing zones are generally 5–6 m in true thicknesses, varying from 2 m to more than 20 m locally. All zones are still open at depth.

The numerous subparallel mineralized zones are characterized by gold-bearing quartz–dravite–arsenopyrite veinlets, contained within quartz–microcline–biotite–dravite–arsenopyrite–pyrrhotite auriferous replacement zones. Sulphide concentrations within the auriferous zones vary from 2% to 5%, with the main sulphides being arsenopyrite, pyrrhotite and pyrite.

Each mineralized zone was interpreted and modelled in 3D for resource estimation purposes. Five (5) zones were interpreted based on their relative position: zones 50, 60, 61, 70 and 80. Zone 50 is the westernmost zone.

Relationships between the nearby diorite and pegmatite intrusions and the gold mineralization event are still unknown.

Exploration targets have been identified around the Roberto deposit, including the hanging wall veins (HWV), the North Zone (NZ) and the 494 area. Further surface mapping and prospecting on the Éléonore Project is planned. Most of the claims have yet to be significantly explored.

In the opinion of the QPs, the knowledge of the deposit setting and lithologies, and of the mineralization style and its structural and alteration controls, is sufficient to support Mineral Resource and Mineral Reserve estimation. The mineralization style and setting of the Project deposit is sufficiently well understood to support Mineral Resource and Mineral Reserve estimation. Prospects and targets are at an earlier stage of exploration, and the lithologies, structural, and alteration controls on mineralization are currently insufficiently understood to support estimation of Mineral Resources.

1.6. Exploration

The work completed before Goldcorp acquired the Project in 2006 was performed by Virginia Gold Mines Inc. ("Virginia"). Exploration activities from 2001 to 2006 included prospecting, gridding, mapping, ground induced polarization (IP) and magnetic surveys, a Hummingbird electromagnetic (HEM) survey, grab and rock chip sampling, soil sampling, trench and channel sampling, and core drilling.

Goldcorp acquired Virginia in March 2006. Since that date, Goldcorp has performed additional geological mapping; core drilling; metallurgical testwork; mineral resource and mineral reserve estimates; baseline environmental, geotechnical and hydrological studies; and a pre-feasibility study on the Project.

In the opinion of the QPs, the exploration programs completed to date are appropriate to the style of the Roberto deposit and the prospects on the Project.

1.7. Drilling

Approximately 597,625 m have been drilled in 1,878 core holes on the Property since 2004. Of these, a total of 348 holes (104,532 m) were completed by Virginia and 1530 holes (493,093 m) by Goldcorp.

All boreholes were drilled from surface and underground on sections spaced approximately 25 m apart in most parts of the deposit. A borehole spacing of 25

m by 25 m was achieved over the bulk of the orebody to a depth of approximately 650 m below surface. Between 650 m and 850 m below surface, borehole spacing increases to roughly 50 m by 50 m. Below 850 m, down to approximately 1,200 m, a borehole spacing of 100 m by 100 m is usually implemented.

Standardized logging forms and geological legends were developed for the Project. Geotechnical logs were completed in sequence prior to geological logging. Geological logging used standard procedures and collected information on mineralization, lithological breaks, alteration boundaries, and major structures. All drill core is photographed.

Core recovery is acceptable for all drill programs, and averages 100% over the life of the Project.

Upon completion, drill hole collars were surveyed using a differential global positioning system (GPS) instrument by a registered surveyor. Underground drill holes are surveyed using a Leica TS15 robotized station.

Down-hole surveys were carried out by the drill contractor for dip and deviation using a FlexIt instrument. Although this instrument is subject to the effects of magnetism in surrounding rock types, the rocks hosting and underlying the Roberto deposit are only very weakly magnetic.

Drill data are typically verified prior to Mineral Resource and Mineral Reserve estimation by running a software program check.

Sample intervals were determined by the geological relationships observed in the core and vary between 0.3 m and 1.25 m. An attempt was made to terminate sample intervals at lithological and mineralization boundaries.

The collection of specific gravity data was initiated after the Project was acquired by Goldcorp. The data was collected by Goldcorp personnel. Core samples about 10 cm long were measured, weighed dry and then wet, and the specific gravity of the sample calculated. The specific gravity database contains 11,923 specific gravity results that were determined on core samples. A specific gravity of 2.77 was used for all veins.

The specific gravity database is currently sufficient to provide a reliable assessment of the variability of the specific gravity across the gold deposit and across the various rock types.

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected during the exploration

and infill drill programs completed by Virginia and Goldcorp are sufficient to support Mineral Resource and Mineral Reserve estimation.

1.8. Sample Analysis and Security

Exploration and infill core samples were analyzed by independent laboratories using industry-standard methods for gold analysis. A number of different laboratories have been used on the Project. Since January 2007, ALS Chemex in Val-d'Or in Quebec has been the primary laboratory, and holds ISO 17025 and 9001/2008 certifications.

Metallurgical testwork has been done at a number of laboratories, but was primarily performed by SGS.

Sample preparation for samples that support sometimes Mineral Resource and Mineral Reserve estimation has followed a similar procedure for all Virginia and Goldcorp drill programs. The preparation procedure is in line with industry-standard methods for a clastic sediment- hosted stockwork-disseminated gold deposit in an orogenic setting.

Samples were dried and crushed to better than 70 to 90% passing 2 mm. A split of the crushed material was then pulverized to 85% passing 75 µm. Gold assays were determined on a 50 g sample using fire analysis followed by an atomic absorption spectroscopy (AAS) finish. For assay results equal or above 3.0 g/t Au, samples were re-assayed with a gravimetric finish. ALS Chemex reports an upper limit of 10 g/t Au and a detection limit of 0.01 g/t for AAS analyses. No other elements are routinely assayed.

The collected sample data adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits.

Virginia and Goldcorp maintained a quality assurance and quality control (QA/QC) program for the Project. This comprised the submission of analytical standard reference materials (SRMs), duplicates and blanks. QA/QC submission rates meet industry-accepted standards of insertion rates. No material sample biases were identified by the QA/QC programs.
The results of the QA/QC programs did not indicate any problems with the analytical programs, therefore the drill core gold analyses are suitable for inclusion in Mineral Resource and Mineral Reserve estimation.

Sample security has relied upon the fact that the samples were always attended or locked in the logging facility. Chain-of-custody procedures consisted of filling out sample submittal forms that were sent to the laboratory with sample shipments to make certain that all samples were received by the laboratory. Current sample storage procedures and storage areas are consistent with

industry standards.

The QPs are of the opinion that the quality of the gold, copper, and silver analytical data are sufficiently reliable to support Mineral Resource and Mineral Reserve estimation and that sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards.

1.9. Data Verification

A number of data verification programs and audits have been performed over the Project's history by independent consultants in support of technical reports and by Goldcorp personnel in support of mining studies. Data verification checks were performed as follows:

- G.N. Lustig Consulting Ltd (2006): review of sampling and assay data on the Project; no material biases or errors noted.
- G.N. Lustig Consulting Ltd (2008): review of check assays performed by SGS Laboratories on 3,285 pulp samples originally assayed by ALS Chemex; the laboratories were considered to have satisfactory agreement.
- Smee Associates (2007): review of QA/QC program and sampling procedures; observations concluded that the sampling and quality control program were running smoothly and were compliant with mineral exploration best practices.
- Goldcorp (2006 to date): database validation checks; no material biases or errors noted.

The QPs consider that a reasonable level of verification has been completed, and that no material issues would have been left unidentified from the programs undertaken. The data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, and the quality of the analyses and the analytical database, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation.

1.10. Metallurgical Testwork

Metallurgical testwork has included chemical analyses, acid neutralization potential tests, semi-qualitative petrography, X-ray diffraction (XRD) study, comminution testwork (including standard Bond, crushing work index (CWi), abrasion index (Ai) and ball mill work index (BWi) tests), bench-scale flotation tests, Knelson/Laplante gravity-recoverable gold testwork, grade variability recovery testwork, establishment of a reagent suite, evaluation of intensive cyanide leach processing of flotation concentrates, cyanide leach tests on gravity tailings, cyanide detoxification testwork, settling tests, filtration tests and paste backfill tests.

Samples selected for testing were representative of the various types and styles of mineralization within the Roberto and Zone du Lac zones. Samples were selected from a range of depths within the deposit. Samples were taken to ensure that tests were performed on sufficient sample mass.

Crushing and grinding testwork was completed on three different batches by SGS Lakefield Research Limited (SGS). The samples came from the Roberto, Roberto East and Zone du Lac zones. The Bond ball mill work index values increase with depth from 16.7 kwh/t to 20.6 kwh/t, and the abrasion index falls between the 78th and 80th percentiles compared to the SGS database at 0.466. Éléonore ore is considered moderately hard and abrasive.

Éléonore ore contains iron sulphides such as arsenopyrite. Most of the rock will have a net acid generating potential and will also leach arsenic when in contact with neutral pH water. Under these conditions, the waste rock and the process tailings will have to be managed within an engineered containment system, as required by Directive 019 from the Quebec Environment & Sustainable Development Ministry.

Overall, circuit recovery will vary with the gold feed grade, with a weighted-average recovery of 93.5% projected over the LOM based on current economic assumptions.

1.11. Mineral Resource Estimate

The current Vulcan (version 8.2) block model produced for this Resource Estimate is a combination of two different block models. The lower portion of the deposit (the “Lower Mine”; below real elevation of -650 m) is exactly as it was presented in the previous Resource Estimate. The upper portion of the deposit (the “Upper Mine”; above real elevation of -650 m) was reinterpreted and a new interpolation was produced. A transition area of approximately 100 m between the Upper Mine and the Lower Mine was reinterpreted in order to link both interpretations together and avoid a fringe effect. The Lower Mine composites contained within this transition area were used for interpolating the Upper Mine model. The resulting is a single block model combining two generations of interpolations. In order to avoid confusion, the reader should understand that the Mineral Resource Estimate presented herein combines the results of both the Upper and Lower Mine resource estimate.

Mineral Resources are based on a total of 839 core drill holes and 143 surface channels, totalling 147,390 assay results, collected between September 2004 and September 5, 2013. The close-out date for the database supporting the estimation was September 5, 2013.

The mineralized zones were interpreted based on alteration, mineralization, structures and assay results. Major lithologies and alteration styles were also interpreted on section and plan views. The interpretations consist of four principal zones (5010, 5050, 6000 and 7000) and 24 secondary zones.

For the Lower Mine (as per the 2010 resource estimate), the interpretation of the geology and mineralized zones was first undertaken on a series of east–west cross-sections spaced 25 m apart, and then reconciled on level plans spaced 100 m apart. A minimum width of 1.0 m and cut-offs varying from 3 g/t to 5 g/t were used. The interpretation was then digitized and 3D solids were built. For the Upper Mine (new re-interpretation), the previous interpretation was updated based on new available information. The interpretation of the geology and mineralized zones was first undertaken on a series of level plans spaced 12.5 m apart and then reconciled on cross-section views spaced 25 m apart. A minimum width of 2.5 m and cut-offs varying from 1 g/t (LG) to 3 g/t (HG) were used. The dilution envelope was created roughly 25 m around the LG zones.

Drill hole samples were composited inside the mineralized solids into equal 2-metre down-hole length intervals. Capping was applied to anomalously high gold grades before compositing.

Unidirectional variography studies were completed using the 2-metre DDH composites of the capped gold assay data for the five (5) main mineralized zones. Ellipsoid radiuses were established using a combination of the ranges determined from the variography, the drill hole distribution, and the geological model comprehension. The investigations yielded the best-fit model along an orientation that roughly corresponds to the strike and dip of the ore shoots observed at Éléonore. The results of the linear variographic investigations for the DDH composites are consistent with the geological features of the deposit.

The variogram parameters obtained were used for the ordinary kriging estimation of all zones (except the mineralized envelope: “ME”), and the anisotropy was rotated to fit the orientation of the veins. The ME veins were estimated using an inverse distance cubed (ID3) interpolation.

Visual inspection confirmed that the block model honoured the drill hole data. A nearest-neighbour (NN) model using 5 m composites and an inverse distance squared (ID2) model were produced to check the global and local bias of the mineral resource model indicated that the global means of resource model matched well with the verification models, with differences within acceptable limits in most of the main HG veins.

For the Mineral Resource Estimate (exclusive of Mineral Reserves), all estimated blocks were classified into the Inferred category. The classification was primarily

based on drill hole density deemed insufficient for establishing either Indicated or Measured Mineral Resources.

Considering the geometry and shape of the orebody, the Éléonore deposit is amenable to underground mining using long-hole stoping methods. Based on the pre-feasibility work, an operating cost of approximately CA$111.51/t and a recovery of 93.5% were considered reasonable. Using a gold price of US$1,300/oz with a CAD/USD exchange rate of 1.05, the cut-off grade required to support reasonable prospects of economic extraction is approximately 3.0 g/t Au.

Mineral Resources take into account geologic, mining, processing and economic constraints, and have been confined within geological boundaries; they can therefore be classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. The QP for the Mineral Resource estimate is Christine Beausoleil, P.Geo, an employee of Goldcorp.

The Mineral Resources (exclusive of Mineral Reserves) for the Éléonore gold deposit were estimated at 13.25 Mt grading an average of 9.63 g/t gold in the Inferred Mineral Resource category, for 4.1 Moz of gold using a cut-off grade of 3.0 g/t Au. The effective date of this Mineral Resource Estimate is 31 December 2013. Table 1-1 summarizes the results.

Table 1-1: Mineral Resource Estimate (exclusive of Mineral Reserves) for the Éléonore Project, effective date of December 31, 2013 (C. Beausoleil, P.Geo.)

Classification Category	Tonnage (Mt)	Au (g/t)	Contained Gold (Moz)
Measured	-	-	-
Indicated	-	-	-
Measured & Indicated	-	-	-
Inferred	13.25	9.63	4.1

Summary Notes to Accompany Mineral Resource Table

1. Mineral Resources are reported exclusive of Mineral Reserves.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources were estimated using a price of gold of US$1,300/oz.
4. Mineral Resources are reported using a 3 g/t Au cut-off grade, which is based on assumptions of a US$1,300/oz gold price, long-hole stoping underground mining methods, a total mining cost of CA$111.51/t, and a life-of-mine metallurgical recovery of 93.5%.
5. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal contents.
6. Complete notes are available in Item 14 of the Report.

1.12. Mineral Reserve Estimate

Mineral Resources classified as either Indicated or Measured were considered during conversion to Mineral Reserves. The requirements for Mineral Resources to be converted to Mineral Reserves are:

- Only Measured and Indicated Mineral Resources can be included;
- Dilution is included in the Mineral Reserve estimate;
- Mineral Reserves are supported by an economic mine plan.

The QP for the Mineral Reserve Estimate is Denis Fleury, P.Eng., an employee of Goldcorp.

Mineral Reserves are reported at a gold price of US$1,300/oz Au and an assumed CAD/USD exchange rate of 1.05, and have an effective date of December 31, 2013.

Mineral Reserves are summarized in Table 1-2. All the Mineral Reserves are classified as Probable. Currently no Mineral Resources that fall within the area of the surface crown pillar (from surface to 55 m depth) are included in the Mineral Reserves. A complete study (hydrogeological and geotechnical) of the potential recovery of the surface pillar will be necessary to support the conversion of these Mineral Resources to Mineral Reserves.

Table 1-2: Mineral Reserve Statement for the Éléonore Project, effective date of December 31, 2013 (Denis Fleury, Eng.)

Level	Stope			Development			Total			
(m)	Tonnage Mt	Au M grams	Grade g/t	Tonnage Mt	Au M grams	Grade g/t	Tonnage Mt	Au M grams	Au Moz	Grade g/t
80-230	5.16	33.71	6.5	0.62	3.48	5.6	5.78	37.19	1.2	6.4
260-440	5.80	34.54	6.0	0.58	2.97	5.1	6.38	37.51	1.2	5.9
470-650	4.85	30.35	6.3	0.38	2.04	5.4	5.23	32.39	1.0	6.2
680-860	1.14	10.95	9.6	0.24	1.98	8.3	1.38	12.93	0.4	9.4
890-1040	0.43	4.30	10.0	0.07	0.60	8.6	0.50	4.90	0.2	9.8
1070-1220	0.03	0.23	7.5	0.01	0.06	6.5	0.04	0.29	0.0	7.3
TOTAL	**17.41**	**111.53**	**6.4**	**1.89**	**13.67**	**7.2**	**19.30**	**125.20**	**4.0**	**6.5**

Mineral Reserve table:

1. Mineral Reserves are estimated using a gold price of US$1,300/oz, and an economic function that includes variable operating costs and metallurgical recoveries. These assume processing costs of CAD$29.89/t, mining operating costs of CA$52.41/t, site services costs of CA$8.76/t, and general and administrative costs of CA$20.45/t, for a total life-of-mine estimated operating cost of CA$111.51/t (average over LOM).
2. Mineral Reserves are reported using a cut-off grade of 3.0 g/t Au.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal contents.
4. Tonnage and grade measurements are in metric units. Gold ounces are reported in troy ounces;
5. The entire Mineral Reserves (exclusive of Mineral Resources) is currently classified as Probable.
6. Reserves below 650mLv are based on January 2012 R&R Evaluation document.

1.13. Proposed Mine Plan

The proposed mine plan was developed by Goldcorp personnel.

Open stope mining (down-hole drilling) and longitudinal retreat with consolidated backfill (paste backfill mixed with crushed waste rock) is planned. A transverse open stope approach will be used where the mineralized lenses are wider than 7 m.

For mine scheduling purposes, the vertical extent of the orebody was subdivided into two parts: the upper part of the orebody located between 55 m and 650 m below surface and the lower part of the orebody located between 650 and 1,130 m below surface. Dividing the orebody into two parts will accelerate the production start-up. Initial production will be at the nominal rate of 3,500 t/d of ore, with two mining horizons on the 440 and 650 levels. Subsequently, the 230, 860 and 1,040 levels will be put into production. Studies to increase and sustain the production rate will be conducted as more drilling information becomes available. Based on the current Mineral Reserves, the planned operation has a 10-year mine life. The production will reach 7,000 tons/day for at least 4 years.

Intensive diamond drilling is ongoing to increase the reserves and extend the mine life to a minimum of 20 years.

One production shaft and one surface decline are progressing well while the exploration shaft (Gaumond shaft) has been completed to 715 m deep. The Gaumond shaft has a diameter of 7 m. It was used to develop the 650 level, to provide an exploration drilling platform for the deeper portion of the ore body, and to ensure an initial production rate of 3,500 t/day with the potential to reach 7,000 tpd. The surface ramp will accelerate development on the levels. All the material that will come from levels deeper than 650 will be brought to the loading station on the 650 level by trucks via the ramp until the production shaft is completed. The production shaft is currently in development. It will improve production efficiency in the Lower Mine (below the 860 level).

The ramp is currently used as the air exhaust and will continue to do so when completed. The main ventilation raise will be the Gaumond shaft. From the shaft, the air will be distributed into two internal ventilation raises, one located in the North Zone and one in the South Zone, each of which will bring fresh air to work places. Using a conservative approach, the fresh air requirement to accommodate the mobile equipment fleet was estimated at 283 m^3/s (600 kcfm) at the proposed production rate of 3,500 t/day. Currently, ventilation on demand is operational and this helps to reduce this preliminary estimation.

Stope widths will vary between 2.5 m and 25 m. The length and height of stopes will be 25 m to 50 m long and 30 m high respectively. Ground support will consist of various combinations of rebar bolts, friction bolts, screen and shotcrete depending on the rock quality and particular requirements of each heading.

Stopes will be backfilled with paste fill. On average, 2,500 t/day of paste backfill will be needed to meet the nominal production targets of 3,500 t/d from two mining horizons. The required strength of the backfill usually varies between 100 kPa and 525 kPa depending of the stope. Rock fill will be used in transversal secondary stopes.

One loading system is progressing well on the 690 level. Mine production will be hauled using 55 t trucks from the ore/waste pass to the loading station located on the 650 level. The rock will be directly loaded from the ore/waste passes into haulage trucks and then dumped onto a grizzly (fitted with 406 mm (16″) square openings) located over the storage bins. A rock-breaker will be located over the bins to break oversized rock. The ore bin and the waste bin will each have a capacity of 5,000 t. The bins will feed the rock unto a conveyor that will transport it to the measuring loading box. At that point, the rock will be automatically loaded into the skip. The skip capacity of the Gaumond shaft will be 10 t. When the production shaft will be completed a second loading station will be built on the 1440 level for the lower mine production.

Currently, no mining is planned above 55 m in order to mitigate risks associated with potential water inflow from the Opinaca Reservoir and to respect the preliminary recommendation for the dimensions of the surface crown pillar. Due to the presence of open subhorizontal decompression joints encountered mainly within the first 150 m below surface, and the proximity of the reservoir, the management of ground water infiltration is considered paramount for the successful project implementation. A 11,000 m^3/day (2,000 US gpm) dewatering system was selected to handle the expected peak water inflow into the mine. The system can be upgraded easily.

A fully-mechanized mining equipment fleet is planned. Equipment will include scoop trams, dump trucks, mine service and personnel vehicles, jumbo drills, bolting platforms, scissor lifts, land cruiser and forklifts.

The mine and fleet designs are appropriate for the Mineral Reserves defined and the selected throughput rate.

There is potential to extend the mine life and potentially sustain the 7,000 t/d throughput rate if some or all of the Inferred Mineral Resources identified within the LOM production plan can be upgraded to higher confidence Mineral Resource categories, and eventually to Mineral Reserves. Mineralization remains open at depth, with the deepest drill hole encountering mineralization at 1,400 m depth; the current mine plan extends to 1,130 m depth.

1.14. Future Development Strategy

The Gaumond exploration shaft has a nominal 3,500 t/d ore-hoisting capacity, and a maximum hoisting capacity of 7,000 t/d (20 hrs/day). Goldcorp is excavating a second shaft at Éléonore, which will have a nominal 8,500 t/d hoisting capacity (17 hrs/day). The objective is to transfer all the ore handling system to the production shaft during ore production from the lower mine.

The current plant is designed for a throughput of 3,500 t/d with a ramp-up period from 2015 to 2017, to reach 7,000 t/day in 2017, which is commensurate with the current Mineral Reserves. However, Goldcorp has designed the plant to be able to expand to 7,000 t/d earlier in the production mine life if Inferred Mineral Resources can be upgraded to Mineral Reserves.

1.15. Proposed Process Plant

The mill will be designed to operate at 7,000 t/d (2.55 Mt/year) for 365 days per year. The comminution circuit will consist of three stages of crushing followed by a single stage of ball mill grinding. The primary crusher (jaw crusher), the secondary crusher (standard cone crusher) and the tertiary crushers will be located at surface. Two short head cone crushers will be needed to handle a

7,000 t/d daily throughput. The fine-crushed ore will be ground using a single-stage ball mill connected in a closed circuit with cyclones. A portion of the cyclones underflow will be directed to a gravity concentration circuit consisting of a Knelson concentrator and an Acacia Reactor to recover liberated native gold.

The cyclones overflow (grinding circuit product) will be directed to the flotation cells to separate the sulphides into a low-mass sulphur concentrate. A thickener will control the density of the flotation tail slurry. The flotation tails will be leached with cyanide for 36 hours while going through five leach tanks. The flotation concentrate will be thickened and reground so that 80% (P80) is smaller than 10 µm using a fine grinding mill; then it will be pre-aerated with oxygen for 18 hours prior to be leached with cyanide for 48 hours in five additional leach tanks. The gold in solution will be recovered in carousel carbon-in-pulp (CIP) circuits (one for each leach circuit).

The carbon from each CIP circuit will be stripped as required in Zadra Process, and the gold recovered from that final stage of the mineral processing circuit will be poured into gold bars at regular intervals. The carbon will be regenerated and returned to the CIP circuits for reuse.

The tails from each leaching circuit will be detoxified in a conventional cyanide destruction circuit (SO_2/Air), and then filtered. Finally, tailings will either be added to the paste backfill, but only the non-sulphides tailings will be stored in a covered shed before being transported by hauling truck to the tailings management facility.

The tailings facilities will be completely lined, and all water touching the tailings will be collected and treated. The exposed surface of the tailings will be kept to a minimum, made possible by the choice of filtered tailings that allows for progressive reclamation. The tailings design envisages a storage capacity of 26 Mt. This is sufficient for the current LOM.

In the opinion of the QPs, the process design is based on a conventional gold plant flowsheet. Reagents, power and consumable requirements have been appropriately estimated and are included in the project operating costs.

1.16. Infrastructure Considerations

Mine construction activities are underway and engineering and procurement activities are mainly completed. A global cumulative construction progress of 70% has been reached at the end of the 2013. Main activities are to complete mill mechanical, electrical and instrumentation of the main ore process facilities in order to start preoperational verifications and commissioning as planned in the third quarter of 2014. A preoperational verification activity has been initiated at the industrial water treatment plant that will be fully operational by the end of June

2014. Construction activities for the administration, garage and warehouse buildings are going in progress and will be completed by June 2014 and phase 2 of the permanent camp that includes recreational, gymnasium and kitchen areas will be completed by the end of 2014. Phase 1 of the permanent camp was completed during the fourth quarter of 2013 and an additional 658 man camp is used to support on-going construction activities. Engineering, procurement and construction management has reached a cumulative progress of approximately 44%.

1.16.1. Access Road

A permanent road with two permanent bridges has been completed, extending from the Sarcelle hydroelectric facility to the Éléonore Project. The Sarcelle station can be reached via a 40 kilometre gravel road, starting at the 396 kilometre marker along the James Bay Highway. All of the material, supplies, and food for the construction and operational phases will be transported along this access route.

1.16.2. Airstrip

The landing airstrip is located approximately 2.5 kilometres north of the camp site and is accessible via an access road between the campsite and parking area. It is proposed that a building be constructed to accommodate travellers, airstrip operations personnel and navigational control equipment at the airstrip. Petroleum installations will also be installed in the parking area.

Workers are brought on site via a permanent year -round air strip located approximately 1.5 kilometres north of the camp.

1.16.3. Power Supply

The Éléonore mining operations and processing plant is fed through a 120 kV overhead electrical power line supplied and installed by Hydro-Québec from the existing distribution point at the Eastmain power generation substation. A 120/25 kV substation on site distributes the power required for the mining infrastructure.

1.16.4. Water

The drinking water supply for the site is drawn from three wells. The collective capacity of the wells is on the order of 300 m^3/day, which is adequate to supply a crew of 1,300 workers given a unit consumption of 220 litres/person/day.

The domestic waste water (sewage) is treated in a specific treatment area. Two ponds are dedicated to biologically treat the water.

The industrial wastewater treatment plant will include the water treatment system, pumps, and all the collection basins required to store and manage the industrial

site water and the mine water. There are four types of effluents that will require treatment:

- Mine water: water from the underground mine.
- Process water: water purges from the mineral processing plant.
- Tailings area run-off water: water from the tailings management facility.
- Industrial zone run-off: run-off water from the industrial area of the site.

1.16.5. Fuel

For fuel storage and services, three different installation points have been selected. It is important to note that there will be daily fuel tanks at each campsite building with an emergency generator. The installation points are:

- The airstrip;
- The depot (mine site);
- The fuel station.

1.16.6. Infrastructure

Underground infrastructure will comprise: the Gaumond shaft (715 m), a circular shaft 1,500 m deep; a 4.5 km-long access ramp; a shaft loading station; ore and waste passes; ore and waste storage bins; a rock breaker/grizzly arrangement; a transfer drift; an exhaust raise; and a mine dewatering system.

The main process infrastructure will consist of: a jaw crusher; a secondary cone crusher, tertiary crushers; a single-stage ball mill; cyclones; a Knelson concentrator; an Acacia reactor; flotation and leach tanks; carousel CIP circuits; a Zadra stripping circuit; and a refinery. Completing the process infrastructure are the following: an Actiflo installation (water treatment plant); a water pumping station; tailings filter presses; and a paste backfill system.

Surface infrastructure will include: an airport terminal and gatehouse; a concrete plant; the main camp; an administration building; a service garage for surface and mine; warehouse facilities; an assay laboratory; process and water treatment plants; an oxygen plant; fuel and propane storage areas; the Gaumond shaft; a tailings storage facility; a waste/ore rock storage area; and the surface ramp.

Workforce requirements are estimated at 600 workers to support the estimated nominal throughput of 7,000 t/d.

In the opinion of the QPs, the existing and planned infrastructure, the workforce considerations and operating hours, the existing power, water and communications facilities, the methods whereby goods are transported to the mine, and any planned modifications or supporting studies are well-established or the requirements to establish such are well understood by Goldcorp and can support the disclosure of Mineral Resources and Mineral Reserves.

1.17. Markets and Contracts

Goldcorp's bullion is sold on the spot market by Goldcorp's in-house marketing experts. It is expected that the same process will be used for gold produced at Éléonore.

The terms contained within the existing sales contracts are typical and consistent with standard industry practices, and are similar to contracts for the supply of doré elsewhere in the world. There is an expectation that the same style of contracts will apply to gold produced from Éléonore.

1.18. Capital and Operating Cost Estimates

Capital and operating cost estimates were prepared by Goldcorp staff. Capital costs are based on the latest mine construction data and budgetary numbers/quotes provided by suppliers. The 2011-2013 capital costs are based on the actual costs and the 2014 capital costs are based on the budget.

Capital cost estimates were based on a combination of quotes, vendor pricing, and experience with similar-sized operations.

Operating costs were based on estimates from the first principals for major items, and included allowances or estimates for minor costs.

The capital cost estimates include direct and indirect costs. The total capital cost estimate is US$1.85 B. Sustaining capital is estimated at US$310 M. The combined total capital and sustaining capital cost estimate is US$2.16 B.
The estimated average annual operating cost is US$106.20/t milled.

1.19. Financial Analysis

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented in this report.

Forward-looking statements in this section include, but are not limited to, statements with respect to the future price of gold, the estimation of Mineral Reserves and Mineral Resources, the realization of Mineral Reserve estimates, the timing and amount of estimated future production, costs of production, capital expenditures, costs and timing of the development of new deposits, success of exploration activities, permitting timelines, currency exchange rate fluctuations, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

Additional risk can come from actual results of current exploration activities; actual results of current reclamation activities; conclusions of economic evaluations; changes in the project parameters as plans continue to be refined; possible variations in Mineral Reserves, grade or recovery rates; failure of plant, equipment or processes to operate as anticipated; accidents, labour disputes and other risks of the mining industry; and delays in obtaining governmental approvals.

The financial analysis of the project was carried out using a discounted cash flow model prepared in Microsoft Excel. The model was constructed using annual cash flows in 2013 constant dollar terms. No provisions were made for the effects of inflation or de-escalation for the value of tax losses carried forward.

Considerations used in the estimate include:

- Probable Mineral Reserve totalling 19.3 Mt grading 6.49 g/t Au.
- Inferred Mineral Resources were treated as “waste” in the financial evaluation.
- This mineralization represents upside potential for the Éléonore Project, if some or all of the Inferred Mineral Resources identified within the LOM open pit production plan can be upgraded to higher confidence Mineral Resource categories, and eventually to Mineral Reserves.
- Gold price of US$1,300/oz.
- Exchange rate of 1.05 (CAD/USD).
- Tax rate of 40%.
- A royalty payable on production from Éléonore is set at 2.20% on the first three (3) million ounces of gold, and increases by 0.25% per million ounces thereafter, up to a maximum of 3.50%.
- An annual payment related to the Cree Collaboration Agreement.
- The total capital cost of the Éléonore project was estimated at US$2.16 B ($1.85 B in direct capital costs and an additional US$310 M of sustaining capital).
- Operating costs were estimated at an overall unit cost of CA$111.51/t was estimated, which comprises CA$29.89/t for processing costs, including backfill and tailings treatment and transportation; CA$52.41/t for mining costs; CA$8.76/t for infrastructure; and CA$20.45/t for G&A costs.
- The gold refining charge was estimated at US$1.75/oz Au.
- Bullion delivery charges were estimated at CA$1.67/oz Au.

The Éléonore Project is expected to yield an after-tax internal rate of return (IRR) of 3.15%. The life of mine (LOM) based on the current Mineral Reserves is 10

years and the payback period is 8 years.

Goldcorp have performed a conceptual review of the likely impact of production rate increases on the mine plan prior to commencement of major plant and infrastructure construction activities. In this concept study, mining activities were assumed to occur on four mining fronts which would result in a doubling of the production rate from the current 3,500 t/d from two fronts to 7,000 t/d. Goldcorp have stated that the company's objective will now be to focus on drilling activities from underground, when the shaft reaches the appropriate depth, to delineate additional mineralization that can be used to achieve the increased production rate. The company has determined that construction activities will assume that the 7,000 t/d rate can be achieved, and that the plant and infrastructure will be sized and built accordingly.

Cash flow fluctuations during the LOM primarily result from fluctuations in the sustaining capital and mill head grade. Negative cash flows are projected at the end of the mine life and correspond to expected reclamation costs.

1.20. Sensitivity Analysis

A sensitivity analysis was carried out for the base case scenario described above to test the sensitivity of the Éléonore Project's after tax IRR to a 15% and 30% change in gold price, average grade, operating costs and initial capital costs.

The Project is most sensitive to the gold price. It is less sensitive to changes in capital costs and operating costs.

1.21. Conclusions

There is sufficient support from the updated results for Goldcorp to move to a production phase starting in 2014

1.22. Preliminary Development Schedule

The Gaumond shaft was excavated to a depth of 715 m and the change-over to production shaft was completed in Q1 of 2013. The excavation of the surface ramp commenced in February 2011 and was completed up to level 650 in January 2014. Some of the permanent infrastructure, such as ventilation raises, ore/waste passes, ore and waste bins to support the production start-up, should be completed in Q2 and early Q3 2014.

1.23. Recommendations

In order to continue with project development and prepare for the production phase, intensive exploration and delineation drilling must be carried out over the next 3 years.The first objective of the drilling will be to convert resources to

reserves. Drilling at depth and laterally is also required to add new resources.
The suggested program involves a rate of approximately 25,000 m per year of exploration drilling to test the deposit's extensions and add new resources, another 50,000 m per year of infill drilling (at 25 m spacing) to convert Indicated Resources to Reserves, and 40,000 m per year of final delineation or definition drilling (production drilling) for stope delineation. This amounts to a total of 115,000 metres of drilling per year, representing an estimated annual budget of CA$17.25 M.

In parallel, exploration drilling in three areas of interest is recommended, as described below.

In the HWV area, 8,000 m of surface drilling is proposed at a total cost of CA$1.4 M, to support a potential Mineral Resource estimation of the mineralization in these shear and alteration zones found near the shaft and exploration ramp; the work is necessary because these zones may become accessible during mine development activities.

In the NZ area, mineralization crops out at surface and may represent a potential open pit target. Additional drilling is warranted, totalling CA$1.5 M for 10,000 m.

In the 494 area, 8,000 m of underground drilling is proposed at a total cost of CA$1.2 M to drill-test a potential mineralized corridor located between the 494 area and a surface showing (Trench #10) with similar mineralized features.

2. INTRODUCTION

The Éléonore project is located in the northeast corner of the Opinaca Reservoir in the James Bay region of Quebec, Canada (Figure 2-1). The Éléonore Project hosts the Roberto gold deposit comprising the Roberto, East Roberto, and Zone du Lac lenses.

The operating entity for the Project is a wholly-owned subsidiary of Goldcorp Inc.: Les Mines Opinaca Ltée. For the purposes of this Report, “Goldcorp” is used to refer interchangeably to the parent and subsidiary companies.

Goldcorp Inc. is a publicly owned company based in Vancouver, British Columbia, Canada. Goldcorp Inc. is a leading gold producer engaged in the operation, exploration, development and acquisition of precious metal properties in Canada, the United States, Mexico and Central and South America. Goldcorp is listed on the New York Stock Exchange (NYSE: GG) and the Toronto Stock Exchange (TSX: G).

2.1. Terms of Reference

This Report supports the disclosure of Mineral Resources and Reserves for the Project, based on the latest geological interpretation used to generate the block model of October 2013, and also supports the disclosure of the Éléonore Project 2014 Budget. Goldcorp will be using the Report in support of the 2013 Annual Information Form filing.

All measurement units used in this Report are metric, and currency is expressed in American (US$) or Canadian dollars (CA$).

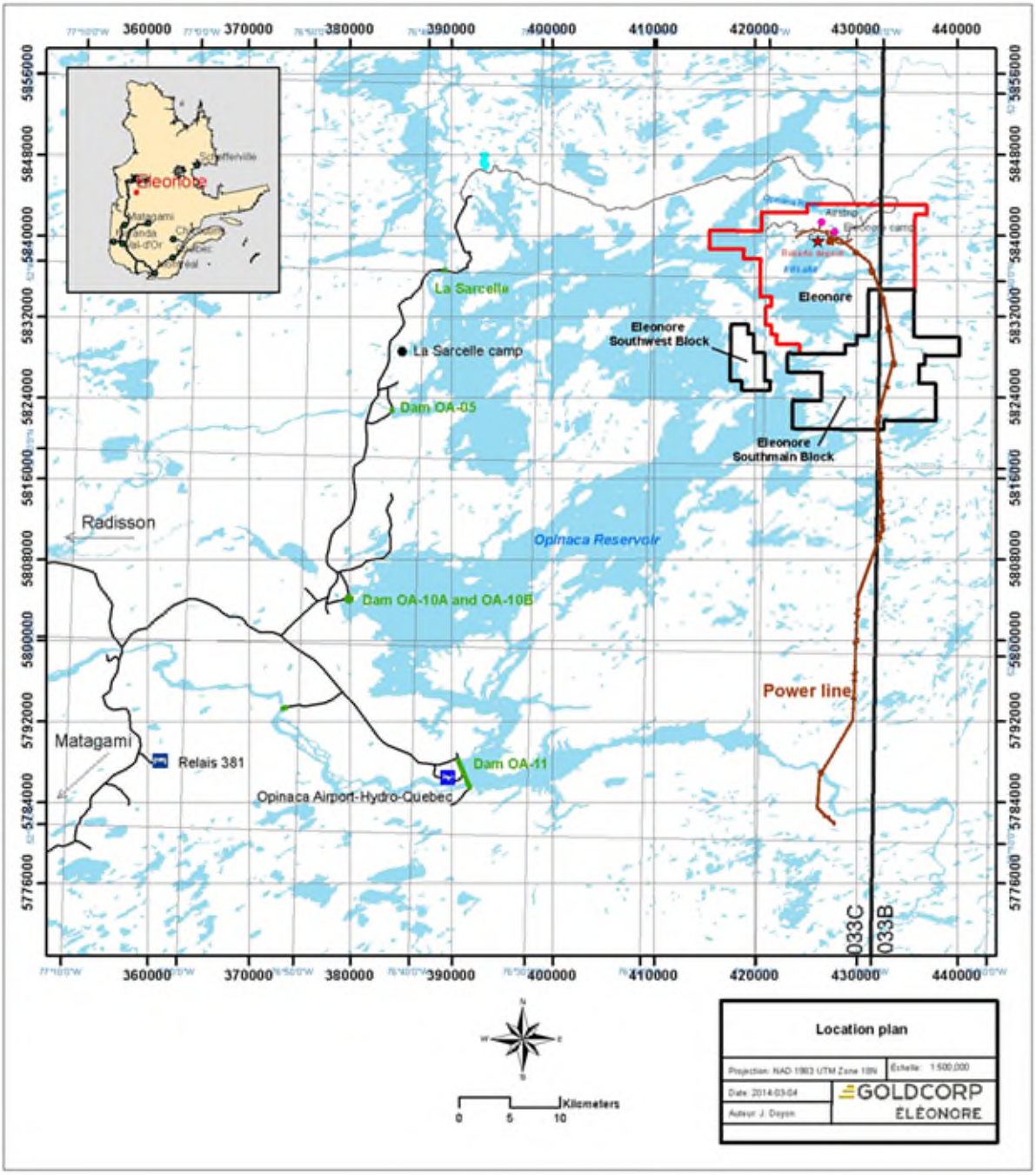


Figure 2-1: Project location map (2013)

2.2. Qualified Persons

This technical report has been prepared by Goldcorp staff. The following persons are the Qualified Persons ("QPs") for this Technical Report as defined by the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), in conformity with generally accepted guidelines of the CIM for "Exploration Best Practices" and "Estimation of Mineral Resources and Mineral Reserves Best Practices". The QPs are Goldcorp employees working on the Éléonore Project and at the head office in Vancouver, British Columbia.

- Denis Fleury, Eng., Chief Engineer, Éléonore Project
- Christine Beausoleil, P.Geo., Chief Geologist, Éléonore Project
- Tony Brisson, P.Geo., Exploration Manager, Éléonore Project
- Andy Fortin, Eng., Process and Site Services Manager, Éléonore Project
- Luc joncas, P.Eng., Mining Manager, Éleonore Project

2.3. Effective Dates

- The effective date for the Mineral Resource Estimate is December 31, 2013.
- The effective date of the Mineral Reserve Estimate is December 31, 2013.
- The effective date for the financial analysis supporting the Mineral Reserve disclosure is January 26, 2014.

The overall effective date of this Report is therefore taken to be the date of the financial analysis supporting the Mineral Reserves and is January 26, 2014.

2.4. Information Sources and References

This Report is based in part on internal company reports, maps, published government reports, and public information, as listed in Section 27 "References" of this Report. Specialist input from other disciplines, including legal, process, geology, geotechnical, hydrological and financial, was sought to support the preparation of the Report.

2.5. Technical Reports

Goldcorp has previously filed the following technical reports for the Project:

Michaud, C., Chen, E., Simoneau, J., Fortin, A., and Belanger, M., 2012: Éléonore Project, Quebec, Canada, NI 43-101 Technical Report. Technical Report prepared by Goldcorp Inc., effective date January 26, 2012.

Simoneau, J., Prud'homme, N., Bourassa, Y., and Couture, J-F., 2007: Mineral

Resource Estimation, Éléonore Gold Project, Quebec. Technical Report prepared by SRK Consulting Inc. for Goldcorp Inc., effective date 9 August 2007.

Prior to Goldcorp's interest in the Project, Virginia Gold Mines Inc. had filed the following reports on the Project:

Cayer, A., Savard, M., Tremblay, M., Ouellet, J. F. and Archer, P., 2006: Technical Report and Recommendations, Summer and Fall 2005, Exploration Program, Éléonore Property, Québec. Virginia Gold Mines Inc. GM 62341

Savard, M., and Ouellette, J.-F., 2005: Technical Report and Recommendations May 2005 Drilling Program, Éléonore Property, Québec. Virginia Gold Mines Inc.. GM 62117.

Cayer, A., and Ouellette, J.-F., 2005: Technical Report and Recommendations, June 2004 – February 2005 Exploration Program, Éléonore Property, Québec. Virginia Gold Mines Inc. GM 61851.

Cayer, A., and Ouellette, J.-F., 2004: Technical Report and Recommendations, June 2003 – May 2004 Exploration Program, Éléonore Property, Québec. Virginia Gold Mines Inc. (June 2004).

3. RELIANCE ON OTHER EXPERTS

This report was compiled through the efforts of Goldcorp staff under the supervision of Qualified Persons, as described in Section 2 (Introduction).

The environmental studies, permitting, and social or community impact (Item 20) were prepared by Martin Duclos, Sustainable Development and Environment Manager of Éléonore Goldcorp project, and this section was supervised by Luc Joncas, Mining Manager, Éléonore Project, Goldcorp Inc.

The Market Studies and Contracts (Item 19), the Capital and Operating Costs (Item 21), and the Economic Analysis (item 22), were prepared by Claude Laflamme, Finance Manager of Éléonore Goldcorp project, and these sections were supervised by Luc Joncas, Mining Manager, Éléonore Project, Goldcorp Inc.

4. PROPERTY DESCRIPTION AND LOCATION

The Éléonore Project is located in the Lake Ell area, in the northeastern part of the Opinaca Reservoir of the James Bay region, in the Province of Quebec, Canada. The Project is located approximately 350 km north of the town of Matagami and 825 km north of Montreal. The Project contains the Roberto gold deposit.

The mine surface and underground infrastructure are currently under construction. The Roberto deposit is situated at approximately UTM Zone 18N (NAD83) 5839829 N and 427470 E.

4.1. Project History

Goldcorp acquired the Éléonore Project from Virginia Gold Mines Inc. ("Virginia") in 2005, under a plan of arrangement. Under the agreement, shareholders of Virginia received 0.4 of a Goldcorp common share and 0.5 of a share in a new public exploration company, Virginia Mines Inc., for each issued and outstanding Virginia share.Virginia Gold Mines Inc. was therefore 100% acquired by Goldcorp and retained the Éléonore Project.

Virginia Mines Inc. retained all other assets of Virginia Gold Mines Inc., including net working capital, cash to be received prior to closing from the exercise of Virginia Gold Mines Inc. options and warrants, its non-Éléonore exploration assets, and a sliding scale 2% net smelter return royalty on the Éléonore property. The royalty is applicable to the entire Eleonore property.

4.2. Mineral Tenure

The Éléonore Project comprises 369 contiguous claims (Figure 4-1) totalling 19,274 ha. The claims are 100% owned by Les Mines Opinaca Ltée, an indirectly wholly-owned Goldcorp subsidiary. Claim details are included as Appendix A. All claims are in good standing.

A request for a 284-hectare Mining Lease covering the Roberto deposit was sent to Quebec's Ministry of Natural Resources in 2011 and is currently under final revision. In 2013, following the normal procedure, eleven (11) claims affected by the future Mining Lease were suspended. A portion of these claims remaining beyond the Mining Lease boundary will be re-activated once the Mining Lease will be delivered.

A block of four claims totalling 208 ha located in the central area of the property was purchased in 2011 by Goldcorp through an agreement with Wemindji Exploration. The four claims are now 100% owned by Les Mines Opinaca Ltée.

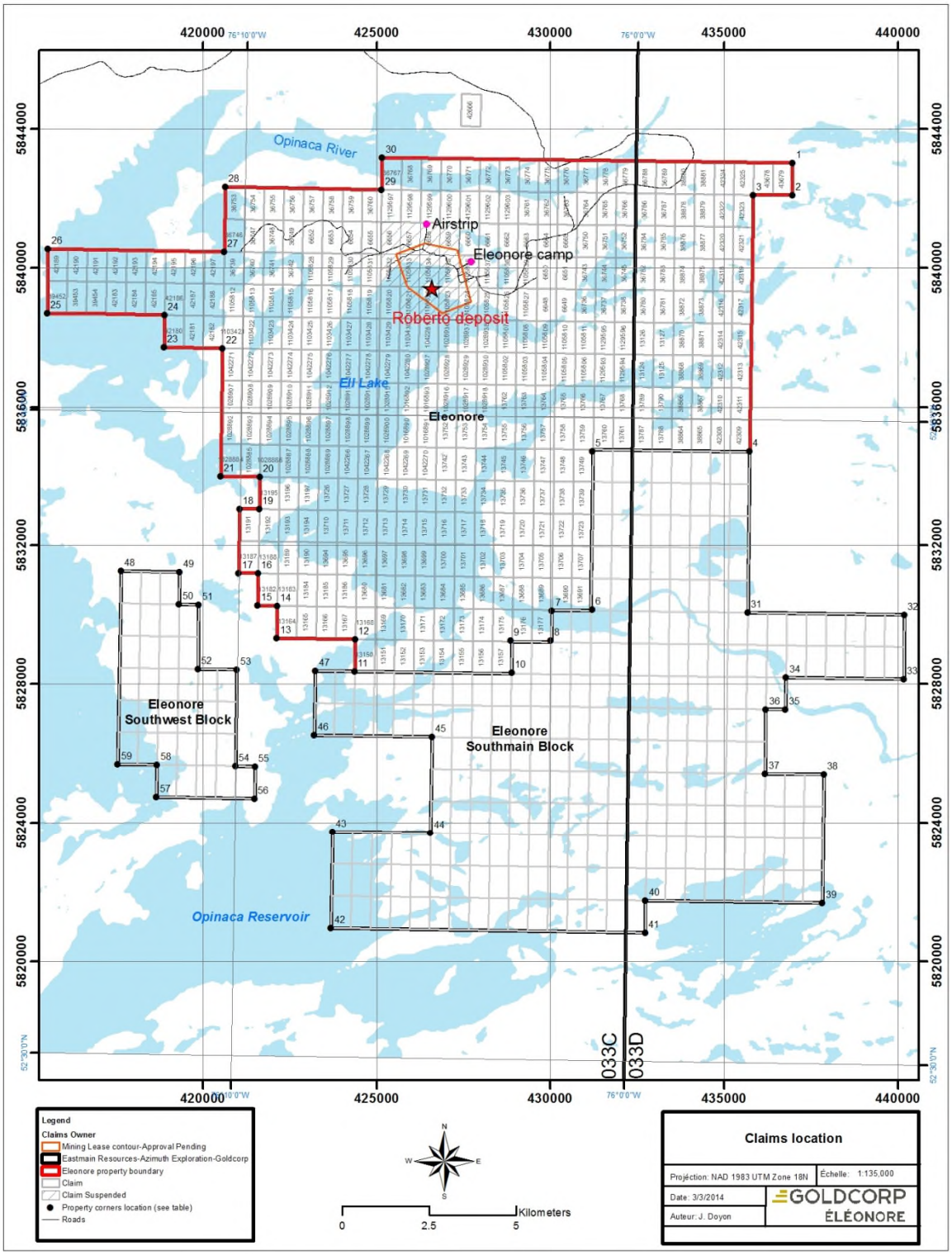


Figure 4-1: Mineral Tenure Plan

Table 4-1: Description of the Claim Blocks

Claim Block	Number of Claims	Northing*	Easting*	Area (ha)
Éléonore Main	369	5,835,701	427,943	19,188
Éléoore South**	248	5,826,281	431,701	12,896
Éléonore Southwest**	34	5,827,724	419,201	1,768
Total	651			33,852

Contiguous with the Project are claims that are part of a tripartite joint venture between Eastmain Resources, Azimut Exploration and Goldcorp, and which form the Éléonore South and Éléonore Southwest properties (refer to Figure 4-1). These 282 claims cover a total area of 14,760 ha. The properties are not considered to be part of the Éléonore Project as the properties are independent projects that are being managed by Eastmain Resources.

The perimeter of the property has not been legally surveyed. Under Quebec law, claims in the James Bay area are map-staked. The map-designated cell coordinates constitute the legal limits of the claims. Physical claim limits can be established by surveying and positioning the map designated coordinates of the claims in the field.

Claims are valid for a two-year period and can be renewed every two (2) years. Renewal fees are $127 per claim if renewed 60 days before the expiry date and $254 if renewed within the 60 days renewal window. In order to maintain tenure, exploration work equivalent ranging from $135 to $2500 per claims are required depending on the number of terms of renewal a claim has undergone. Table 4-2 details the work requirements per renewal period north of the 52 parallel. Work expenditures that are in excess of the amount required for the term can be transferred to other contiguous claims that are within 4.5 km from the center of the claim or can be credited toward future renewals.

Table 4-2: Work Requirement per Renewal Period

Term	Surface area of claim		
	Less than 25 ha	25 to 45 ha	More than 45 ha
1	$48	120$	$135
2	$160	$400	$450
3	$320	$800	$900
4	$480	$1,200	$1,350
5	$640	$1,600	$1,800
6	$750	$1,800	$1,800
7 or more	$1,000	$2,500	$2,500

4.3. Surface Rights

The Éléonore Project is located entirely in Cree territory, or Eeyou Istchee, on Category III lands belonging to the Quebec government and subject to the James Bay and Northern Quebec Agreement (JBNQA). Infrastructure for the mining project required the acquisition of surface leases issued by the Ministry of Natural Resources (MNR).

The leases were obtained for all infrastructure planned in the Project area (Table 4-3). They are automatically renewed by the MNR every year.

For road infrastructure construction, an approval must be issued by the MNR. Necessary consents have been obtained for the entire planned road network including the permanent road (# 001 160 10 000).

Quarries and sand pits needed for construction purposes require, as appropriate, an authorization without a lease (ASB), a non-exclusive lease (BNE) or an exclusive lease (BEX). As appropriate, ABS, BNE or BEX were obtained for all pits and quarries currently in operation. A number of requests for new quarries and sand pits are being reviewed or are under application (Table 4-4).

The Project is also in the territory of the Municipality of James Bay (MBJ). The following uses are permitted:

- Industry (I),
- Leisure and recreation (L),
- Resources (R)
- Conservation (C).

A building permit must be issued by the MBJ before the start of construction. This permit was obtained in 2011 and is required to be updated annually. The update is due during the first quarter of 2014, and at the Report filing date, was under evaluation.

Table 4-3: Surface Lease Summary Table

Site	Object	Lease number
Communication tower	Communication tower	822018 00 000
Airport	Surface modification for the weather station	0831 10 000
Airport	Modification for the apron extension	000020 10 000
Global project	Grouping of leases surrounding Éléonore under a global lease	000 880 10 000
Airport	Surface modification for the weather station (Revocation of the previous one)	000020 10 000
Tailings management facilities	Tailings management facilities	001171 10 000
Tailings management facilities	Clay stock pile	001173 10 000
Tailings management facilities	Peat moss stock pile	001174 10 000
Global project	Modification of the global lease to incorporated the Water treatment plant.	000880 10 000
Permanent road	Revocation of the lease for the construction of a gate at the entrance of the road	001198 10 000
Airport	Modification for the apron extension	000020 10 000
Waste rock pad	Modification for the extension of the pad.	000527 10 000
Global project	Modification of the global lease for the extension of the waste rock pad	000880 10 000
Domestic waste	Construction of a landfill.	001396 10 000

Table 4-4: Quarry and Sand Pits

Quarry and Sand Pit Names	Type of Lease	Lease Number	Renewal Date
Carrière C-05	MRN BEX	BEX 1097	2017-01-09
Carrière C-07	MRN BEX	BEX 1125	2014-03-20
Carrière C-04	MRN BEX	BEX 1153	2017-05-14
Carrière C-11	MRN BEX	BEX 1126	2017-05-14
Sablière A-01	MRN ASB	ASB 5432	2014-03-20
Sablière DG-R25	MRN ASB	ASB 5433	2014-03-31
Sablière DG-R25	MRN ASB	ASB 5434	2014-03-31
Sablière R-30-B	MRN ASB	ASB 5439	2014-04-30
Sablière R-36-B	MRN ASB	ASB 5440	2014-04-30
Sablière R-44	MRN ASB	ASB 5441	2014-04-30
Carrière C-02	MRN BEX	BEX 1129	2017-05-07
Sablière R-34-A	MRN ASB	ASB 5557	2014-12-17
Sablière R-36-C	MRN ASB	ASB 5558	2014-12-17
Sablière R-40	MRN ASB	ASB 5559	2015-01-19
Sablière A-01	MRN ASB	ASB 5560	2015-03-20
Carrière C-07	MRN BEX	Under study	Under study
Sablière R-30-A	MRN BNE	Under study	Under study
Sablière R-38-B	MRN BNE	Under study	Under study

4.4. Royalties

A royalty is payable to Virginia Mines Inc. The amount is calculated as a percentage based on mine production and gold price (Table 4-5). The royalty is applicable to the entire Eleonore property.

An example of the royalty calculation is (see Table 4-5):

- Mine production of 5,250,000 gold ounces would require payment of a 2.75% royalty based on production, being 2% on the first 3 Moz of gold produced, then successive 0.25% interest payments on each next 1 Moz of gold produced, i.e. 2%+ 0.25% + 0.25% + 0.25% = 2.75%
- And assuming the ounces were sold at a gold price of US$670/oz, then the final total royalty would be 3.025% (2.75% + (2.75 x 0.1)).
  The maximum royalty percentage is fixed at 3.5%. Advance payment of royalties to Virginia Mines Inc. of $US100,000 per month commenced on April 1, 2009.

4.5. Agreements

Goldcorp have entered into a confidential agreement on February 21, 2011 with the Grand Council of the Crees (Eeyou Istchee), the Cree Regional Authority and the Cree Nation of Wemindji, termed the Opinagow Collaboration Agreement. Under this agreement, an annual payment is made. The QPs have reviewed the agreement and there are no terms in the agreement that would have a negative impact on the Project. The annual payment is incorporated into the Project financial model for the Project.

4.6. Permits

The current permitting status of the Project and the kinds of permits likely to be required to support project development are discussed in Section 20.

4.7. Environment

The current status of the environmental project, community consultation studies, current and proposed environmental studies and environmental permitting are discussed in Section 20.

4.8. Social Licence

The social and environmental licence for the Project is discussed in Section 20.

Table 4-5: Royalties Payable, Based on Production and Gold Price

Percentage Payable, Based on Production and Gold Price	
Percentage	Number of Gold Ounces Produced
+2.00%	On first 3 Moz of gold
+0.25%	On ounces produced between 3 Moz and 4 Moz of gold
+0.25%	On each additional 1 Moz of gold
Percentage Adjustment Based on Market Gold Price	
Percentage	Number of Gold Ounces Produced
-10%	If ≤$US350/oz Au
-5%	If >$US350/oz Au but ≤$US400/oz Au
0%	If >$US400/oz Au but ≤$US450/oz Au
+5%	If >$US450/oz Au but ≤$US450/oz Au
+10%	If >$US500/oz Au

4.9. Comment on Section 4

In the opinion of the QPs, the following interpretations and conclusions are appropriate:

- Goldcorp holds 100% of the Éléonore Project. The Project comprises 369 contiguous claims totalling 19,188 ha;
- Additional ground in the Éléonore South and Éléonore Southwest properties is held under joint venture, and because the joint venture is managed by a third -party, the properties are not considered to be part of the Éléonore Project:
- Surface rights are held by Les Mines Opinaca;
- A sliding-scale royalty is payable to Virginia Mines Inc., and is capped at 3. 5%. Advance royalty payments commenced in April 2009;
- An annual payment is required to the Cree Nation under the Opinagow Collaboration Agreement; this payment is included in the Project financial model;
- Permits obtained by the company to explore and undertake project development are sufficient to ensure that activities are conducted within the regulatory framework required by the local, provincial, and federal governments.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1. Accessibility

The closest towns to the Éléonore Project are Matagami and Chibougamau, which are both located approximately 350 km to the south.

A permanent road with two permanent bridges has been completed, extending from the Sarcelle hydroelectric facility to the Eleonore site. The Sarcelle station can be reached via a 40-km-long gravel road starting at the 396 km marker along the James Bay Highway ("Route de la Baie-James"). All the material, supplies and food for the construction and operational phases will be transported along this access route (Fig. 5.1).

Workers are brought on site via a permanent year-round air strip located approximately 1.5 km north of the exploration camp. The airstrip can accommodate planes capable of transporting up to 30 people, such as a Dash 8-100.

5.2. Climate

The climate is typical of Northern Canada and is a temperate to sub-arctic climate. Average summer temperatures between June and September vary between 10°C and 25°C during the day, and 5°C and 15°C at night. Winters can be cold, with temperatures ranging from -60°C and -10°C. Precipitation varies throughout the year, reaching an average of 2 metres annually. Exploration activities are currently conducted year-round, but can be temporarily halted during spring thaw and fall freeze-up. Mining activities are expected to be conducted year-round.

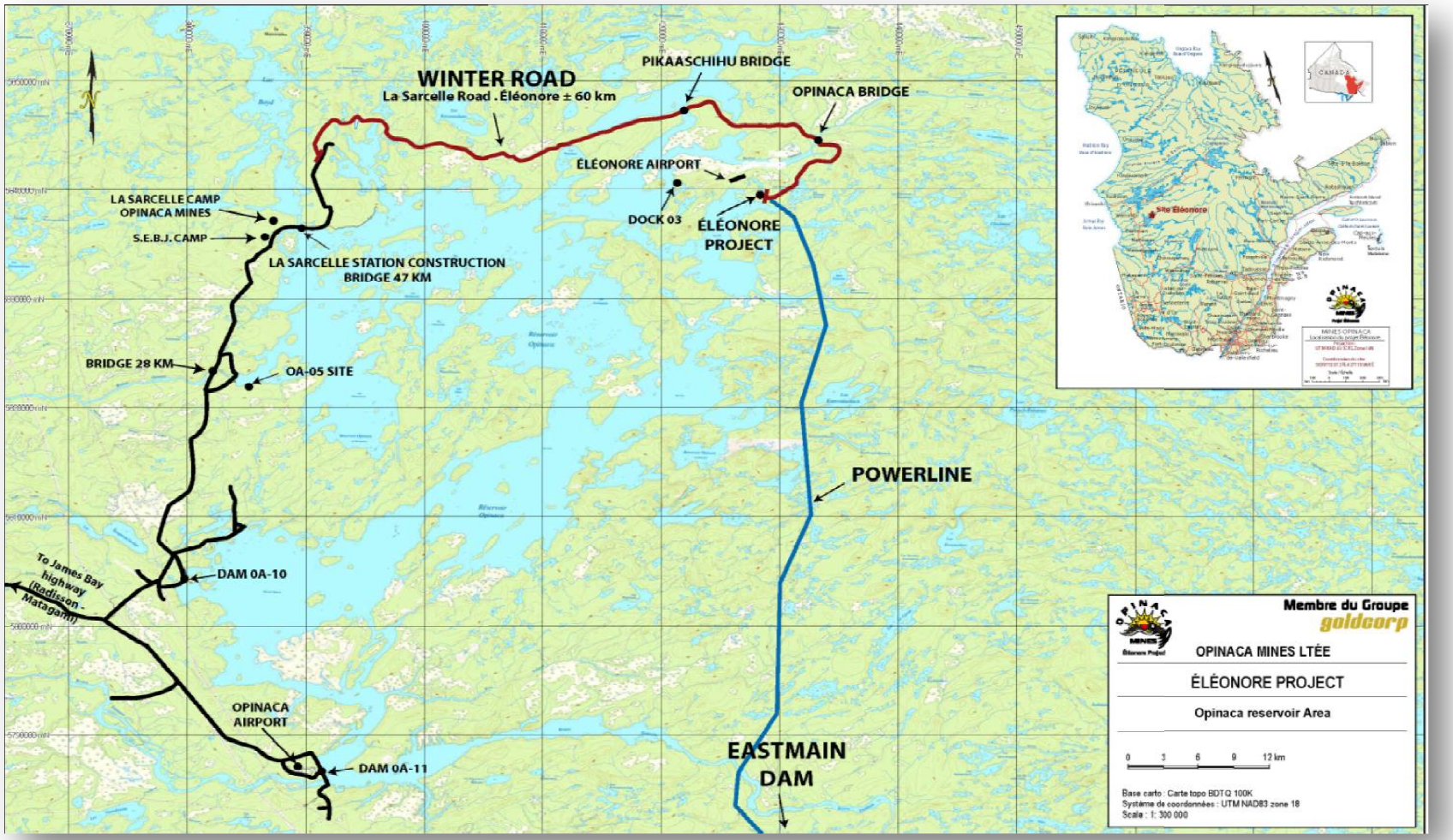


Figure 5.1: Accessibility of the Éléonore Project

5.3. Local Resources and Infrastructure

Matagami and Chibougamau offer extensive community, health, and transportation services. Matagami is located on the James Bay Highway.

The James Bay area is surrounded by extensive hydroelectric facilities and associated infrastructure, the closest of which are the Sarcelle hydroelectric facility, located 40 km due west of the Project site on the Opinaca Reservoir, and the Eastmain Dam, located 70 km to the south.

The site has the following infrastructure at the time of writing this report:

- Airport with illuminated runway
- Gaumond shaft (sinking in progress)
- A surface ramp (under excavation)
- Production shaft (sinking in progress)
- Process facilities (currently under construction)
- Water Treatment Plant (currently under construction)
- Tailings management facility
- Administration, warehouse and garage buildings (currently under construction)
- Accommodation at the permanent camp (403 rooms)
- Accommodation camps for the construction component (777 rooms)
- Cafeteria

Figure 5.2 is an aerial view of the current campsite and mine development area. The planned infrastructure to support mining operations is discussed in Section 18.

5.4. Physiography

The physiography of the region is typical of the Canadian Shield and includes many lakes, swamps and rivers. Outcrop is limited due to the presence of swamps and glacial deposits.

The area is characterized by a gently undulating peneplain relief. The elevation of the few hills of this rolling landscape ranges from 215 m to 300 m above sea level. The area is drained by Lake Ell, which is itself part of the Opinaca Reservoir.

Vegetation is typical of taiga and includes sparse spruce forests separated by large swampy areas devoid of trees. The entire area was burned by forest fires a few years ago.

Figure 5.2: Aerial photograph of the Éléonore camp and mine development area

Note: Photograph dated March 2014

6. HISTORY

The first recorded exploration in the Éléonore area was by Noranda, in 1964. Noranda identified a copper showing located within the Ell Lake diorite intrusion; this showing is located approximately 6 km southwest of the Roberto deposit.

In 2001, Virginia Gold Mines Inc. ("Virginia") completed regional reconnaissance grab and channel sampling around Noranda's Ell Lake copper showing; this work identified a number of new showings. A series of mineralized corridors consisting of stockworked gold and chalcopyrite-bearing quartz veinlets were outlined within dioritic to tonalitic intrusions. In addition, a number of mineralized and partially-rounded erratic blocks, located about 300 m from the mineralized corridors, returned significantly elevated copper, gold, and silver values.

Ground magnetic and inverse polarization (IP)/resistivity geophysical surveys were completed in 2002 and resulted in the discovery of several new Au-Cu-Ag occurrences in the diorite intrusion. During the program, an intensely-altered boulder of quartzo-feldspathic metasediments with disseminated arsenopyrite and pyrrhotite was identified. Subsequent investigation of this boulder and the glacial train in the area indicated that the source area was located a few kilometres to the northeast along the contact zone between the diorite intrusion and a cluster of quartzo-feldspathic sediments lying directly north of the intrusion.

Additional ground geophysical surveys, including IP/resistivity (94 line-km), magnetics (81 line-km) and Hummingbird electromagnetic (HEM; 26 line-km) were performed on 200-m spaced lines during 2003–2004. A soil geochemical Modified Mercalli Intensities (MMI) survey (949 samples) was also conducted.

In late 2003, prospecting and reconnaissance mapping were completed ahead of mechanical trenching. The trenches were excavated to test geological, geophysical, and geochemical targets. Grab and channel sampling of the trenches indicated anomalous gold values of over 1 g/t. The first two samples of the Roberto Zone (#809786 and #809785) were collected at the very end of this exploration program and yielded values of 10.25 and 2.26 g/t Au. The program also identified additional gold occurrences in the diorite intrusion in the southwest portion of the grid.

A helicopter-borne, detailed magnetic survey (45 line-km on two 50-m line-spaced grids) was carried out in early February 2004 over the northern half of the Ell Lake intrusion and the cluster of metasediments lying to the north.
During June–August 2004, additional trenching was performed on the Roberto Zone. Virginia commenced core drilling in September 2004, and by November 2005, a total of 247 core holes (85,960 m) had been drilled on the Éléonore property. Drilling completed by Virginia successfully extended the mineralization

found at surface to a depth of 800 m below surface. It also extended the mineralization beyond the Roberto Peninsula into the Bay Area and on the north shore of Ell Lake as well as to the south.

In addition, during 2004–2005, a total of 125.5 km of grid lines were cut on the land while a further 127 km of grid was added to the Ell Lake grid system. This work was followed by an IP survey that covered a total of 226.3 line-km.

In 2005, a large B-horizon survey encompassing a good portion of the property was completed. A total of 1,244 samples were taken at a spacing of 50 m along lines spaced between 500 m and 1,000 m.

An in-principle agreement to acquire the Project was reached between Goldcorp and Virginia Gold Mines Inc. in November 2005 followed by a buyout offer shortly thereafter. Goldcorp took control of the Éléonore property on March 31, 2006.

Since acquisition, Goldcorp has performed till sampling, lake-bottom sediment sampling, surface mapping and trenching, additional core drilling, Mineral Resource estimation and geological modelling. Mine construction activities are underway.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1. Regional Geology

The Roberto deposit is located in Archean rocks of the Superior Province of Canada in the transition zone between the Opinaca Subprovince and the La Grande Subprovince (Figure 7-1). The contact between the two subprovinces is not well known and generally corresponds to regional-scale deformation zones and a sharp change in the metamorphic gradient. In some areas, the contact is masked by late intrusions of one or the other subprovince (Bandyayera and al, 2010).

The Opinaca Subprovince basin is a sedimentary basin dominated by migmatized paragneisses and diatexites from the Laguiche Complex and intruded by syn to post- tectonic tonalite, granodiorite, granite and pegmatite intrusions from the Janin and Boyd intrusive suites. The metamorphic grade increases from amphibolites facies near the margins to granulite facies toward the center of the basin (Moukhsil et al., 2003). The paragneisses are strongly metamorphosed and folded rocks that retained few of their original structures (Bandyayera and al, 2010). Unit ages vary between 2,844 Ma, similar to the La Grande basement rock ages, and 2,672 Ma and 2,647 Ma corresponding to paragneiss fusion episodes (Goutier et al., 2002; David et al., in preparation).

The “S-shaped” La Grande Subprovince surrounds the Opinaca Subprovince on its west and north sides, spanning a distance of 450 km in the east-west direction and of 250 km in the north-south direction. The La Grande Subprovince is an assemblage of volcano-plutonic rocks composed of 85% intrusive rocks and 15% volcano- sedimentary units, the latest forming the volcano-sedimentary units of the La Grande River and Eastmain River green belts (Gouthier et al., 2001; Hocq, 1994). These assemblages overlay an older tonalitic basement (2.79 to 3.39 Ga). Metamorphic grade increases from the greenschist facies to the amphibolites facies toward the contact with the Opinaca Subprovince (Gauthier and Larocque, 1998; Moukhsil, 2000).

The Project area is overlain by rocks of the Eastmain Group of the La Grande Subprovince (Figure 7-2). At its base, the Eastmain Group consists of the Bernou Formation (2,722 Ma) and the Kasak Formation (2,704 Ma) Formations, which are composed of basalts and intermediate to felsic tuff (Moukhsil et al., 2003; Bandyayera et al., 2010).

Discordantly overlying these two formations are the Pilipas and Low formations, consisting of conglomerate, greywacke and wacke. This volcano-metasedimentary sequence is intruded by synvolcanic and syn- to late-tectonic tonalite, granodiorite and diorite intrusions.

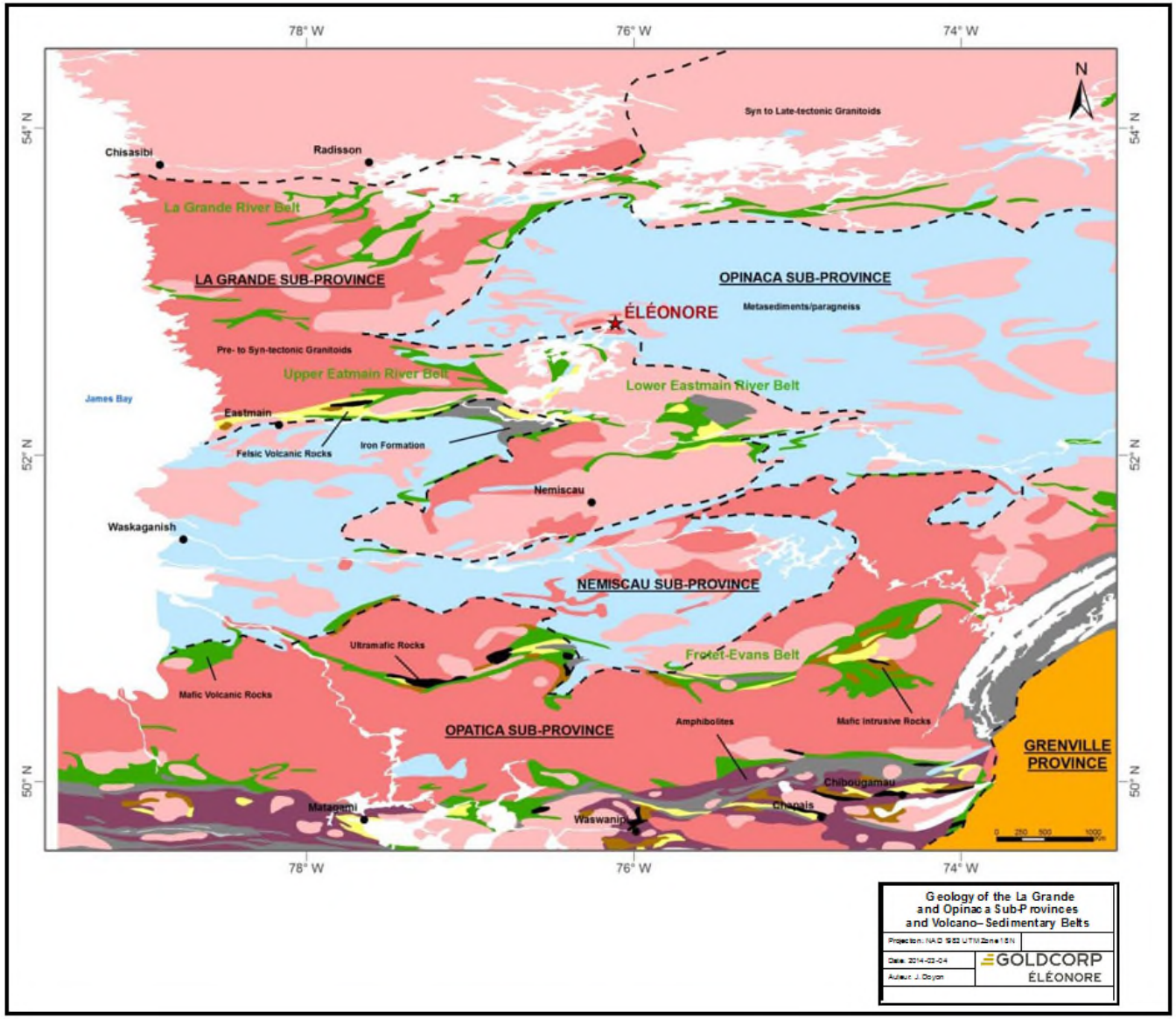


Figure 7-1: Geology of the La Grande and Opinaca Subprovinces and volcano-sedimentary belts (2013)

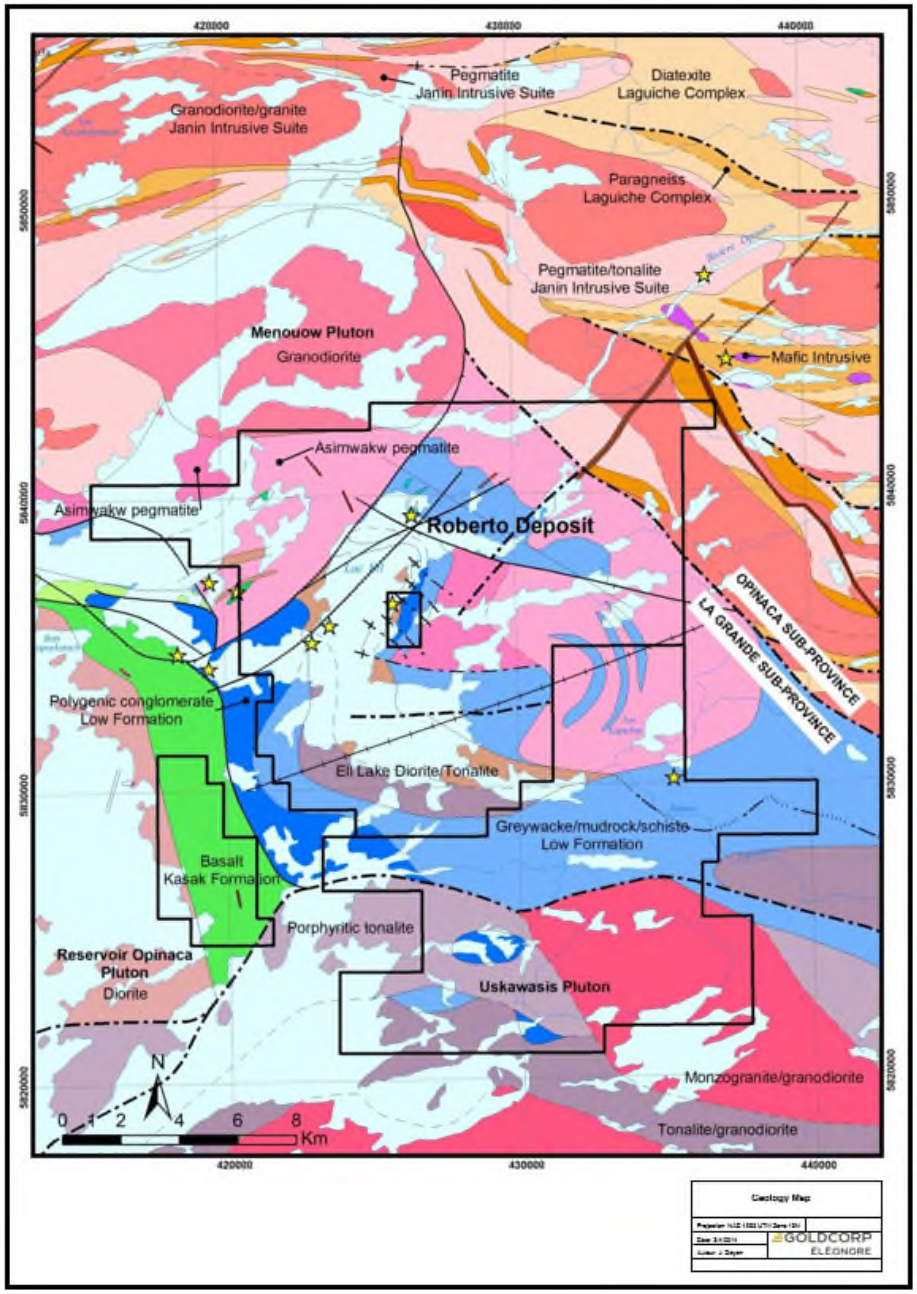


Figure 7-2: Project geology map (2011)

Note: The Vortex showing indicated on the plan is on ground that is not held by Goldcorp. Thick black solid lines indicate Goldcorp tenure boundaries. The small tenure square in the centre of the figure was acquired by Goldcorp during 2011 and is now part of the Goldcorp Éléonore project

Some intrusions dated between 2,709-2,704 Ma could be synvolcanic with the Kasak Formation while others in the 2,710-2,769 Ma range are considered syn- tectonic intrusions (Moukhsil et al., 2003; Bandyayera et al., 2010).

Regional faults are mainly present in the La Grande Subprovince and are oriented north–south, east–west, northwest–southeast, and northwest–southeast. In outcrop, the faults can be recognized by either a strong tectonic banding or by the presence of intense shear zones with mylonitization. In the Opinaca Subprovince, faults and shear zones are mainly located along fold limbs. Outcrop within the two subprovince areas has been extensively eroded by repeated glaciations.

7.2. Local Geology

The Éléonore property straddles the contact between the Opinaca and La Grande subprovinces (refer to Figure 7-2). The contact is located in the northeast corner of the property along a northwesterly trend that is defined by a strong shear zone, a change in the magnetic grain, and an increase in the metamorphic gradient (Bandyayera et al, 2010).

7.2.1. Metasedimentary Units

Rock units from the Opinaca Subprovince occur in the northeastern corner of the Project. Lithologies are dominated by granite, granodiorites and heterogeneous assemblages of pegmatites, tonalites and granites from the Janin Intrusive Suite intermixed with partially migmatized paragneiss from the Laguiche Complex. The structural grain is oriented in a northwesterly direction evolving to an east west grain toward the east part of the Project area.

The La Grande Subprovince rock units make up most of the Project area west of the contact between the subprovinces and host the Roberto deposit. Lithologies are dominated by metasedimentary units of the Low Formation. Various types of conglomerates, either clast-supported or matrix-supported with monomictic to polymictic clast composition, make up the base of the Low Formation and can reach significant thicknesses. The units form large outcrops along the west side of the property border.

The upper part of the Low Formation is made up of massive, finely- to thickly-bedded greywacke; greywacke that contains aluminosilicate porphyroblasts; conglomeratic greywacke; conglomerates; local arenites; mudstone; and cherty units. Immediately outside the western Project limits, rock units consist of basalts and intercalated intermediate to felsic tuff units attributed to the Kasak Formation. The Low Formation is interpreted to be in discordant contact with the Kasak Formation.

7.2.2. Intrusive Units

The crescent-shaped Ell Lake diorite intrusion is 10 km long and occupies the centre of the Project. Few observations of the contact between the Ell Lake diorite intrusion and the Low Formation have been recorded.

Asimwakw pegmatites, as named by Bandyayera in 2010, intrude the metasedimentary units and dominate topographic highs. The pegmatites generally have a northeasterly trend that does not appear to be solely the results of preferential erosion during periods of glaciation.

Large granodiorite, granite and pegmatite intrusive units are located on the southeastern side of the property. The Uskawasis pluton, consisting of monzonite, granite and granodiorite is located just south of the property boundary.

Proterozoic diabase dykes that have a northeasterly orientation transect the property.

7.2.3. Structure

Metasedimentary units appear to form an open fold with a northeasterly-trending axis. A large mylonitic high-strain zone that also trends to the northeast roughly follows this fold axis. A large northwest-trending, subvertical regional fault, observed both in drill core and in outcrop, is located just east of the Roberto deposit. A 1 km dextral displacement is interpreted for this fault from magnetic data. The magnetics also indicate the presence of a number of additional faults within the Project boundaries.

Rock units in the Project area have been deformed by three deformation phases of which D2 was the principal phase and formed a penetrative regional foliation. This deformation is expressed differently in the two subprovinces, being regular with an east-west trend of domes and basins in the Opinaca Subprovince (Remick, 1977) and with volcano-sedimentary units deformed around the resistant intrusions in the La Grande Subprovince. A strong east–west to northwest–southeast schistosity and a mineral foliation in intrusive rocks is visible in the La Grande units.

D1 Deformation is more visible in the volcano-sedimentary units of the La Grande Subprovince and expressed by P1 folds with a north northeast–south–southwest axis. The folds are locally refolded by D2 folds with northwest–southeast axes. In the Opinaca Subprovince units, this deformation is faint and expressed by mineral lineations.D3 deformation is discrete in both subprovinces and appears as folding of S2 and as crenulation cleavages.

7.3. Roberto Deposit

The Roberto deposit has historically been divided into the Roberto, Roberto East, Zone du Lac, North and Hangingwall Zoneszones. This nomenclature is based on their geographical location and the main alteration types observed. All these zones are made up of many individual mineralized lenses.

The host rock of the mineralized zones is typically a thinly-bedded greywacke (approximately approx. 10 cm beds) greywacke near the contact with a massive greywacke unit, and locally, with a thin conglomerate unit (Figure 7-3). A section through the deposit showing the geology and mineralized zones (displayed in red) is presented in Figure 7-4. The steeply east-dipping Roberto East fault, marked by a thin black tourmaline marker band, forms the eastern limit of the mineralized vein cluster.

The structural hanging wall of the mineralized zones is characterized by a greywacke containing centimetre-scale aluminosilicate porphyroblasts overlain by a thin conglomerate unit. The aluminosilicate-bearing greywacke and the conglomerate appear tightly folded with axis generally oriented in the east– west direction and refolded by the F2 event. This folding is in sharp contrast with the generally north– south-trending bedding in the mineralized zones. The structural footwall of the mineralized zones is characterized by greywacke, locally exhibiting a higher metamorphic grade, which contains a higher amount of pegmatite dykes and quartz veins.

The mineralized zones consist of a stockwork of gold-bearing quartz–dravite–arsenopyrite veinlets, contained within quartz–microcline–biotite–dravite–arsenopyrite– pyrrhotite auriferous replacement zones (Ravenelle et al, 2010). Mineralization shows variable proportions of disseminated arsenopyrite and pyrrhotite. Traces of pyrite, sphalerite, bornite, and chalcopyrite are also present locally.

The mineralized zones are visually recognizable. They are light to dark brown in colour due to microcline and tourmaline alterations, with generally abundant sulphide concentrations. These darker colours differentiate the mineralized zones because the wall rocks are usually light to medium-grey colour and have a low sulphide content.

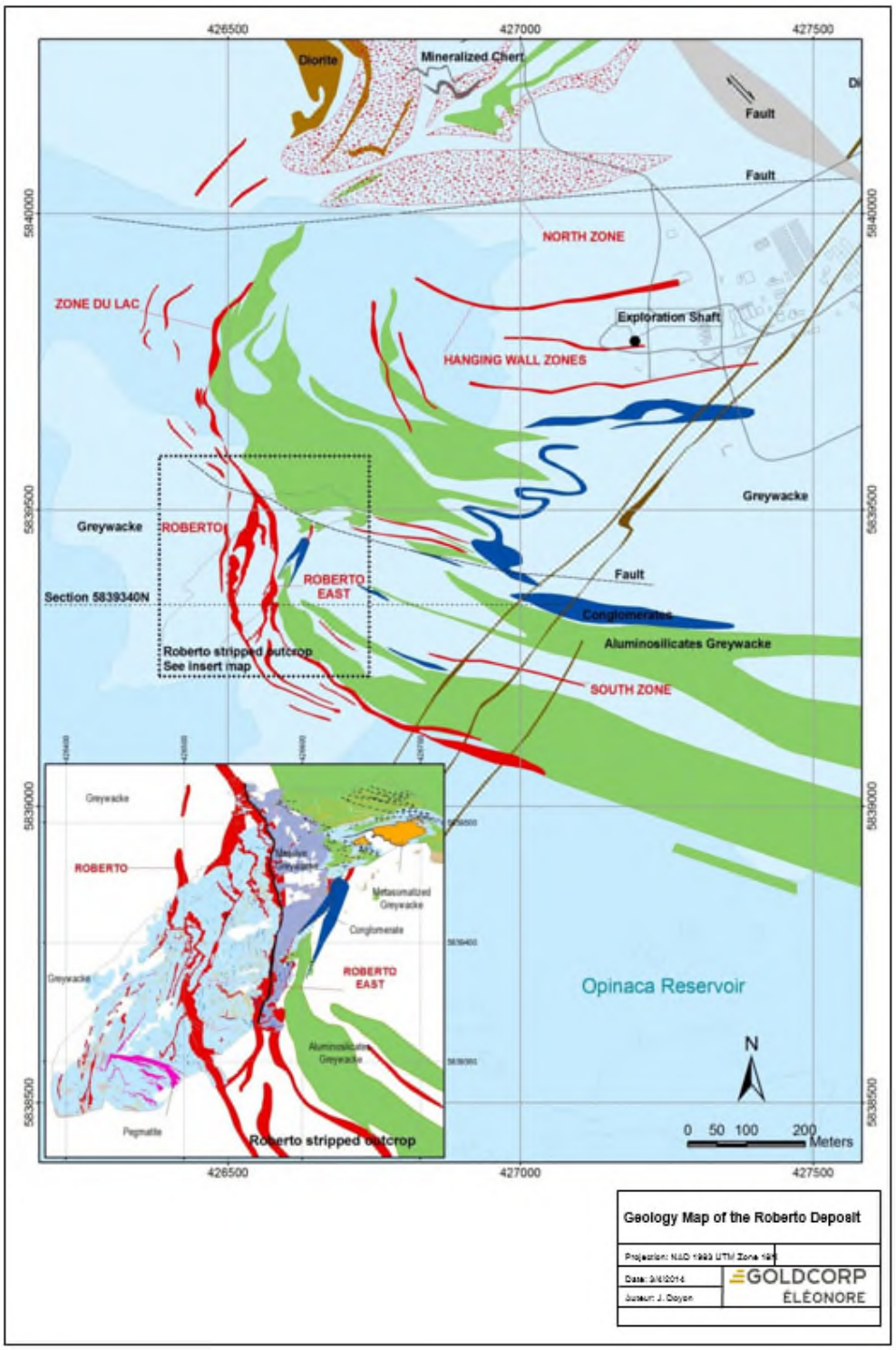


Figure 7-3: Geology map of the Roberto deposit (2013).

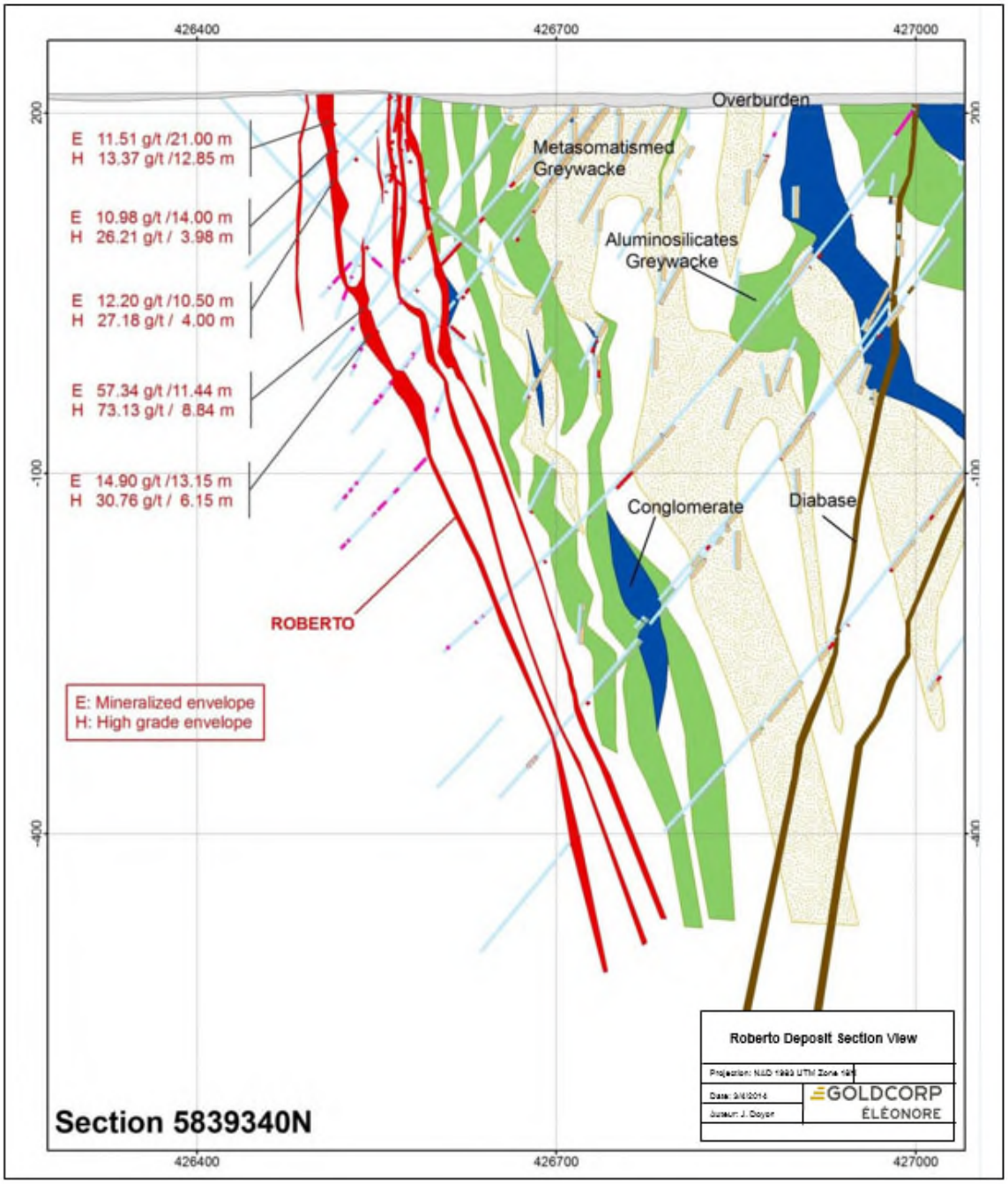


Figure 7-4: Roberto deposit section view, looking north (2013). Vein envelopes are based on drilled thicknesses.

The sulphide concentration within the mineralized zones varies between 1% and 5%, and primarily consists of arsenopyrite, pyrrhotite and pyrite. The “waste” rock may contain sulphides, usually pyrrhotite, but this is in lesser amounts, from trace to 2%, and occurs mostly in the structural hanging wall.

The mineralized zones are generally 5 m to 6 m in true thickness, varying between 2 m and more than 20 m locally. Mineralization is considered to pre-date the final deformation phase (Ravenelle, 2010).

The mineralized zones are folded with increased thicknesses in the hinge of the folds while limbs are fairly straight and continuous. Transposition of the sedimentary beds post-mineralization may also explain some of the thickening of the mineralized zones.

The Roberto gold zones dip steeply to the east and rake (plunge) steeply to the northeast. All zones remain open at depth and along strike.

7.4. Exploration Targets

Outside the Roberto deposit area, exploration is being undertaken on the Old Camp showing located in the Project area, 5 km west of the Roberto deposit (refer to Figure 7-3). In this location, shear zones that carry pyrite, arsenopyrite and chalcopyrite and are associated with quartz–tourmaline veins are located in the Ell lake diorite near the contact with the metasedimentary units. Grab samples have returned anomalous high-grade gold values. Similar gold values and settings are observed on the east side of the Ell Lake intrusion.

7.5. Comments on Section 7

In the opinion of the QPs, the knowledge of the deposit setting, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation. The mineralization style and setting of the Project’s deposits is sufficiently well understood to support Mineral Resource and Mineral Reserve estimation.

8. DEPOSIT TYPES

The Roberto deposit is considered to have many aspects in common with classic greenstone-hosted quartz–carbonate vein deposits, but represents a clastic sediment- hosted stockwork-disseminated end member. Canadian end-member examples of greenstone and clastic-hosted quartz–carbonate vein deposits include Pamour and Timmins.

The following description is based on Dubé and Gosselin (2007).

Greenstone-hosted quartz–carbonate vein deposits are a subtype of lode gold deposits, and are defined as structurally controlled, complex epigenetic deposits that are hosted in deformed and metamorphosed terranes. They are distributed along major compressional to trans-tensional crustal-scale fault zones in deformed greenstone terranes of all ages, but are more abundant and significant, in terms of total gold content, in Archaean terranes.

Although dominantly hosted by mafic metamorphic rocks of greenschist to locally lower amphibolite facies, deposits can be hosted in metamorphosed sediments.

Greenstone-hosted quartz-carbonate vein deposits are typically associated with iron- carbonate alteration. The relative timing of mineralization is syn- to late-deformation and typically post-peak greenschist-facies or syn-peak amphibolite-facies metamorphism.

Deposits consist of simple to complex networks and arrays of gold-bearing, laminated quartz–carbonate fault-fill veins in moderately to steeply dipping, compressional brittle-ductile shear zones and faults, with locally associated extensional veins and hydrothermal breccias. Individual vein thickness varies from a few centimetres up to 5 m, and their length varies from 10 up to 1000 m. The vertical extent of the known deposits is commonly greater than 1 km and may reach as much as 2.5 km.

Gold is mainly confined to the quartz–carbonate vein networks but may also be present in significant amounts within iron-rich sulphidized wall rock. The sulphide minerals typically constitute less than 5 to 10% of the volume of the mineralized zones. The main minerals are native gold with, in decreasing amounts, pyrite, pyrrhotite, and chalcopyrite and occur without any significant vertical mineral zoning. Arsenopyrite commonly represents the main sulphide mineral in clastic-hosted deposits. The Au/Ag ratio typically varies from 5 to 10.

8.1. Comment on Section 8

The Roberto deposit is the first example of a greenstone- and clastic-hosted quartz– carbonate vein deposit identified in the La Grande Subprovince.

Features of the deposit that indicate it is part of the greenstone-hosted quartz–carbonate vein deposit continuum include:

- Spatially associated with major crustal features developed in deformed greenstone rocks; mineralization is considered to pre-date the final deformation phase;
- Associated with large, district-scale carbonate alteration;
- Gold mineralization hosted in laminated fault-fill quartz–carbonate veins of varying thicknesses;
- Association of gold with arsenopyrite, pyrrhotite and pyrite.

Features of the Roberto deposit that are atypical of the general greenstone-hosted quartz–carbonate vein deposit continuum include the fact that it is characterized by stockwork and replacement-style mineralization hosted in amphibolite facies turbiditic metagreywacke and paragneiss.

9. EXPLORATION

The main focus of the exploration activities on the project have been to advance the Roberto deposit to a development decision, and therefore the greater Éléonore Project area has not been subject to significant exploration work in the last six years. However, high-quality exploration targets exist, both near the Roberto deposit and on other parts of the concession, and these warrant further investigation. Table 9-1 summarizes the exploration activities on the Éléonore Project other than drilling.

9.1. Grids and Surveys

The coordinate system used for all of the exploration, drilling and support of Mineral Resources and Mineral Reserves is the Universal Transverse Mercator (UTM) coordinate system using the North American datum of 1983 (NAD 83). The UTM Zone is Zone 18 North. Data acquired prior to Goldcorp's acquisition of the project was also in UTM coordinates, however the datum was NAD 27 and in order to be converted the following translation had to be calculated:

- Conversion to NAD 83: North= +228.407m, East= +22.643m and Elevation= -4.423m.
- The GPS survey data were directly downloaded into the Project acQuire database.
- In 2006, an air photo/LiDAR survey was completed over the property by XEOS of Quebec. The survey covered two areas:
- Sector A covered the entire Éléonore claim group at 60 cm resolution, with 4 m topographic contours;
- Sector B covered the Roberto area at 25 cm resolution, with 1 m topographic contours.

In 2008, an air photo survey was conducted over the northern portion of the property to provide better topographic information for infrastructure planning. The survey was conducted by Haut-Mont during the summer of 2008.

Table 9-1: Summary of exploration work carried out on the Éléonore Project

Year	Type	Survey	Area	Company	Amount	Comment/Result
2006	Air Photo		Éléonore	XEOS		Northern Area: 1 m topography contours. Property: 4 m contours
	Geophysics	VTEM	Éléonore	Geotech	3123,5 km	MAG + EM. Anomalies associated with Iron Formation. No significant anomalies associated with Roberto.
2007	Geophysics	Induced polarization	Roberto	TMC	5,7 km	Survey was completed by Geosig following year
	Trenching	Outcrop stripping	Roberto	Goldcorp		Roberto Outcrop was stripped of the overburden over an area of 400 m by 175 m, exposing the main Zones of the deposit.
	Geochemistry	Photo interpretation	Éléonore	Inlandsis		Interpretation of the glacial cover and potential dispersion trends over the property.
2008	Geophysics	Induced polarization	Roberto	Geosig	15,6 km	Strong chargeability anomaly associated with Roberto deposit.
	Geochemistry	Till sampling	Éléonore	Inlandsis	496 samples	5 district anomalous sectors identified, for which additional work is recommended.
	Air photo		Roberto	Haut-Mont		Air photo of northern part of Éléonore.
	Trenching		Roberto North	Goldcorp	9 trenches	A series of trenches in the North area to help understand geological controls on mineralization.
2010	Geochemistry	Lake sediment sampling	Éléonore	Inlandsis	653 samples	Northern half of the reservoir was sampled. Anomalies associated with Roberto and Old Camp.

9.2. Geological and Structural Mapping

Virginia Gold Mines Inc. performed several mapping and sampling programs from 2001 to 2005. Systematic sampling and detailed mapping in 2002 near the Ell Lake showing led to the discovery of a sulphide-bearing quartzo-feldspathic metasedimentary boulder that returned high gold values on assay. Prospecting and reconnaissance mapping successfully exposed surface outcrops of alteration zones and gold mineralization.

From the summer of 2006 to 2011, small mapping and sampling programs were carried out by Goldcorp at 1:20,000 scale in various areas of the Project. During this period, more than 1,000 outcrops were visited and sampled by Goldcorp geologists. This work was used as exploration vectors.

9.3. Geochemistry

Prior to Goldcorp's interest in the Project, Virginia Gold Mines Inc. conducted a soil geochemistry MMI survey in 2003 that yielded 949 samples. Together with other exploration activities, including geophysics and trenching, the aim of the survey was to locate the source of the gold-bearing boulder discovered in 2002.

A till sampling survey was completed during the summer of 2008. The survey was conducted under the supervision of Rémi Charbonneau of Inlandsis. The survey covered the entire Project area and consisted of 496 samples collected at 100 m to 200 m spacing along lines distributed at every 1 km to 1.5 km. A set of 37 duplicate samples were selected for visible gold and other characteristic mineral grain counts, covering the area down-ice from the Roberto deposit.

The program outlined six exploration targets, named sectors A to H. The highest priority targets were considered to be Sector A, which is up-ice from the Roberto deposit, and Sector B, which has two distinct dispersal trains. Sector E was the third-ranked target, located 5 km down-ice from Roberto where till sampling suggested the potential of gold mineralization due to the abundance of pristine-shaped visible gold grains, counts of 5 to 20 scheelite grains, high-grade gold results, and associated Sb, Sn and Cu anomalies.

A lake-bottom sediment sampling survey was completed over the northern portion of the Opinaca Reservoir during the summer of 2010. A total of 653 samples collected with a 75 m to 100 m spacing along north–south lines distributed every 200 m to 300 m. The samples were submitted to multi-element analysis by MMI (Metal Mobile Ion) partial dissolution, followed by sensitive ICP-MS determination for 53 elements, including Au with a 0.05 ppb detection limit. The survey revealed three sectors of interest including Southeast Roberto, Old Camp and the North sector.

9.4. Geophysics

In 2002, Virginia Gold Mines Inc. ("Virginia") performed a ground magnetic survey (33.6 line-km) and an induced polarization (IP) survey (24.6 line-km) over a grid covering 8 km^2 in the Ell Lake area. Several IP conductors were detected.

In March 2003, a fixed-wing magnetic survey was carried out over an area of 23 km by 9.5 km, covering the contact between the Ell Lake intrusion and the Opinaca and La Grande subprovinces. In the meantime, in order to identify the source of the gold- bearing boulder, Virginia performed a ground magnetic survey (81 line km) and IP survey (65 line km) on 200 m spaced lines on the northwest and north shores of Ell Lake.

In February 2004, a helicopter-borne, detailed magnetic survey (45 km^2, 50-m line spacing) more accurately outlined the northern contact between the Ell Lake intrusion and the subprovinces and detected a number of magnetic features suggesting alteration zones in the surrounding sediments.

In January 2005, Virginia carried out an IP survey that covered a total of 226.3 line-km on the northeastern side of Ell Lake. The survey was performed by Geosig Inc

In 2006, a helicopter-borne magnetometer and electromagnetic VTEM survey (time domain) with an in-loop configuration was flown over the entire Project. The survey was conducted by Geotech Ltd and totalled 3,123.5 line km. Line spacing was 75 m and the survey was oriented at N360 degrees. A more detailed survey was conducted over the Roberto deposit area with line spacing of 50 m, which was oriented at N90 degrees.

Anomalies were generated from the VTEM survey. The anomalies located inside the reservoir are large weak conductors and are interpreted to be related to the strong concentration of conductive superficial sediments found in the reservoir. The long anomalies located to the east of Roberto are interpreted to be caused by sediments, such as iron formation and or argillitic sequences that have significant sulphide concentrations. Some of the exploration drilling has since targeted some of the sediment-hosted anomalies and results indicate that they do not carry significant gold mineralization.

A surface induced polarization (IP) survey was conducted totalling some 21.3 line-km between 2007 and 2008 on the Roberto deposit area. The purpose of the survey was to determine the response of the Roberto mineralization to a pole-dipole IP configuration and to verify the efficiency of this method for exploration targeting.

In order to penetrate the conductive layer of the sediments in the reservoir and the stripped area over the main orebody, the pole-dipole configuration used a 50 m separation.

TMC Geophysique from Val d'Or completed the initial six lines (5.7 km) of the survey in 2007. Geosig Inc. of Quebec completed the survey in 2008, undertaking 12 lines for15.6 km.
The survey was successful in identifying anomalies related to the Roberto mineralization and the pole-dipole configuration with 50 m spaced electrodes is the recommended configuration for future surveys.

9.5. Trenches

During the summer of 2001, Virginia Gold Mines Inc. ("Virginia") performed channel sampling along the shoreline of Ell Lake and identified several Au-Cu-Ag occurrences. Best results include: 382 ppb Au over 8.5 m and 359 ppb Au over 8 m. In 2002, channel sampling in the diorite intrusion identified similar occurrences: 1400 ppb Au over 1 m, 860 ppb Au over 1 m.

In 2003, the prospecting and reconnaissance mapping was followed by mechanical trenching in order to test several geological, geophysical and geochemical targets. Samples yielded anomalous to sub-economic grade gold values over metric to decametric widths (0.81 g/t Au over 37 m and 2.42 g/t Au over 5.25 m).

Virginia conducted a mechanical stripping program in the late summer and early fall 2004 in order to test the lateral extensions of the Roberto Zone. Channel samples were taken perpendicular to the main schistosity at 10 to 20 metre intervals, across the entire ore zone and a few metres further into the country rock. The program began on an outcrop sampled in late 2003 which had returned elevated gold values. A total of three trenches were excavated in the Roberto area.

An IP anomaly was identified in 2003, and three trenches were excavated in the area in 2004, which uncovered the Roberto East Zone.

Regional trenches were also excavated to reproduce grab samples gold values and to test geophysical, geological and geochemical anomalies. Channel samples from 25 trenches were taken. The mineralization encountered in the trenches was later targeted by drilling.

The trenches on Roberto Island were enlarged in 2006 and 2007, creating a single exposed outcrop measuring 400 m by 100 m, oriented to the northeast.

Lithologies, alterations, veins and mineralized zones were mapped in detail during the summer of 2007, thereby improving the understanding of the distribution of the mineralized zones and the structural setting.

The outcrop was extensively sampled and assayed for gold and trace elements. Mineralized zones in the form of 3D solids interpreted from drilling data were reconciled with the surface geological interpretation.

In 2008, nine (9) new trenches were opened in the North Area and an old Virginia Mines trench was enlarged. These trenches are smaller in scale than the Roberto Main trenches. Mapping of the trenches improved the understanding of the geological framework in the North Area of the deposit.

Mapping of lithological contacts, structures and veins was done using the same high- precision global positioning system (GPS) instrument used to locate drill hole collars, ensuring a precise map without distortion. The locations of channel samples were also measured with this GPS. All data were imported into ArcGIS software to create a final map of the deposit (Figure 9-1).

Channel samples are treated in the same way as drill core samples. They were laid into HQ-size (63.5 mm) core boxes and brought into the coreshack. The lithologies, alterations, veins and structures were recorded in detail using acQuire software, applying procedures identical to those used for drill core samples (refer to Sections 10.2 and 10.3).

Trench and channel samples were not used for mineral resource modelling purposes.

9.6. Theses

In 2013, J.-F. Ravenelle completed a PhD study funded through a Natural Sciences and Engineering Research Council (NSERC) industry grant, the Institut National de la Recheche Scientifique (INRS), the Geological Survey of Canada (GSC), Goldcorp Inc., and Virginia Mines Inc.:

Ravenelle J.-F., 2013, Amphibolite Facies Gold Mineralization: An example from the Roberto deposit, Eleonore property, James Bay, Quebec. Doctoral thesis, Institut national de la recherche scientifique, Centre Eau Terre Environnement, 283 pages.

Also in 2013, Arnaud Fontaine started a new PhD study funded through a Natural Sciences and Engineering Research Council (NSERC) industry grant, the Institut National de la Recheche Scientifique (INRS), the Geological Survey of Canada (GSC), and Goldcorp Inc., with the title ‘Genesis of the Roberto world class gold

deposit, Superior Province, Quebec, Canada'.

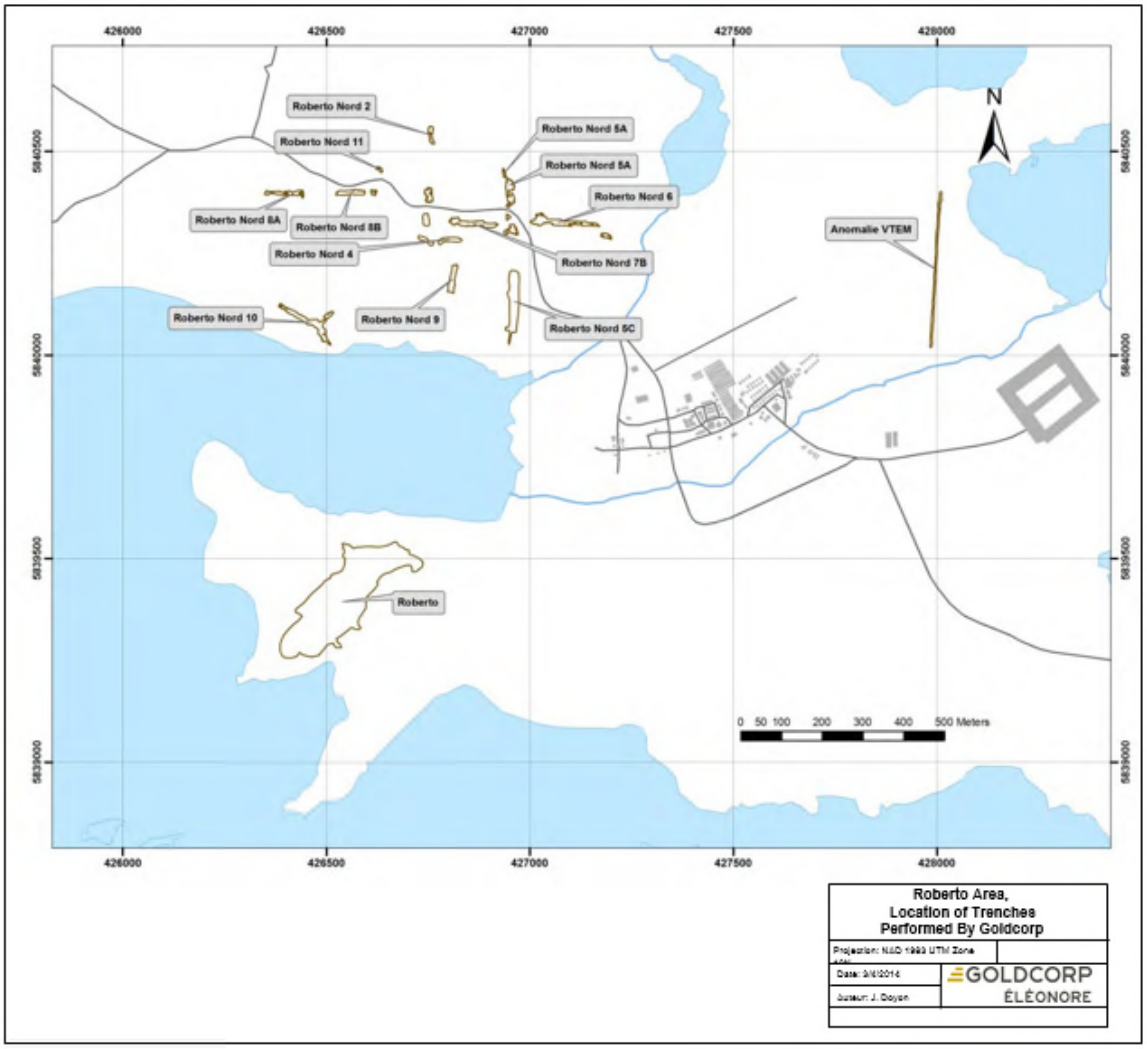


Figure 9-1: Roberto area, location of trenches excavated by Goldcorp (2013)

9.7. Exploration Potential

Exploration potential remains at depth in the Roberto deposit. Mineralization has been drill tested to 1,400 m, in the deepest drill hole to date. Since 2013, the Gaumond shaft has provided a drill base for additional work on the deposit at depth.

Other targets have been identified around the Roberto deposit, including the hanging wall veins (HWV), the north low-grade zone (NLG) and the 494 area.

The HWV are part of the alteration zones surrounding the Roberto deposit. The veins are generally small and erratic, but include alteration haloes which can

range in width from 1 to 5 m wide. Because these zones are close to infrastructure, the HWV may be a potential future source of additional mill feed for the Project if additional drilling supports Mineral Resource estimates for the veins.

The NLG is a wide alteration zone found near-surface. With additional drilling, there is also potential for this zone to support Mineral Resource estimates.

The 494 area is located north of the Roberto deposit and represents the upper extension of a high-grade zone located 1,000 m below surface; it was discovered by hole No. 494 in 2007.

Limited exploration has been undertaken outside the immediate Roberto deposit area, and the remaining claims retain good exploration potential.

9.8. Comment on Section 9

In the opinion of the QPs, the exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project. The exploration and research work supports the interpretations of genesis and affinity.

10. DRILLING

At the end of 2013, a total of 1,878 surface and underground drill holes (approximately 597,625 m) had been completed by Virginia Gold Mines Inc. ("Virginia") and Goldcorp. Details of the various drilling programs are summarized in Table 10-1. Drill hole collar locations in the Roberto area are shown on Figure 10-1 and drill holes collars over the entire property are shown on Figure 10-2.

Drilling includes 556,042 m of exploration and delineation drilling; 25,395 m of geotechnical drilling; 1,850 m for hydrological/water bore purposes; and 1,119 m for metallurgical purposes. Condemnation drilling as well as drilling to support the locations of planned infrastructure was completed from May 1, 2010 to October 31, 2012, for a total of 13,219 m. Drill hole collar locations for these holes are shown on Figures 10-3 to 10-5.

Following the closeout date for the database, another 337 additional drill holes (72,141 m) were completed. Of this total, 66 holes represented drilling for grouting purposes along the exploration and planned production shafts (2,862 m), hydrogeological holes from surface (756 m), and service holes (1,763 m).

Exploration drilling in 2013 aimed to better define the mineralized zones between 250 m and 800 m below surface. This drilling was ongoing at the Report filing date. Results of the drilling appear to confirm the continuity of the geological model as interpreted. Since January 2013, exploration and delineation drilling is exclusively done from underground infrastructures.

Table 10-1: Drill Hole Summary Table

Company	Date	Core Holes Completed	Length (m)
Virginia*	September 2004 to March 2006	329	105,537
Opinaca	April 2006 to December 2006	127	54,726
Opinaca	January 2007 to December 2007	161	76,803
Opinaca	January 2008 to December 2008	174	85,815
Opinaca	January 2009 to December 2009	36	19,773
Opinaca	January 2010 to December 2010	94	29,099
Goldcorp	January 2011 to December 2011	213	52,473
Goldcorp	January 2012 to December 2012	260	60,792
Goldcorp	January 2013 to December 2013	484	112,607
Total		**1,878**	**597,625**

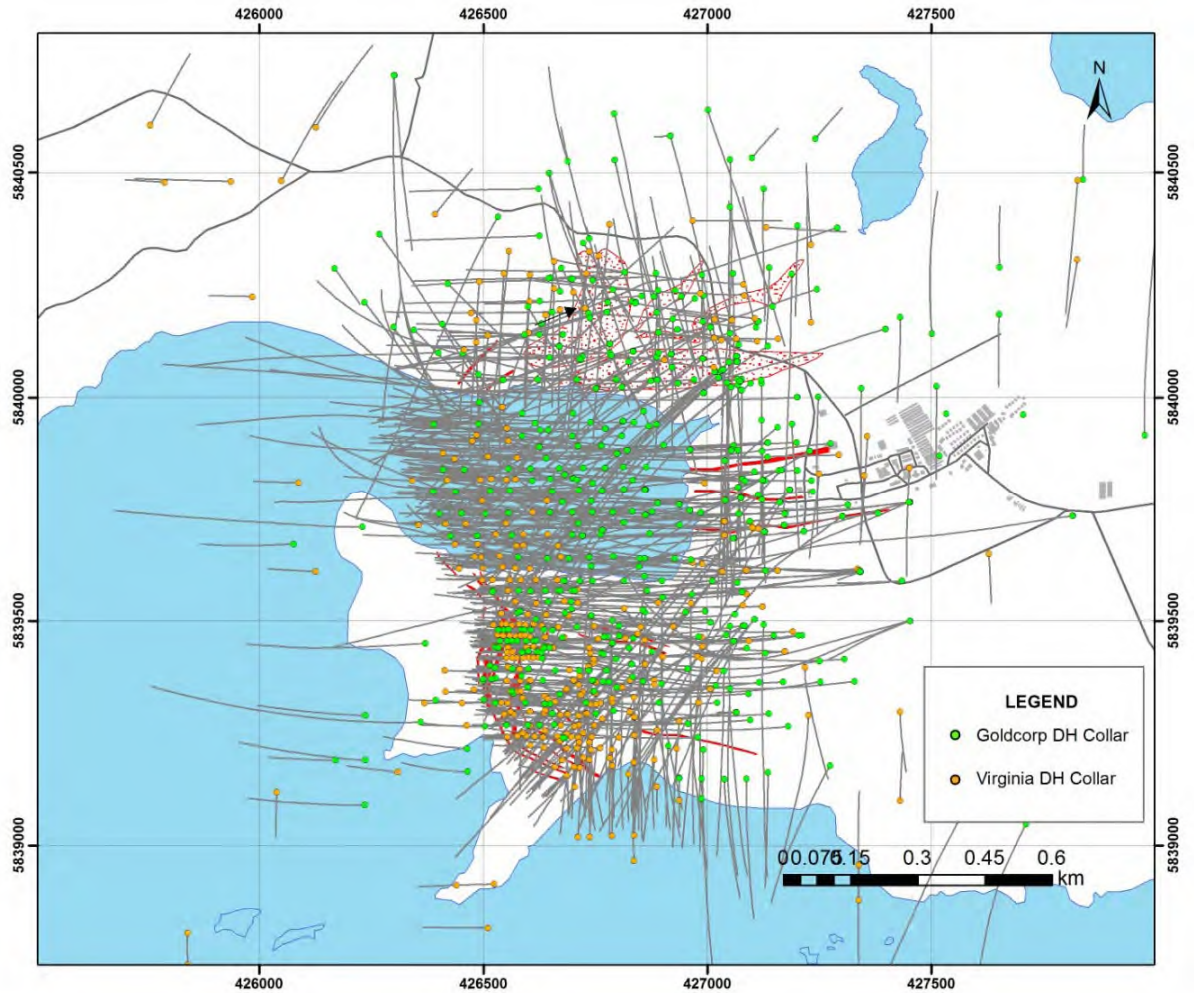


Figure 10-1: Drill hole location plan, exploration and infill drilling programs (2013)

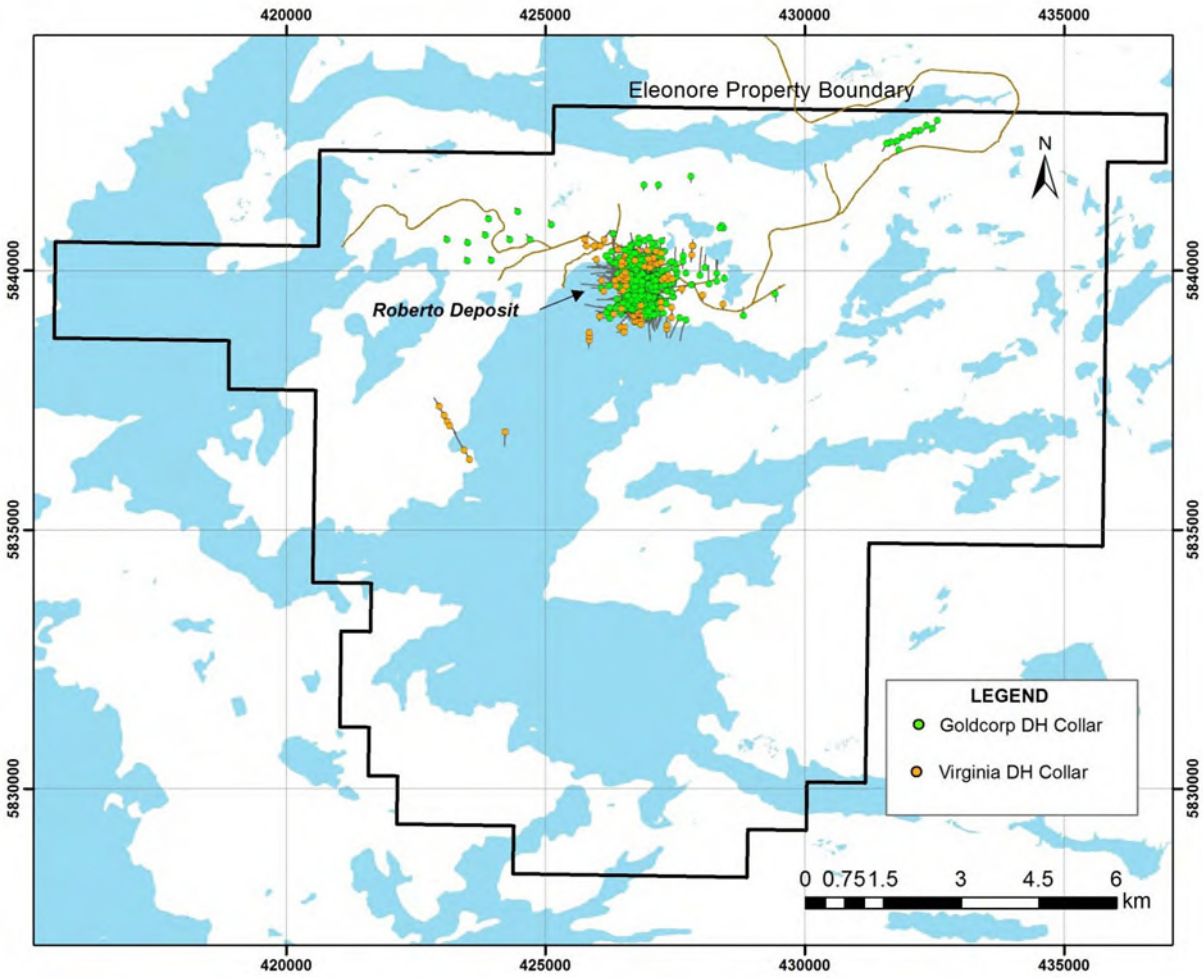


Figure 10-2: Regional drill hole plan- Éléonore property (2013)

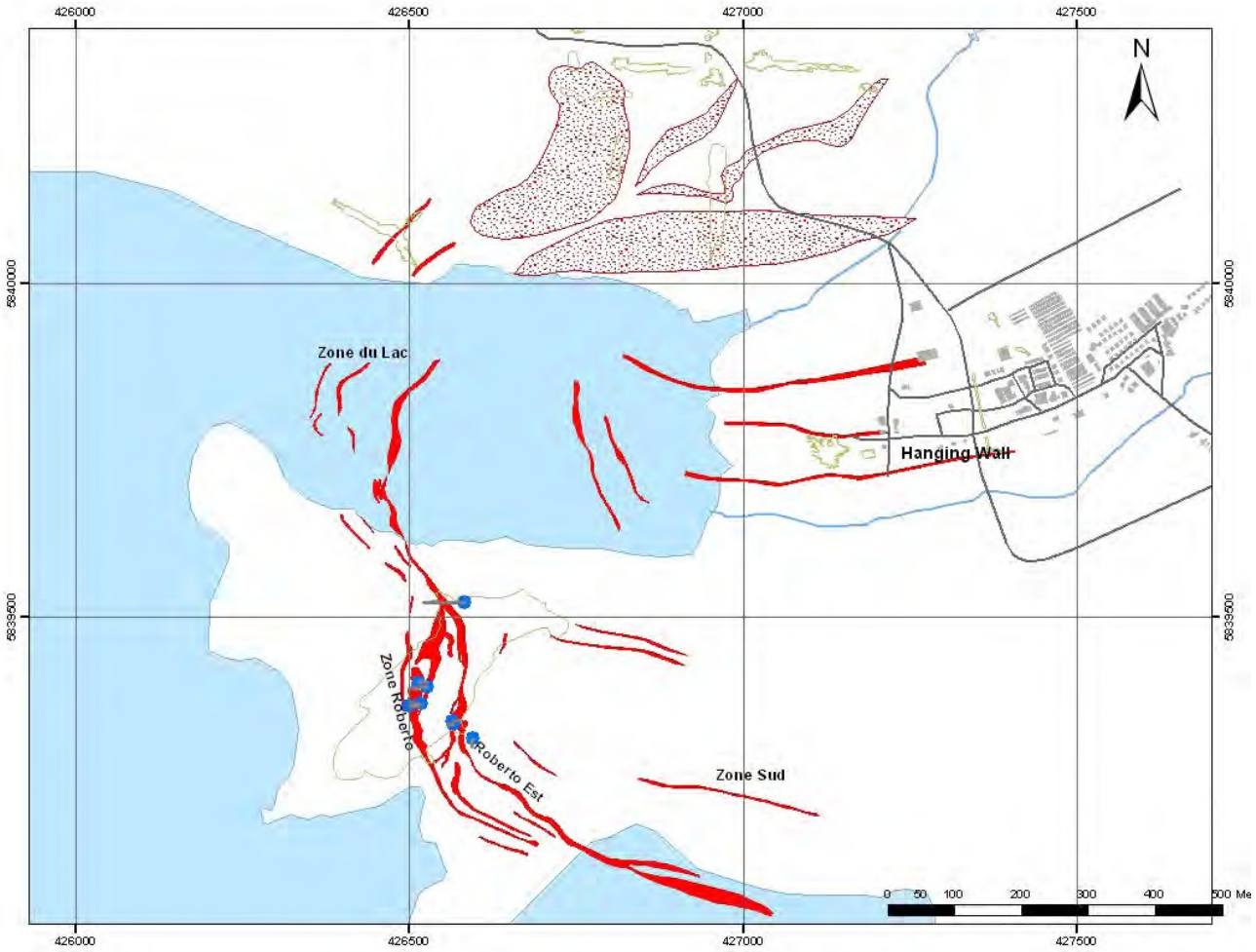


Figure 10-3: Metallurgical drilling collar location plan (2013)

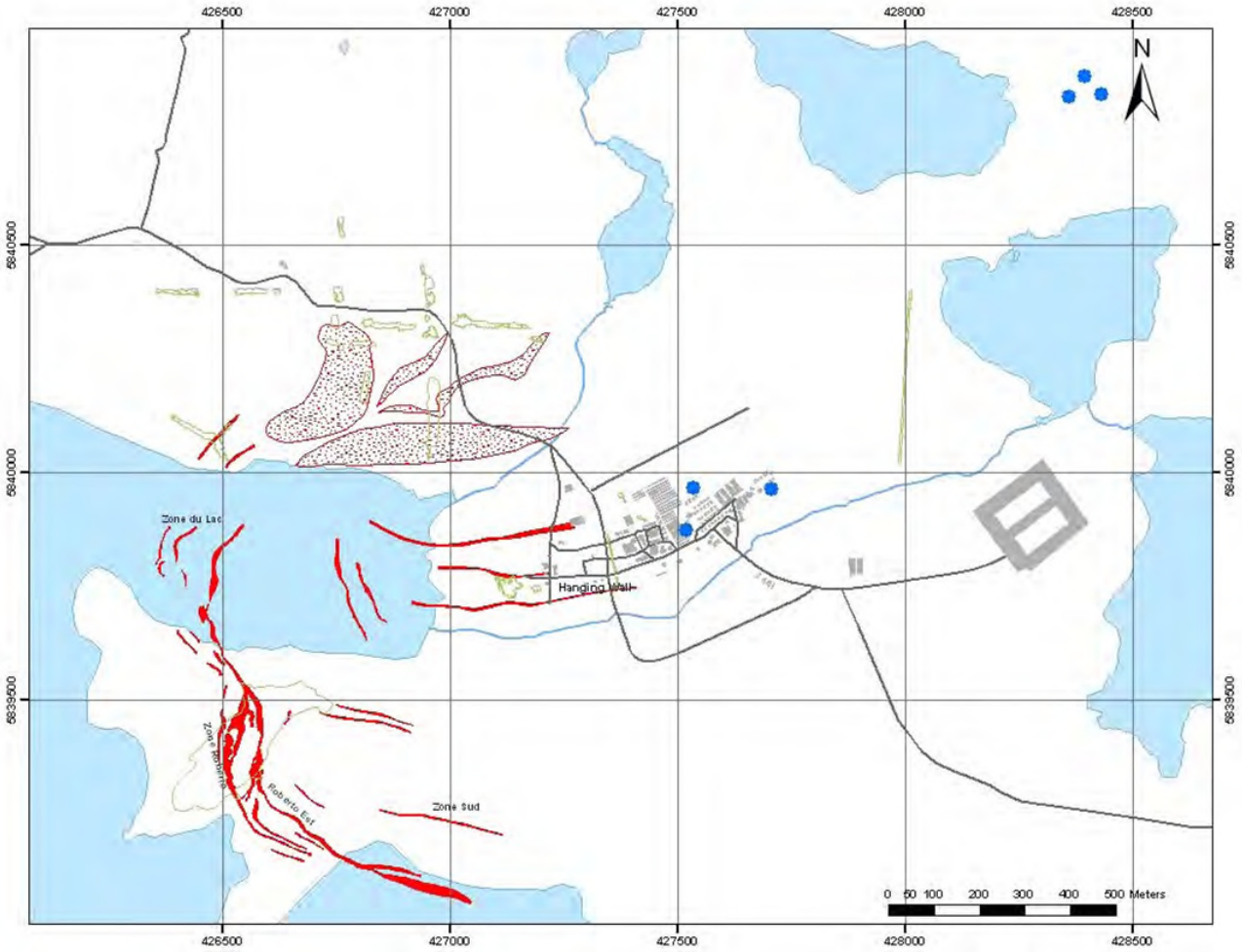


Figure 10-4: Hydrological drilling collar location plan (2013)

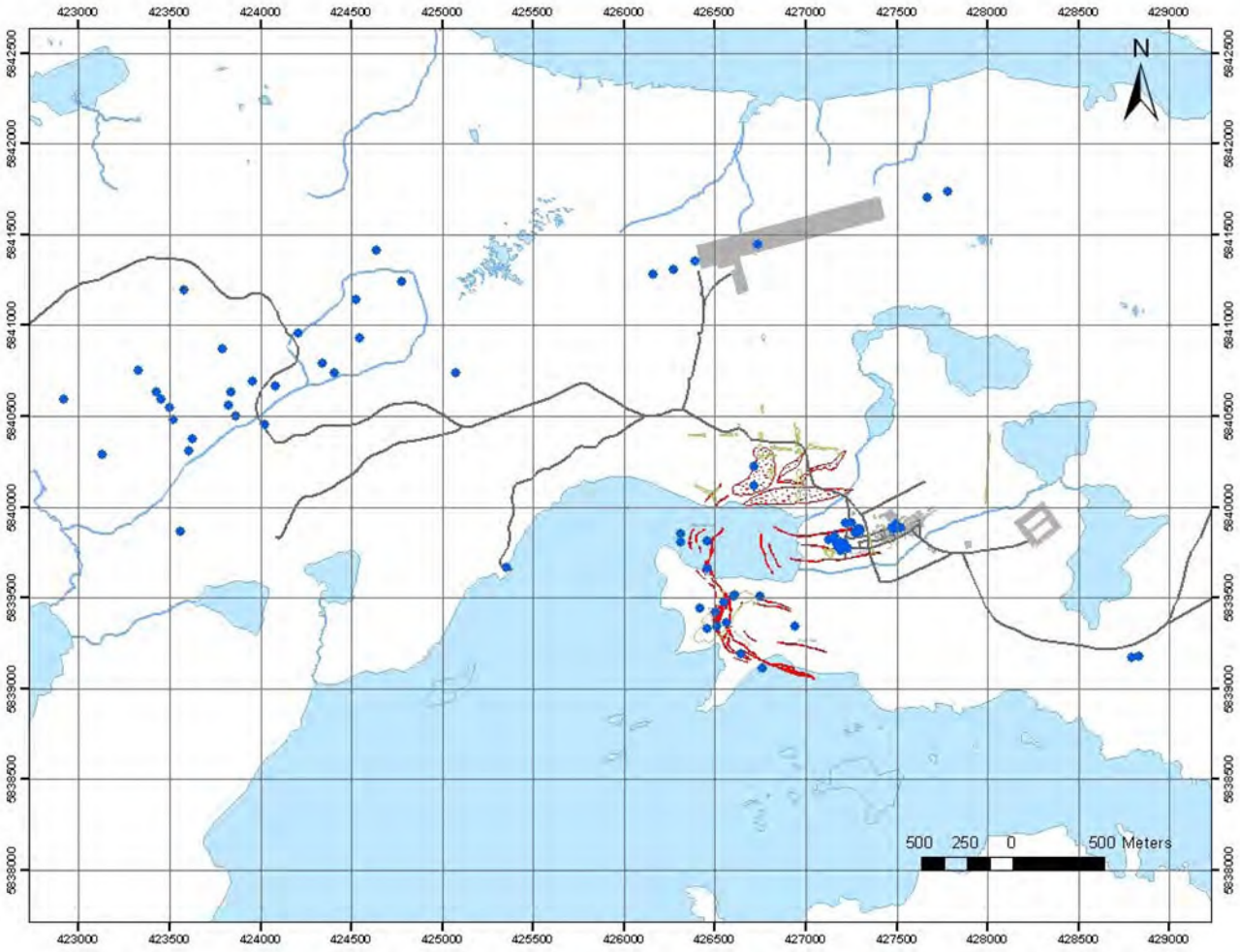


Figure 10-5: Geotechnical/condemnation drilling collar location plan (2013)

10.1. Virginia Gold Mines Drilling

Virginia Gold Mines Inc. ("Virginia") launched an initial drilling program in September 2004. The aimed was to investigate the Roberto Zone and continue testing geophysical, geochemical and geological anomalies that yielded interesting results or could not be explained by channel sampling. At first, two rigs were used, then a third was called in. A total of 69 holes were drilled in the Roberto area for a total of 17,118 m. Drilling successfully confirmed the northward extension of the zone below the lake. A total of 29 holes were drilled on regional targets for a total of 5,585 m. Many of the holes encountered the mineralization and alteration that had been identified in trenches. Several drill holes failed to explain the geophysical and geochemical anomalies they were testing.

The winter 2005 drilling program was the continuity of fall 2004 program. Three rigs were used for the winter 2005 program, and the objective was to test the extension of the Roberto Zone at depth and to the north and south.

A total of 53 holes (17,530 m) were drilled to test the Roberto Zone, and another 11 holes (2,313 m) were drilled to test IP anomalies. Drilling successfully extended the Roberto Zone at depth, to the north and to the south.

During the remainder of 2005, Virginia continued to drill with three to four drill rigs and continued to expand and define the orebody. An additional 70 drill holes were completed for a total of approximately 43,365 m. Between January and March 2006, Virginia drilled a total of 96 drill holes for approximately 19,627 m. These represented infill drill holes that aimed to better define the near-surface portions of the ore zone.

Drilling operations for the 2004 to 2006 programs were carried out by Chibougamau Diamond Drilling Ltd (Forage Chibougamau Ltée). Drill log descriptions were recorded by geologists from Services Techniques Geonordic.

The positions of the holes drilled by Virginia are shown on Figures 10-1 and 10-2. An example of a drilling cross-section is presented in Figure 10-6.

10.2. Drilling Methods

All core diamond drilling completed on the property consists of wireline diamond drilling recovering NQ size (47.6 mm) drill core, except for definition drilling where BQ size (36.4 mm) is used.

Chibougamau Diamond Ddrilling Ltd has been the sole surface core diamond drilling contractor since the beginning of the Project. The numbers of rigs numbers on site have varied from one to six. Since the beginning of underground drilling in 2012, Machines Roger International Inc. has been the sole underground core diamond drilling contractor, using one to ten electrical drill rigs. The companies responsible for the geotechnical drilling are AlXtreme, Forages Giroux, Forages SL Inc. and Technic-Eau.

Since the end of 2012, all drilling has been carried out from the nearest underground infrastructure.

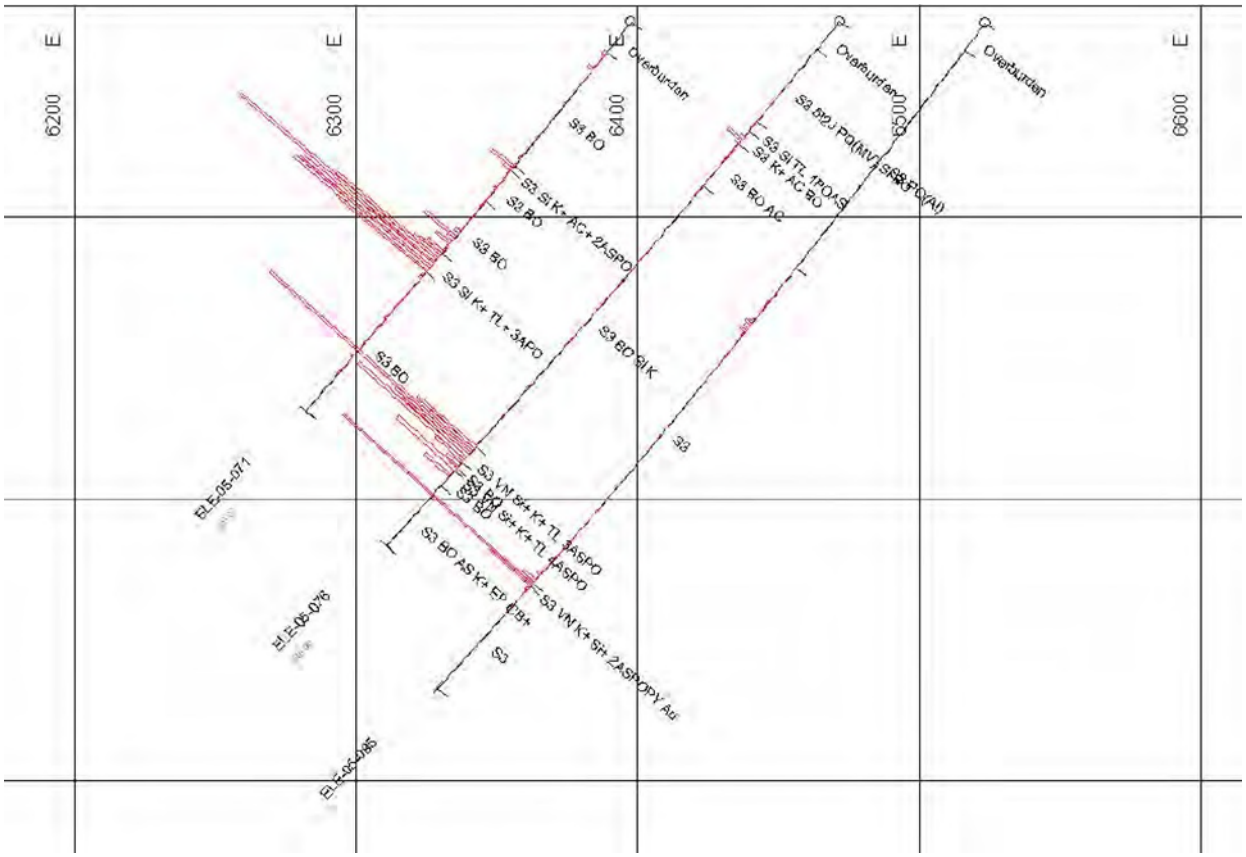


Figure 10-6: Example of a cross-section (2800N) of Virginia's drilling (from the technical report and recommendations for the May 2005 drilling program on the Éléonore property)

Note: S3: Wacke; S3 Vn: Wacke with quartz vein; S3 PQ (Al-Mv): Wacke with muscovite and aluminosilicate porphyroblasts; AC: actinolite; AS: arsenopyrite; BO: biotite; MV: muscovite; TL: tourmaline; Si+: silicification; K+: potassic alteration; CB+: carbonatization.

10.3. Field Procedures

Drill cores are placed into wooden core boxes at the drill site. The core boxes are labelled and closed with metal wire or rubber bands by the drilling contractor. An orientation device, either an Acetool or Corient, is attached to the tube of some drill rigs. The core base is marked by the drillers, based on the tool readings. This technique allows orientation of the core in real space and provides structural measurements that represent actual ground conditions.

Drill core is retrieved at the end of every shift from each drill site either by Goldcorp employees or drill contractor personnel. Core boxes are placed onto racks located in front of the logging core shack. To avoid mixing drill hole data, each drill rig has its own well-labelled storage racks.

10.4. Geological Logging

Core boxes are opened immediately upon arrival at the core shack, and the geologist responsible for that drill rig logs a summary description of the drill core (Quick Log). This information is then plotted on the corresponding drill section together with the drill hole deviation in order to facilitate tracking of drilling progress within the drill hole.

The core boxes are subsequently brought into the geotechnical core shack, where the core is correctly repositioned in the core boxes by locking the core pieces to each other. Core is then carefully measured. For boreholes on which an orientation device was used, the core is repositioned using the driller's marks and a green line is traced indicating the bottom of the hole.
Core boxes are permanently labelled using aluminum tags that include the drill hole number, box number and the depth interval start and end points.

Once it is measured and oriented, the drill core is moved to the logging core shack, where a geologist records a detailed description of the lithologies, structures, mineralization, alteration, and veining. Logging forms have pre-defined pick lists for codification of the geological descriptions and built-in validation to minimize wrong entries. To ensure the geologists log the core samples in a consistent manner, the data entry procedures were documented and assembled into a manual. Core library and posters are available for geologists-in-training; these geologists are also tutored by a more experienced geologist until logging proficiency is attained.

After completion of the core description, the geologist is responsible for marking the samples on the core. Photos of the core for the entire drill hole length are then taken, with four boxes photographed per picture. Specific gravity measurements and point load testing are carried out before the core boxes are moved to the core-cutting facilities.

Once the core samples have been cut, the boxes containing the remaining core halves are placed in an outside permanent core rack.

10.5. Recovery

Core recovery and RQD is measured and calculated for each core. Core losses are recorded in the drill log. Rock units intersected by drilling are generally solid, yielding an effective core recovery of 100%.

10.6. Collar Surveys

Until December 2005, surface drill hole collars were surveyed using Garmin hand-held global positioning system (GPS) instruments, or were chained from already-completed boreholes.

In January 2006, a land surveyor was mandated to install a Trimble TSC1-V7.50 GPS fitted with differential correction from a base station located on site. Measurement precision is two centimetres, both in the horizontal and vertical axes. From that date, all new collars were surveyed using this system. Core holes drilled in 2004 and 2005 were easily resurveyed as casings were left in place, with the exception of 19 core holes that had been drilled with rigs set on lake ice during the winter of 2004 and 2005. For these, collar locations may be inaccurate by a few metres.

Procedures for drill hole collar surveys include paper tracking of final collar surveys that must be signed by the surveyor.

The location of proposed drill hole collars is assessed using a Trimble GPS and marked with a picket. Front sights are implanted at 15 m, 30 m and 55 m with the GPS and double-checked with a compass. Drilling is visually aligned using the front sights.

Upon completion of the final collar survey, the planned collar coordinates and the surveyed collar coordinates are compared and any discrepancies investigated.

Underground drill holes are surveyed using a Leica TS15 robotized station.

10.7. Down-Hole Surveys

Down-hole surveying has been performed routinely on every drill hole since Project inception, using a rented FlexIt SmartTool electronic instrument.

During drilling, a single-shot test is taken approximately 15 m past the casing to determine the initial drill orientation. Following this, single-shot tests are taken approximately every 60 m to 100 m at the end of drilling shifts. If in the process of drilling the drill rods need to be removed from the bore hole, the FlexIt tool is used in multi-shot mode to conduct tests every 3 m or 6 m.

Once a drill hole is deemed complete, the FlexIt tool is used in multi-shot mode to conduct tests every 3 m or 6 m as the drill rods are being pulled. Multi-shot data are downloaded into the drill hole database and verified by the geologist. Station data that produces anomalous azimuth readings due to excessive local magnetic interference are flagged to prevent their use in plotting the drill trace. Multi-shot

readings are used in favour of single-shot readings and are flagged to be used when plotting the drill trace.
The magnetic north azimuths are corrected automatically in the acQuire database to the true north azimuth using a correction of -17 degrees.

The FlexIt tool is sensitive to the magnetic properties of the rocks encountered in the drill hole. However, other than the diabase dykes, diorite and an occasional mineralized unit in the northern part of the property, rocks underlying the Roberto deposit are very weakly magnetic and do not generally affect the azimuth readings taken by the down-hole survey tools.

In 2010, a test was performed on one drill hole, DDH ELE-10-00695-M01, over a distance of 1,338 m using a north-seeking Gyro instrument with the aim of comparing the results with those obtained with the FlexIt tool. The results showed an overall difference of 0.25% between both surveys at a distance of 1,338 m down hole.

10.8. Geotechnical and Hydrological Drilling

These holes were logged and sampled as per normal procedures.

10.9. Metallurgical Drilling

All the metallurgical core samples from drilling are identified in appendices of SGS Lakefield metallurgical test work reports. The drill samples represent the Roberto and Zone du Lac orebodies from surface to 750 metres depth with various gold head grades, ranging from 2 to 20 g/t Au.

10.10. Condemnation Drilling

Drilling was undertaken on the construction site in areas where infrastructure is planned to be built to confirm the low potential for economic mineralization. These holes were logged and sampled as per normal procedures.

10.11. Drill Spacing

Drilling has been conducted over the Roberto deposit on a 1,500 m by 1,500 m area. The drilling pattern was designed to sample the deposit orthogonally to the interpreted strike and dip of the gold mineralization. The majority of the core holes were drilled with an inclination varying between -45° to -63°.

All core holes were drilled on sections spaced approximately 25 m apart in most parts of the deposit. Drill hole spacing of 25 m by 25 m occurs over the bulk of the orebody to a depth of approximately 650 m below surface. Between 650 m and 850 m below surface, borehole spacing increases to about 50 m by 50 m.

Below 850 m, down to approximately 1,200 m, a core hole spacing of 100 m by 100 m is usually observed. Only a few drill holes were drilled below 1,200 m. The deeper boreholes intersected the mineralized horizons at a depth of approximately 1,400 m below surface. For definition drilling, drill hole spacing is generally 12.5 m by 12.5 m inside the existing 25 m drill spacing as permitted by the mine development schedule.

10.12. Drill Sample Length/True Thickness

Sample intervals are determined by the geologist logging the core. Samples do not cross geological boundaries; they respect lithological, alteration, mineralization and structurally interpreted boundaries. The minimum sample length is 0.30 m, and the maximum is 1.5 m in waste and 1.0 m in mineralized material.

True thickness interval lengths are defined as being perpendicular to the strike and dip of the mineralization at the point of bore hole intersection. It is the shortest distance between the hanging wall and the footwall points of intersection of the bore hole with respect to the strike and dip of the mineralization. Due to the irregular shape of the orebody, there is no predetermined angle for this.

10.13. Drill Intercepts

Typical drill hole orientations are as shown in Figure 7-4 in Section 7. Table 10-2 shows a selection of intersections through the low-grade envelope (which contains the high grade envelope) to illustrate typical grades within the deposit.

Table 10-2: Drill Intercept Summary Table (V: Virginia; G: Goldcorp)

Hole ID	Collar Information					Down Hole Information				
	East	North	Elev.	Azimuth	Dip	From (m)	To (m)	Intercept	Drilled Length (m)	Area
ELE-04-00034 (V)	426654.5	5839440.9	10217.2	273	-68.0	40.00	47.40	30.99	7.40	Main
ELE-05-00061 (V)	426686.2	5839248.1	10214.9	183	-50.0	91.23	94.70	6.29	3.48	South
ELE-05-00106 (V)	426880.9	5839191.3	10213.4	183	-45.0	123.00	131.00	5.24	8.00	South
ELE-05-00113 (V)	427092.2	5839456.1	10216.6	268	-58.0	543.01	546.00	41.78	2.99	Main
ELE-06-00162 (V)	426614.6	5839243.3	10215.8	272	-51.0	51.75	55.00	22.06	3.25	Main
ELE-06-00194 (V)	426475.2	5839903.4	10214.6	268	-46.0	62.40	71.00	13.68	8.60	Bay
ELE-06-00204 (V)	426722.5	5839195.4	10215.6	225	-45.0	12.71	28.00	14.50	15.29	South
ELE-06-00338 (G)	426660.7	5839390.0	10215.3	269	-50.0	78.00	81.49	3.32	3.49	Main
ELE-07-00392 (G)	427009.7	5839590.1	10214.4	258	-55.0	488.12	494.00	10.49	5.88	Main
ELE-07-00403 (G)	426876.2	5839744.7	10213.8	265	-55.0	665.80	667.00	10.15	1.20	Bay
ELE-07-00506 (G)	426668.9	5840115.0	10218.6	268	-50.0	102.00	114.00	2.25	12.00	North
ELE-08-00575 (G)	426835.0	5839461.1	10214.2	225	-52.0	442.35	446.25	3.22	3.90	South
ELE-08-00620B (G)	427171.3	5839714.1	10217.4	265	-56.0	937.79	943.15	12.01	5.36	Main
ELE-08-00627 (G)	426454.1	5840098.5	10216.9	145	-50.0	95.00	103.00	13.28	8.00	North
ELE-09-00649B (G)	427214.7	5839700.1	10217.0	263	-56.0	913.30	919.00	3.01	5.70	Main
ELR-05-00037 (G)	426434.8	5839864.1	10216.4	273	-53.0	25.00	40.00	4.10	15.00	Bay
ELR-05-00038 (G)	426574.6	5839865.4	10216.1	272	-52.0	220.00	238.00	5.64	18.00	Bay

10.14. Comment on Section 10

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar and down-hole survey data collected in the exploration and infill drill programs completed by Virginia Gold Mines Inc. and Opinaca are sufficient to support Mineral Resource and Mineral Reserve estimation as follows:

- Core logging meets industry standards for gold exploration within a sediment-hosted stockwork-disseminated gold deposit in an orogenic setting.
- Collar surveys have been performed using industry-standard instrumentation.
- Downhole surveys were performed using industry-standard instrumentation.
- Recovery data from core drill programs are acceptable.
- Geotechnical logging of drill core meets industry standards for planned underground mining operations.
- The drilling pattern provides adequate sampling of the gold mineralization for the purpose of estimating Mineral Resources and Mineral Reserves.
- Drilling is ideally perpendicular to the strike of the mineralization. Depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths are typically greater than true widths.
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.
- Drill orientations are shown in the example cross-section included in Section 7 as Figure 7-4, and can be seen to appropriately test the mineralization.
- Drill hole intercepts, as summarized in Table 10-2, appropriately reflect the nature of the gold mineralization. The table demonstrates that sampling is representative of the gold grades in the deposit area, reflecting areas of higher and lower grades
- No material factors were identified with the data collection from the drill programs that could affect Mineral Resource or Mineral Reserve estimation.

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1. Sampling Methods

11.1.1. Geochemical Sampling

Till samples, which were approximately 15 kg each, were processed using a shaking table to extract 150 g to 250 g of dense minerals. The dense fraction was submitted for analyses by fire assay for Au and ICP-MS for 54 elements.

Several drill holes on several adjacent sections were selected for top to bottom geochemical sampling. All channel samples were sent for geochemical analysis.

The preparation, packaging, tracking procedures and quality control program developed for drill core samples (refer to Section 11.1.3) also apply to geochemical sampling but samples are marked for geochemical analysis.

11.1.2. Trench and Channel Sampling

The preparation, packaging, tracking procedures and quality control program developed for drill core samples (refer to Section 11.1.3) also apply to channel samples. The only difference is that sample tags are inserted in the field and the whole channel sample is sent to the laboratory.

11.1.3. Drill Sampling

The sample preparation used by Virginia Gold Mines Inc. ("Virginia") is similar to the one currently used by Goldcorp. Virginia's method is described below; the only difference was the use of a hydraulic splitter to split the core from country rocks adjacent to ore zones.

Since mid-2007, exploration drill cores have been systematically sampled from top to bottom. For in-fill drilling with 25 m spacing or below, sampling is systematically done on complete mineralized envelope with 7.5m closure on each side of the envelope. Sampling is designed to reflect the general geology, all significant alterations and significant mineralization found on the property. Sample lengths vary between 0.3 m and 1.50 m.

Sample intervals are marked on the cores using a red marker. The geologist draws a red line on the core marking where it needs to be cut. The geologist also inserts the sample tag and marks the corresponding sample number on the core.

All exploration drill cores are cut using a diamond saw. One half of the core is put into a plastic bag along with a portion of the sample tag and the other half of the core is left in the core boxes. When cutting the core, the sampling technician must always follow the red line and must also always sample the same side of the core. The BQ size definition drilling core are wholly bagged and send to the lab following the same protocol as the exploration core.

Sample tags are pre-stamped indicating where standard reference materials (SRMs), blanks, duplicates and quarter splits should be inserted. Tags for sample SRMs are inserted in the core box beside the previous sample tag. This procedure alerts the sample technician that there is an SRM, a blank, a duplicate or a quarter split to be inserted into or taken from the core boxes.

The sample bags are closed shut in such a way as to avoid losing any material. The bags are pre-labelled by the sampling technician. The samples are then put into sequence on a table so that standards and blanks may be inserted into the boxes, under the supervision of the senior technician, and they are readied for shipping.

The geologist enters the required sampling information in the acQuire database and in the sample booklets.

11.2. Metallurgical Sampling

The drill core samples were crushed to pass 6 mesh. A designed mass was riffled out of each -6 mesh sample and combined according to zone and elevation. Approximately 6 kg was riffled out of each blended elevation composite and submitted for standard Bond ball mill grindability testing. The balance of each 6 mesh elevation composite was further crushed to pass 10 mesh and was separately rotary split onto 1 kg test charges for metallurgical testing and head & chemical analysis.

The whole and half-core PQ core size received was specifically designated for the comminution testing.

One kilogram samples was submitted for screen metallics protocol analysis for gold and silver at +/- 150 mesh (106 um). The entire screen oversize (3-5% mass) was fire assayed to extinction. Duplicate 25 to 30 g aliquots were riffled from each screen undersize and submitted for fire assay for gold and silver.

11.3. Density/Specific Gravity Determinations

Collecting specific gravity data was initiated after Project acquisition by Goldcorp. The protocol is as follows. Once the geological logging is completed and before the core is sampled, pieces of about 10 cm are measured, weighed dry and then weighed wet. Data are recorded in the acQuire® logging database and specific gravity is automatically calculated. If the calculated measurement is above the limits set for the Éléonore rocks, then the system flags the entry in red and the measurement is taken again.

A total of 15,369 measurements of specific gravity measurements were completed. A measurement was taken in the middle of each mineralized zone greater than 1 m and at 3 m and 6 m from the host rock. Point load tests were also taken on the same samples as well as every 50 m along the borehole length. The table 11-1, summarise the specific gravity per zones.

11.4. Analytical and Test Laboratories

The laboratories used during the various exploration, infill and step-out drill analytical programs were:

- From 2004 to December 2006, core samples were prepared and analyzed at Laboratoire Expert (Expert) in Rouyn-Noranda as the primary laboratory. This laboratory was not certified before 2006.
- Since January 2007, preparation and samples assay are performed by ALS- Chemex (ALS) in Val d'Or, Quebec, which is accredited for ISO 17025 and 9001/2008.
- Until June 2010, SGS Laboratory in Rouyn-Noranda acted as the secondary (umpire) laboratory for both the Expert and ALS Chemex programs. The laboratory has ISO9001 certification and holds a certificate of accreditation to conform to ISO/IEC 17025.
- All samples selected by Virginia Gold Mines Inc. for multi-element analysis were sent to Actlabs laboratory in Toronto.

Metallurgical testwork has been completed at a number of laboratories. Laboratories used are summarized in Table 11-2. Metallurgical laboratories are not typically accredited or certified.

Table 11-1: Specific Gravity Summary

Zones	Number of Samples	Average Specific Gravity
Auriferous Zones	3,246	2.77
Wacke (S3)	8,161	2.76
Conglomerates (S4)	78	2.70
Argillites (S6-S9)	14	2.95
Paragneiss (M4)	99	2.74
Diorite (I2J)	77	2.78
Pegmatite (I1G)	248	2.63
Total	**11,923**	**2,76**

Table 11-2: Metallurgical Testwork Laboratories

Laboratories	Orebody Samples	Metallurgical Testwork	Metallurgical Reports
SGS Lakefield Research (Ontario)	Zone Roberto	Comminution; Gravity (GRG); Flotation; Cyanidation of whole ore / gravity tailing	SGS Lakefield Research (Ontario)
SGS Lakefield Research (Ontario)	Zone Roberto at different elevations and grades	Comminution; Gravity (GRG); Flotation; Cyanidation of gravity tailing /flotation tailing	SGS Lakefield Research (Ontario)
SGS Lakefield Research (Ontario)	Zone du Lac and Roberto	Comminution; Gravity (GRG); Flotation; Cyanidation of gravity tailing /flotation tailing	SGS Lakefield Research (Ontario)
SGS Lakefield Research (Ontario)	Zone Roberto	Communition and flowsheet comparison (Gravity tails flotation prior cyanidation / Gravity tails cyanidation prior flotation) and Cyanide Destruction	Project 11385-006 – Final report (Sept 28, 2010)
CANMET Mining and Minerals Sciences Laboratories	Zone Roberto (Flotation concentrate produced in 2009 by SGS)	Cyanidation of gold from ultrafine sulphide concentrate	Design of a Leaching technology to Extract Gold from Éléonore Mine Ultrafine Sulphide Concentrate (Project 603546, May 2010)
Golder Pastec (Ontario)	Zone Roberto (Flotation tails and concentrate produced in 2009 by SGS)	Rheological characterization; Dewatering testing; Potential backfill recipes	Material testing of the Opinaca Éléonore Mine Tailings, (June 15, 2010)
FLSmith (Utah)	Zone Roberto (Flotation tailings produced in 2010 by SGS)	Sedimentation, Filtration and Rheology Testing	Sedimentation, Filtration and Rheology tests on Gold Tailings – Éléonore Project (August 2010)

11.5. Sample Preparation and Analysis

11.5.1. Expert Sample Preparation Procedures

At the laboratory, samples were received and sorted on the floor, manually recorded, and later entered in an Excel® spreadsheet and a partial LIMS (GEMS®). Labels were printed out for pulp bags. Drill core was dried if it was judged to be necessary in an oven without temperature control.

Samples were reduced to about 6 mm in a Denver jaw crusher and then reduced again to 2 mm in a roll crusher. Crushed samples were placed in an aluminium pan and into a riffle splitter. Split sample weight was about 200 g and the sample split was then pulverized in a TM ring and puck pulverizers. A 30 g aliquot was weighed and about 125 g of flux and litharge added in the crucible. The sample was fused in a 28- pot electric furnace. The fused samples were poured in moulds and lead buttons are separated from the slag by hammering.

11.5.2. ALS Chemex Sample Preparation Procedures

Upon arrival at the laboratory, samples are sorted on benches or on the floor, logged into ALS database tracking system (GEMS®), and identified with a bar code. Most of the procedures are tracked by the GEMS® software. Samples are then dried in a forced air dryer under controlled temperature conditions.

Samples are reduced with “TM Engineering Rhino & Terminator” crushers in a single pass or in multiple passes to obtain a primary crushed material that is better than 70 to 90% passing 2 mm. Crushed samples are placed in an aluminum pan and fed into a riffle splitter.

Split samples are pulverized in TM 300 g or LM-2 pulverizers with greater than 85 of the pulverized sample passing through a 75 µm screen.

A 50-gram aliquot is used and 200 grams of flux are added to the sample. Samples are fused in 84-pot auto-pour fusion furnaces fitted with a digital temperature control system. Lead buttons are separated from the slag in a template.

11.5.3. Actlabs Sample Preparation Procedures

A 0.5-g sample was digested with aqua regia (0.5 ml H2O, 0.6 ml concentrated HNO3 and 1.8 ml concentrated HCl) for 2 hours at 95°C. The sample was cooled then diluted to 10 ml with deionized water and homogenized**.**

11.5.4. Expert Analytical Procedures

Gold assays were performed by standard fire assay with an atomic absorption spectrometry (AAS) finish using a Varian instrument. For assay results equal or above 3.0 g/t Au, samples were re-assayed with a gravimetric finish. Before Goldcorp's Project interest, Virginia Gold Mines Inc. was routinely analyzing every sample above 500 ppb Au by gravimetric finish.

Expert reported a detection limit of 5 ppb for AAS determination and 0.03 g/t Au for gravimetric analyses.

No other elements were routinely assayed.

11.5.5. ALS Chemex Analytical Procedures

Gold assays are performed by standard fire assay with an AAS finish. For assay results equal or above 3.0 g/t Au, samples are re-assayed with a gravimetric finish. ALS Chemex reports an upper limit of 10 g/t Au and a detection limit of 0.01 g/t Au for AAS analyses.

No other elements are routinely assayed.

11.5.6. Actlabs Analytical Procedures

The samples were analyzed using a Perkin Elmer OPTIMA 3000 Radial ICP for the 30-element suite.

11.6. Quality Assurance and Quality Control

11.6.1. Expert

At Expert, no quality assurance or quality control (QA/QC) was undertaken during sample preparation. The monitoring of gold accuracy included the insertion of one RockLabs standard reference material (SRM) per 28 samples, the insertion of one pulp duplicate every 12 samples, and the insertion of one analytical blank in each batch of 28 samples. SRMs were rejected if the analysis was at two standard deviations from the mean. SRM and blank data were reported in the certificates of analysis.

11.6.2. ALS Chemex

The quality control samples are automatically inserted into the sampling queue by the GEMS system. Sample preparation quality control includes a sizing of the crusher and pulp products at the beginning of each shift for each machine and QC data is captured and plotted on charts. Each batch of 84 fusion samples contains one analytical blank, three pulp duplicates and two Rocklab SRMs for gold accuracy monitoring. Results are reported in the assay certificates and imported into the Project acQuire database.

11.6.3. Actlabs

A matrix standard and blank was run every 13 samples. A series of USGS geochemical standards were used as controls.

11.6.4. Virginia Gold Mines Inc.

From August 2004 to December 2005, Virginia Gold Mines Inc.was inserting approximately one standard and one blank in every batch of 50 samples and randomly inserting for each mineralized zone a supplementary standard and blank. From November 2004, a quarter splits of 10% of all samples from mineralized zones were sent to SGS Laboratory in Rouyn-Noranda for analysis and the pulps was were re-assayed by Expert.

11.6.5. Goldcorp Eleonore (Opinaca)

Each sample batch of 50 contains two SRMs, two blanks, two duplicates and one quarter- split sample. For SRMs, the sampler must obtain the corresponding SRM from the senior technicians who are responsible for ensuring that the type of SRM being inserted is appropriate. Once inserted, the sample bag is treated like any other sample.

QA/QC samples types and positions are predetermined by the acQuire program based upon the ending two digits of sample number and check sample types within a set of 100. Standard grade types are determined by the technician who runs the acQuire program and stamps the sample tags before being used by the logging geologist who inserts them into the core. Example of QA/QC samples positions:

08 Standard
12 Blank
20 Replicate of 19 (Pulp Lab Duplicate)
28 Standard
32 Blank
40 Replicate of 39 (Pulp Lab Duplicate)
49 ¼ Split of 48 (Field Duplicate)
58 Standard
62 Blank
70 Replicate of 69 (Pulp Lab Duplicate)
78 Standard
82 Blank
90 Replicate of 89 (Pulp Lab Duplicate)
99 ¼ Split of 98 (Field Duplicate)

Between 2004 and June 2006, SRMs from RockLabs were used. They were gradually replaced by standards from Canadian Laboratories (CDN Labs).
In June 2007, in-house SRMs were created from Éléonore sample rejects and in April 2008, in-house SRMs were created from PQ drill core and prepared by CDN

Labs. The SRMs were bagged in 120 g pouches by CDN Labs. These standards went through a round-robin analysis process and have been certified by Smee & Associates Consulting Ltd.

Blank material consists of commercially-available marble chips used for landscaping. Duplicate samples are used at every stage of the sampling protocol. A sampling duplicate, a coarse reject duplicate and a pulp duplicate are inserted in the sample stream.

Laboratory duplicates are a repeat of the previous sample and are a split of the sample pulps. This duplicate is performed by the laboratory and is part of the internal controls.

To create the coarse reject duplicate, an empty sample bag containing the duplicate sample tag is stapled to the sample to be duplicated in the sample stream. This alerts the laboratory staff to process a second split of the crushed material. This sample then goes to the pulverization and assaying stage in the same manner as any other sample.

The sampling duplicate, which consists of splitting the remaining half core left in the core box into quarters, and selecting a quarter-core sample, is inserted as a regular sample in the batch. This sample goes through all the sampling stages and therefore has the most variability of all the duplicate samples.

Digital assay certificates received from the laboratory are imported through a built-in routine in the acQuire® software on a daily basis. Control sample graphs are automatically displayed during the import process and standard, blank or duplicate failures are flagged in red. The database manager analyzes the graphs and takes the decision whether or not to import or not the assay results based on a set of fixed rules. If the batch fails the QA/QC rules, the laboratory is requested to re-run the batch.

When importing batches into acQuire, if adjacent standards fall between 2 and 3 standard deviations on the same side of the mean in the immediate group of standards (either preceding or following), the standards are classified as failures. Standards are accepted within three standard deviations for a single standard failure and blanks within 50 ppb. The following fixed rules are applied:

1) When a standard/blank within a batch fails outside of the two standard deviations or above 50 ppb, ensure that the standard type is accurate and has not been improperly labeled in acQuire. If, by comparing the resultant value to known standard values it can be accurately ascertained that the standard has been mislabeled, print the acQuire error graph, write the current date, error and action taken to fix the error on the top of the page and file in the QA/QC binder. Rerun the import routine with the corrected standard type.

2) When a batch fails due to poor duplicate results, confirm the amount of error. The duplicate error in acQuire is set for the course reject at 90% of the duplicates to be within a 20% precision. If the results are beyond this, there may be a sample mix-up at the lab and the wrong samples were reported duplicated. If this is the case, import the file and reject the batch, the batch must be re-run. Print the acQuire error graph, write the current date, error and action taken to fix the error on the top of the page and file in the QA/QC binder. Send a note to the lab with the reason for the batch failure with the appropriate batch file. Unless for otherwise obvious reasons, a batch cannot fail due to duplicate failures.

3) When a batch fails due to a single standard/blank failure, the grade of the samples within the batch must be examined:

    a) If there are no grade intersections of interest, i.e. all the samples are below 0.5 g/t and there are no intersections expected and the other QA/QC passed, the batch may be imported and accepted with the error. Print the acQuire error graph, on it write the check sample number, the current date, the reason the batch was imported with the error and the word “Passed” and file in the QA/QC binder.

    b) If there are grade intersections of interest and the standard/blank falls within it, import the file and reject the batch, the batch must be re-run. Print the acQuire error graph, on it write the check sample number, the current date, the reason the batch was not imported and the word "failed" and file in the QA/QC binder. If a blank fails, the cleanliness of the system needs to be checked and the rejects must be used. If the standard fails, the accuracy of the analysis must be verified and the pulp is to be used. Send a note to the lab including the batch file, the standard numbers, where they are to be inserted, the reason for the failure and whether the rejects or pulps are to be used. In the file folder where the batch files are kept, rename the batch file to include the suffix "_failed". This will enable the geologist importing the batch files to recognize the rerun batch when it is resent.

4) When a batch fails due to two or more standard/blank failures import the file and reject the batch, the batch must be re-run. Print the acQuire error graph, on it write the check sample number, the current date, the reason the batch was not imported and the word “failed” and file in the QA/QC binder. Send a note to the lab including the batch file, the standard number, where they are to be inserted, the reason for the failure and whether the rejects or pulps are to be used. In the file folder where the batch files are kept, rename the batch file to include the suffix “_failed”. This will enable the geologist importing the batch files to recognize the rerun batch when it is resent.

5) When checking failures of a batch, be aware that the error may be caused by the lab reversing the tray and consequently the resultant assays will be in reverse order to the sample numbers. The current QA/QC should catch this.

6) If at any time a batch is reviewed and the assays do not reflect the current geological interpretation, review the sampling process. If the sample numbers in acQuire and the samples with the core match, the batch may be rerun to confirm the analysis. If not, there may be a problem with the sampling process.

7) Current standards are highlighted in yellow in table 11.1, other have been used in the past, before 2013. Standards are chosen to represent a range of results. Low grade, medium-low, medium, average and high grade. As there are four standards per one hundred, the use of the standards is rotated throughout the year.

Table 11-1: Standards used by Goldcorp

Standard ID	Standard value	Standard Deviation	Standard Range	Acceptable Minimum	Acceptable Maximun
LG3N	0.619	0.038	0.076	0.543	0.695
MLGN	2.5	0.095	0.19	2.31	2.69
3NMG	5.4	0.135	0.27	5.13	5.67
ELE3	8.55	0.205	0.41	8.14	8.96
H2GP	13.015	0.346	0.692	12.323	13.707
P5	0.525	0.021	0.042	0.483	**0.567**
STD	0.583	0.026	0.052	0.531	**0.635**
LG2P	0.63	0.04	0.08	0.52	**0.68**
LG	0.68	0.04	0.08	0.6	**0.76**
P7A	0.77	0.03	0.06	0.71	**0.83**
1P5A	1.37	0.06	0.12	1.25	1.49
1P5	1.58	0.08	0.16	1.42	1.74
2B	2.03	0.12	0.24	1.79	2.27
STD1	2.604	0.042	0.084	2.52	2.688
MLGP	2.949	0.124	0.248	2.701	3.197
5C	4.74	0.14	0.28	4.46	5.02
5B	4.83	0.19	0.38	4.45	5.21
UGCO	4.92	0.195	0.39	4.53	5.31
2MGP_old	5.049	0.137	0.274	4.775	5.323
5D	5.06	0.25	0.5	4.56	5.56
2MGP	5.21	0.026	0.412	4.79	5.62
14	7.47	0.16	0.31	7.16	7.78
MG	7.86	0.22	0.44	7.42	8.3
STD2	8.367	0.217	0.434	7.933	8.801
STD3	8.543	0.175	0.35	8.193	8.893
ELEP	8.585	0.263	0.526	8.059	9.111
10B	8.64	0.49	0.98	7.66	9.62
10A	9.78	0.265	0.53	9.25	10.31
HG	12.54	0.335	0.67	11.87	13.21
H3NG	13.51	0.34	0.68	12.83	14.19
15A	14.83	0.61	1.22	13.61	16.05
TH2P	19.2	0.365	0.73	18.47	19.93

11.7. Databases

The main Project database is acQuire®. Geological, assay, down-hole survey and drill collar data are uploaded in electronic format to the database.

Upon completion of drill hole logging, geologists are responsible for verifying the survey and logging data and have to sign off that all data have been entered in acQuire. Data entry is double-checked by the database manager who then locks the drill hole once assay data have been imported. The drill hole is then flagged as “finalized” in the database.

Once the drill hole is “finalized”, no changes can be made to the drill hole data, unless the database manager changes the status of the drill hole. Any changes to finalized drill holes are documented and kept on record. Only the database manager can perform this step.

11.8. Security

11.8.1. Sample Security

From the moment the core boxes are delivered to the core logging facility by the drilling contractor and up to their delivery to the laboratory, the samples remain in the custody of personnel under the direct supervision of either Virginia Gold Mines Inc. (2004 to 2006) or Goldcorp (2007 to present) personnel.

Sample shipping procedures changed slightly in January 2007, when a decision was made to increase the batch size from 24 to 50 samples.

The individual plastic sample bags are sealed at the sampling facility with a stapler. The samples are bagged in sequence, in groups of five, and inserted into rice bags. A batch is made up of 50 samples, or ten rice bags. A group of six batches is assembled on a pallet for shipment for a total of 300 samples.

A sample shipping form, with a unique identification number, detailing the contents of each batch is filled out by the core sampler. It is verified by the senior technician and entered in the acQuire® logging database. The pallet is then wrapped with plastic and identified with the shipment number.

Once or twice a week, the samples are transported directly to the laboratory in Val d’Or in a locked truck with drivers employed by Goldcorp. The sample shipment form follows the shipment at all times and the transportation waybill is signed by the laboratory supervisor. A copy of the waybill is returned to the site and filed.

11.8.2. Sample Storage

Rejects and pulps from assay sample preparation are archived in a well organized, secured facility in Rouyn-Noranda that is supervised by Goldcorp personnel.

Drill core is stored on site in core racks organized by drill hole number.

11.9. Comment on Section 11

The QPs have made the following observations:

- Sample preparation for samples that support Mineral Resource estimation has followed a similar procedure for all Virginia Gold Mines Inc. and Goldcorp drill programs. The preparation procedure is in line with industry-standard methods for sediment-hosted stockwork-disseminated gold deposits in an orogenic setting.
- Exploration and infill core samples were analysed by independent laboratories using industry-standard methods for gold analysis.
- Typically, drill programs include the insertion of blank, duplicate and SRM samples.
- QA/QC submission rates meet industry-accepted standards of insertion rates. The QA/QC program results do not indicate any problems with the analytical programs, therefore the gold analyses from the core drilling are suitable for inclusion in Mineral Resource and Mineral Reserve estimation.
- Data that were collected were subject to validation, using in-built program triggers that automatically checked data on upload to the database. Verification is performed on all digitally-collected data on upload to the main database, and includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards. Independent data audits have been conducted, and indicate that the sample collection and database entry procedures are acceptable.
- Sample security has relied upon the fact that the samples were always attended or locked in appropriate storage facilities. Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory.
- Current sample storage procedures and storage areas are consistent with industry standards.
- The specific gravity database is currently sufficient to provide a reliable assessment of the variability of the specific gravity across the gold deposit and across the various rock types.

The QPs are therefore of the opinion that the quality of the gold analytical data are sufficiently reliable to support Mineral Resource and Mineral Reserve estimation and that sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards.

12. DATA VERIFICATION

12.1. Smee (2007)

Smee and Associates Consulting Ltd. was retained by Goldcorp in February, 2007 to review core handling, data collection, QC protocols and database design.

Smee and Associates concluded:

"The sampling and quality control program appears to be running smoothly and is in compliance with mineral best practices. Only minor adjustments are contemplated"

Adjustments, including in-house standards and the ability to compare failed and accepted standards, have been implemented.

12.2. SRK Consulting (2007)

During preparation of a technical report on the Project in 2007, SRK used Gemcom software to review the Project database, primarily for items such as missing or overlapping intervals, and intervals that were longer than the drill hole depth.

SRK personnel also interviewed project personnel on all aspects of the field program, and visited several outcrop exposures to ascertain the geological setting of the project area and to witness the location of exploration work.

SRK also reviewed drill core from several boreholes intersecting gold mineralization in all areas of the gold deposits. The purpose of this review was to ascertain the geological and structural controls on the distribution of the gold mineralization and to verify the geological descriptions.

SRK concluded:

"In the opinion of SRK, Goldcorp used industry best practices to explore for gold on the Éléonore project. The exploration data was collected with care and is appropriately managed to ensure the safeguard of exploration information about the Éléonore gold deposits. The resulting exploration data is generally reliable for resource estimation."

12.3. G.N. Lustig Consulting Ltd (2007)

All borehole sample and assay data collected from the first 2004 borehole up to December 2006 was verified by G.N. Lustig Consulting Ltd (GNL). The verification performed by GNL was done by re-creating the database from the

original documents. Sample numbers and intervals were entered directly from the sample tags in a

Microsoft Excel spreadsheet and merged with assay results imported from the original digital assay certificates obtained directly from the laboratory.

The resulting database was cross-checked with the acQuire database. A list of errors, such as missing samples, overlaps, gaps, or wrong assay results, was provided to Goldcorp and the errors were corrected.

12.4. G.N. Lustig Consulting Ltd (2008)

In July 2007, a total of 3,285 drill core pulp samples representing approximately 20% of the year to date (2007 program) were sent to SGS, who acted as an external check for the primary analyses performed by ALS Chemex. GNL reviewed the results and concluded:

"The external duplicate analyses indicate no significant overall relative bias between SGS and ALS Chemex gold analyses. The results of the analyses of standard reference material along with the duplicates indicates that the SGS analyses are accurate and free from any significant bias, with the exception of fire assays between 3 and 10 g/t where gravimetric gold determinations would usually have been used. Overall the analyses where both labs used AAS resulted in SGS being 2.28 % higher than ALS. Where ALS used gravimetric and SGS used AAS, the ALS analyses were 4.9% higher and where both labs used a gravimetric finish SGS was higher by 0.472%. The results have confirmed that the ALS results have no relative bias where the analytical methods are the same. The bias in the 3-10 g/t range appear to be a low bias on the part of SGS due to the analytical method used, confirmed by the routine analyses of a standard in that grade range that was biased low by 3.7%."

12.5. Goldcorp

Upon the acquisition of the Éléonore Project by Goldcorp, drilling information was reviewed in-house.

A new data-management system (acQuire) was installed to help standardize and verify data collected in the field. A set of procedures were established for finalizing geological logs.

12.6. Comment on Section 12

The QPs consider that a reasonable level of verification has been completed by external consultants and dedicated database management staff, and that no material issues would have been left unidentified from the verification programs undertaken.

The QPs, who rely upon this work, have reviewed the appropriate reports, and are of the opinion that the data verification programs undertaken on the data collected from the Project, adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation:

- No material sample biases were identified from the QA/QC programs
- Sample data collected adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits
- Drill data are typically verified prior to Mineral Resource and Mineral Reserve estimation, by running a software program check.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

The metallurgical test program was overseen by Goldcorp's metallurgical team. Extensive metallurgical studies were carried out on samples taken from the various ore zones of the Roberto deposit. Gold recovery studies were completed by SGS Lakefield Research. FLSmith and Golder Associates were involved in the sedimentation and filtration tests as well as in the paste backfill tests. A complete list of the laboratories used for metallurgical assessment purposes was provided in Table 11.2.

13.1. Metallurgical Testwork

13.1.1. Metallurgical Testwork 2006

A total of 190 samples were submitted for testwork. The samples were taken from the assay laboratory reject samples, representing approximately 181 m of diamond drill core from the Roberto and Roberto East zones. The samples came from 44 diamond drill holes spread across the strike of the orebody over a distance of about 375 m. The deepest sample was sourced at a depth of 1,009 vertical metres below surface.

In the absence of a geological model to explain the likely lithological controls that could impact the metallurgical response, the samples for the metallurgical test program for the upper, middle and the lower parts of the deposit were prepared according to the ore sequence expected to be delivered to the processing plant. Thus, the samples were composited over three vertical intervals.

Table 13-1 outlines the composition of the samples used for the various tests. In all, 256 kg of material were subjected to three different tests in that part of the testing program.

Representative head samples were removed from each of the 27 composites during the sample preparation phase. These samples were submitted to a variety of chemical analyses.

Composite samples from the Roberto and the Roberto East lenses showed no appreciable differences in their Au:S (gold to sulphur) and Au:As (gold to arsenic) ratios. The total sulphur content was 50% higher in the Roberto lens than in the Roberto East lens. The proportion of non-sulphide sulphur, presumably as sulphate, was found to be close to 32% in both lenses.

Table 13-1: Sample requirements for the proposed test program.

Zone	Roberto			Roberto East			Total Mass kg
	16 kg Bulk Composite for Scouter Flotation Test (60% Roberto: 40% Roberto East)						16
UPPER	28 kg for Physical Testing			28 kg for Physical Testing			56
	4 kg **Low** Grade for Met Testing	4 kg **Med.** Grade for Met Testing	4 kg **High** Grade for Met Testing	4 kg **Low** Grade for Met Testing	4 kg **Med.** Grade for Met Testing	4 kg **High** Grade for Met Testing	24
MIDDLE	28 kg for Physical Testing			28 kg for Physical Testing			56
	4 kg **Low** Grade for Met Testing	4 kg **Med.** Grade for Met Testing	4 kg **High** Grade for Met Testing	4 kg **Low** Grade for Met Testing	4 kg **Med.** Grade for Met Testing	4 kg **High** Grade for Met Testing	24
LOWER	28 kg for Physical Testing			28 kg for Physical Testing			56
	4 kg **Low** Grade for Met Testing	4 kg **Med.** Grade for Met Testing	4 kg **High** Grade for Met Testing	4 kg **Low** Grade for Met Testing	4 kg **Med.** Grade for Met Testing	4 kg **High** Grade for Met Testing	24
Total	**128 kg**			**128 kg**			**256**

The nature of the elevated non-sulphide sulphur content in samples, which otherwise show little visible evidence of oxidation, requires further study through additional analysis, using fresh samples.

Other than elevated arsenic levels, inductively-coupled plasma and heavy metal analyses showed no evidence of any potentially significant contaminants such as mercury, cadmium, lead or antimony.

The acid neutralizing potential (NP) of the rock was found to be low, despite a relatively high paste pH, typically around 9.7. The zone-weighted Net NP is about -25 kg $CaCO_3$/kt, so the potential for acid generation from the ore and tailings is considered to be high.

Elevation composite samples from the Roberto and Roberto East ore zones

were submitted to semi qualitative petrographic and X-ray diffraction (XRD) examination.

The results were:

Pyrite, pyrrhotite and arsenopyrite are present in approximately equal amounts in both ore zones, and this ratio does not appear to change with depth;

- The chemical analysis of the Roberto East zone showed slightly lower arsenopyrite contents;
- There is no obvious difference in the sulphide mineral liberation characteristics between the Roberto and Roberto East ore zones; and there seems to be no appreciable change with depth;

Fine gold inclusions, less than 5 µm in size, were observed within the sulphides.

13.1.2. Metallurgical Testwork 2007

Samples representing approximately 20 m of PQ size drill core from the Roberto and Roberto East zones were submitted to physical testing (PQ core size is 85 mm in diameter). The samples were taken at depths ranging from surface to 200 m below surface.

Samples representing approximately 400 kg of NQ size drill core spread across the orebody strike length were used to complete an optimization study and to define the Bond Ball mill grindability index.

These samples were composited over three vertical intervals and, within these intervals, over seven grade ranges for variability testing. The optimization study and Bond Ball mill grindability index were undertaken on composites from the Roberto and the Roberto East zones, and on composites made of a combination of ores from these two zones.

The grade composites and elevations for the Roberto and the Roberto East zones were selected for variability testwork as shown in Table 13-1.

13.1.3. Metallurgical Testwork 2008

Two types of samples were used in the metallurgical analysis: PQ core samples and samples taken from pails and boxes. The bulk PQ core samples and the samples from pails represented the Roberto, Roberto East and Zone du Lac zones.

The PQ core samples from Roberto and Roberto East were used in the pilot plant testwork to produce enough concentrate for the cyanidation process tests. The

Zone du Lac PQ core samples were not included in the bulk pilot-plant composite but portions of the cores were used in a series of comminution tests.

The bulk pilot plant composite included 90 boxes of Roberto PQ core samples and 15 boxes of Roberto East PQ core samples. These were combined, blended and crushed to -1/4 inch (-6.35 mm). In total, approximately 1,400 kg of samples were available for the pilot-plant run.

The Zone du Lac PQ core samples were used in several standard comminution tests. Samples from 18 pails and 3 boxes were used for the metallurgical test program. The comminution and metallurgical tests samples taken from the various zones and elevations of the orebody are shown.

Grades for the head samples used for the gold and silver assays, for both the 2007 and the 2008 testwork, are shown in Table 13-8. The Roberto 2008 gravity- recoverable gold (GRG) and the Roberto East 2008 GRG gold head grades were significantly higher than those determined in the 2007 test program.

Additional samples from the Roberto and Roberto East GRG composites were also submitted for S-2 analysis (proportion of sulphur available as sulphides) in order to confirm their sulphide contents. A sub-sample of the Zone du Lac composite was submitted for As, S, S-2 and semi-quantitative ICP scan analysis.

13.2. Comminution Tests

Crushing and grinding testwork was completed on three different batches by SGS Lakefield Research Limited (SGS). The samples came from the Roberto, Roberto East and Zone du Lac deposits. All samples were submitted for standard Bond tests, crushing work index (CWi) tests, abrasion index (Ai) tests and ball mill work index (BWi) tests.

This work was commissioned to evaluate the variations that could be expected in the crushing and grinding characteristics of the ores and to confirm the comminution configuration and power requirements for process plant design.

Bond ball mill work index values increase with depth. Zone du Lac ore yielded a BWi of 16.7 for samples taken between 0 and 250 m below surface, and 20.6 for samples taken at depths greater than 750 m. The BWi of the Roberto ore ranged between 18.2 for samples taken between 0 and 250 m below surface, and 19.5 for samples taken at depth greater than 750 m.

The 2008 data indicated significantly harder ores (compared with the database) when using the non-standard closing screen size of 200 mesh (75 um). There

is a clear trend indicating that the ore becomes harder with depth. The Roberto ore BWi ranged between 18.2 for samples taken between 0 and 250 m below surface and 19.5 for samples taken at depths greater than 750 m below surface. The Zone du Lac BWi ranged between 16.7 for samples taken between 0 and 250 m below surface and 20.6 for samples taken at depths greater than 750 m.

In terms of Crushing Work Index, a CWi of 14.1 obtained from the Zone du Lac ore appears to be significantly harder than the SGS crushing work index database average of 10.3, the Roberto ore at 8.9 and the Roberto East ore at 9.5. When compared to other values in the SGS database, the Roberto and Roberto East ores fall at the 54th and 57th percentiles respectively, whereas the Zone du Lac ore falls at the 79th percentile.

The Zone du Lac Abrasion Index (Ai) was 0.4223, slightly less abrasive than the Roberto and the Roberto East Ai of 0.4659 and 0.4668 respectively. The Éléonore ores exhibit similar degrees of abrasiveness and fall between the 78th and the 80th percentiles compared with other values in the SGS abrasive index database.

13.3. Gravity

Knelson/Laplante gravity-recoverable gold tests were conducted on the Roberto, Roberto East and Zone du Lac composite samples during 2006 and 2008, as shown in Table 13-10. The optimum grind size identified (P80 = ~65 µm) was selected as the final stage grind target for the subsequent tests.

The gravity-recoverable gold (GRG) tests consist of passing 20 kg through the Knelson concentrator, collecting a concentrate and tailing sample. The Knelson concentrate and tailing samples were filtered and submitted for size-fraction analysis for Au. The remainder of the tail was decanted to 65% solids, split into two equal parts and each half ground in a 10 kg rod mill for 12.5 minutes. For stage 1, the gravity separation, sampling and size-fractional assaying procedure was repeated. For stage 2, the Knelson tail was split into two equal parts and each part ground at 65% solids in the 10 kg rod mill for 62.5 minutes. For stage 3, gravity separation was performed as per the above stages repeating the gravity separation, sampling, and size-fractional assaying procedure.

From tests performed in 2006 by SGS Lakefield, the gravity-recoverable gold response indicated that a large proportion of the gold was liberated/recovered at a primary grind size finer than ~570 µm and coarser than ~123 µm. Given the apparent degree of gravity-recoverable gold, the conclusion was that it would be advisable to include gravity separation in the proposed Éléonore process

flowsheet.

In 2006, Goldcorp decided to process samples through a gravity recovery circuit prior to completing downstream cyanidation and/or flotation testwork done by SGS Lakefield. A Knelson MD-3 concentrator was used as the primary gravity-gold recovery unit in these tests. The Knelson concentrate was recovered and further upgraded using a Mozley C-800 laboratory mineral separator. A final concentrate mass recovery ranging between 0.1 and 0.3% was targeted. Gravity concentrates were assayed to extinction for gold by standard fire assay methods. The Knelson and Mozley tailings were recombined, blended and divided into representative charges for downstream testwork.

The presence of gravity-recoverable gold in the ore has several key implications for the process design. Firstly, the cost of recovering gold via gravity is relatively low provided that a significant volume of gold can be recovered. Secondly, coarse free gold can be difficult to recover in flotation and may be too coarse to effectively leach by the cyanidation process. Losses in traditional cyanidation or flotation circuits can be attributed to coarse gold. To prevent or to minimize these losses, gravity processes can be used.

The ores of the Éléonore Project were shown to contain various amounts of gravity- recoverable gold, and testwork has indicated that a gravity recovery process will be required to minimize losses in either a flotation process or a cyanidation process

13.4. Flotation

13.4.1. Collector Suite Optimization

Conventional bench-scale flotation tests were completed using ground ore that had been processed for gravity gold recovery. The batch flotation tests were run in 2 kg Denver flotation machines. Three different reagents, combined in various proportions, were used in the flotation tests done by SGS Lakefield in 2008.

The use of PAX alone (potassium amyl zanthate) and PAX + Cytec R208 (dithiophosphate) essentially yielded identical results. PAX + 3418A (phosphene-based collector) yielded a slightly higher mass pull and marginally better gold recovery. However, given the marginally better results and the cost of the 3418A compared to that of the R208 (approximately three times higher), and given that the flotation tailings were planned to be cyanide-leached, it was decided to continue with the PAX + R208 for the remaining tests in this program.

13.4.2. Grind Size Optimization

Having established the reagent suite that would be applied in all subsequent

tests in 2008, 2009 and 2010 done by SGS Lakefield, grind optimization tests were completed on both the Roberto and the Roberto East Zone composites.

The Orebody Composite tests performed in 2006 were completed on 10-kg (dry equivalent) charges of gravity tailings.

Data presented in Figure 13-4 indicate a marked improvement in gold recovery at a grind of P80 = 64 µm when compared with results of tests completed at coarser grinds. Finer grinding appears to yield a significant increase in rougher mass recovery: 8.6% at 48 µm compared with 5.2% at 64 µm, with no corresponding improvement in gold recovery. A flotation grind target of P80 = 65 µm was selected for the grade variability recovery tests undertaken in this program.

13.4.3. Flotation Testwork Results

At 77%, the flotation gold recovery achieved from the Zone du Lac (2008) gravity tailings compares very well with the Roberto (2007) and Roberto East (2007) test results of 82% and 78% respectively. Overall gravity + rougher flotation gold recovery from the Zone du Lac ore, at approximately 80%, was only slightly lower than the Roberto (2007) at 82.3% and the Roberto East (2007) at 83.2%.

Sulphide sulphur recoveries were similar in all three ores ranging between about 93% from the Roberto East ore to about 96% from both Roberto and Zone du Lac ores.

13.5. Cyanidation

13.5.1. Cyanidation Testwork on Flotation Concentrates

A series of tests was completed, from the metallurgical testwork done in 2007, 2008, 2009 and 2010 by SGS Lakefield to assess the potential of using intensive cyanide leach processing of flotation concentrates generated from the Éléonore ore.

The mineralogical characterization of the concentrate consisted of 65% gangue minerals, 23% pyrrhotite (hexagonal), 9.6% arsenopyrite, 2.2% pyrite and 0.08% chalcopyrite with trace amounts of galena (0.05%) and sphalerite (0.03%).

The rougher flotation concentrate generated from the flotation tests was used. The Gekko Systems test protocol was applied in all cases. Test conditions applied in all cases were as follows:

- Pulp Density = 10% solids (w/w)

- Cyanide Concentration = 10 g/L NaCN (maintained)
- pH = ~11 (maintained with lime if required)
- Dissolved Oxygen = ~20 mg/L (maintained with hydrogen peroxide)
- Lead Salt Addition = 2,000 g/t $Pb(NO_3)_2$
- Retention Time = 48 hours.

Regrinding the flotation concentrate prior to cyanidation resulted in a significant increase in gold extraction. Gold extraction was about 81% after 48 hours without regrinding (P80 = 40 um) and about 96% after 48 hours after regrinding (P80 = 11 um). Cyanide consumption was generally high. Pre-aerating in Test CN-40 resulted in a drastic decrease in cyanide consumption: from 48 kg NaCN/t of leach feed at 11 um without pre-aeration in Test CN-26 to about 18 kg NaCN/t of leach feed at 11 um with pre-aeration in Test CN-40.

13.5.2. Cyanidation Testwork on Gravity and Flotation Tailings

A series of cyanide leach tests was completed, from the metallurgical testwork done in 2007–2008 on the gravity tailings generated from tests completed on the Roberto Composite (2007) and the Roberto East Composite (2007) respectively. A grind size range between about 190 µm and about 65 µm was assessed.

Standard test conditions were applied as follows:

- Pulp Density = 40% solids (w/w)
- Cyanide Concentration = 0.5 g/L NaCN
- pH = ~10.5–11 (maintained with lime)
- Pulp Preparation = 1–3 hours (until DO levels was 5 mg/L or higher)
- Lead Salt Addition = 200 g/t as $Pb(NO_3)_2$
- Retention Time = 48 hours.

There was a clear correlation between cyanidation feed grind size and gold extraction for both testwork on gravity and flotation tailings. There was a marked improvement in gold recovery with finer grind. The cyanide consumption of the flotation tailings cyanidation was negligible compared to the gravity tailing cyanidation testwork due to the gold/sulphides removal by flotation.

Table 13-2: Metallurgical results from the process

At the outset of the program, it was decided to evaluate three basic flowsheet configurations:

- Option 1: Gravity separation + Cyanidation of the gravity tailings;
- Option 2: Gravity separation + Flotation of the gravity tailings + Cyanidation of the flotation concentrate;
- Option 3: Gravity separation + Flotation of the gravity tailings + Cyanidation of the flotation tailings + Regrind and intensive cyanidation of the flotation concentrate.

Based on the data presented in Table 13-2, the metallurgical advantage of Option 3 is clear. Applying simple cyanide leach conditions, the flotation tailing yielded approximately 65–75% of its contained gold content. Overall, the flotation tailing cyanidation circuit, at the selected design grind of about 65–70 µm, contributed about 10–15% of the overall circuit recovery and required a minimal amount of additional reagents.

Gold recoveries from two different flowsheet configurations, labelled FS-1 and FS-2 in Figure 13-6, were evaluated and compared.

The Option 3 flowsheet is referred as flowsheet 2 (FS-2) and included essentially the same gold recovery unit processes as FS-1 but in a different order. In FS 1, cyanidation followed gravity separation, and flotation followed gravity tailing cyanidation. This scenario included cyanide detoxification between cyanidation and flotation and had the flotation concentrate ultra-fine regrind and re-leaching operations placed at the end of the circuit.

The gold recovery obtained with both flowsheets was similar, ranging between 92.4% and 93.1% recovery. At this level, the only valid distinction that can be made on recovery concerns the grade of the tails. The combined tail of FS-1 was 1.12 g/t Au, as compared with 1.01 g/t Au for FS-2.

Goldcorp considered that FS-1 may be somewhat more complex in terms of operation than FS-2, because the cyanide detoxification step is placed within the gold recovery circuit. It may also run the risk of sulphide depression (by surface oxidation) prior to flotation.

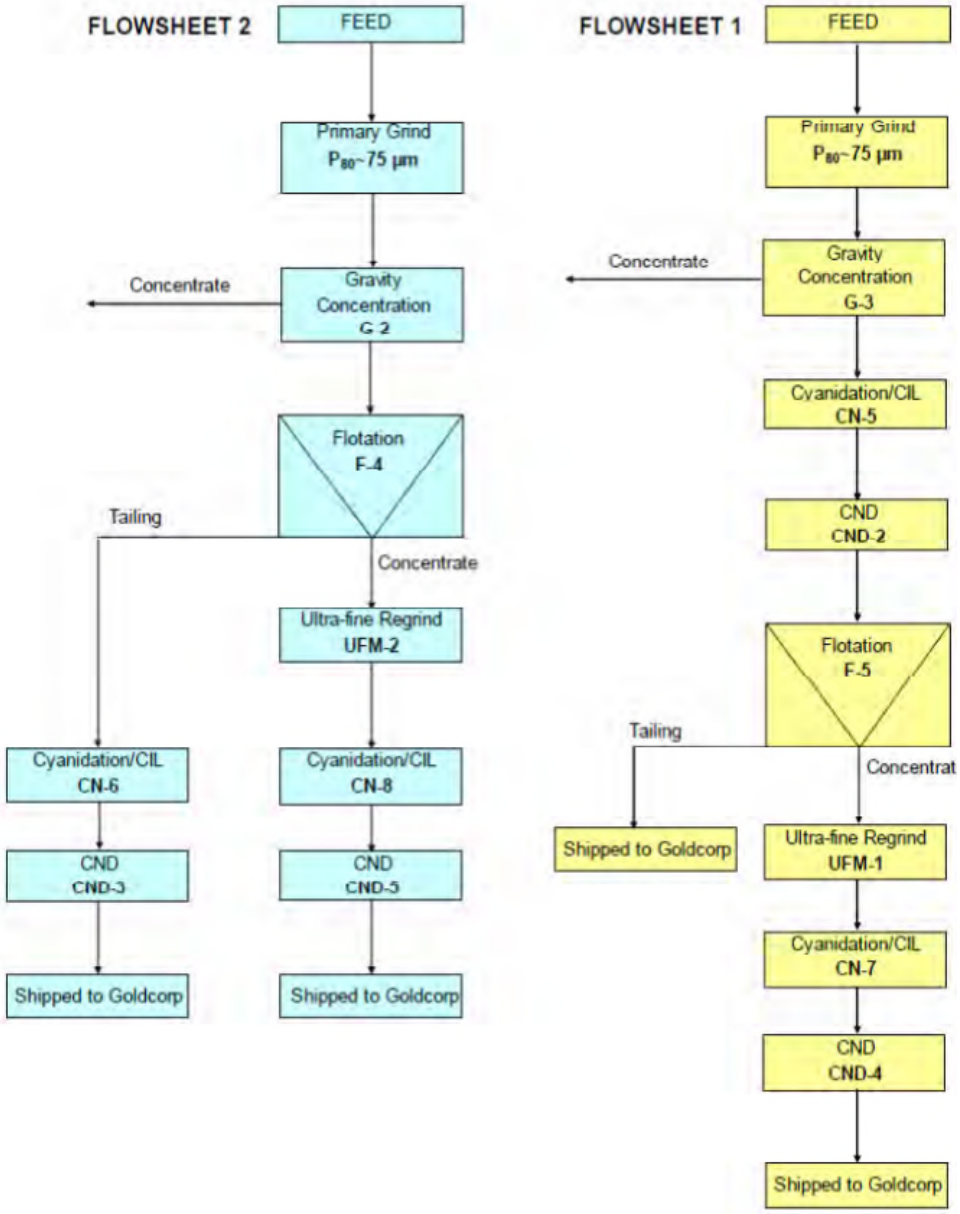


Figure 13-1: Comparative process flowsheets

While there is some evidence suggesting that sulphide recovery kinetics may in fact be somewhat delayed using FS-1 as compared with FS-2, due to oxidation occurring in the cyanide leach and detoxification circuits; in FS-1, the effect appears to have been sufficiently overcome by the introduction of sulphide activator (copper sulphate and/or sodium sulphide) in the subsequent flotation. Consequently, FS-2 was selected as the proposed flowsheet option for the Éléonore Project.

13.6. Cyanide Detoxification

The tailings pulp from each of the cyanidation tests (CN-5 and CN-7) from FS-1 and (CN-6 and CN-8) from FS-2 were individually subjected to a preliminary cyanide detoxification evaluation. The purpose of this set of tests was threefold:

- To determine the amenability of the cyanide tailing pulps to cyanide detoxification using the SO_2/air process;
- To determine approximate reagent requirements;
- To generate circuit end products for environmental testing at an external laboratory.

In each case, the feed pulp was placed in a vessel of suitable size. The required amount of copper sulphate was determined based on chemical analysis of the cyanide tailing solution(s). All pulps were treated in batch mode with $Na_2S_2O_5$ and air to reduce the concentration of weakly acid-dissociable cyanide (CN_{WAD}) in solution to approximately 1 mg/L. The oxidation reduction potential (ORP) of the pulp was monitored with a Pt/Ag/AgCl combination electrode, while the residual CN_{WAD} concentration in the solution phase was analysed using the picric acid method.

At the end of each cyanide detoxification test, a solution sample was taken for analysis of cyanide (via the cyanide-nitroprusside test (CNT) and CN_{WAD}) and the critical metals of interest (Cu and Fe).

The QPs note that it is critical that the residual cyanide levels in the process tailings stream be reduced below the threshold prescribed under regulations prior to placing the tailings into a tailings storage facility. SO_2/Air and copper sulphate will be used to break down the cyanide complexes as they are discharged with the slurry to the filtration plant. It is expected that water will be reclaimed from the filtration plant and the tailings storage facility for re-use in the process facilities.

13.7. Sedimentation and Filtration Tests on Tailings

Samples of flotation tails were sent to FLSmidth (FLS) in 2010 to carry out dewatering tests. FLS performed settling tests as well as filtration tests to determine the dewatering characteristics of the flotation tails and the size of the equipment that would be required for the various dewatering options.

13.7.1. Thickening Tests

Flocculant screening showed that an anionic polyacrylamide flocculant with a high molecular weight and low–medium-charge density produced the best settling rates and overflow clarity. HyChem AF-306 was used for this test campaign. The test results showed that to obtain the best flocculation, the thickener feed solids concentration should be 12.5 wt% solids. The necessary feed dilution in the full-scale thickener can be accomplished internally using an FLS E-Duc® feed dilution system. A summary of the thickener testing results are shown in Table 13-22.

FLS recommended using a minimum unit area of 0.03 m^2/Mt/d, and an underflow density of 70–72 wt% solids. The thickener underflow will be used as feed to the filter. The minimum diameter for a thickener able to treat the output of a 3,500 t/d mill was determined to be 11.6 m. This diameter increases to 16.4 m for an expandable mill throughput of 7,000 t/d.

13.7.2. Filtration Tests

Two filtration options were tested to define equipment requirements:

- Filtration Option 1: Vacuum filtration (horizontal belt filter or low submergence drum filter)
- FiltrationOption 2: Pressure filtration for recessed chamber filter press, PneumaPress® filter.

Vacuum filtration testing indicated that the sample can be filtered as produced with a FLS Horizontal Belt Filter or low submergence drum filter. The results showed that vacuum filtration produces a filter cake with 15 wt% residual moisture at a rate of 1,134 kg/m^2/hr. For a 3,500 t/d mill, a total of 130 m^2 of filtration area would be required. Disk filters cannot be used due to their fast cake formation and geometry limitations.

Recessed chamber pressure filtration testing indicated that the sample can be filtered as produced using a FLS automatic recessed chamber filter press with 15 wt% residual cake moisture with a dry cake bulk density of 1,478 kg/m^3. Two pressure filtration options were proposed for dewatering the gold tailings stream:

- Pressure Filtration Option 1: 2020 x 2020 filter press.
- Pressure Filtration Option 2: 1500 x 1500 filter presses.

The advantage of Option 2 was that one filter press is operational during maintenance.

- PneumaPress® pressure filtration testing indicated that the gold tailings stream can be filtered as produced using a FLS PneumaPress® filter with 15 wt% residual cake moisture at a filtration rate of 2,369 kg/m^2/hr. For a 3,500 t/d mill throughput, a total of 67 m^2 of filtration area would be required. This does not include any safety factor.

13.8. Paste Backfill

In 2010, Golder PasteTec performed a series of paste backfill tests on two new samples provided by Opinaca. These samples were characterized for size distribution and mineralogy.

For the other tests, a blend was prepared with 10% concentrate and 90% flotation tails. The blend sample was submitted to rheological tests that indicated a slump of 175 mm at 76.5% solids which is typical of paste backfill. Other rheological tests were done including a water bleed, yield stress over time tests and a plug yield stress analysis.

Dewatering tests were done on the blended material. It was determined that this sample could be dewatered to 67% solids by simple thickening to 76% solids with centrifuge dewatering and to 81% solids using vacuum filtration.

In settling tests, an anionic flocculant with very high molecular weight was selected based on preliminary tests. Further detailed testing showed that the optimal feed density was 25% solids or less and that 15–20 g/t of flocculant should be used for thickening. Using 15 g/t of flocculant, an underflow density of 67% solids was reached after a one-hour residence time.

The underflows from the settling tests were centrifuged until no further densification was observed and 75–76% solids was reached. Leaf filtration tests were done using the underflow of thickener at 67% solids. The filter capacity simulations ranged between 600 kg/h/m^2 and 900 kg/h/m^2 by varying the cycle time duration. This yielded 81.2–80.6% solids.

Two series of unconfined compressive stress (UCS) testing were completed to define the composition of the backfill. The first series was used to study various parameters such as the type of binder and the impact of adding sand to the

backfill. These tests were conducted using a 5% binder and a constant 175 mm slump. The results showed that the most beneficial factor for the UCS were the use of blast furnace slag (BFS) and the addition of sand.

In the second series of tests, blends of BFS and normal Portland cement (NPC) were used as well as unblended NPC with different binder contents (2.5% and 3.5%) and unblended sand. The NPC binder gave slightly higher resistance in the early days but the binder with high BFS contents gave a much higher resistance after 28 days when compared with the other options.

The samples achieved better backfill strengths at 7 days than previous tests for the same composition: previous result of 0.3 MPa compared with current result of 0.7 MPa for the new samples. The best binder identified so far is 90/10 BFS-NPC. The increase in sulphides content in the paste backfill mix did not deteriorate its strength although the time was too short to observe the long term effects. Long term effect of sulphides should be tested prior to selecting the final paste backfill composition. The increased addition of sand improved the backfill strength. Golder PasteTec did however recommend not exceeding 50% of the blend to maintain the non-segregating properties of the paste.

13.8.1. Paste backfill constituent evaluation

In 2013, Golder Paste Tec was mandated to test the tailings from the Éléonore Project. The results of the previous testing programs were presented in 2010, in which the dewatering, rheological and unconfined compressive strength (UCS) properties of the tailings were assessed (Figure 13-2).

The 2013 testing program consisted of expanding on the UCS testwork, providing additional information with which to optimize the paste backfill mix recipes by varying the constituents (binder (90/10), flotation tails, fine sulphide concentrate and aggregate). In addition, rheological characterization and a small flow loop test were added to assess the anticipated flow properties of a paste aggregate fill (PAF) mixture.

This program demonstrated that the aggregate addition in the paste has a positive impact on the binder consumption comparing with the fine sulphide concentrate where more binder is required beyond 15% to get an UCS of 0.5 MPa after 28 days of cure. The paste backfill composition needs to be optimized during operations, but the results established an initial paste backfill composition of 17.5% aggregate, 12% fine sulphide concentrate, 3.5% binder and 67% flotation tails.

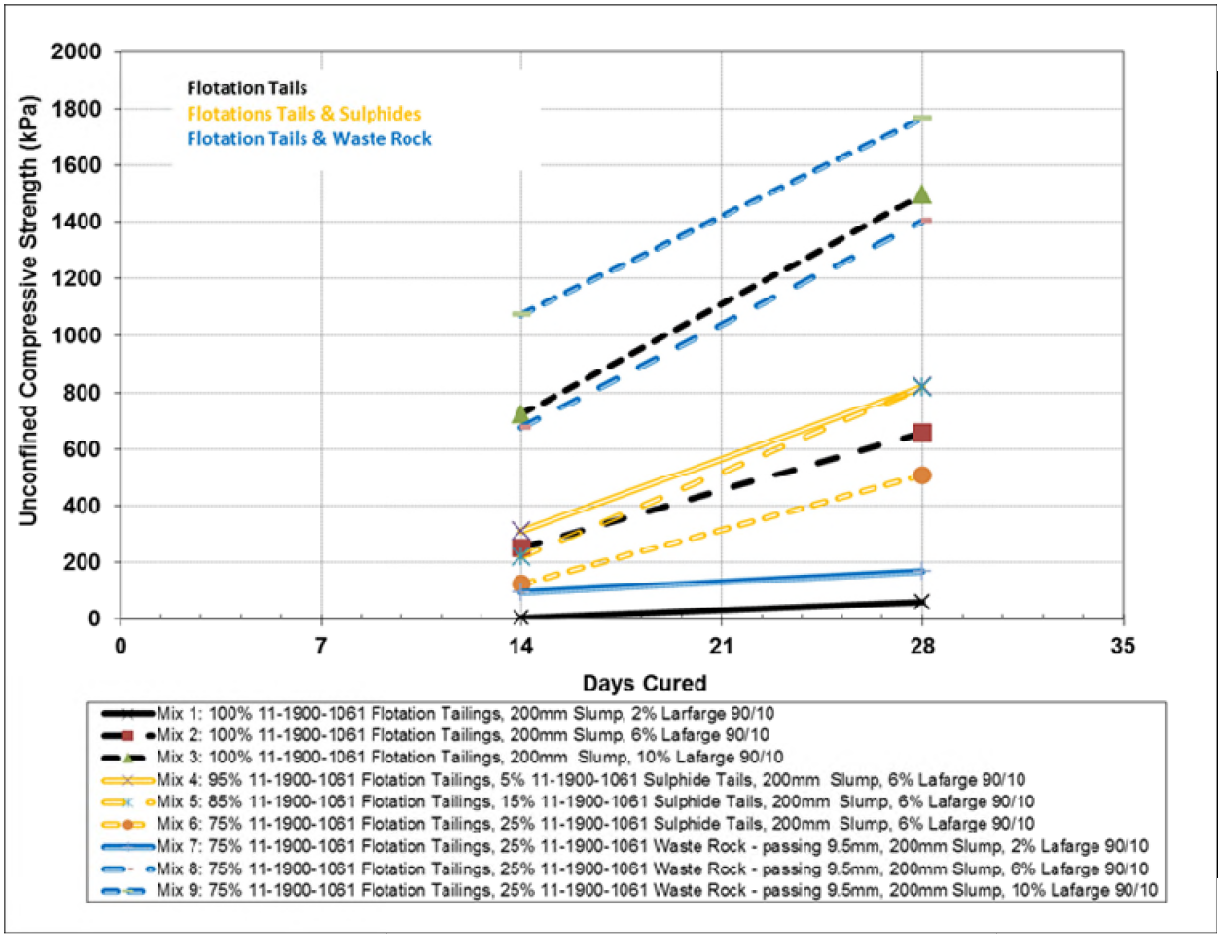


Figure 13-2: UCS versus different paste backfill compositions

13.9. Projected Gold Recovery

Based on the flowsheet selected, grade variability recovery testwork was conducted with 37 grade variability composites. The tests followed the gravity separation + gravity tailing flotation + flotation tailing cyanidation + flotation concentrate regrind and intensive cyanidation flowsheet (Figure 13-3).

The overall gold recovery from the 37 grade variability composites ranged between 90% and ~97%, averaging 92.6%. There is no obvious correlation between head grade and gold recovery (Figure 13-4).

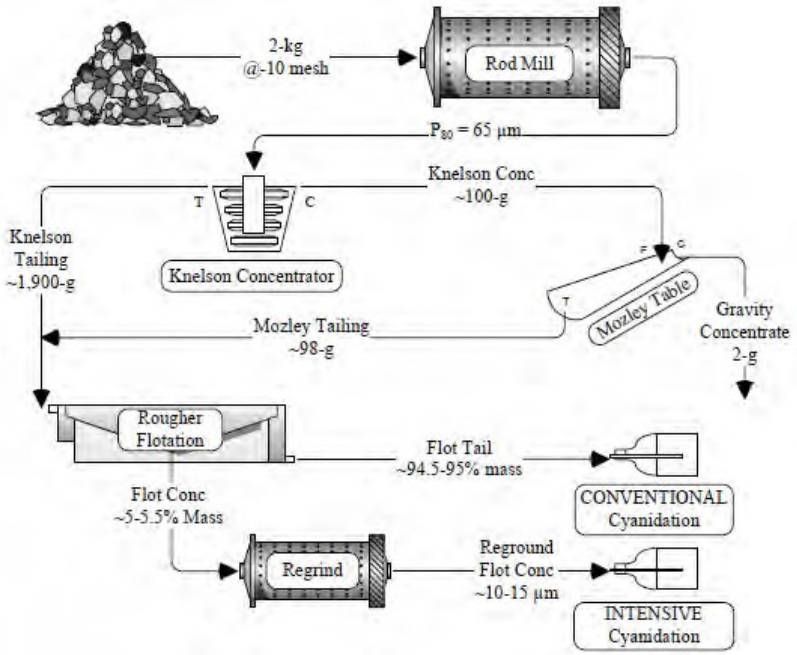


Figure 13-3: Grade variability recovery composite testwork flowsheet

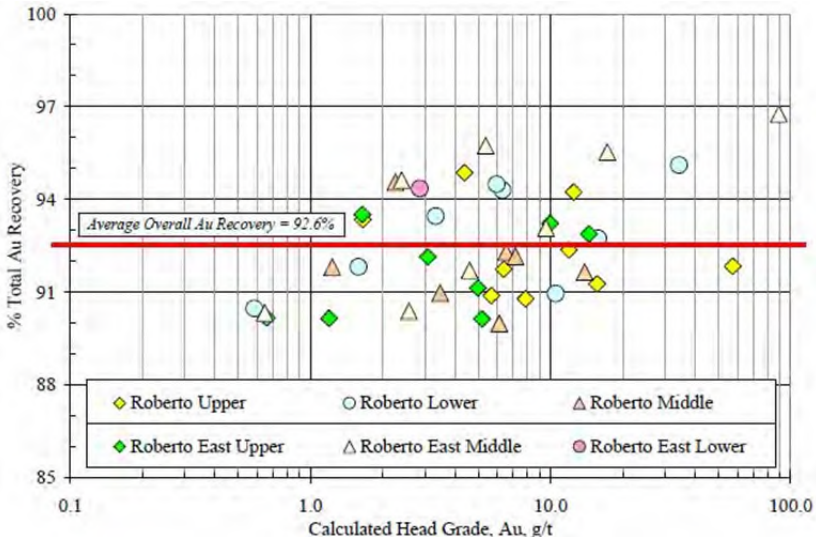


Figure 13-4: Projected gold recovery versus head grade

Overall, gravity separation contributed 20% of the total gold recovery while intensive cyanidation of the flotation concentrate yielded 61.2% and cyanidation of the flotation tailing gave an additional 11.4%.

Based on a review of all test data generated in this program, the best opportunity to increase overall gold recovery may lie in further optimization of

the intensive cyanidation and the gravity circuits.

For this reason, Goldcorp expects to obtain an overall gold recovery ranging between 93 and 94% during the operations phase.

13.10. Comment on Section 13

In the opinion of the QPs, the following interpretations and conclusions are appropriate:

Metallurgical testwork and associated analytical procedures were performed by recognized testing facilities, and the tests performed were appropriate to the type of mineralization.

Samples selected for testing were representative of the various types and styles of mineralization at Éléonore. Samples were selected from a range of depths within the deposit. Sufficient samples were taken to ensure that tests were performed on sufficient sample mass.

Testwork has established the most appropriate grind size for plant design. A grind size of 65 µm is planned. Éléonore ores are classified as moderately soft and abrasive. There was a clear correlation between cyanidation feed grind size and gold extraction from the testwork on gravity and flotation tailings. There was a marked improvement in gold recovery with finer grind.

The ores of the Éléonore Project were shown to contain various amounts of gravity-recoverable gold, and testwork has indicated that a gravity recovery process will be required to minimize losses in either a flotation process or a cyanidation process.

Assumed life-of-mine (LOM) gold recovery assumptions are based on appropriate testwork, and will average between 93 and 93.5% over the LOM.

Other than elevated arsenic levels, ICP and heavy metal analyses showed no evidence of any potentially significant contaminant elements such as mercury, cadmium, lead or antimony.

The overall gold recovery could be affected by non-respect of the sulphide concentrate regrind size at 10–15 µm or if there is less gold associated with iron sulphides than expected. The risk remains very low knowing that Goldcorp will use the proven Isamill technology to achieve the grind expected and, based on current knowledge, the gold association with iron sulphides is constant through the orebody from surface to a depth of 750 m.

14. MINERAL RESOURCE ESTIMATES

The Mineral Resource Estimate herein was prepared by Christine Beausoleil, P.Geo, Chief Geologist Interim, Éléonore Mine, using all available results. Mineral Resources are not Mineral Reserves as they have no demonstrated economic viability. The result of this study is a single Mineral Resource Estimate, comprising Inferred Resources, for all mineralized zones (see below for details). The effective date of this Mineral Resource Estimate is December 31, 2013.

14.1. Basis of Estimate

The coordinate system used for resource modelling is UTM NAD 83, Zone 18N. The 3D topography surface, dated June 2008, was used as the upper constraint of the mineral resource model. A 3D surface, representing the bottom of overburden, was also modelled and rock units located above this surface were excluded from the Mineral Resource Estimate.

The Mineral Resources are based on a total of 839 core drill holes and 143 surface channels, for a total of 147,390 assay results. The samples were collected between September 2004 and September 5, 2013. The closeout date for the database used in the estimation is September 5, 2013. Holes and channels more than 30 metres away from the Low Grade Envelopes were excluded from the Project in order to lighten the database.

Drilling has been conducted over a 1,500 m by 1,500 m area on the Éléonore deposit. The drilling pattern was designed to sample the deposit orthogonally to the interpreted strike and dip of the gold mineralization. A drill hole spacing of 25 m by 25 m was applied over the bulk of the orebody to a depth of approximately 650 m below surface. Between 650 m and 850 m below surface, borehole spacing increases to about 50 m by 50 m. Below 850 m, down to approximately 1,200 m, a core hole spacing of 100 m by 100 m is usually observed. Only a few drill holes were drilled below 1,200 m. The deeper boreholes intersected the mineralized horizons at a depth of approximately 1,400 m below surface. For definition drilling, drill hole spacing is generally 12.5 m by 12.5 m inside the existing 25 m drill spacing as permitted by the mine development schedule.

The current Vulcan (version 8.2) block model produced for this Resource Estimate is a combination of two different block models. The lower portion of the deposit (“Lower Mine”; below real elevation -650 m) is exactly as it was presented in the previous Resource Estimate. The upper portion of the deposit (“Upper Mine”; above real elevation -650 m) was reinterpreted and a new interpolation was produced. A transition area of approximately 100 m between the Upper Mine and the Lower Mine was reinterpreted in order to link both interpretations together and avoid a fringe effect. The Lower Mine Composites contained within this

transition area were used for interpolating the Upper Mine model. The resulting is a single block model combining two generations of interpolations. Only the Upper Mine parameters are discussed herein; the reader is referred to the report entitled "Éléonore Gold Project Quebec, Canada NI 43-101 Technical Report", published March 30, 2012 (available on SEDAR) for information regarding the Lower Mine parameters. It is important to note, however, that Table 14-4 presents the combined results of both the Upper and Lower Mine Resource Estimate.

The block model uses a block size of 5 m x 5 m x 5 m (Easting, Northing, Elevation) with sub-blocks of 0.5 m x 0.5 m x 0.5 m (Easting, Northing, Elevation). The block dimensions reflect the dimension of the mineralized zones and plausible mining methods. The Mineral Resource model was not rotated. The model covers an area delineated by these coordinates:

- North: 5,838,800 to 5,840,300N
- East: 426,000E to 427,330E

Elevation: 250 m to -750 m (i.e., approximately 1,000 m below surface).

An average specific gravity of 2.77 was used for all veins.

The Mineral Resource estimate was prepared in accordance with the Canadian Institute of Mining Metallurgy and Petroleum (CIM) Definition Standards (2010) and CIM Best Practice Guidelines.

14.2. Geological Models

The mineralized zones were interpreted based on alteration, mineralization, structures, and assay results. Major lithologies and alteration styles were also interpreted on section and plan views. The interpretations consist of four principal zones (5010, 5050, 6000 and 7000) and 24 secondary zones. The numbering of the zones is from the footwall to the hanging wall. Zone 5050 comprises the Roberto style of mineralization, whereas zones 5010, 6000, and 7000 comprise the Roberto East and Zone du Lac style of mineralization (Figure 14-1).

Within the Vulcan block model, there are three types of mineralized solids: high-grade (HG) solids, mineralized envelopes or low-grade (LG) solids, and mineralized envelopes for dilution (ME). Twenty-eight (28) zones were defined inside the ME. Within each LG zone, one or more internal HG veins were modelled. In some cases, an HG zone exists without an LG zone surrounding it. Inside the HG zones, nineteen (19) internal dilution volumes were modelled.

For the Lower Mine (as defined in the previous resource estimate), the interpretation of the geology and the mineralized zones was first undertaken on a

series of east–west cross-sections spaced 25 m apart and then reconciled on level plans spaced 100 m apart. A minimum width of 1.0 m and cut-offs varying from 3 g/t to 5 g/t were used. The interpretation was then digitized and 3D solids were built.

For the Upper Mine (new re-interpretation), the previous interpretation was updated using new available information. The interpretation of the geology and mineralized zones was first undertaken on a series of level plans spaced 12.5 m apart and then reconciled on cross-section views spaced 25 m apart. A minimum width of 2.5 m and cut-offs varying from 1 g/t (LG) to 3 g/t (HG) were used. The dilution envelope was created roughly 25 m around the LG zones.

For the Upper Mine, the interpretation was conducted using Vulcan software. Solids previously created in Gems for the Lower Mine were imported into Vulcan (2010) and merged with the Upper Mine solids. The 3D model shapes were appropriately checked and validated.

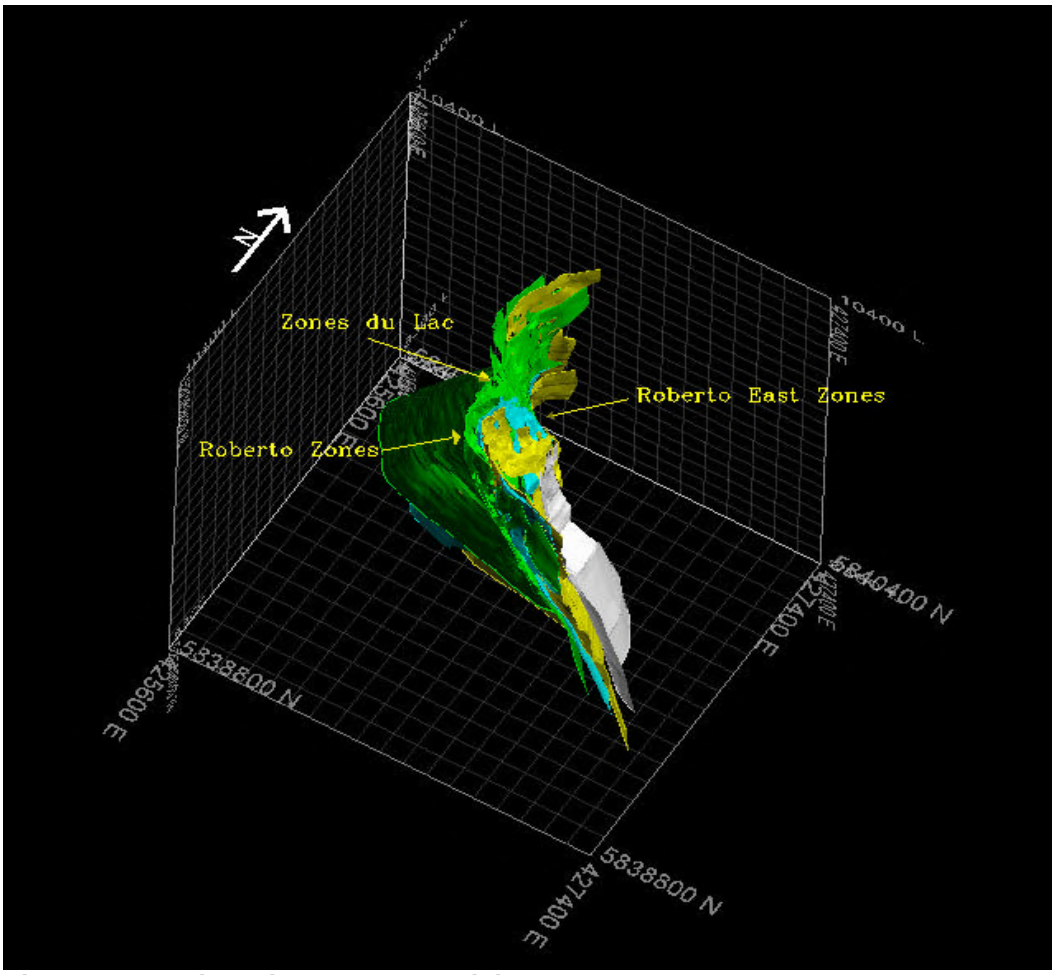


Figure 14-1: Mineral resource model

14.3. Exploratory Data Analysis

An exploratory data analysis (EDA), in the form of summary statistics, histograms, and cumulative probability plots, was performed on uncapped composites for gold to determine suitable geological constraints to mineralization. Table 14-1 summarizes the statistics for the uncapped composites for gold. The reader should note that only six (6) zones out of 29 contain a sufficient quantity of information for such an analysis and therefore only those six zones are presented. Zone "100" is the dilution envelope.

Table 14-1: Statistics for uncapped composites

Zones	Samples	Minimum	Maximum	Mean	Standard deviation	CV	Variance	Skewness	Log samples	Log mean	Log variance	Geometric mean
100	126 680	0.00	3 190	0.24	9.77	40.45	95.42	285.90	126 680	- 3.07	2.33	0.05
5 010	2 185	0.01	494	5.90	18.86	3.20	355.70	16.27	2 185	0.61	2.50	1.83
5 050	8 660	0.00	365	6.22	11.93	1.92	142.40	7.98	8 660	0.61	3.17	1.83
5 070	1 599	0.00	2 460	6.29	63.62	10.12	4 048.00	36.12	1 599	- 0.26	3.80	0.77
6 000	2 475	0.00	242	4.01	10.00	2.50	100.10	12.77	2 475	0.15	3.03	1.16
7 000	2 238	0.00	97	3.07	5.93	1.93	35.15	5.90	2 238	- 0.09	3.27	0.91

14.4. Grade Capping and Compositing

Outliers were identified primarily based on the lognormal cumulative probability curves of assay grades (Table 14-2). Each principal zone was evaluated separately. The high-grade cap threshold was determined at a grade where the curve showed a clear break and grades started becoming erratic as compared to the main grade populations. Verifications with histogram and decile methods were also conducted. High-grade samples were cut to the cap values prior to compositing. Cap values and the impact on the mean grade of each vein are summarized in Table 14-3.

Samples generally vary from 0.30 m to 3.00 m (mean = 1.06 m; mode = 1.00 m). In an attempt to retain the integrity/resolution of the original samples, and considering the relative narrowness of the HG veins in some areas, the length of the composite was set at 2 metres.

Drill hole samples were composited inside the mineralized solids into equal 2-metre down-hole length intervals. For HG and LG composites, residues were retained in the database since Vulcan allows composites to be weighted by length during the interpolation. Composites were calculated by length-weighted averaging of gold assays within each interval. Non-assayed intervals and lost core, except for those in the overburden or within the wedge areas were replaced with the background grade of zero. Wedge duplicates, wedge lost core, and navi drill lost core were ignored.

Table 14-2: High-grade probability curves (Au assays)

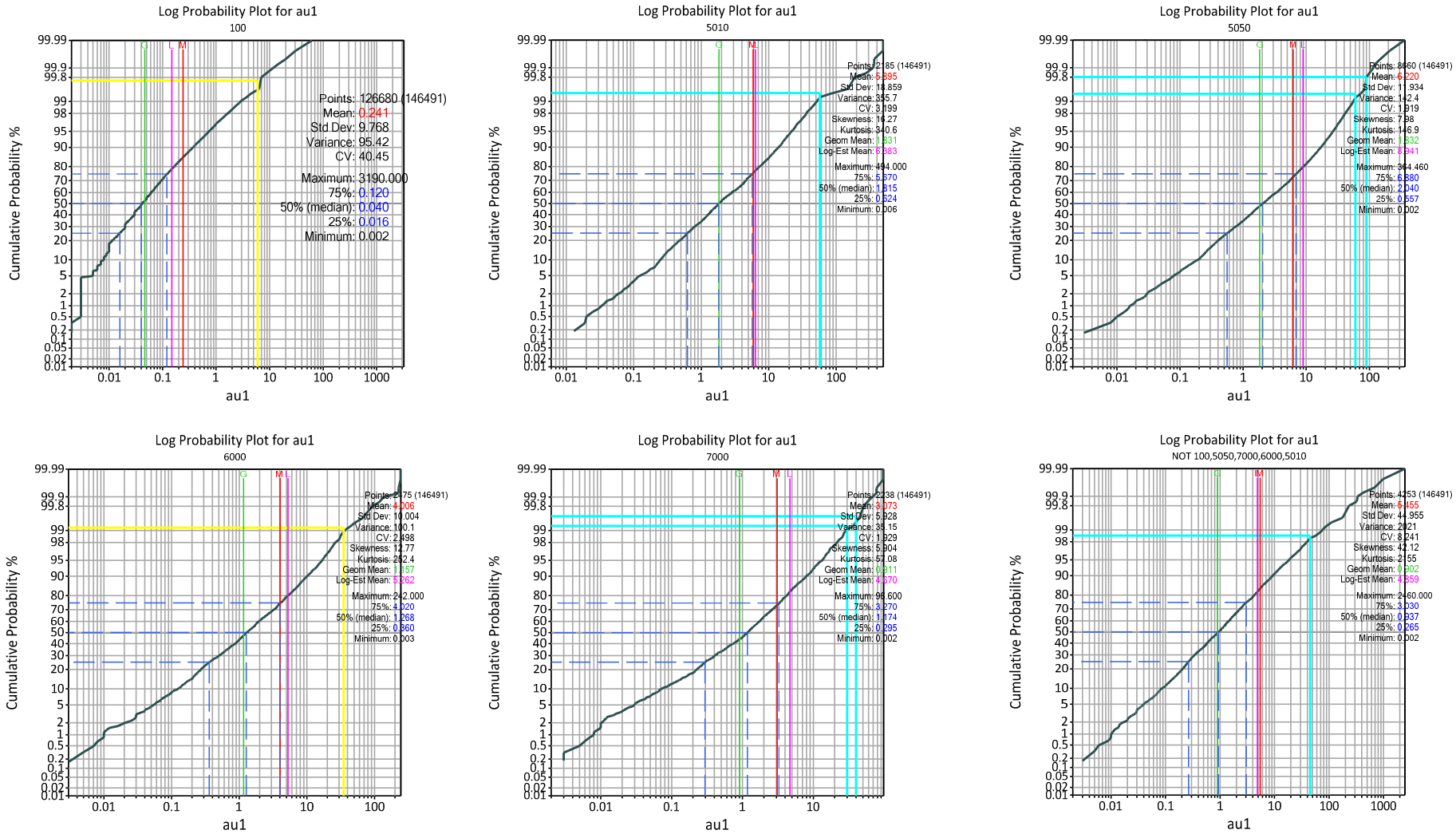


Table 14-3: High-grade capping on Au assays

Zones	Type	n Samples	Min (g/t)	Max (g/t)	Mean (g/t)	Std dev	CV	Var	Histogram	Prob plot	Decile	Capping 2013	Cap Mean	Cap CV	%metal loss	%data Cap	# data cap
100	Dilution Envelope	126680	0.002	3190	0.241	9.768	40.45	95.42	10 g/t	6 g/t	3 g/t	7 g/t	0.17 g/t	2.92	19.81%	0.22%	279
5010	High Grade	2185	0.006	494	5.895	18.86	3.199	355.7	60 g/t	57 g/t	47 g/t	60 g/t	4.99 g/t	1.72	11.44%	0.50%	11
5050	High Grade	8660	0.002	364.5	6.22	11.93	1.919	142.4	62 g/t	60 g/t	51 g/t	90 g/t	5.64 g/t	1.67	1.13%	0.20%	17
6000	High Grade	2475	0.003	242	4.006	10	2.498	100.1	30 g/t	35 g/t	33 g/t	70 g/t	3.61 g/t	1.82	3.98%	0.20%	5
7000	High Grade	2238	0.002	96.6	3.073	5.928	1.929	35.15	32 g/t	30 g/t	27 g/t	45 g/t	2.93 g/t	1.83	1.60%	0.22%	5
other_zones	Various	4253	0.002	2460	5.455	44.95	8.241	2021	50 g/t	45 g/t	64 g/t	100 g/t	3.32 g/t	2.12	24.67%	1.34%	57

14.5. Variography

Composites within interpreted geological zones were used to generate variography and ultimately determine search ellipsoids.

Unidirectional variography studies were completed using the 2m DDH composites of the capped gold assay data for the five (5) main mineralized zones. Attempts were made to model the other zones but were unsuccessful, due mainly to the time constraint, the large variation in vein shape orientation, and the lack of data in selected vein areas where relatively consistent orientations were observed.

Ellipsoid radiuses were established using a combination of the ranges determined from the variography, the drill hole distribution, and the geological model comprehension.

The Snowden SuperVisor® (version 8) software was used to model unidirectional log variograms. Maximum continuity was found in the vein dip direction, plunging 70° to the north, with a range of 150 m. The unidirectional variogram for the Roberto deposit is shown as Figure 14-2 at two different lag distances.

The results of the linear variographic investigations for the DDH composites are consistent with the geological features of the deposit. The investigations yielded the best-fit model along an orientation that roughly corresponds to the strike and dip of the observed ore shoots at the Roberto deposit.

The variogram parameters obtained were used for the kriging estimation of all zones (except ME), and the anisotropy was rotated to fit the orientation of the veins. The ME veins were estimated using an inverse distance cubed (ID3) interpolation

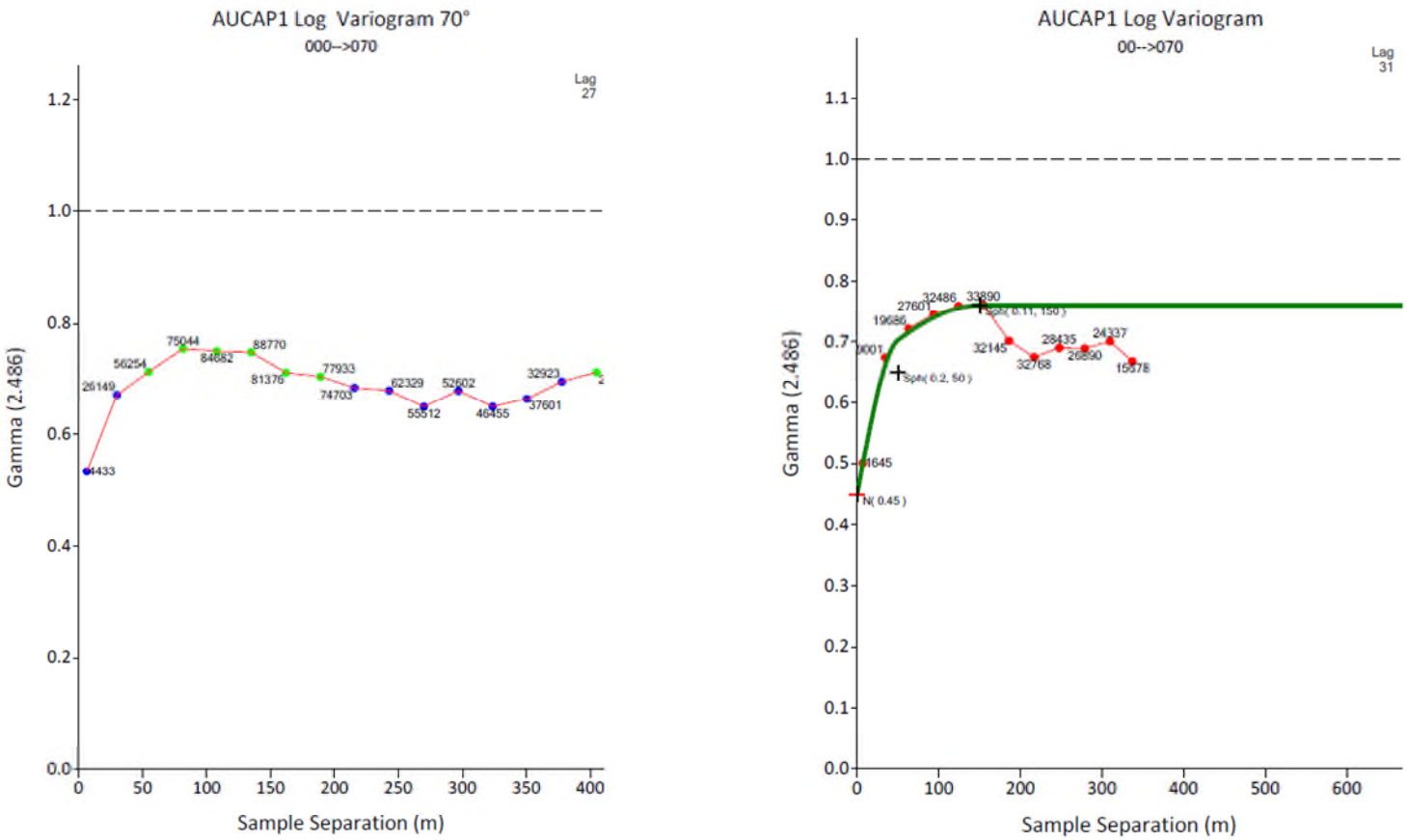


Figure 14-2: Unidirectional variograms showing results for two different lag distances.

14.6. Estimation Parameters

The block model was set up as a sub-block model. Each block could have only one rock code. The blocks were divided into three groups: HG, LG, and ME, according to the zone type.

Block grade was estimated with a hard boundary. Composites were only used for the estimation of blocks with the same rock code, except where one vein was the splay of another vein; in that case, composites were shared.

The HG, LG, and ME blocks were estimated independently, with a three-pass plan. The first two passes used relatively short search radiuses to interpolate the blocks close to the drill holes. The third pass was defined to populate the remaining blocks within the entire vein solid.

Ordinary kriging was used to estimate grades within the main HG and LG zones while grade estimation in the mineralized envelope (ME) was completed using inverse distance cubed (ID3) methods.

Parameters used for grade estimation are summarized in Table 14-4.

Table 14-4: Resource model estimation parameters

Zone ID	Domain	Estimation	Estimation pass	Search radius (m) - rotated to fit est. domain			Number of composites			HG threshold (au g/t)	HG restricted search radius			Search type	Estimation method
				Major	Semi	Minor	min	max	max/hole		x	y	z		
all	all	Au (HG)	1	75	30	20	4	10	3	none	none	none	none	ellipsoidal	OK
all	all	Au (HG)	2	150	55	20	3	9	3	none	none	none	none	ellipsoidal	OK
all	all	Au (HG)	3	300	110	40	3	12	4	none	none	none	none	ellipsoidal	OK
all	all	Au (LG)	1	75	30	20	4	10	3	none	none	none	none	ellipsoidal	OK
all	all	Au (LG)	2	150	55	20	3	9	3	none	none	none	none	ellipsoidal	OK
all	all	Au (LG)	3	300	110	40	3	12	4	none	none	none	none	ellipsoidal	OK
Dilution	all	Au	1	75	75	25	8	18	6	1	15	15	5	ellipsoidal	ID3
Dilution	all	Au	2	150	150	50	6	18	6	1	15	15	5	ellipsoidal	ID3
Dilution	all	Au	3	300	300	100	10	20	10	1	15	15	5	ellipsoidal	ID3

Notes : Au (HG) = zone grade estimation inside the interpreted solids at COG 3g/t
Au (LG) = zone grade estimation inside the interpreted solids at COG 1g/t and outside HG

14.7. Block Model Validation

Visual inspection confirmed that the block model respected the drill hole data.

A nearest-neighbour (NN) model using 5 m composites and an inverse distance squared (ID2) model were produced to check the global and local bias of the mineral resource model; results indicated that the global means of the resource model matched well with the verification models, with differences within acceptable limits in most of the main HG veins.

The trend and local variation of the mineral resource model grades were also reviewed against NN model with swath plots in northing, easting and elevation directions. It was noted that, above the 600 level (600 m below surface) and at the core of the deposit, the mineral resource model showed similar trend in grades as the NN model with the expected smoothing. The two models departed slightly at depth and at peripheries where drill holes became sparsely spaced. This indicated that there is some risk associated in estimating Mineral Resources in areas of limited drilling, particularly at depth.

14.8. Classification of Mineral Resources

The resource classification definitions used for this report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their document “CIM Definition Standards for Mineral Resources and Reserves”.

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.

For the Éléonore Project Mineral Resource Estimate (exclusive of Mineral Reserves), all estimated blocks were classified into the Inferred category. The main basis for the classification was a drill hole density deemed insufficient for establishing either Indicated or Measured resources.

Inferred Mineral Resources were defined for blocks within 60–70 m from the drill hole, with an occasional block influence as far as 80 m where the mineralization trend was demonstrated by multiple adjacent holes.

Indicated Mineral Resources were defined for blocks estimated through the first pass (minimum two drill holes within 75 m x 30 m x 20 m search radii) and within 30 m from drill hole. All the Indicated Mineral Resources were promoted to Probable Reserves and are therefore absent from the current resource estimate, which is reported exclusive of mineral reserves.

In order to classify the resources as Measured, an average diamond drill hole spacing of 12.5 metres must be used, and samples must come from a minimum of three drill holes or underground development in mineralization. At the time the report was compiled these requirements were not achieved except in a few rare instances.

14.9. Reasonable Prospects of Economic Extraction

Considering the geometry and shape of the orebody, the Roberto deposit is amenable to underground mining using long-hole stoping methods. Based on the pre-feasibility work, a recovery of 93.5% and an operating cost of approximately CA$111.51/t (mining: CA$52.41/t; processing: CA$29.89/t; services: CA$8.76/t; G&A: US$20.45/t) were considered reasonable. Using a gold price of US$1,300/oz with a CAD/USD exchange rate of 1.05, the cut-off grade required to support reasonable prospects of economic extraction is approximately 3.0 g/t Au.

14.10. Mineral Resource Statement

The Qualified Person for the Mineral Resource estimate is Christine Beausoleil, P.Geo., an employee of Goldcorp.

Given the density of the processed data, the search ellipse criteria, and the specific interpolation parameters, the current Mineral Resource Estimate can be classified as Inferred resources. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves.

The Mineral Resources (exclusive of Mineral Reserves) for the Éléonore Project were estimated at 13.25 Mt grading an average of 9.63 g/t gold in the Inferred Mineral Resource category using a cut-off grade of 3.0 g/t gold (Table 14-5).

Mineral Resources are reported at a commodity price of US$1,300/oz gold. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-5: Mineral Resource Estimate (exclusive of Mineral Reserves) for the Éléonore Project, effective date of December 31, 2013 (C. Beausoleil, P.Geo.)

Classification Category	Tonnage (Mt)	Au (g/t)	Cont'd Gold (Moz)
Measured	-	-	-
Indicated	-	-	-
Measured & Indicated	-	-	-
Inferred	13.25	9.63	4.1

Notes to accompany Mineral Resource table:

1. The Qualified Person (QP) for the Mineral Resource Estimate, as defined by NI 43-101 / Regulation 43-101, is Christine Beausoleil, PGeo (Chief Geologist Interim for Goldcorp), and the effective date of the estimate is December 31, 2013.
2. The entire Mineral Resource (exclusive of Mineral Reserves) is classified as Inferred. The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred resources as Indicated or Measured and it is uncertain if further exploration will result in upgrading them to the Indicated or Measured mineral resource category.
3. These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
4. While the results are presented undiluted and in situ, the reported mineral resources are considered to have reasonable prospects for economic extraction.
5. CIM definitions and guidelines were followed in the estimation and reporting of these Mineral Resources.
6. The resource was estimated using Vulcan 8.2. Mineral Resources are based on a total of 839 core drill holes and 143 surface channels, totalling 147,390 assay results, collected between September 2004 and September 5, 2013.
7. A minimum true thickness of 2.5 m was applied for the Upper Mine and 1.0 m for the Lower Mine, using the grade of the adjacent material when assayed or a value of zero when not assayed.
8. Supported by statistical analysis and the high-grade distribution within the deposit, a top cut varying from 45 g/t to 100 g/t (7g/t for the dilution envelope) was applied to assay grades prior to compositing grades for interpolation into model blocks using Ordinary Kriging and ID3 methods, and was based on 2 m composites within block model made of 5m long x 5m wide x 5m high blocks.
9. Three passes for each of the geological zones were used for interpolation.
10. A bulk density of 2.77 g/cm^3 was used for all types of lithological material in the block model.
11. Mineral Resources are reported using a 3 g/t Au cut-off grade, which is based on assumptions of a US$1,300/oz gold price, long-hole stoping underground mining methods, a total mining cost of CAD$111.51/t (mining: CAD$52.41/t; processing: CAD$29.89/t; services: CAD$8.76/t; G&A: CAD$20.45/t), a CAD/USD exchange rate of 1.05, and an LOM metallurgical recovery of 93.5%.
12. Ounce (troy) = metric tons x grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
13. The number of metric tons was rounded to the nearest thousand. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101 / Regulation 43-101.
14. The QP is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issue that could materially affect the Mineral Resource Estimate.

14.10.1. Comment on Section 14

The QP is of the opinion that the Mineral Resources for the Éléonore Project, which have been estimated using core drill data and channel samples, were estimated according to industry best practices and conform to CIM requirements (2010).

Key areas of uncertainty that may materially impact the Mineral Resource include:

- Commodity price assumptions;
- Metal recovery assumptions;
- Hydrogeological constraints;
- Rock mechanics (geotechnical) constraints.

15. MINERAL RESERVE ESTIMATES

Mineral Reserves have been estimated from the geological resources block model produced by the geology department in October 2013. The requirements for Mineral Resources to be converted to Mineral Reserves are as follows:

- Only Measured and Indicated Mineral Resources can be considered;
- Dilution is included in the Mineral Reserve estimate;
- Mineral Reserves are supported by an economic mine plan.

The Mineable Shape Optimizer (MSO) software was used to create the stope designs. MSO is a module available in the Studio 5 software, which was developed to optimize underground stope design. Using the block model as an input, MSO identifies the optimal shape, size and location of stopes for underground mine design. Stope sections produced in MSO are imported into Studio 5 software. Studio 5 (Underground Module) software is used to create solids (mineable stopes) from the section produced with MSO, estimating tonnage and grade for every stope. Finally, Studio 5 and EPS (another module of Studio 5) are used to optimize the mining sequence.

15.1. Estimation Procedure

The stope design software models 5 m slices across pre-defined orientations through the orebody, and then combines these data to optimize the stope direction in relation to the orebody strike. An iterative process then creates the final optimal shape that maximizes the value of the stopes, while still respecting geometry constraints and input parameters. Input parameters include a specific gravity value of 2.77 and a 3 g/t Au cut-off grade.

For the Éléonore Mine, separate cases for the north, centre and south zones were run for every level (Figure 15-1).

The optimal stope shapes are created taking into account orebody geometry; they are not restricted to the standard rectangular or oblique stope shape. Constraints used on stope geometry are as indicated in Table 15-1.

Studio 5 is used to join the 5-m-long initial stopes that were produced with MSO into stopes of 50 m long. Studio 5 was also used to create tonnage and grade reports and to perform mine scheduling.

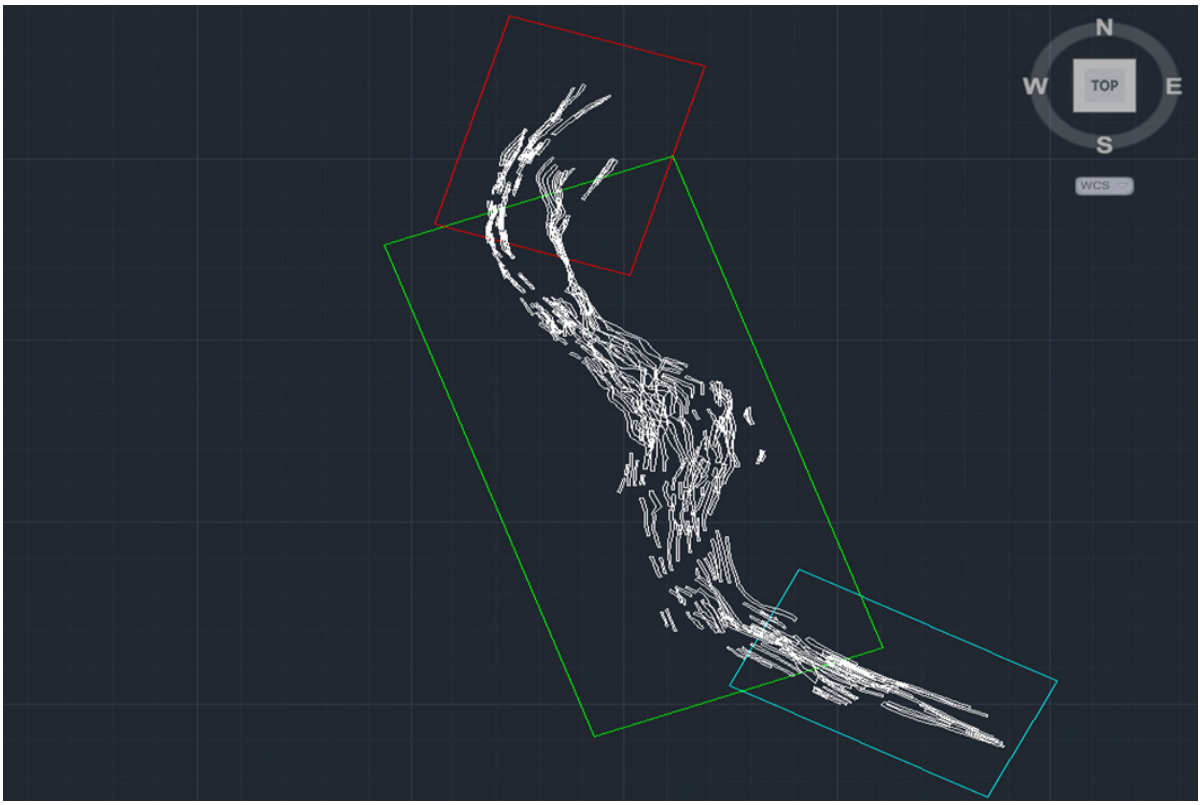


Figure 15-1: Locations of the North and South zones defined for MSO simulations

Table 15-1: Geometry constraints

Geometry Constraints	
Minimum Width	3 m
Maximum Width	100 m
Minimum Waste Pillar Width	1 m
Near Wall Dilution	0 m
Far Wall Dilution	0 m
Minimum Dip Angle	45°
Maximum Dip Angle	135°
Maximum Strike Angle	45°
Maximum Strike Angle Change	20°
Maximum Side Length Ratio	3
Level Spacing	15 m
Section Spacing	5 m

15.2. Dilution

Two types of dilution were considered during the reserves estimation: internal and external.

External dilution is discussed in detail in Section 16.4.1,

Internal dilution is evaluated directly from the Life of Mine (LOM) plan and specific to each and every stope. The tonnes and grades associated with internal dilution are incorporated into the block model and stope designs. This is described as Class 5 of the LOM and is of very low grade (average grade of 0.2 g/t Au). The average internal dilution in tonnes is 4% over a sample of 1,300 stopes; in practice, every effort will made to minimize this dilution.

It is expected that short-term to mid-term planning teams during mining operations will re-assess the stope design geometry, length and inclination to minimize any such additional dilution.

15.3. Mining Widths

Mining widths are primarily a function of the geometry of the orebody. The selection of a long-hole stoping mining method (transversal and longitudinal) and the choice of mining equipment will allow mining to reach a mining width of 3 m before dilution. The use of smaller equipment to reduce the minimal width is under study to determine if this option is economical.

15.4. Mining Extraction and Ore Losses

Every stope was evaluated for the proposed mine plan and included if appropriate, in the preliminary mining sequence and in the mining schedule.

Where recovery of mineralized material requires additional data, that material was not included in the Mineral Reserves and was reported as Mineral Resources only. Currently no Mineral Resources that fall within the area of the surface crown pillar (from surface to 55 m depth) are included in the Mineral Reserves. Additional hydrogeological and geotechnical data will be required to support conversion of these blocks.

15.5. Conversion Factors from Mineral Resources to Mineral Reserves

Mineral Resources classified as either Indicated or Measured were considered during the conversion to Mineral Reserves.

The economic analysis used to define the Mineral Reserve combines the results from long-term and mid-term planning. The Geology Department issues block

model data which are used by the Engineering Department to build a long-term mine plan and Mineral Reserves. The work consists of analyzing the block model with MSO, Studio 5 and EPS to create mineable stopes and produce a preliminary mining sequence. At this stage of the project, the mining sequence in the Upper Mine was optimized according to development priorities and rock mechanic requirements, but the mining sequence in the Lower Mine was based on the original scenario (NI 43-101 of January 2012) using the longitudinal retreat mining method.

15.6. Mineral Reserve Statement

Mineral Reserves have been modified from Mineral Resources by taking into account geologic, mining, processing and economic parameters and are therefore classified in accordance with the NI 43-101 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Reserves incorporate considerations of minimum mining width, dilution and mining sequence.

The Qualified Person for the Mineral Reserve estimate is Denis Fleury, P.Eng., an employee of Goldcorp.

Mineral Reserves are reported using a gold price of US$1,300/oz. The cut-off grade was fixed at 3 g/t Au and the exchange rate between the American and Canadian currencies was fixed at CAN/US$ = 1.05. The Mineral Reserve for the Éléonore Project is summarized in Table 15-2.

Table 15-2: Reserves by mining horizon

Level	Stope			Development			Total			
(m)	Tonnage Mt	Au M grams	Grade G/t	Tonnage Mt	Au M grams	Grade G/t	Tonnage Mt	Au M grams	Au M oz	Grade G/t
80-230	5.16	33.71	6.5	0.62	3.48	5.6	5.78	37.19	1.2	6.4
260-440	5.80	34.54	6.0	0.58	2.97	5.1	6.38	37.51	1.2	5.9
470-650	4.85	30.35	6.3	0.38	2.04	5.4	5.23	32.39	1.0	6.2
680-860	1.14	10.95	9.6	0.24	1.98	8.3	1.38	12.93	0.4	9.4
890-1040	0.43	4.30	10.0	0.07	0.60	8.6	0.50	4.90	0.2	9.8
1070-1220	0.03	0.23	7.5	0.01	0.06	6.5	0.04	0.29	0.0	7.3
TOTAL	**17.41**	**111.53**	**6.4**	**1.89**	**13.67**	**7.2**	**19.30**	**125.20**	**4.0**	**6.5**

Notes to accompany Mineral Reserve table:

1. Mineral Reserves are estimated using a gold price of US$1,300/oz and an economic function that includes variable operating costs and metallurgical recoveries. These assume processing costs of CAD$29.89/t, mining operating costs of CAD$52.41/t, site services costs of $8.76/t, and general and administrative costs of CAD$20.45/t, for a total life-of-mine estimated operating cost of CAD$111.51/t (average over LOM).
2. Mineral Reserves are reported using a cut-off grade of 3.0 g/t Au.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal contents.
4. Tonnage and grade measurements are in metric units. Gold ounces are reported in troy ounces.
5. The entire Mineral Reserves (exclusive of Mineral Resources) is currently classified as Probable.
6. Reserves below 650mLv are based on January 2012 R&R Evaluation document.

All the Mineral Reserves are classified as Probable Mineral Reserves. , The Mineral Resources located in the surface crown pillar, between the surface and a depth of 55 m were not included in the actual reserves. A complete study (hydrogeological and geotechnical) on the recovery of the surface pillar would be required to support the conversion of these Mineral Resources to Mineral Reserves.

15.7. Comment on Section 15

The QPs are of the opinion that the Mineral Reserves for the Éléonore Project, which have been estimated using core drill and development data, appropriately consider modifying factors, have been estimated using industry best practices, and conform to NI 43-101. Intense diamond drilling is underway on the Project to convert the actual and growing Inferred Resources to Indicated Resources; at this point, if the program is proved successfull and grade consistant with actual resource, it appears that this work could increase the LOM Reserves by approximately 5 years.

Factors that can affect the Mineral Reserves estimates are:

- Low recovery at the mill because of a possible change in the hardness of the rock. The grinding can be longer if the hardness of the rock changes deeper in the mine. No laboratory tests on the rock resistance have been performed on drill core from an elevation lower than 500 m.
- More water infiltration from the surface or underground than expected. Water infiltration could come from the geological structures and from the diamond drill holes. Some of the old diamond drill holes were not correctly grouted or simply not grouted, and therefore even if all current holes are correctly grouted, there is likelihood that some non-grouted holes could be intercepted during production. In this situation, if the water infiltration is very high, Goldcorp could be obliged to leave the broken rock in the stope, and immediately commence stope backfilling perations.
- In situ stress in the rock. Currently no in situ stress measurements were performed in the area of Éléonore. These measurements should be done in 2014 when development headings will be available in the 650 area. Changes in the mining sequence and pillar dimensions may be necessary if higher in-situ stresses are indicated from this program.
- Rock burst. It is hard to predict the future behaviour of the rock mass with the available information. Ramp and shaft station excavation have triggered minor, small seismic events. These seismic events appear to be associated with changes in geology. Seismic events may become bigger and really dangerous if the planned mining sequence is not

followed, and could compromise the Mineral Reserve estimates.

- Deviations in the drill holes necessary to support production may cause more dilution. The production hole diameter will be 102 mm (4 in) but the length will be 27 m (90 feet). Controlling the deviation will be an important factor in reducing unplanned dilution. If additional dilution results, the gold cut-off grade may need to be increased.
- Paste backfill strength. Laboratory tests have indicated good results but in cases where a stope has water infiltration issues, or where the stope is in an area of cold temperature (i.e. in the upper mine levels), then paste backfill strengths can be lower than expected. This could result in unplanned dilution, and therefore require an increase in gold cut-off grade.
- Stope dilution and recovery factors are based on assumptions that will be reviewed after mining experience.
- Stope stability is also an important factor with some stopes having considerable span and thickness.
- As always, changes in commodity price and exchange rate assumptions will have an impact on the cut-off grade.

16. MINING METHODS

The proposed mining method is a mix between open stoping and longitudinal stoping. The geometry of the mineralized lenses varies depending on the location. Generally, the lenses in the center of the orebody are wider than the one in the northern and southern areas so it becomes more appropriate to use transverse open stoping.

A trade-off study was performed to determine the optimal mining rate. Golder evaluated production from two different levels, and set production parameters for four mining activities (drilling, blasting, mucking and backfilling). The results indicated a production rate between 1,800 t/shift to 2,500 t/shift would be required from two production levels at the same time. The Goldcorp engineering team built different scenarios using all the mining activities and all the stopes to validate the production rate. The results showed that it would be possible to produce an average of 1,800 t/day from one mining horizon with the currrent mineral reserves.

16.1. Geotechnical Considerations

Golder is geotechnical engineering firm for the Project. The following subsections are summarized from Golder's work, in particular from Castro et al. (2009). The Goldcorp engineering team started to update Golder's information in 2013. The development of different infrastructures and galleries below 300 metres deep allows rock mass characterization in the deeper part of the mine (below 300 m).

16.1.1. Rock Mass Quality

Geotechnical site investigations of the upper part of the mine were carried out during 2007 and 2008. The main findings from these investigations were as follows:

- Over 85% of the drill core recovered and logged during the site investigations was composed of greywacke (or wacke) with minor occurrences of pegmatite, diorite and diabase intrusive. Paragneiss units were encountered infrequently and may correspond to high grade metamorphic greywacke.
- All rock types encountered at Éléonore can be described as strong to very strong (uniaxial compressive strength, UCS > 100 MPa).
- Rock mass quality for the greywacke varies with depth, being lower near surface (upper 150 m) due to the occurrence of open subhorizontal fractures, as follows:
- 0–50 m: Q = 1.0 to 9.8 (Qavg = 3.4), RMR76 = 52 – 64, Poor to Fair rock mass quality;
- 50 m–150 m: Q = 1.8 to 48 (Qavg = 10.2), RMR76 = 63 – 80, Fair

to Good rock mass quality;

- 150 m–300 m: Q = 4.8 to 96 (Qavg = 20.5), RMR76 = 70 – 84, Good rock mass quality.
- A total of 73% of open structures recorded during the geotechnical investigations belonged to subhorizontal joint sets. Open structures with apertures of up to 35 cm were located in the first 150 m below surface. Three additional subvertical joint sets were identified with east–west, northwest–southeast and northeast–southwest directions.
- Core mapping showed that the fracture frequency per metre decreases from 3.5 m to 0.5 m between surface and 500 m depth. The same observation can be made in the fracture spacing. From surface to 500 m depth, the spacing between fractures increases from 0.5 m to almost 3 m.

Geotechnical mapping was performed in the summer of 2013 by Goldcorp engineering team. A total of 254 linear metres of mapping was done mainly in the ramp between levels 410 and 530, levels 410 and 440, and finally on the level 650. The orientation of the main join sets are the same as the one identified by Golder. Only the subhorizontal joint set is less represented. This can be explain by the orientation of the mapping, although deeper in the mine subhorizontal structures are still present but their aperture and frequency decrease. Finally, the frequency of the joints can be compared to Golder's results.

Main structures were identified from drill core data and underground mapping (see Figure 16-1). A total of four faults were modelled. They all have a northwest–southeast orientation. Only one of them can be characterised as a typical fault. It contains chlorite and gauge material. The others are a suite of NW–SE epidote–quartz breccia zones. They are modelled as discrete structures but in reality they are not always internally continuous. They often look like damage zones.

To the southeast of the mine, there is a prominent NE–SW trending zone with abundant diabase dyke intersections. In detail it appears that there are several subparallel dyke segments. They are stronger than the host rock. The geotechnical significance of these dykes remains to be determined.

Many fault zones have been logged in core; however even with the benefit of oriented core it is difficult to determine the extent and significance of minor faults. They seem to be bounded by the major faults, but there is no evidence as to how they interact with each other, so at present they are modelled without mutual offset though clearly they must terminate or be offset where they intersect. Where they are intercept the development, minor modifications to the support may be necessary.

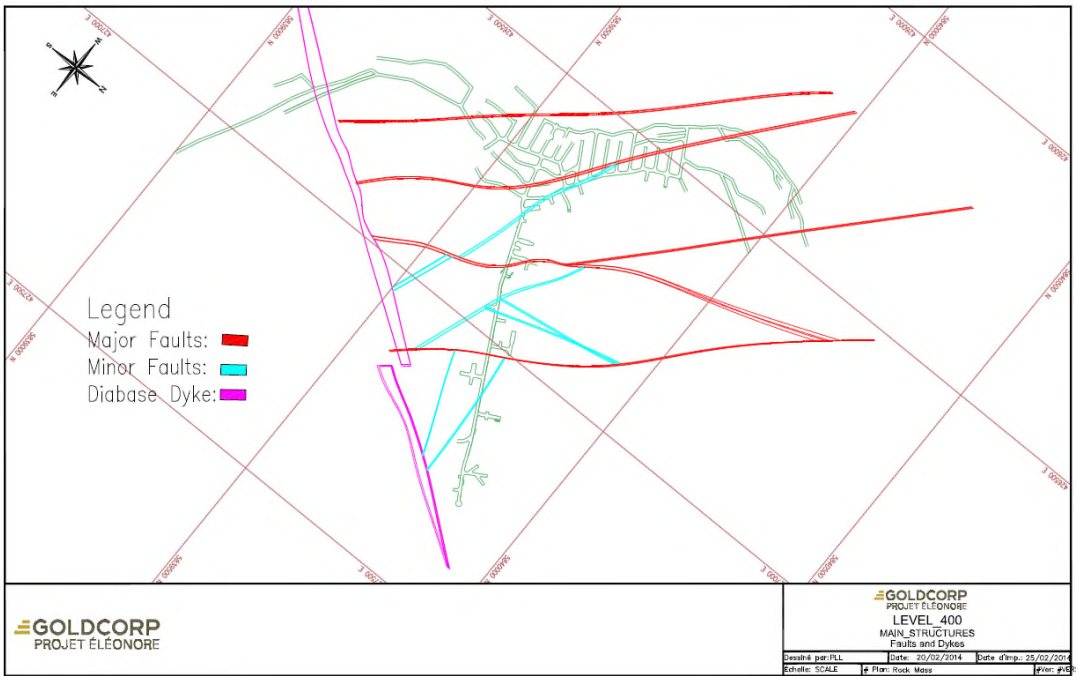


Figure 16-1: Plan view of the main geological structures intercepted on the 410m level

16.1.2. Stope Design

The selected mining method is long-hole stoping (down hole drilling) using longitudinal retreat with consolidated backfill and primary and secondary transverse stoping (Open stope). Paste backfill mixed with crushed waste rock is envisioned.

The orebody can be divided into three areas. In the southern and the northern areas are the longitudinal stopes and in the centre, where the lenses are wider, are the transverse stopes. Based on the work completed by Golder in 2009 (Golder, 2009) and subsequent reviews undertaken by Goldcorp, the following maximum stope dimensions can be considered for mine planning:

- Stope width = up to 20 m, minimum 2.5 m
- Stope height = 30 m (sublevel floor to floor)
- Stope strike length = up to 50 m

These dimensions are based on different calculations. Depending of the depth and the geology, the dimensions will vary and support may need to be added. As additional information becomes available on geology and ground conditions, the stope sizing recommendations will be reviewed to validate the preliminary assumptions.

16.1.3. Ground Support

Systematic ground support is installed in all excavations such as drifts, raises and ramps. A typical reinforcement pattern of 1.2 m x 1.2 m is in the back (roof) of the excavations with #6 and #9 wire mesh screen for permanent and temporary excavations respectively. The walls are also supported to avoid the formation of unstable wedges.

Depending on the size of the excavation, rebars and split sets of different lengths are used to reinforce the back (2.4m to 3m long) and walls (1.5 m long) of permanent accesses, such as the ramp and main levels. Temporary support such as super-swellex and split set are installed before the permanent support (cables, shotcrete and/or concrete).

Shotcrete is used to control local areas of weaker or fractured ground including pegmatite dykes, fault/shear zones or higher stress areas, during the excavation of large openings (garage, rock breaker room, refuge stations, etc.) and during construction.

Cable bolting is used for large excavations and at intersections with spans greater than 9.5 m. Cable bolt support will be installed in the back of the stopes in all the transverse stopes. Cable bolting in the longitudinal stopes above 600 metres is considered, but more analysis will be required as the geological and stress information become available.

16.1.4. Backfill

Open stopes will need to be backfilled for long-term stability and to accommodate the long hole retreat mining method and the primary-secondary transverse stopes. Consolidated backfill will be used to avoid pillars between stopes. Paste fill (mill tailings and binder) mixed with crushed waste rock is expected.

Golder's report recommends that for the end walls of the 10 to 15 m wide longitudinal stopes to be self-supporting, the cemented backfill will require a minimum strength (UCS) of approximately 300 kPa for sublevel intervals of 30 m. UCS of 340 kPa will be required for 15 m wide stopes having a double stope height of 60 m. Hence, to satisfy these requirements, preliminary paste fill design should target a strength of about 0.5 MPa at 28 days.

To mine ore zones less than 10 m wide located directly below the fill, preliminary assessments indicate that a minimum paste fill strength of 2.5 MPa will be required.

Analysis and strength testing of potential paste fills were conducted after the completion of the 2009 Golder report. In practice, UCS strength of 440 kPa was targeted after a 28-day curing time, which constitutes a conservative approach with respect to the recommendations of Castro et al., 2009. Various binder

proportions were tested.

In July 2013 Goldcorp mandated Graham Swan (Swan, 2013) to revised Golder's preliminary recommendations. The mining method changed (transverse stoping) in the centre area of the orebody. Swan's recommendations for longitudinal stopes 3 to 7 metres wide and 20 to 40 m long are a UCS between 100 kPa and 170 kPa, and for transverse stopes 5 to 12 m wide and 20 to 30m long, 270 kPa to 525 kPa. He also evaluated a special case with multiple panels in a secondary stope; in this specific case the UCS strength needs to be 1.5 MPa.

Further investigations will be carried out in order to decrease backfill costs. Since backfill is cost-sensitive to binder, it is anticipated that a reduction in binder proportions will result in cost savings. This could be achieved by replacing a portion of the paste fill binder with crushed waste rock (aggregate) while still aiming to maintain the same paste fill strength. Another advantage of using aggregate in the paste fill will be the reduction of the volume of development waste rock stockpiled at surface.

The paste backfill mixture will consist of 67% mill tailings, 17.5% -½" aggregate, 12% fine sulphide concentrate, and 3.5% binder. If no aggregate is used in the paste fill, then the percentage weight of the binder will be around 5%.

16.1.5. Surface Crown Pillar

A preliminary crown pillar analysis was carried out, complete with numerical modelling and empirical rules assessment. The results of this analysis show that for a 15 m stope span, the minimum recommended crown pillar thickness was 75 m. In the case of a 10 m stope span, the minimum crown pillar thickness could be decreased to 40 m.

These recommendations took into account the high risk of water infiltration from the overlying reservoir.

16.1.6. Hydrological Considerations

The Roberto deposit is located under the Opinaca Reservoir whose water level is controlled by Hydro-Québec. The highest water level in the reservoir is at 215.8 metres above sea level while the critical water level is at 216.4 metres above sea level (Hydro-Québec data).
Due to the presence of open subhorizontal decompression joints encountered mainly within the first 150 m below surface, and the proximity of the reservoir, management of ground water infiltration is considered paramount for the successful implementation of the Éléonore Project.

Hydrogeological site investigations were carried out in 2007–2008, in conjunction with the geotechnical design study. The hydrogeological modelling study indicated potential peak water inflows of approximately 181,000 m^3/d (33,000 US gpm).

This was deemed unmanageable without implementing considerable mitigation measures.

Geotechnical and hydrogeological investigations for the preparation of the Gaumond shaft were carried out during mid-2008 and provided additional insight into the hydrogeological characteristics of the bedrock.

Based on the data collected during the hydrogeological and geotechnical site investigation campaigns, three zones with different hydraulic conductivity characteristics have been identified:

- Shallow bedrock between 0 and 150 m below surface: high coefficient of permeability (3 x 5-10 m/s);
- Intermediate bedrock between 150 and 500 m below surface. The intermediate bedrock has a generally lower coefficient of permeability (6.4 x 10-8 m/s) than the shallow bedrock, but intersects a more permeable zone between 300 and 360 m below surface (1 x 10-6 m/s);
- Deep bedrock, more than 500 m below surface: low coefficient of permeability (1 x 8-10 m/s) based on a literature survey as no data were collected below 500 m.

The hydrogeological model, which was based on the October 2008 mine design, forecast peak total water inflows of 126,000 m^3/d (23,000 US gpm). That plan suggested that mitigation measures such as avoiding mining in some areas and injecting grout in infrastructure excavations located in the shallow bedrock could help reduce the water inflow.

In the second half of 2010, Goldcorp consulted with Peter White, P.Eng., regarding water mitigation strategies, mine water inflows, and mine water pumping capacity.

A review of all relevant work and studies was completed, combined with an analysis of the 2010 field testing results. The 2010 field work confirmed the rock permeability.
Based on drilling and grouting work near the Gaumond shaft and under the “Roberto Island” area, ground at depth was found to be generally competent with the exception of various permeable fracture zones. It was demonstrated that these zones could be effectively grouted.

Furthermore, a thorough and intensive grouting injection program in the first 40 m of the Gaumond shaft, managed by Mr. White, also led to excellent results at a location where significant water inflows had been encountered.

At the end of this process, and taking into consideration the latest mine design prepared by Goldcorp's special team, which incorporates Golder's 2009 minimum crown pillar thickness, Mr. White recommended implementing the following mitigating measures and pumping capacities:

- Grout exploration drill holes;
- Cover drilling, probe drilling and cement grouting in advance of mine development headings, where appropriate;
- Cement grouting operations to control significant water inflows expected to be encountered during mine development/production activities;
- Design/construct expandable underground water handling and surface water treatment facilities with increasing capacities to discharge and treat 10,000 m^3/d, 20,000 m^3/d, and up to 40,000 m^3/d (1,840, 3,670 and 7,340 US gpm).

In February 2013, the total water inflow was approximately 2,000 m^3/day. At the time, development was as follows: 11.8 km of underground workings (vertical and horizontal) had been excavated; the Gaumond shaft had been completed to 715 m; and the surface ramp from surface to the 710 level (710mLv) as well as sublevels from 650mLv to 200mLv and the production shaft were under development (the shaft was at 718 m in February 2014). During the summer of 2013, a surface program to characterize the flow between the reservoir and the orebody was completed (Verreault, 2013). A slug test was performed on the shore and a seepage flux was performed on the shore and deeper in the reservoir. At that time, the normal flowing direction was respected. The water was flowing from the aquifer towards the reservoir. Monitoring will continue as the development of the mine advances.

The mining plan proposed in herein does not include mining the ore zone above 55 m below surface. Mining of the upper part of the mine, the 230m level and above (mining horizon 1), with a higher coefficient of permeability, will start in 2014. This horizon provide more operational flexibility and knowledge on the water inflow regime so that appropriate mitigation measures may be implemented, if required, to manage the extra water or to minimize the water inflow coming from this zone.

16.2. Project Infrastructure

Details of the proposed Project infrastructure are included in Section 18.

16.3. Proposed Mine Plan

The Mineral Reserves mine plan addresses the recovery of mineralization between 55 m and 1,130 m below surface. Figure 16-2 shows a longitudinal view of the main infrastructures and the Mineral Reserves.

For mine scheduling purposes, the vertical extent of the orebody was subdivided into two parts: the upper part of the orebody located between 55 m and 650 m below surface (between 55mLv and 650mLv), and the lower part of the orebody below a depth of 650 m (under 650mLv).

Dividing the orebody into two parts was designed to accelerate the production start-up. The Gaumond shaft is 715 m deep. It was used to develop 650mLv, to provide an exploration drilling platform for the deeper portion of the orebody and to start the production of the upper part of the mine. The mine production will begin at a rate of 3,500 t/d (expected in October 2014). To achieve this production target a surface ramp, the excavation of which started in February 2011, was necessary. The ramp started from the surface and will finish at the bottom of the known mineralization. If deeper mineralization is found, the excavation of the ramp would continue to allow access to this material. Figure 16-3 presents a diagram of the life-of-mine (LOM). The excavation of the main infrastructures and the production start-up are also represented in the same figure.

The selected mining method will consist of long-hole stoping (down-hole drilling) on longitudinal retreat with consolidated backfill and transverse stoping (Figures 16-4, 16-5 and 16-6).

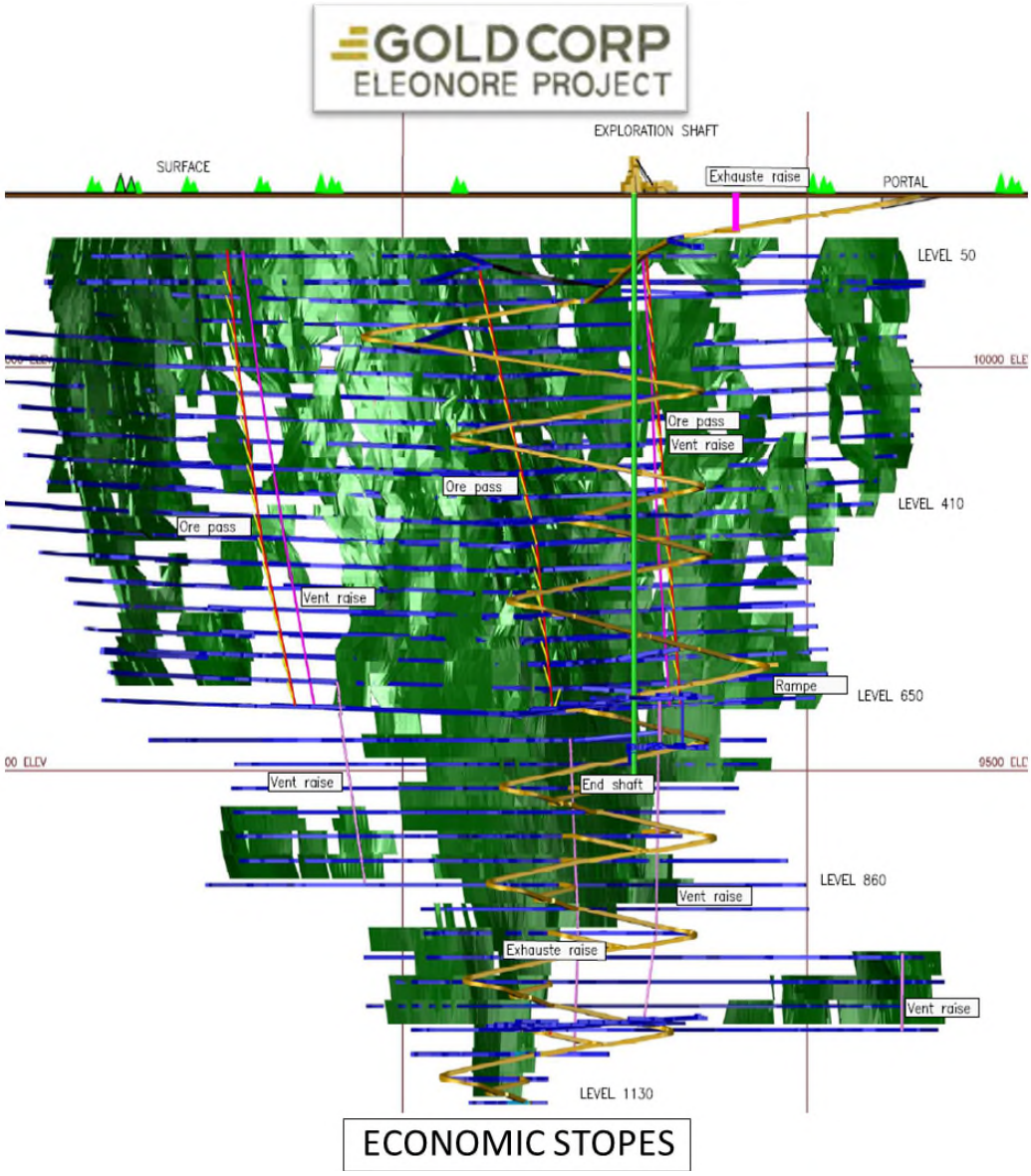


Figure 16-2: Longitudinal view of economic stopes and proposed major infrastructure

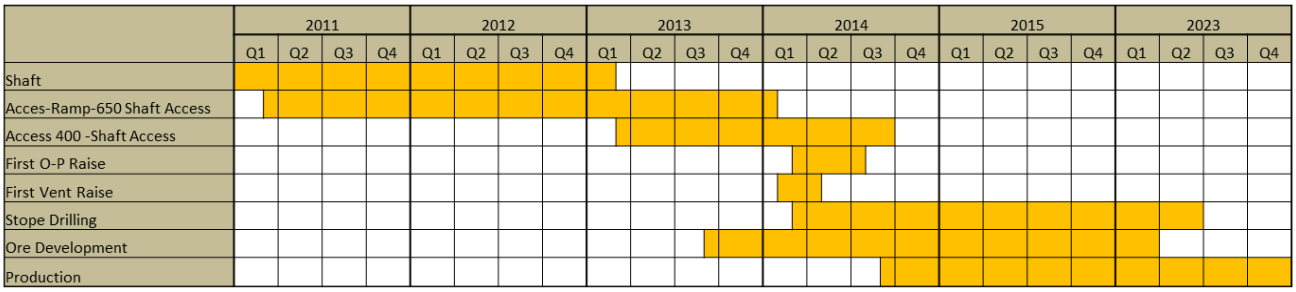


Figure 16-3: Life-of-Mine (LOM) diagram **Note: Current plans are for processing to commence in Q4 2014.**

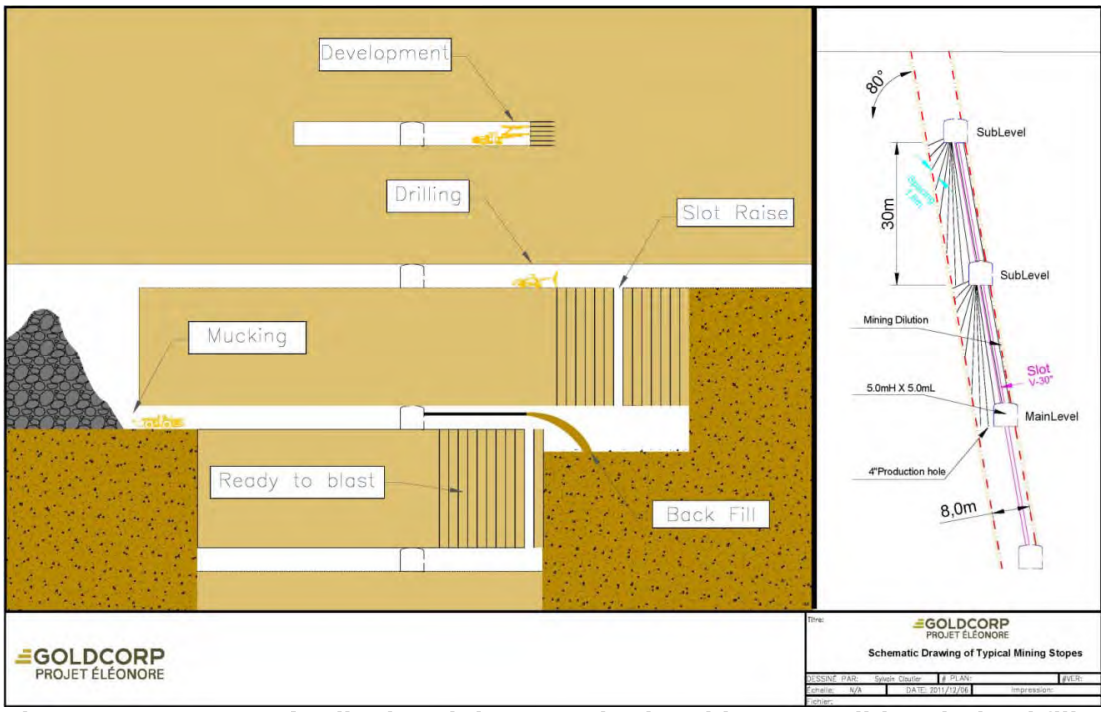


Figure 16-4: Longitudinal mining method with consolidated backfill – Longitudinal section

However, a transverse-stoping approach may be used where the mineralized lens is wider. The vertical distance between mining levels will be 30 m, from floor to floor. Figures 16-5 and 16-6 show the general trend of the mining sequence in longitudinal and transverse stoping.

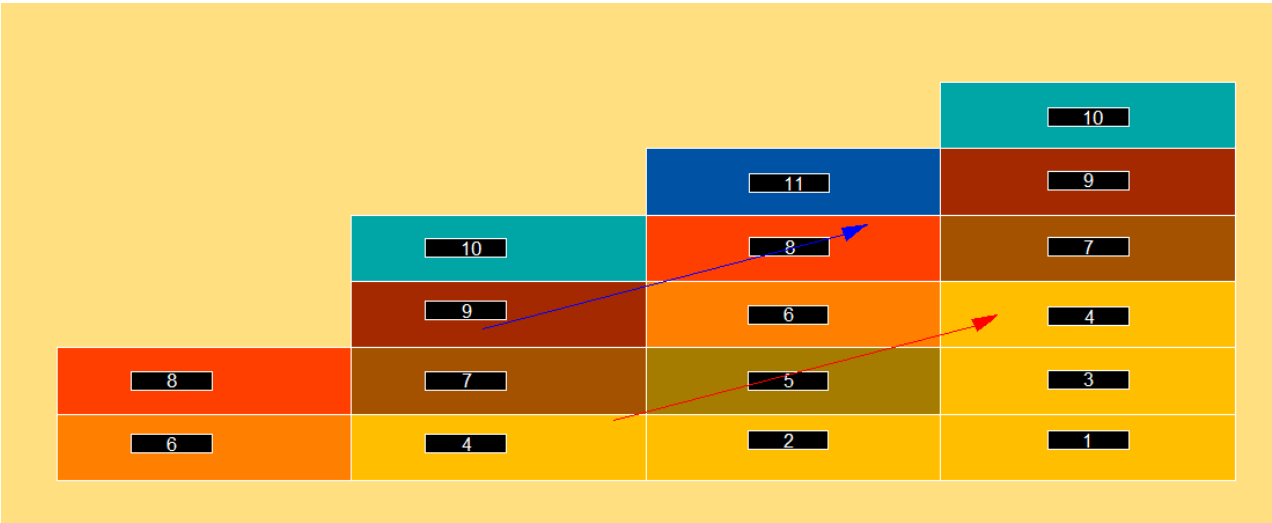


Figure 16-5: Longitudinal mining sequence – Longitudinal section

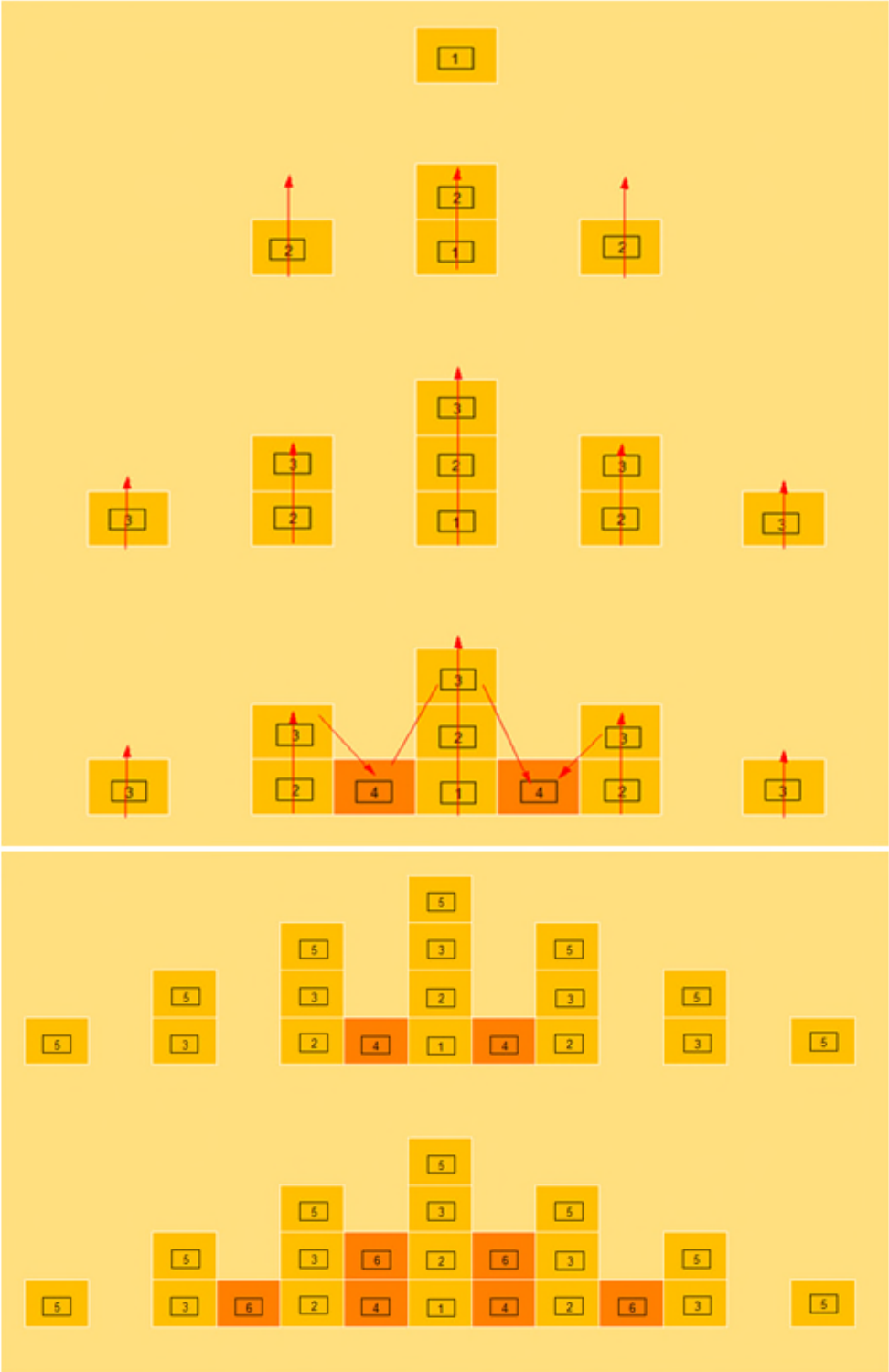


Figure 16-6: Transverse mining sequence – Longitudinal section

16.3.1. Production Rate

Golder evaluated the production rate, using 500mLv and 650mLv; Goldcorp later changed the two first mining levels. Mining will start from 440mLv and 650mLv. Figure 16-4 shows the general layout of a mining level.

On each level the lenses will be subdivided into three parts, the North Zone, the Central Zone and the South Zone. Each zone will have its own ore/waste passes to reduce the distance between the stopes and the ore pass. Two ventilation raises will be excavated (north and south). This will improve productivity as well as operational flexibility. Figure 16-7 represents a typical level.

The Goldcorp engineering team evaluated the production rate by scheduling all the mining activities in a mining sequence. Table 16-1 shows the mining activities related to every stope.

Table 16-1: Mining activities scheduled over the LOM

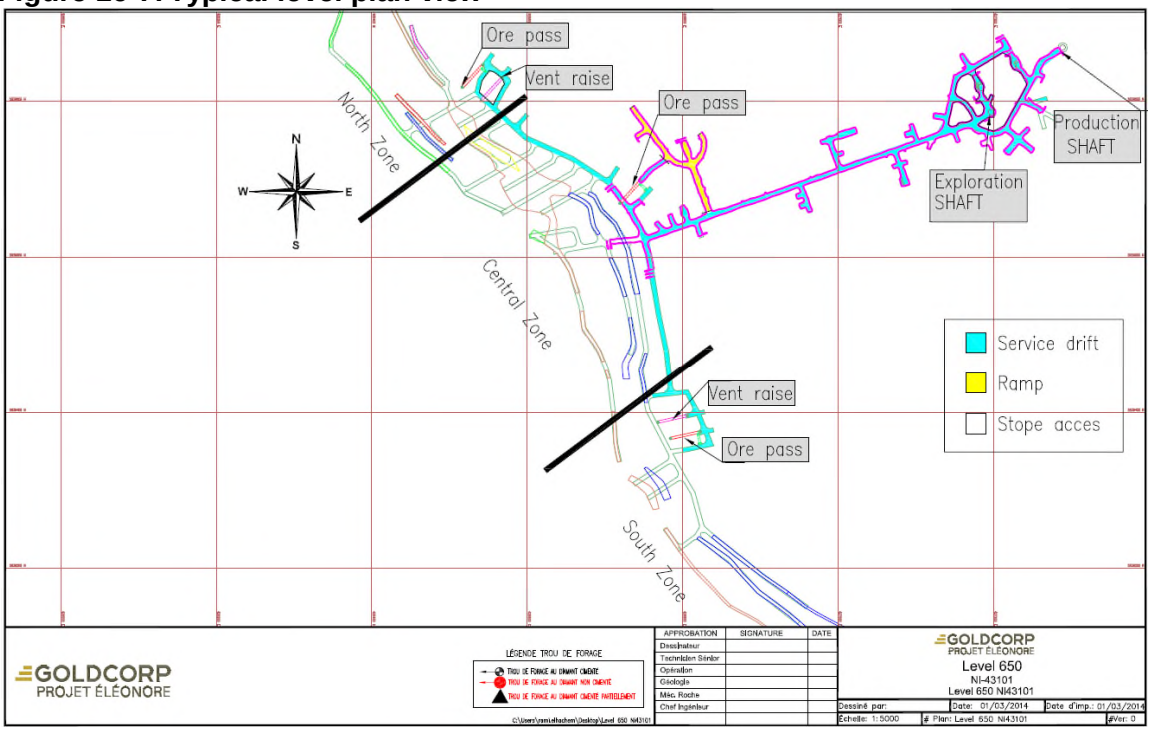
NAME	DESCRIPTION	DRIVEN BY	Units	TOTAL UNITS CALCULUS
PREP	Preparing to receive V30	Duration	days (d)	Fixed duration (3d)
SLOT	Creating a slot raise	Duration	days (d)	Fixed duration (3d)
DRILL	Drilling the stopes	Rate	drilled meters(dm)	**dm=[Mined Tonnes]/10**
BLAST	Loading and blasting the stope	Duration	days (d)	Fixed duration (3d)
STOPE	Mucking the Stope	Rate	Recuperated tonnes	**rMt=[Mined Tonnes]*0.90**
FILLPREP	Preparation required before pastefill	Duration	days (d)	Fixed duration (3d)
PASTEFILL	Filling stope with paste	Rate	Pastefill tonnes (pft)	**pft=[Insitu Volume]*1.8**
CUREC	Curing paste for 14 days	Duration	days (d)	Fixed duration (14d)
CUREL	Curing paste for 28 days	Duration	days (d)	Fixed duration (28d)

Nine activities have been taken into consideration. All the stopes were modelled in Studio 5 software. The schedule was then completed with EPS software. The results are similar to the one from Golder. It shows that for one independent mining horizon (example: from the 440 to the 230 levels) the average production rate is 1,800 t/d, excluding the beginning and end of the sequence. The maximum rate could be as much as 2,500 t/d per horizon. Therefore, a nominal production rate of 3,500 t/d of ore is a realistic target assuming that at least two independent mining horizons are mined.

For the first two years of production only the upper part of the mine will be in production. Production will be at the nominal rate of 3,500 t/d with the two mining horizons on the 440 and 650 levels. Based on the ground conditions encountered during the proposed exploration program and hydrogeological investigations, a third mining front may be subsequently started on the 230 level (see Section 16.3.2). Ore will be transported to surface via the Gaumond shaft. Currently, no mining is planned above a depth of 55 m to mitigate the risks associated with potential water inflow as per Golder's preliminary recommendations regarding the thickness of the sill pillar (see Section 16.1.5).

In the beginning of the third year, it is expected that sufficient infrastructure will be available to allow exploitation of the 860 and 1040 levels. Exploration drilling from underground will be conducted to test for additional mineralization that may support Mineral Resource estimation and potential conversion to Mineral Reserves. Current results indicate a really high potential in the lower mine. Currently, with the available information and the Mineral Reserve estimate, increasing the production to 7,000 t/d can be achieved and sustained for almost 4 years. Depending of the result of the exploration drilling program, this production rate might be extended. At this stage of the project, it is expected that all the ore and waste of the lower mine (860 level and above) will be trucked up to the 650 loading station. If the Mineral Reserve continues to grow, all the production of the mine (upper and lower mines) could be transferred to the production shaft.

Figure 16-7: Typical level plan view



Lateral development started on 65mLv and 690mLv in Q1 of 2013 when the Gaumond shaft was completed. One team developed 650mLv, and a second team developed the ramp downward from 650mLv level to 1130mLv. The development of mining horizon 2 started from the surface ramp when the levels were reached (380mLv, 400mLv and 440mLv). The ramp break-through, at the end of January 2014, helped increase the development rate. Currently, six teams are excavating on levels 650mLv, 620mLv and 590mLv (mining horizon 3); on levels 440mLv, 400mLv, 380mLv and 350mLv (mining horizon 2); on levels 230mLv and 200mLv (mining horizon1); and finally down the ramp from 650mLv to 1130mLv.

16.3.2. Projected Mine Life

The pre-production period is projected until Q3 of 2014. With the current Mineral Reserve, the mine will have a 10-year life, with production planned from the end of 2014 to the end of 2023. Table 16-2 shows the lateral and vertical development for the pre-production period over the planned life-of-mine (LOM). Figure 16-8 shows the annual ore tonnage planned from development and stopes respectively. Table 16-3 shows the projected ore tonnage and average gold grade per year over the LOM.

Table 16-2: Lateral and vertical development of the pre-production period and the LOM

	PreProduction				Life of Mine (LOM)										Capital			
	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	CAPEX	SUBSTAINING	OPEX	TOTAL
	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)
Horizontal Development																		
Ramp	0	830	1687	1514	1772	1243	0	0	0	0	0	0	0	0	**5 194**	**1 853**	**0**	**7 047**
Long. Drift	0	0	0	0	3 070	5 900	7 602	10 810	10 374	11 372	8 156	6 928	7 616	2 112	**0**	**0**	**73 940**	**73 940**
Transv. Drift	0	0	0	233	5 126	2 853	2 651	1 809	2 421	2 027	0	0	0	0	**3 651**	**0**	**13 470**	**17 121**
Haulage Drift	0	0	0	231	3 876	3 468	2 132	1 749	2 222	1 513	2 411	2 194	2 742	0	**3 368**	**19 170**	**0**	**22 538**
Acces Drift	0	0	0	0	1 534	1 365	1 037	609	731	582	1 715	1 545	2 074	16	**1 302**	**9 906**	**0**	**11 207**
Infrastructure	0	0	0	19	83	112	462	390	10	61	300	296	464	0	**84**	**2 112**	**0**	**2 196**
Total Horizontal Development		**830**	**1 687**	**1 998**	**15 461**	**14 941**	**13 885**	**15 367**	**15 758**	**15 555**	**12 581**	**10 964**	**12 895**	**2 128**	**13 598**	**33 041**	**87 410**	**134 049**
Vertical Development																		
Shaft	63	576	149	0	0	0	0	0	0	0	0	0	0	0	**788**	**0**	**0**	**788**
Vent raise	0	0	0	0	396	38	334	235	508	245	436	0	454	0	**246**	**2 399**	**0**	**2 645**
OP and WP	0	0	0	0	740	467	446	291	1285	0	0	0	0	0	**418**	**2 811**	**0**	**3 229**
Total Vertical	**63**	**576**	**149**	**0**	**1 136**	**504**	**780**	**526**	**1 794**	**245**	**436**	**0**	**454**	**0**	**1 452**	**5 211**	**0**	**6 663**

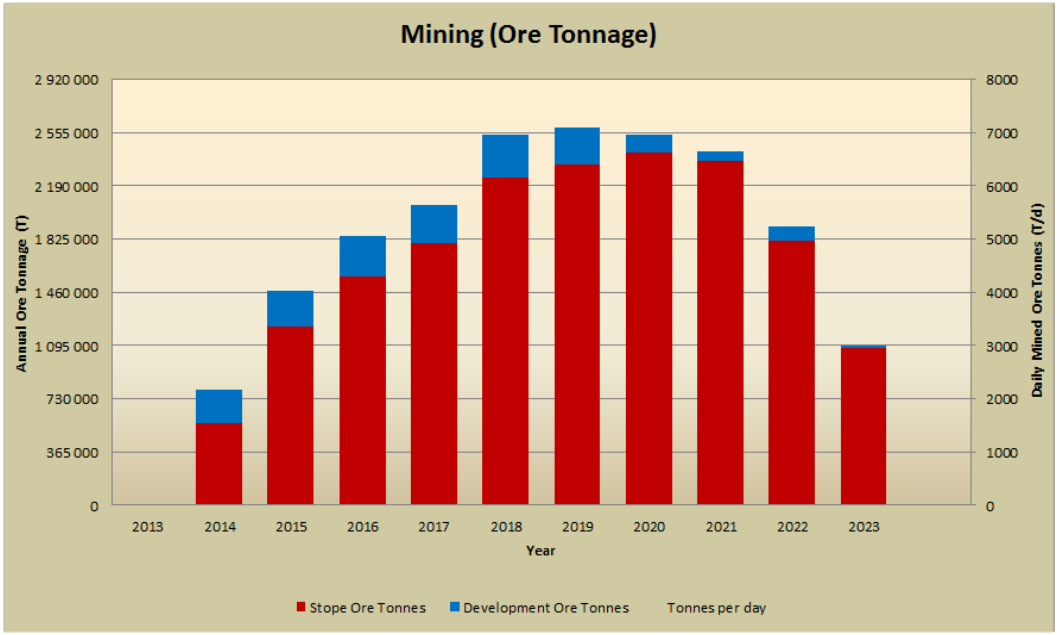


Figure 16-8: Annual ore tonnage projections from development and stopes

Table 16-3: Projected ore tonnage projections from development and stopes

	PreProduction					Life of Mine (LOM)										Capital			
	2010	2011	2012	2013	Jan-Sep 2014	Sep-Dec 2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	CAPEX	SUBSTAINING	OPEX	TOTAL
	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)
Horizontal Development																			
Ramp	0	830	1687	1514	1162	610	1243	0	0	0	0	0	0	0	0	**5 194**	**1 853**	**0**	**7 047**
Long. Drift	0	0	0	0	1148	1323	5 900	7 602	10 810	10 374	11 372	8 156	6 928	7 616	2 112	**1 148**	**0**	**72 792**	**73 940**
Transv. Drift	0	0	0	233	3 418	1708	2 853	2 651	1809	2 421	2 027	0	0	0	0	**3 651**	**0**	**13 470**	**17 121**
Haulage Drift	0	0	0	231	3 137	739	3 468	2 132	1749	2 222	1513	2 411	2 194	2 742	0	**3 368**	**19 170**	**0**	**22 538**
Acces Drift	0	0	0	0	1301	232	1365	1037	609	731	582	1715	1545	2 074	16	**1 302**	**9 906**	**0**	**11 207**
Infrastructure	0	0	0	19	65	18	112	462	390	10	61	300	296	464	0	**84**	**2 112**	**0**	**2 196**
Total Horizontal Development	**0**	**830**	**1 687**	**1 998**	**10 231**	**5 230**	**14 941**	**13 885**	**15 367**	**15 758**	**15 555**	**12 581**	**10 964**	**12 895**	**2 128**	**14 746**	**33 041**	**86 262**	**134 049**
Vertical Development																			
Shaft	63	576	91	0	0	0	0	0	0	0	0	0	0	0	0	**730**	**0**	**0**	**730**
Vent raise	0	0	0	0	246	150	38	334	235	508	245	436	0	454	0	**246**	**2 399**	**0**	**2 645**
OP and WP	0	0	0	0	418	322	467	446	291	1285	0	0	0	0	0	**418**	**2 811**	**0**	**3 229**
Total Vertical	**63**	**576**	**91**	**0**	**664**	**472**	**504**	**780**	**526**	**1 794**	**245**	**436**	**0**	**454**	**0**	**1 394**	**5 211**	**0**	**6 605**

16.3.3. Ventilation

The Éléonore Mine will be equipped with a Ventilation-On-Demand (VOD) system which will be tracking mobile and electrical equipment as well as personnel throughout the underground network. This system will provide an adequate ventilation rate per mobile unit in compliance with the requirements of Québec's Occupational Health and Safety Act. For the nominal production rate of 3,500 t/d, the quantity of fresh air to be delivered will vary from 450,000 f3/m to 620,000 f3/m, as estimated from the planned mobile equipment list for the Life of Mine.

The ventilation system (for the nominal 3,500 t/d) consists of a pull system. The air is pulled from the ramp at which an airlock (SAS doors) is installed near the surface. The air bypasses over an exhaust raise (diameter 7m) where the main exhaust fan (2000 HP) is installed. To reduce the air velocity in the ramp, an exhaust raise is planned to be excavated following the ramp.

Fresh air is provided through the Gaumond shaft where a low pressure fan (750 HP), to pressurize the shaft head frame, is installed. From this main fresh air shaft, two discharge points located on levels 400 and 650 will be distributing the air all over the mine with the support of two internal raises (3 m diameter). Each of those internal raises will be located in the North and South zones.

The ventilation system will have to be modified to raise the production to 7,000 t/d. The additional fresh air will be delivered with the production shaft. The total quantity will vary from 620,000 f3/m to 850,000 f3/m. Two other discharge points will be added on 1020mLv and 1400mLv. Internal raises will continue below 650mLv.

16.3.4. Explosives

Hydrogeological studies indicate that there is a high risk of mining activity intercepting large water flows. As a consequence, emulsion has been selected as the explosive of choice. Initially, material was stored on surface, prior to completion of additional drilling and a trade-off study to determine whether an underground or surface facility is the preferred option for Project operations. In Q2 of 2014, all explosive products will be stored underground in two explosive storage facilities on 400mLv and 650mLv.

16.3.5. Loading

The main loading system for a production rate of 3,500 t/d will be constructed on 690mLv. Mine production will be hauled using 55 t trucks from the ore/waste pass of each zone to loading stations located on 650mLv. The rock will be directly loaded from the ore/waste passes chutes into haulage trucks and then dumped onto a grizzly (fitted with 406 mm (16") square openings) located over the storage bins.

A rock-breaking system will be located over the bins to break oversized rock. The ore bin and the waste bin will each have a capacity of 5,000 t. The bins will feed the rock onto a conveyor that will transport it to the measuring loading box. At that point, the rock will be automatically loaded into the skip. The skip capacity of the Gaumond shaft is 10-12 tonnes.

All the material that will come from under 650mLv will be hauled using the 55 t trucks via the ramp to 650mLv loading station.

When the production shaft will be completed, a second loading station will be built on level 1440mLv for the lower mine production. At this time all the ore/waste will come from the production shaft. The ore/waste from the bins between 650mLv and 690mLV can be easily rerouted to the production shaft. The plans were made by considering this option.

16.3.6. Equipment Fleet

The mobile diesel equipment fleet will consist of 9.5 yd3 and 15.2 yd3 loaders, 45 tonnes, 55 tonnes and 60 tonnes dump trucks, mine service and personnel vehicles, fully automatic jumbo drills, bolting platforms, scissor lifts, forklifts, boom trucks, and utility trucks. Top hammer drills will be used to drill 100 mm (4”) diameter holes in the stope. Table 16-4 shows the equipment requirements to support the planned 3,500 t/d nominal production rate and the increased production rate at 7,000 t/d.

Table 16-4: Required equipment fleet to support 3,500 t/d and 7,000 t/d production rate

Equipment		Model	Quantity		
			3500 t/day	adding 3500 to 7000 t/day	total at 7000 t/day
Camion	CAT	AD45	0	1	**1**
Camion	CAT	AD55	4	-1	**3**
Camion	CAT	AD60	0	2	**2**
Chargeuse	CAT	R1600G	2	0	**2**
Chargeuse	CAT	R2900G	4	3	**7**
Jumbo	Atlas Copco	M2C; M2D	4	2	**6**
Production Drill	Atlas Copco	SIMBA	2	1	**3**
Bolter	Maclean	MEM-928	7	4	**11**
Scissor Lift	Maclean	SL3	3	3	**6**
Backfill pipe scissor lift	Maclean	SL3	1	0	**1**
Boom truck	Maclean	BT3	4	2	**6**
Cassette Carrier	Maclean	CS/	1	1	**2**
Shocrete machine	Maclean	SS-3	1	0	**1**
Telehandler	CAT	TH255	1	2	**3**
Cement mixer U/G	CAT	725	1	1	**2**
Transport	Toyota	Landcruiser	10	8	**18**
Grader	CAT	12M	1	1	**2**
Back hoe	CAT	420F	1	0	**1**
Loader	CAT	962	1	0	**1**
Loader	John Deere	344J	1	0	**1**
Excavator	CAT	303	0	1	**1**
Tractor	John Deere	6100D	0	1	**1**
Emulsion loader PROD	Mclean	EL-3	1	0	**1**
Emulsion loader DEV	Mclean	EL-3	1	0	**1**
Mobile rock breaker	Mclean	SB-9	1	0	**1**
Block Holer	Mclean	BH-3	1	0	**1**

16.3.7. Dewatering Considerations

The permanent pumping system is design to be upgradable depending of the total water infiltration in the mine and also the mine plan. The system is designed to pump dirty water to the stations above and finally reach the surface. The first phase of the permanent pumping system will start in Q2 of 2014. It consists of two main pumping stations (400mLv and 650mLv). All the water collected above 400mLv will be redirected to the main pumping station on 400mLv. The water collected below 400mLv will be redirected to the main station on 650mLv and pumped to the 400mLv station. Finally, all the water will reach the surface from 400mLv via a pipe in the Gaumond shaft (see figure 16-9). The pumps capacity on level 400mLv are 11,000 m³/day (2000 gallons/minute) and 5,500m³/day (1000 gallons/minute) on level 650mLv.

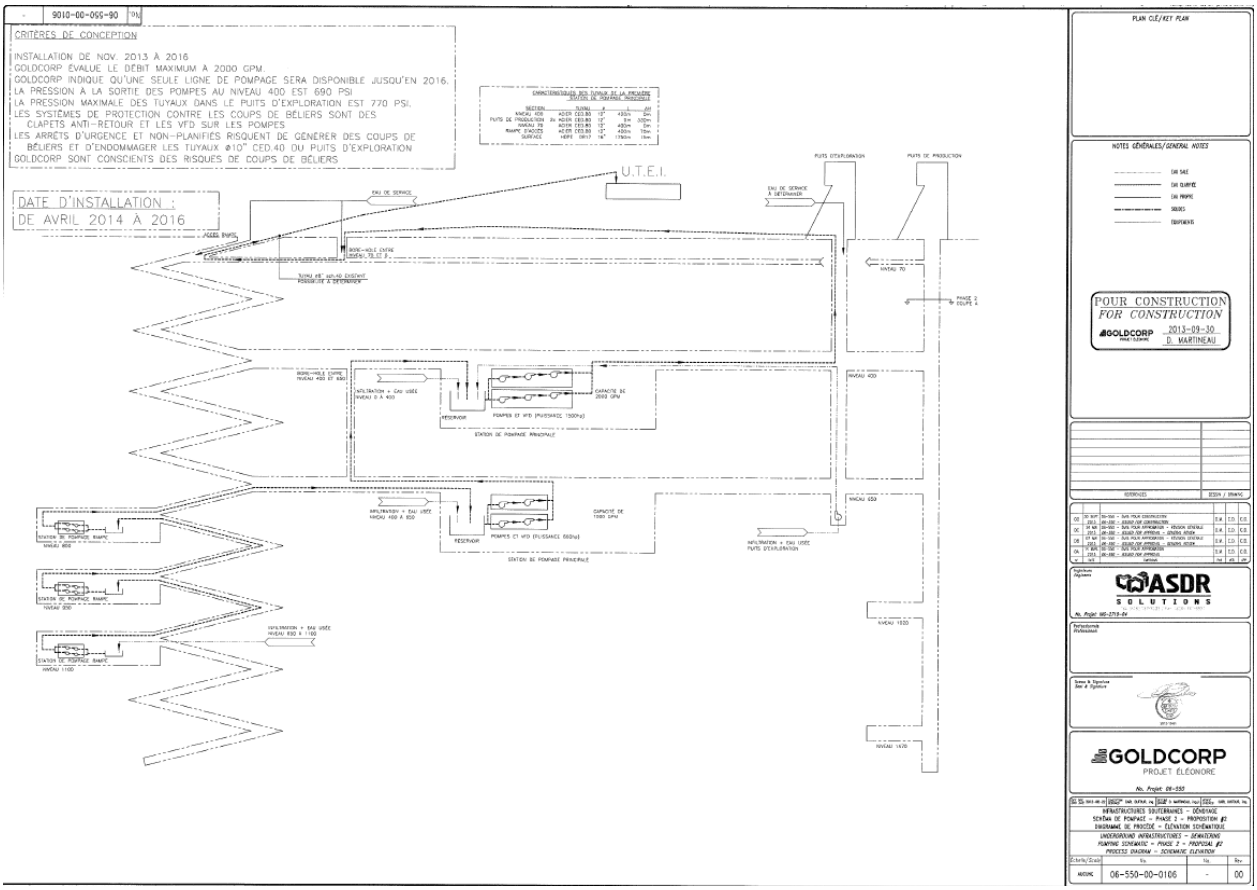


Figure 16-9: Pumping system, Phase 1

The second phase will start when the production of the lower mine will start. At that time, another pumping station will be installed on level 1020mLv. The pump capacity will be 5 500 m³/day (1000 gallons/minute). The system will be the same as the one in Phase 1; the only difference is the water reaching the surface from 400mLv via the production shaft. (Figure 16-10). Table 16-5 provides the pump characteristic.

Finally, in every main pumping station (400, 650 and later 1020) the number of pumps will double. The system is designed to pump in line for a period of time. After a certain amount of hours, the system will be automatically redirected to the other line of pumps. This means that every pumping station has two identical sets of pumps.

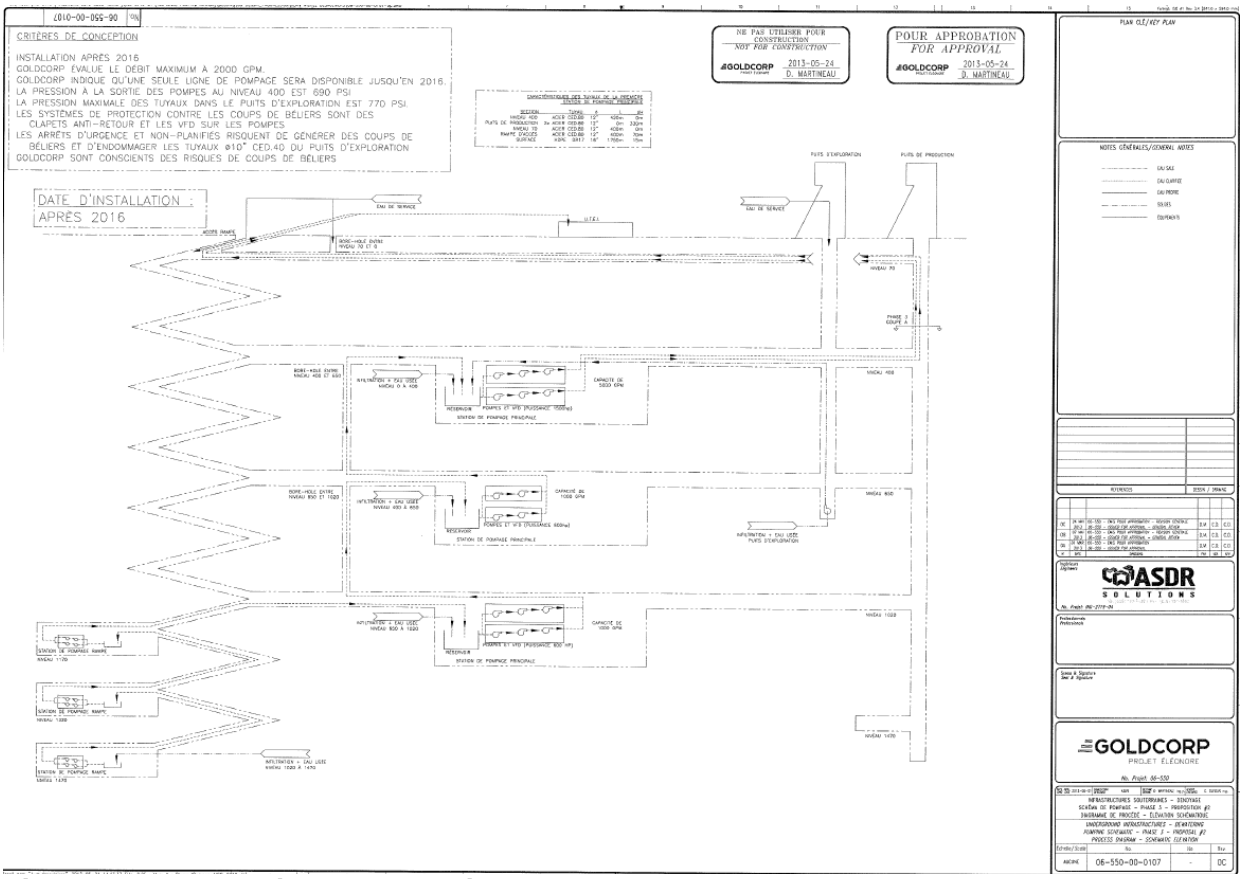


Figure 16-10: Pumping system, Phase 2

Table 16-5: Pump characteristics and their locations

	Level	Pump Description	Quantity	Total Pumping Capacity
Phase 1	400	1500 HP	3	2000 USGPM
	650	600 HP	2	1000 USGPM
Phase 2	1020	600 HP	2	1000 USGPM

16.3.8. Communications System

A Leaky feeder system is the chosen communication system for verbal underground communication. A Wi-Fi system will be available in some areas. Every level will also be equipped with optic fibre in specific areas to support remote activities. Currently the following automated systems have been set up:

- Ventilation-On-Demand (VOD)
- Locating system for workers and equipment;
- Real-time data acquisition from production equipment.

The next steps are:

- Remote control LHV with a mobile and surface control station;
- Paste line instrumentations

16.3.9. Backfill

The excavation will be backfilled with paste fill. On average, 2500 tons per day of paste backfill will be needed to meet the production targets of 3,500 t/d from two mining fronts. The pipeline will have a diameter of 203 mm (8 in) (still in design phase). At 3500 t/d, the mill can fill only one line at a time. When production will reach 5,500 t/d, a second line will be available. The paste plant was designed to provide paste to two different lines at the same time even with two different compositions. Depending on the pressure in the line, the pipe thickness will vary between schedule80 and schedule120 (still in design phase). Figure 16-11 shows the different network depending of the elevation. The ramp section from 40mLv to 140mLv is the most critical part. The pressure will be the highest (1400 to 1800 psi depending of the stope to fill). The installation of the pipes will start at the beginning of Q2 in 2014. The QPs note that if production rate increases are contemplated, a second paste backfill line will be needed.

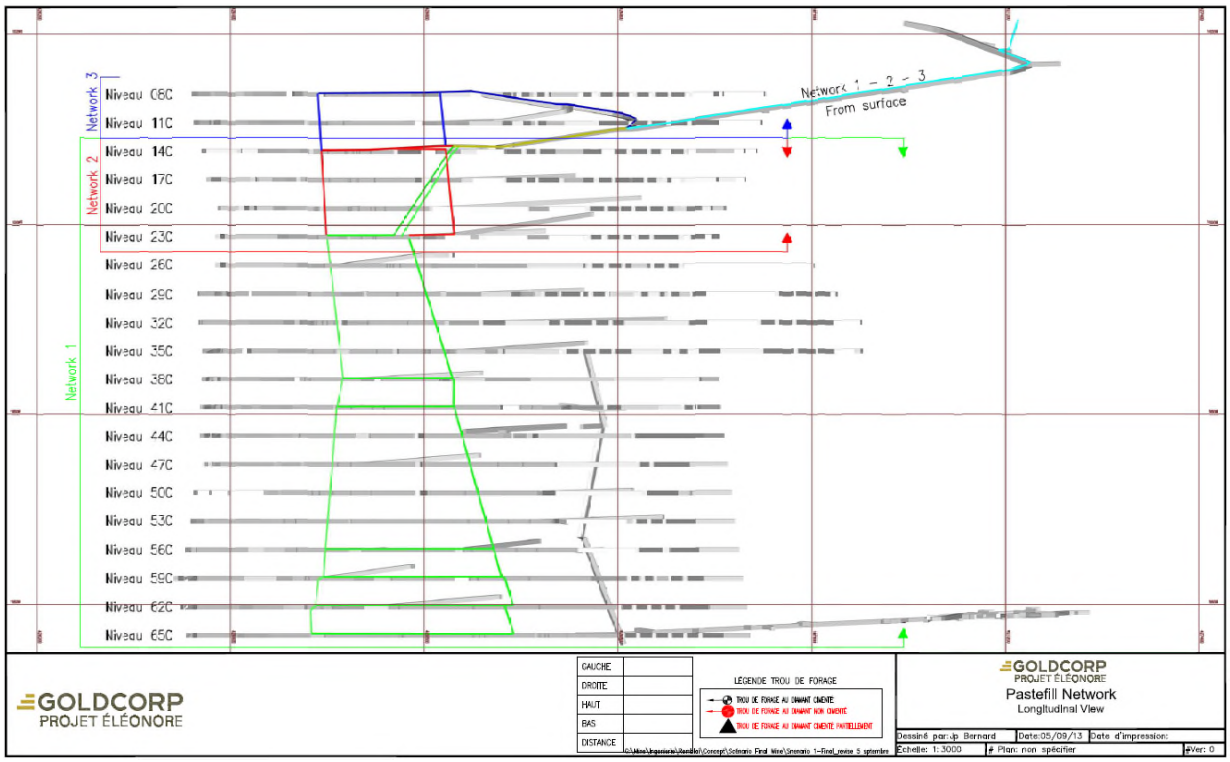


Figure 16-11: Paste backfill networks

The paste backfill mixture will consist of 67% mill tailings, 17.5% -½" aggregate, 12% fine sulphide concentrate, and 3.5% binder. If no aggregate is used in the paste fill, then the percentage weight of the binder will be 5%.

These mixtures are expected to have unconfined compressive strengths (UCS) of 0.5 MPa after 28 days.

16.4. External Dilution

Understanding that no actual stopes have been mined out yet, the approach of external dilution is based on stope thickness (from hanging wall to footwall) and drill and blast experience. It is a logical option that a narrow vein will create relatively more dilution compared to a thicker stope.

As an example, adding 0.3 m to each side of a 3 m thick stope will add 20% dilution compared to 7.5% for an 8 m thick stope. According to the 2014 LOM, the average stope thickness is 4.7 m by number of stopes but 7.2 m by tonnes.

Table 16-6: External dilution in the LOM reserves

Stope Thickness	(m)	3	4	5	6	7	8	9	11	13	15	19
External Dilution	(%)	24	21	16	14	13	11	10	8	7	6	5

16.5. Production Reconciliation

On a monthly basis, the total tonnage of mined (broken) rock will be calculated from excavated volumes. Total ounces will be reported from the mill, and adjusted to include the inventory in ore passes, bins, surface stockpiles, and the rock remaining in the stopes. Ounces will be assigned to the various mined stopes, based on information gathered during the month. For every blast in ore, random muck samples will be collected and analyzed. This assay information will be summarized for each blast/stope and used as a guide in assigning ounces back to the stopes. Cavity Monitoring System (CMS) will be used in the stope to reconcile the tonnage of broken rock versus the planned stope geometry. This will be an essential tool for reviewing external dilution.

16.6. Production Schedule

The proposed mine life, based on the 2013 Mineral Reserve estimate is approximately 10 years, from the end of 2014 to the end of 2023. Figure 16-3, Tables 16-2 and 16-3 included in Section 16.3.2, show the projected mine life production tonnage.

16.7. Comment on Section 16

In the opinion of the QPs, the following comments are appropriate:

- The planned nominal throughput rate of 3,500 t/d is appropriate for the style of the mineralization with two different mining horizons.
- With the delineation of additional Mineral Resources, it is possible to add more than the currently planned mining horizons. Goldcorp's team has demonstrated that a production rate of 7,000 t/d is possible. It will be necessary, however, to increase the Mineral Resources and Mineral Reserves to reach and sustain this rate. And any increase in throughput will require an additional backfill line.
- Stope designs are appropriate for the thickness of the mineralization.
- Geotechnical considerations have been appropriately assessed.
- Water management is critical for Project success, as the orebody is located directly under the Opinaca Reservoir. Mining will not take place within 55 m of the surface due to the presence of the reservoir and open subhorizontal decompression joints mainly encountered within the first

150 m below surface. Additional studies are planned.

- The mine dewatering system is design to be easily upgradable. A permanent system will be in operation shortly and will provide a maximum rate of 11,000 m^3/d.
- Mining equipment selection was based on the mine production schedule and equipment productivities, and included consideration of workforce and operating hours. The fleet is appropriate for the planned production schedule.

17. RECOVERY METHODS

17.1. Process Flowsheet Summary

The plant capacity is controlled by the underground mine capacity. The operation will initially operate at 3,500 t/d. The crushing area is designed for a capacity of 8,500 t/d including waste crushing (1,500 t/d), and the other plant areas were designed for a processing capacity of 7,000 t/d.

The process plant availability was established at 95% based on performance of similar operations with the same type of comminution circuit. The crushing plant will provide a -11 mm feed to the grinding circuit and will also crush waste material for addition in the paste backfill mix.

The comminution process consists of three stages of crushing followed by a single stage ball mill grinding. The primary crusher (jaw crusher), the secondary crusher (standard cone crusher) and tertiary crushers (two short-head cone crushers will be required for a 7,000 t/d throughput rate) will be located on the surface. The fine crushed ore will be ground by a single stage ball mill in a closed circuit with cyclones. A portion of the cyclones underflow will be directed to a gravity concentration circuit consisting of two Knelson Concentrators and an Acacia Reactor to recover liberated native gold. The cyclones overflow (grinding circuit product) will be directed to the flotation cells for iron sulphides separation in a low mass sulphur concentrate. The flotation tail slurry density will be adjusted with a thickener and the material is leached with cyanide for 36 hours in five (5) leach tanks.

The flotation concentrate will be thickened and reground to a P80 of 10-15 µm in a fine grinding mill and will be leached with cyanide for 48 hours in five (5) other leach tanks. The gold in solution will be recovered in carousel carbon-in-pulp (CIP) circuits (one for each leach circuit). The carbon from each CIP circuit will be stripped as required in a Zadra stripping circuit and the gold recovered will be poured in gold bars at regular intervals. The carbon will be regenerated and returned to the CIP circuits. The tails from each leaching circuit will be detoxified in a conventional cyanide destruction circuit, and will be filtered for addition in the paste backfill or stored in a covered shed before transportation to the tailings management facility.

A schematic flowsheet of the proposed process is included as Figure 17-1.

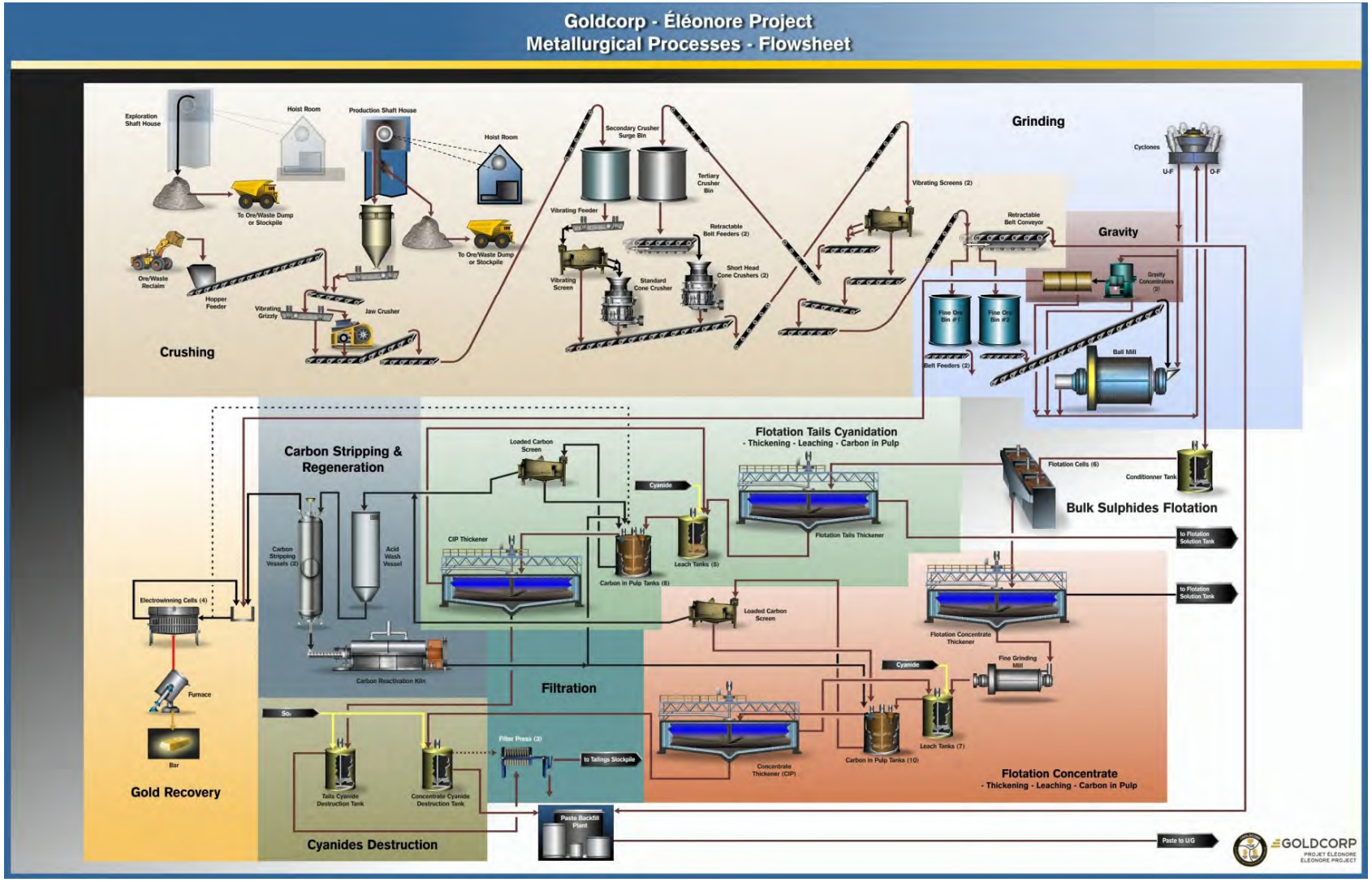


Figure 17-1: General process plant flowsheet.

17.2. Crushing Circuit

The raw material with 450 mm (F80) size will be reclaimed from underground at a feed rate of approximately 220 t/h (skip capacity of the two shafts). The material in the surface bin will be reclaimed with a variable speed apron feeder at a controlled feed rate (depending on the effective crusher system capacity) to achieve 8,500 t/d in 16 hours or less of operation per day (7,000 t of ore and 1,500 t of waste). The reclaimed material will be fed to the 1,200 mm wide conveyor that will discharge ore into the primary crusher feed hopper. Before feeding into the jaw crusher, material will pass through the vibrating grizzly feeder that bypasses approximately 40% of the crusher feed.

The grizzly feeder undersize and the jaw crusher product will be collected on a common conveyor to transfer material to the secondary crusher surge bin. The ore will be reclaimed and fed to a double deck vibrating screen (76 mm and 22 mm openings). The oversize will discharge to the secondary cone crusher that will be equipped with a standard/fine cavity (HP6 with a 450 kW motor). The secondary crusher product and the secondary crusher screen undersize will be collected onto a common conveyor with the tertiary crushers product and transferred to the tertiary crusher double deck vibrating screens (32 mm and 11 mm openings).

The screen oversize will be collected by a conveyor and loaded into a surge bin with two retractable variable speed belt feeders. The feeders will feed the tertiary cone crushers with short head/fine cavity (HP6 with a 450 kW motor). The tertiary crushers product will be recycled back to the tertiary crusher screens. The tertiary crusher screen undersize product at -11 mm will be conveyed and stored in two 3,500 tonnes ‘‘live’’ fine ore bins or into the 2,000 tonnes waste bin enclosed to prevent freezing. These bins will be located at the process plant and will have more than 24 hours of storage capacity.

17.3. Grinding Circuit

The ore stored in the fine ore bins will be reclaimed via two variable speed belt feeders and loaded at a controlled feed rate into the ball mill (6.40 m x 11.9 m with a 9000 kW motor) operating in closed circuit with a cyclone pack that will control the grinding circuit product size. The cyclone pack will have 19 cyclones of 400 mm diameter with 16 in operation and three on standby. The cyclone pack overflow at P80 of 65 µm will be the final grinding circuit product and will be sent to the extraction process. The underflow of the cyclone pack will be recycled to the ball mill and is reground until it is fine enough to go in the overflow of the cyclone pack.

17.4. Gravity Circuit and Intensive Cyanidation

A portion of the secondary ball mill cyclone underflow will be diverted and fed to the gravity concentration circuit. A scalping screen will eliminate coarse solids and ball scats and the screen undersize will feed two gravity centrifugal concentrators (Knelson QS40 with 40" diameter bowl). The tailings from the centrifugal concentrators and the scalping screen oversize will be returned to the ball mill discharge pump box while the concentrate will be flushed periodically to the intensive leaching circuit located underneath.

The gravity concentrate will be accumulated for one day and will be processed in batches using an intensive cyanidation dissolution module. The dissolved gold in solution will be pumped to a gravity pregnant solution tank situated in the gold room for subsequent electrowinning. The gold sludge will be periodically recovered and smelted to pour quality gold bars. The rejects of the intensive leaching circuit will be flushed to the grinding circuit pump box.

17.5. Flotation Circuit

The ground product from the cyclone pack overflow will be passed over a linear trash screen to remove wood pieces, plastic and other small trash material which would otherwise plug the carbon screens in the CIP circuits. It will then enter the flotation conditioner (4.4 m x 5.0 m) with 10 minutes of retention time where it will be conditioned with PAX and R208 collectors to float the sulphides. The conditioner overflow will follow through to the flotation circuit with 40 minutes of retention time. The flotation circuit will be made up of six flotation tank cells of 130 m^3 capacity each. When transferred to the flotation cells, frother (MIBC) will be added to the feed to produce a stable froth and recover 95%+ of the sulphides. The sulphide concentrate (approximately 10% weight recovery) will be transferred to the flotation concentrate cyanidation circuit with two vertical tank pumps (one operating and one standby). The flotation tails will be pumped to the flotation tail cyanidation circuit using horizontal slurry pumps (one operating and one standby).

17.6. Flotation Tails Leaching and CIP Circuit

From the flotation circuit, the flotation tailing will be pumped to the flotation tails high- rate thickener to thicken the slurry to 65% solids. The thickener overflow water will be recycled to the process water tank for reuse and the thickener underflow will be pumped to the flotation tails leaching circuit. There will be five leach tanks operating in series.

The leach feed density will be adjusted by recycling the overflow from the flotation tail CIP thickener and process water to operate at 45% solids. The leaching feed (flotation tails thickener underflow) pH will be adjusted with lime to approximately

10.5 to prevent HCN formation as a safety precaution. Cyanide will be added to the first and third leaching tank and the cyanide addition is controlled with a cyanide analyzer. The flotation tails leaching train will have a total retention time of 36 hours. The slurry will flow by gravity to each tank in steps of 600 mm. Each tank will be equipped with an agitator to maintain the solids in suspension and air will be injected to promote the gold dissolution rate. Interconnecting tank launders will be arranged so that any tank in series can be bypassed without the whole plant having to be shutdown.

After leaching, the slurry will be transferred to the tail CIP circuit. The gold will be adsorbed onto carbon in a Kemix AAC pumpcell system. The adsorption circuit will have six tanks of 120 m^3 each in series with 15 minutes retention time each and will contain a total of 36 t of carbon. The pumpcell mechanism will combine the functions of agitation, interstage screening and slurry transfer in one unit. The suspension of the carbon and slurry mixture will be maintained by the hydrofoil mixer. The interstage screen surface will be kept free of carbon build-up by means of wiper blades attached to the rotating cage. The pumping action of the interstage screen will generate a head differential and the contactors can be placed at the same level. With the contactors being at the same level it will become relatively simple to provide common feed and tailings launders to allow the plant to operate in a carousel mode. This mode of operation has several advantages compared to conventional counter-current CIP circuits:

- Higher carbon loading capability;
- Lower carbon inventory;
- Lower contacting volume;
- No back mixing of slurry during carbon transfer;
- Equal contact time of all carbon (no short circuiting of carbon);
- Carbon level in tank is constant;
- Smaller elution and carbon regeneration circuit required; and
- Lower operating and capital cost.

The tanks will be fed by a launder which, via a series of plug and gate valves, will allow the feed slurry to be diverted to the different head tanks. When the carbon in a head tank has reached the required gold on carbon loading, this tank will be isolated from the adsorption sequence and the loaded carbon will be separated from the slurry by pumping the entire content of the tank over a vibrating screen to recover the carbon for the elution circuit. The screened slurry will flow back to join the adsorption circuit feed. The design carbon concentration in the adsorption tanks will be 50 g/L which will be within the normal pumpcell operating range of 30–60 g/L for a total of 2.5 t of carbon per tank.

After the adsorption circuit, the flotation tails stream will be pumped to the

flotation tail CIP thickener (16 m diameter high rate thickener) through a carbon safety screen for cyanide solution recovery. The thickener overflow will be recycled to the leach feed while the underflow will be sent to the flotation tails cyanide destruction system.

17.7. Flotation Concentrate Leaching and CIP Circuit

The flotation concentrate will be pumped to the flotation concentrate thickener (9.0 m diameter high rate thickener) where it will be thickened to 55% solids. The thickener overflow water will be recycled to the process water tank for reuse and the thickener underflow will be pumped with peristaltic pumps (one operating and one standby) to the IsaMill feed pump box.

The IsaMill (M5000 with an 1,500 kW motor) will be fed with the flotation concentrate thickener underflow and grinding media will be added as required in the IsaMill feed pump box to keep the power input constant to the regrind mill. The flotation concentrate will be reground to a P80 of 10–15 µm before being discharge into a second pumpbox where dilution water recycled from the concentrate CIP thickener overflow and process water will be added to reduce the leach feed density to 30% solids. The IsaMill product will then be pumped to the first of two pre-aeration tanks to pre-condition the slurry for leaching through five flotation concentrate leaching tanks (7.9 m diameter x 8.4 m).

The pH in cyanidation will be maintained at approximately 10.5 as a safety precaution to prevent HCN formation. Lead nitrate and oxygen will be added to reduce the sulphide activity. Cyanide will also be added to the first and third leaching tank and the cyanide addition will be controlled with a cyanide analyzer. The flotation concentrate leaching train will have a total retention time of 48 hours. The slurry will flow by gravity to each tank in steps of 400 mm. Each tank will be equipped with an agitator to maintain the solids in suspension and oxygen will be injected to promote gold dissolution rate. Interconnecting tank launders will be arranged so that any tank in series can be bypassed without the whole plant having to shutdown.

After leaching, the slurry will be transferred by gravity to the concentrate CIP circuit. The gold will be adsorbed onto carbon in a Kemix AAC pumpcell system. In order to achieve the high gold loading on carbon and due to solution concentrations, the concentrate CIP system has been designed to have a 30 minute contact time in each pumpcell tank. The gold loading on carbon is expected to be as high as 25 kg/t of carbon. The adsorption circuit will have ten 30 m^3 tanks in series for a total of 15 t of carbon. The tanks will be fed by a launder which, via a series of plug and gate valves, will allow the feed slurry to be diverted to the different head tanks. When the carbon in a head tank has reached the required gold on carbon loading, this tank will be isolated from the adsorption

sequence and the loaded carbon is separated from the slurry by pumping the entire content of that tank over a vibrating screen to recover the carbon for the elution circuit. The screened slurry will flow back to join the adsorption circuit feed. The design carbon concentration in the adsorption tanks is 50 g/L which is slightly above the normal pumpcell operating range of 30–60 g/L for a total of 1.5 t of carbon per tank.

After the adsorption circuit, the flotation concentrate stream will be pumped to the concentrate CIP thickener (12.0 m diameter high rate thickener) through a carbon safety screen. The thickener overflow will be recycled to the flotation concentrate leach feed for dilution and the thickener underflow at 40% solids will be transferred to the cyanide destruction system.

17.8. Carbon Elution Circuit and Carbon Regeneration

The loaded carbon recovered from the two CIP adsorption circuits will be pumped from the loaded carbon surge tanks to the acid wash tank with a capacity of 6 t of carbon. A dilute solution of 2% nitric acid will be pumped into the acid washing vessel from the bottom and returns to the acid holding tank while the pH is monitored. After the acid wash cycle is completed, the spent acid will be neutralized by adding caustic, drained and pumped to the tailings pump box for disposal.

An exhaust fan will be connected to the acid wash pump box and the acid wash vessel and is used to remove fumes. After the acid wash treatment, the carbon will be rinsed with water. The carbon will then be rinsed again with a small amount of caustic to ensure neutralization. The carbon will next be pumped to the carbon stripping vessel. In the stripping vessel, the gold will be desorbed from the carbon by circulating a caustic-cyanide strip solution at high temperature (140ºC) and pressure (550 kPa) using the Zadra stripping process.

The gold-loaded strip solution, also called pregnant solution, will be cooled with heat exchangers to about 80ºC and accumulated into a pregnant solution tank. Reagents (caustic and cyanide) will be added as needed to the strip solution to have the correct chemistry and conductivity for the carbon stripping. At the end of the elution cycle, the carbon will be rinsed with fresh water and pumped to the carbon reactivation system. After stripping, the carbon slurry pumped from the stripping column will feed a dewatering screen ahead of the reactivation kiln. The water used to pump the carbon slurry to the screen will drain to the quench tank. The dewatered carbon from the screen will be stored in a 6 t surge bin in front of the kiln which will ensure a steady feed during kiln operation. A steam-rich atmosphere will be maintained in the kiln to prevent the carbon from charring. The kiln will discharge into a quench tank filled with water to simultaneously cool and wet the carbon. The kiln will be electrically fired and will have a regeneration

capacity of 6 t per day. The reactivated carbon batch will be pumped to the pumpcell systems after the carbon extraction in the pumpcell is completed. Fresh carbon required to make up for losses of fine carbon must be conditioned prior to use to remove fines, sharp edges and to thoroughly wet the particles. This will be achieved in an attrition tank.

17.9. Electrowinning and Refining

The gold-loaded strip solution, also called pregnant solution, is cooled with heat exchangers to about 88ºC and accumulated into a pregnant solution tank. The pregnant solution will be pumped into the electrowinning cells. Each elution circuit (Flotation Concentrate and Flotation Tails) has its own electrowinning circuit to feed. The pregnant solution is pumped into series of two (2) electrowinning cells in each circuit. The gold in solution will precipitate and adhere to the cathode which will be made of woven-mesh stainless steel. The barren solution will go to the barren solution tank and will be pumped back to the elution column passing through the in-line heater for re-heating to 143ºC. The electrowinning in the gold room is done with two (2) electrowinning cells per circuits. Additionally, a separate electrowinning cell is used to recover the gold in the pregnant solution from the gravity-intensive cyanidation system. There are five (5) cells in total.

The loaded stainless steel cathodes and the sludge accumulated at the bottom of the cell will be cleaned in-situ with a high-pressure washer. The sludge will be directed to a holding tank ahead of a sludge pump feeding a recessed plate filter for filtration. The filtered solids will be discharged from the plate filter in trays and dried in the mercury retort. Because of safety issues, a mercury retort was added in place of a simple drying furnace for the drying of the gold sludge from the electrowinning cells. The dry solids will be cooled and mixed with an appropriate amount of flux and refined. The refining furnace provided will be an induction furnace. Refined gold will be poured to a series of moulds and the slag will be poured into slag moulds.

The slag residue will be processed in a small gravity system comprising a jaw crusher, a cone crusher and a small Knelson concentrator. The tail from this small recovery circuit will be recycled to the grinding circuit.

17.10. Cyanide Destruction

Two cyanide destruction systems will be required for the flotation concentrate stream and the flotation tails stream. The selected cyanide destruction system is the Inco SO_2/air with oxygen. The thickened slurries from their respective CIP thickeners will be pumped to their respective cyanide destruction tank where process water will be added to adjust the slurry density to the operating level. The

systems use SO_2 to destroy the cyanide and oxygen will be sparged into the tanks from the oxygen plant. Copper sulphate will be added as needed to catalyze the cyanide destruction reaction and has been incorporated into the design. The target weakly acid dissociable cyanide (CNwad) content at the output of the destruction system will be <15 ppm. After the cyanide destruction process, the tails streams will be routed to their respective thickeners.

17.11. Filtration Plant

The slurry from cyanide destruction of flotation tails will be fed to the non-sulphide tailings thickener with addition of fresh water and flocculant. The thickener overflow will be pumped to the process water tank. The thickener underflow will be pumped to the non-sulphide tailings filters surge tank. The non-sulphide tailings filters surge tank slurry will be pumped to the pressure filters (two operating, one standby). The filter cake will fall to reversible conveyors either sending cake to the paste backfill plant or to the enclosed non-sulphide tailings stockpile.

The slurry from cyanide destruction of flotation concentrate will be fed to the sulphide tailings thickener with addition of fresh water and flocculant. The thickener overflow will be pumped to the process water tank. The thickener underflow will be pumped to the sulphide tailings surge tank. When the paste backfill plant is in operation (60.8% of the time), the slurry from the surge tank will be pumped to the paste backfill plant. When the surge tank capacity is exceeded, the slurry will be pumped to pressure filter #1. The filtrate will go to the non-sulphide thickener feed tank. The filter cake will be conveyed to the emergency sulphide tailings stockpile.

17.12. Paste Backfill Plant

The filter cake will be convoyed to the filter cake bin. The filter cake multi-screw feeders will then feed a conveyor that will lead to the two paste mixer hoppers (67% of the paste). The sulphide tailings slurry from the surge tank will be pumped to the paste mixer hoppers (12% of the paste). Tailings from the sulphide stockpile will be reclaimed to a transfer bin prior to feed the paste mixer hoppers (3.25% of the paste).

The crushed waste will be fed by vibrating feeders to the two crushed waste conveyors which leads to the paste mixer hoppers (17.5% of the paste). Cement will be fed to the paste mixer hoppers from a silo by screw feeders (0.3% of the paste). Blast furnace slag will be fed to the paste mixer hoppers from a silo by screw feeders (2.3% of the paste). Process water will be added to the mixers to adjust paste density. The paste mixers (two operating) will feed the paste hoppers which will in turn feed the piston pumps (one by mixer). Each piston pump will

send paste to underground through its individual pipeline. Finally, tails will either be added to the paste backfill or stored in a covered shed before being transported to the tailings management facility.

17.13. Fresh and Process Water Supply

The plant water balance is presented in Figure 17-2 in cubic meters per day. This represents an average operating day with 95% availability of the process plant. Fresh water makeup comes from the polishing pond after the water treatment plant.

The process water tank will be received water recycled from the flotation products thickeners, from the stripping circuit heat exchanger and from the paste backfill plant. As required, water is pumped back from the clarification pond after the water treatment plant to complete the process water plant needs. Process water will be distributed in the process plant through two process water pumps (1 operating and 1 standby) at a system pressure of 350 kPa (50 psi). A total process water flow rate of 453 m^3/h is estimated to be required to feed the various process areas like the grinding mills, the cyanidation areas and the cyanide destruction system. There is an excess of process water of 43 m^3/h, which must be bled to the water treatment plant to maintain the plant water balance.

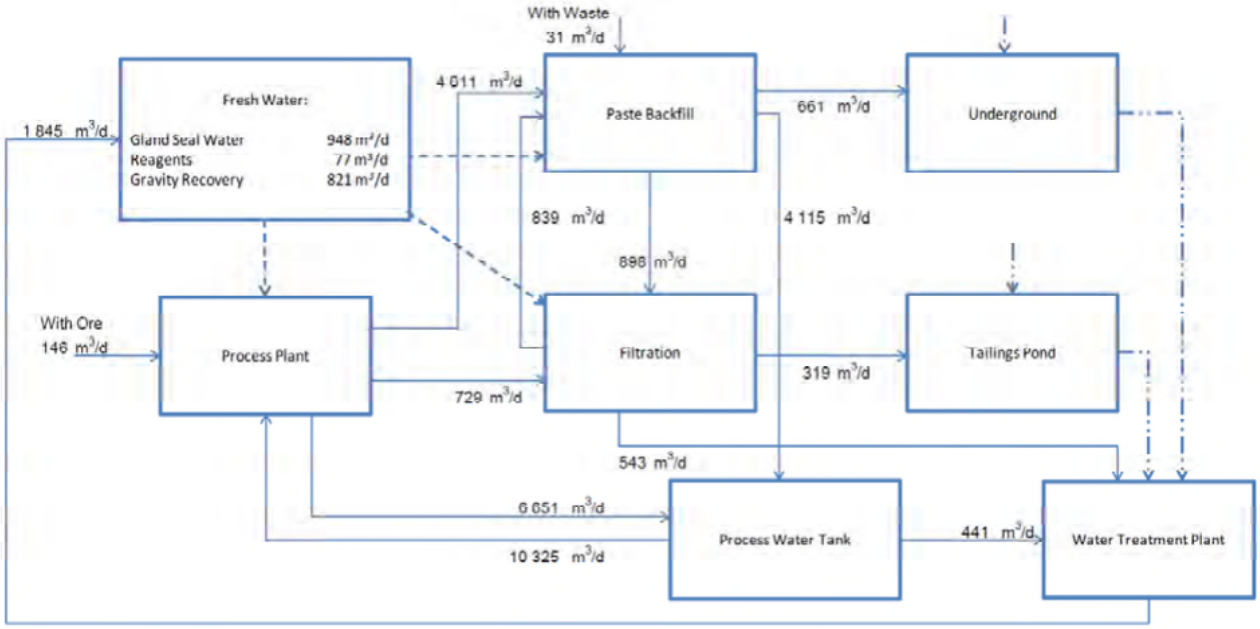


Figure 17-2: Plant water balance

Fresh water will be used in the process plant to prepare reagents and to provide clean gland seal water for the slurry pumps. Clean water is also required for the Knelson gravity concentrator. The fresh water source is the treated water from the water treatment plant. Slurry pumps requiring higher pressure gland seal water will be supplied by two gland seal water booster pumps (one operating and one on standby) increasing the water pressure from 350 kPa to 860 kPa (50 psi to 125 psi). The process plant fresh water requirement is estimated at 42 m^3/h for gland seal water, 36 m^3/h for the Knelson concentrator, and 3.4 m^3/h for reagent preparation and dilution at a system pressure of 350 kPa (50 psi).

17.14. Comments on Section 17

In the opinion of the QPs, the following interpretations and conclusions are appropriate:

- The Éléonore Project will use conventional mineral processing equipment to produce a marketable gold doré.
- The process flowsheet is standard, consisting of three stages of crushing, grinding, gravity concentration, sulphides flotation, cyanide leaching, and gold recovery in a carbon-in-pulp (CIP) circuit.
- The plant is designed to achieve 7,000 t/d for an annual throughput of 2.5 Mt/a and operate for 365 days a year. The precious metals recovery circuit is designed to produce an average of approximately 600,000 ounces of gold annually.
- The designed mill can achieve the recoveries seen in laboratory testwork, which are projected to be in the range of 93.0–93.5% over the LOM.
- The process plant will use electricity as energy, supplied by Hydro-Québec. The power requirement at 7,000 t/d will be about 30 MW.
- The main reagent to recover the gold will be cyanide with several other reagents such as quicklime, NaOH, flotation collectors, SO_2, and copper sulphate. The residual cyanide will be treated before sending it to the tailings management facility and the remaining residual reagents will be treated in the water treatment plant.
- The process plant will use more than 90% of the process recycle water with the filtration tailings approach. The remaining 10% will be fresh water from the water treatment plant. The water management plan will follow industry best practices.

18. PROJECT INFRASTRUCTURE

The main infrastructure will include the Gaumond shaft and the production shaft, a surface ramp, a waste rock storage dump, tailings storage facilities, a process plant, offices, and a permanent camp. Figure 18-1 shows an aerial schematic of the proposed infrastructure layout.

18.1. Underground Infrastructure

18.1.1. Gaumond/Ventilation Shaft

The circular shaft will have an inside diameter of 7 m and will be lined with concrete to a final depth of 715 m. In addition to providing access for underground exploration, the Gaumond shaft will be used to develop the upper levels. The Gaumond shaft will be equipped with two 10- to 12-tonne skips. A cage will be fitted below one of the skip to transport personnel and material from surface to the working levels.

The main hoist will be used for both sinking and hoisting during the pre-production development period and at the beginning of the production period from the upper levels. The Gaumond shaft will have a total hoisting capacity of 7,000 t/d, sufficient to handle both ore and waste during the first two years of production. Ore production will be at a nominal rate of 3,500 t/d.

The main hoist will be located at surface. It is 3.66 m (12 ft.) in diameter and will be powered by two (2) 867-kW (1.162 hp) motors. The hoist rope is 44.5 mm (1.75 in.) in diameter. This hoist is adequate for sinking to a depth of 900 m.

A Maryann auxiliary cage is installed in the Gaumond shaft for emergency egress. The Maryann arrangement consists of a double-deck cage with a total capacity of eight people.

A 2.4 m (8 ft) diameter hoist, powered by a 510 kW (684 hp) motor will be required for the Maryann cage. In case of a Hydro-Québec power failure, a back-up generator will supply electrical power to this hoist to ensure that the Maryann remains operational at all times for emergency evacuation. The hoist rope for the Maryann cage is 25.4 mm (1 in) in diameter and the hoisting speed will be 7.62 m/s (1,500 ft/min).

Two shaft stations will be excavated on 400mLv and 650mLv. The loading station will be located between 650mLv and 690mLv (see section 18.1.4).

Finally, fans will be installed at the surface and the Gaumond shaft will be used as the main ventilation raise.

Figure 18-1: Aerial plan schematic of the proposed infrastructure layout
Note: Distance from left to right across the plan schematic is approximately 1 km

18.1.2. Surface Ramp

The portal of the ramp is located approximately 300 m from the Gaumond shaft and 800 m from the orebody. The first part of the ramp is 7 m wide by 5 m high and have a grade of 15% (8.4°). At the connection with level 400mLv, the ramp dimension reduces to 5.8 m wide and a grade of 17%. The total length of the ramp, between surface and 650mLv is 4.5 km, and an extra 3.5 km of ramping will be needed to reach 1130mLv.

The ramp will provide a drilling platform for exploration and definition of the orebody. It will also accelerate production from the upper part of the orebody. The ramp will expedite the construction of infrastructure on the upper levels because it will be independent of the Gaumond shaft.

18.1.3. Loading Station

The shaft loading station will be located on 690mLv. Ore and waste storage bins of 5,000-tonne capacity each will be excavated below 650mLv. A rock breaker/grizzly system installed at the top of the silos will control the rock size going into the skips. The grizzly openings will be set at 406 mm (16 in). On 690mLv, an automated conveyor will transfer the rock to the measuring box; the

rock will then be dumped into the skips. Figure 18-2 shows the general arrangement of the 690mLv loading station. A second loading station will be excavated on 1440mLv when the production shaft will be completed (Figure 18-3). This station will provide an efficient and more economical way to produce in the lower mine.

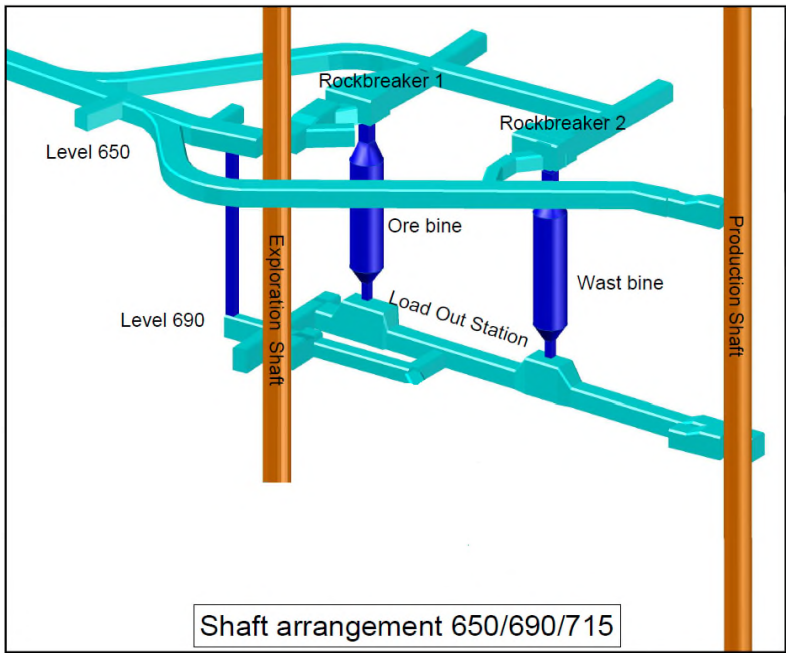


Figure 18-2: General Arrangement of the 650-690 Loading Station

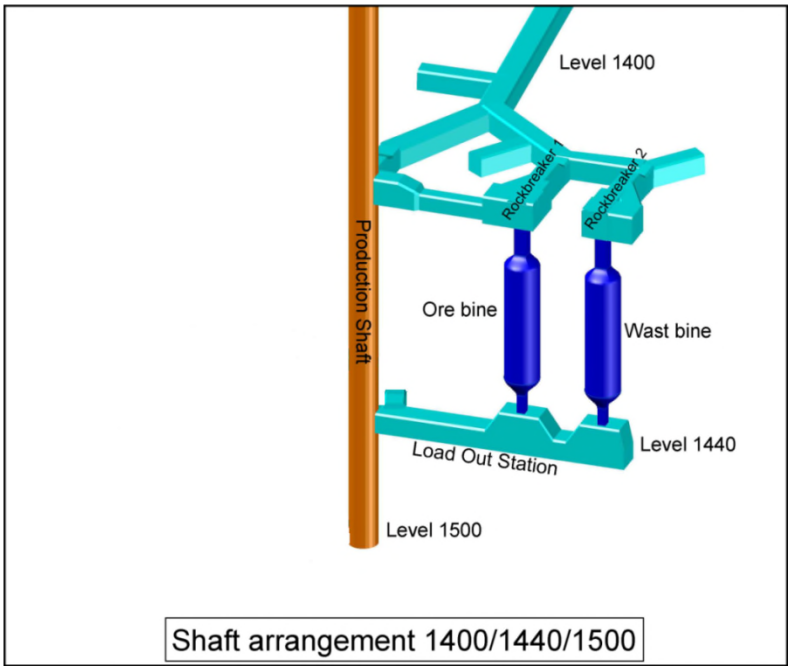


Figure 18-3: General Arrangement of the 1400-1440 Loading Station

18.1.4. Spill Pocket

The Gaumond shaft has a spill pocket. This excavation allows the clean-up of all rocks and mud that will accumulate at the bottom of the shaft.

18.1.5. Production Shaft

The production shaft will be circular with an inside diameter of 7 m. The shaft will be lined with concrete to a final depth of 1,500 m. This shaft will be equipped with two (2) 23-tonne skips (payload), a service cage with two cables, and a Maryann auxiliary cage.

Preliminary hoist calculations indicate that the required hoist diameter will be 6.25 m (20.6 ft) powered by an 8,000 hp motor. The hoist rope diameter will be 61 mm (2.4 in.).

The same hoist will be used for sinking and production. For the production shaft, Goldcorp selected a hoisting capacity of 8,500 t/d (combined ore and waste), which represents about 17 hours of hoisting per day at a hoisting speed of 15.24 m/s (3,000 ft/min).

A service cage will be installed to transport personnel and material to the working levels. The service hoist will have a diameter of 5.5 m (18 ft) and will be powered by a 8,000 hp motor. The hoisting speed will be 9.14 m/s (1,800 ft/min). The hoist rope will have a 44.5 mm (1.75 in) diameter for a maximum suspended load of 50,903 kg (112, 222 lbs).

An auxiliary Maryann cage will be installed in the production shaft for personnel and emergency situations. The Maryann installed in the production shaft will be identical to that installed in the exploration/ventilation shaft. Hence, the equipment will be interchangeable, if required.

Four (4) shaft stations will be excavated from the production shaft, namely at the 400, 650, 1020 and 1400 levels. The shaft stations will be used to develop the main shaft accesses and the mine dewatering pumping stations. Two shaft loading stations are planned at depths of 690 m and 1,440 m respectively. A loading pocket arrangement will be installed on the 650 level, identical to that used in the exploration shaft, and the conveyor will be rerouted.

18.1.6. Ore/Waste Passes

The ore/waste passes will be located close to the ore zones in order to minimize the hauling distance between the mining work places and the dumping areas, and to limit the number of shaft accesses. The current mine layout includes one transfer drift on 650 ore and waste from the ore zone to the shaft.

Ore and waste from the lower mine will be trucked via the ramp up to the loading station on the 650m lever.

Each zone will have its own ore/waste pass (Figure 18-4); this approach will minimize the haulage distance between the stopes and the ore passes. Because there will be only one raise per zone for both ore and waste, it will be necessary to manage the rock in batch mode.

Ore/waste passes will be excavated with an Alimak. The final diameter will be 3.3 m. Using an Alimak raise will allow ground support installation. Fingers will be excavated with conventional raises. Fingers will be 2.4 m x 2.4 m. Figure 18-5 shows the layout of a typical raise and the loading setup; the example will be constructed on the 650 level .

18.1.7. Ventilation Raise

The main ventilation network and the quantity of air needed underground was discussed in Section 16.4.3. At a nominal production rate of 3,500 t/d, the Gaumond shaft will be used as the main air intake. The exhaust raises that connect in the ramp will have a minimum diameter of 7 m. Internal ventilation raises will be 3 m in diameter. These raises will be excavated with raise-boring machines or an Alimak raise. The ventilation raises will be located in the northern and southern areas of the mine. Figure 18-4 shows the location of each ventilation raise on a standard level.

Figure 18-6 shows a longitudinal view of the entire mine with the main infrastructure locations.

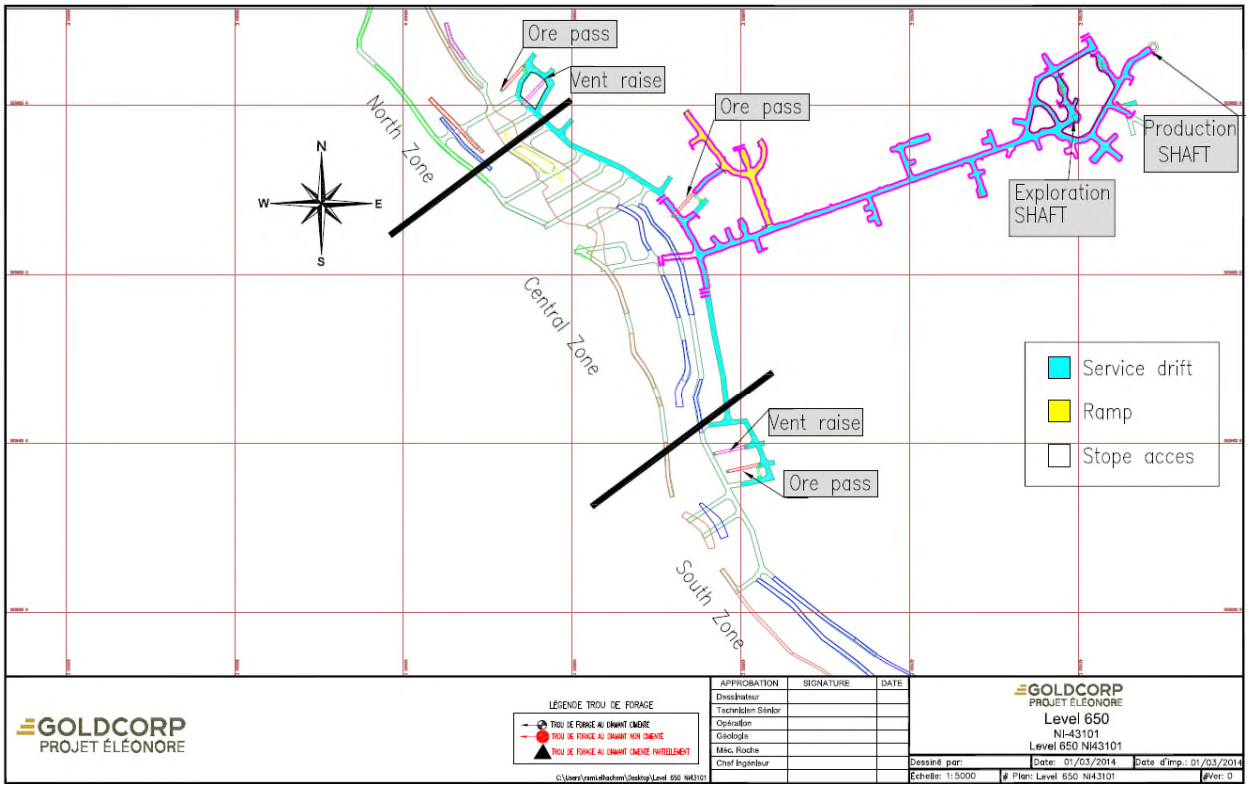


Figure 18-4: General arrangement on a typical level

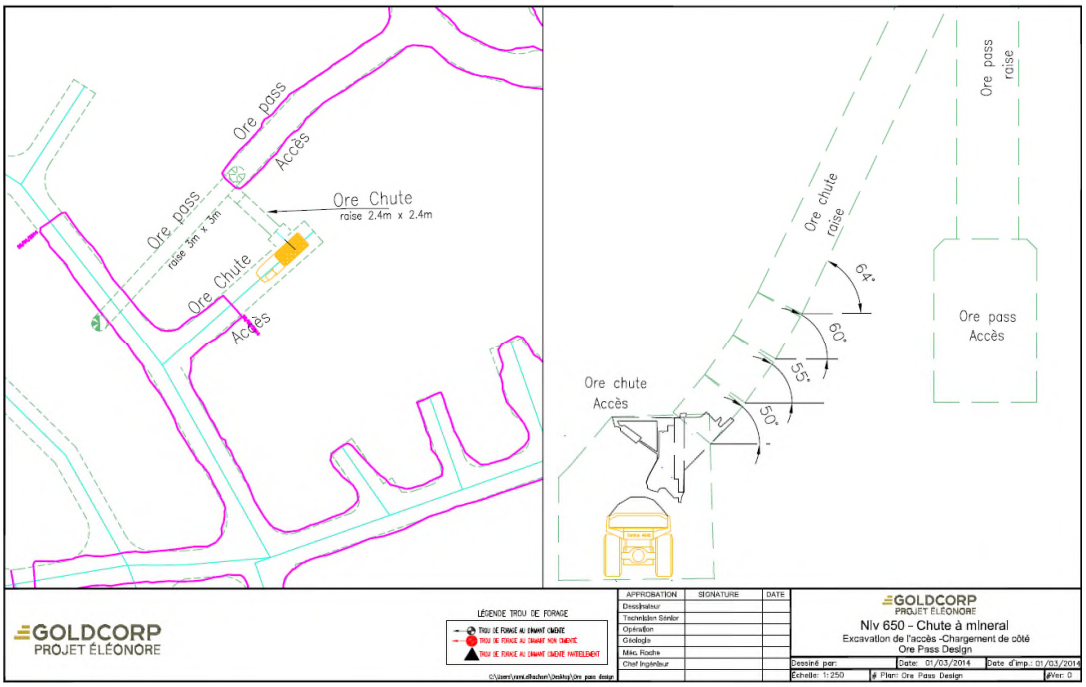


Figure 18-5: Section view of a typical ore pass network

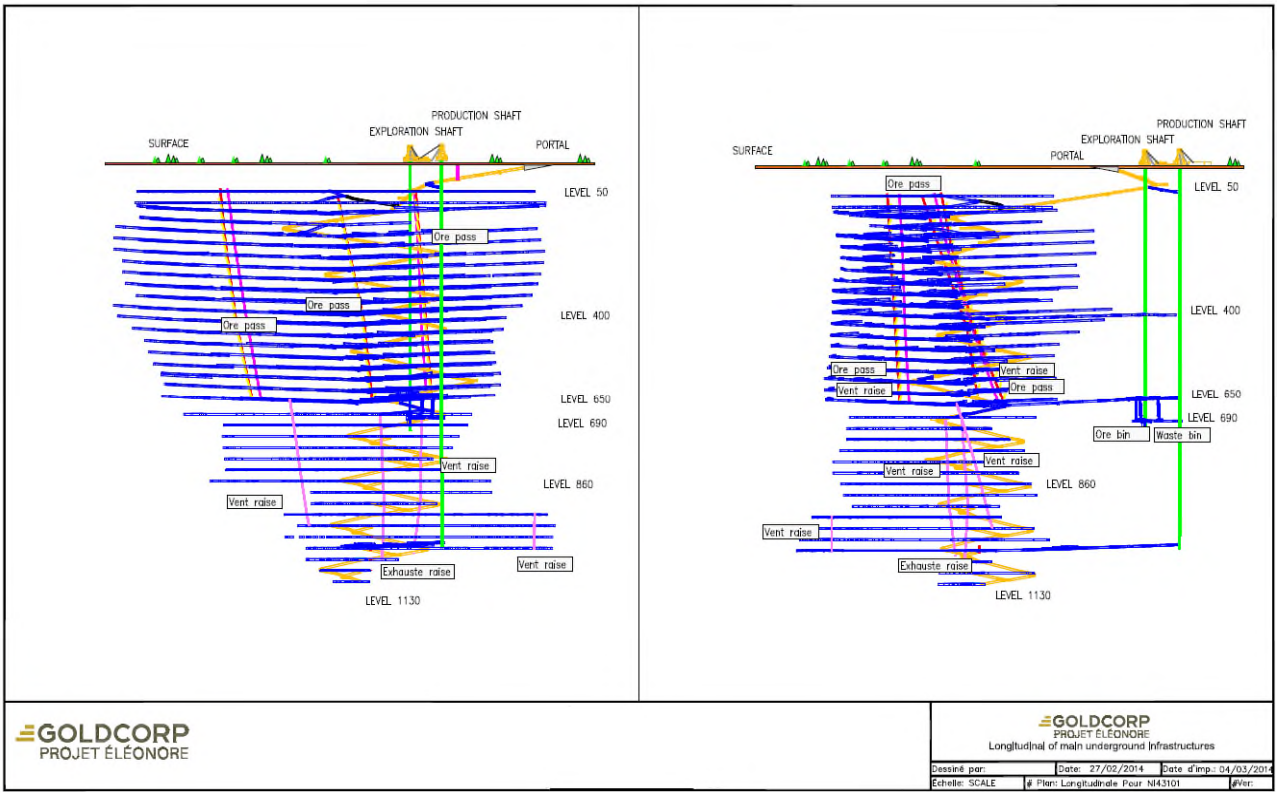


Figure 18-6: Longitudinal section showing the location of the main underground infrastructure

18.2. Surface Infrastructure

18.2.1. Waste Dumps

A waste/ore rock pad, with a volume of 800,000 m^3, was built for stockpiling rocks to support underground development activities and provide ore inventory for the process plant commissioning. Another waste rock pad of 900,000 m^3 was built into the tailings management facility to support the underground development during the ramp-up period. Because waste rock has an acid drainage and arsenic leaching potential, the waste rock pad was completely lined with an HDPE liner (high density polyethylene liner). Water that has been in contact with waste rock is collected in retention basins and treated before being released to the environment.

Over the LOM, approximately 11 Mt of waste is expected to be produced. Some of this material may be stored in a temporary tailings facility, or on an expanded waste rock facility. However, by the end of the mining operation, all waste will have been consumed in backfill and returned underground.

18.2.2. Tailings Facility

The tailings area will be fully lined. Figure 18-7 shows the tailings area design. A low risk tailings impoundment structure will be implemented to store about 26 Mt of filtered tailings at 85% solids over the 20-year LOM. The filtered tailings will be trucked 5 km to the tailings area, which will have a surface area of 80 ha. The filtered tailings will offer a great potential for progressive restoration and will support a significant reduction of risk of dyke or dam failure.

All the water in contact with the tailings area will be contained and pumped to the water treatment plant. The actual size of the tailings management facility will be about 80 ha and shall be developed in four phases approximately over a five-year period.

For years 0 to 10, the tailings management facility will receive mainly non-sulphide tailings from the process plant and waste rock.

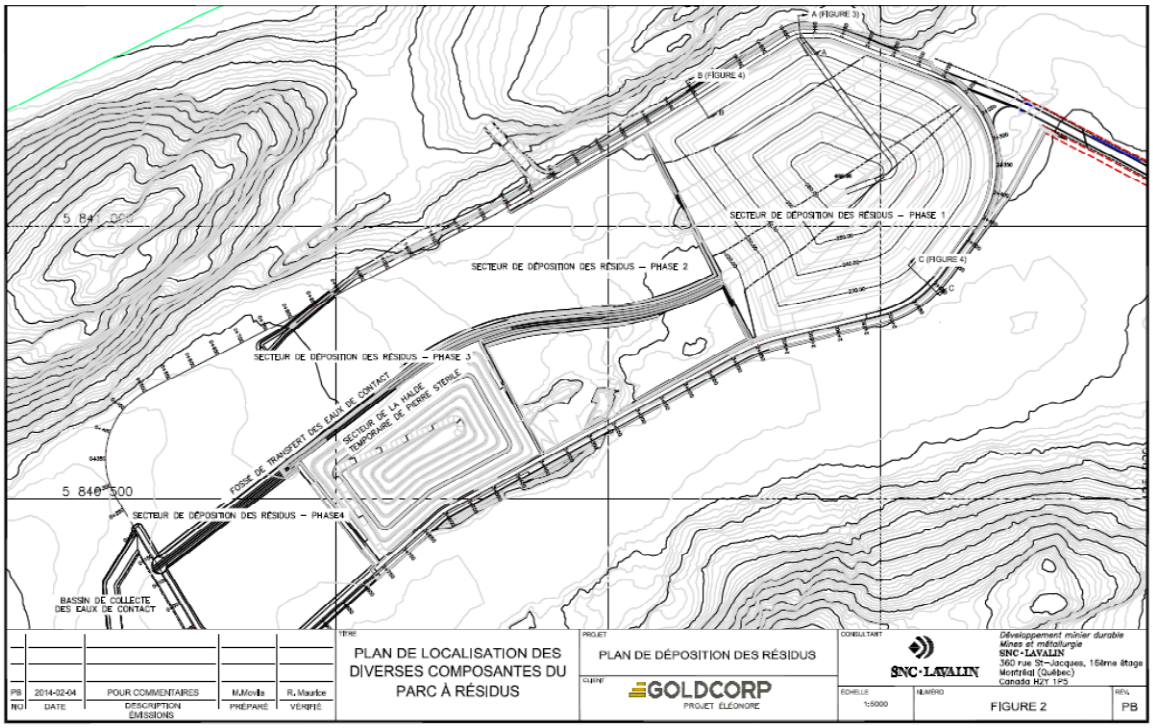


Figure 18-7: Design of the tailings area

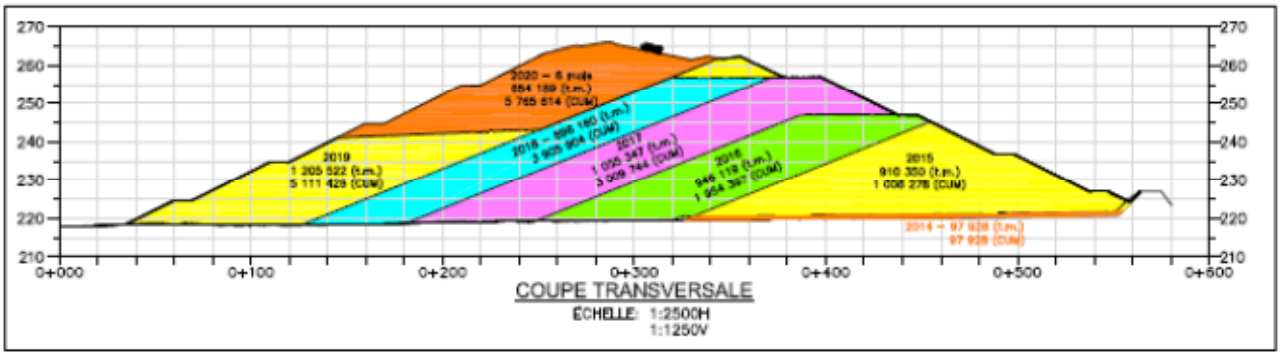


Figure 18-8: First cell deposition design

For years 0 to 10, there will always be approximately 40 ha of exposed tailings that will come into contact with runoff water. As per the Ministry's D019 directive, the water management facility in the tailings area has to be designed to take into account the following:

- A 1:100 year snowfall event, melting over a 30-day period;
- A 1:100 year 24-hour rainfall event during that period.

Based on these criteria, the collection basin required at the tailings area will range from 80,000 to 100,000 m^3, assuming that 4,800 m^3/day (200 m^3/h) is pumped continuously for approximately 30 days during this event. Under the average annual rainfall scenario (i.e. 1:2 year), water will be pumped out of the tailings area over a total of 74 days per year. No water will be pumped from the tailings area during the winter months. In the spring period, during the snow thaw, it is expected that water will be pumped continuously for 15 to 30 days. In the summer period, the water will be pumped intermittently.

The following residues will be stored in the tailings area:

- Non-sulphide residues;
- Waste rock (temporary storage, approximately 2 Mt from year 0 to 10).

The non-sulphide ore residues are considered non acid-generating. Arsenic was formerly identified as a risk element; however, a new static leaching test confirms that arsenic can be leached from the residues, as well as copper and iron. The kinetic leaching test on the residues, after 24 weeks, reveals that there is little arsenic, iron and copper in the water after it came into contact with the residues.

An evaluation of the leaching test data on the waste rock is underway. This study is being undertaken to estimate the quality of the tailings area runoff water based on the mixing of run-off coming into contact with waste rock and the non-sulphide residues.

18.2.3. Main Camp

The camp is located approximately 1.5 km north of the process site. Permanent housing has already been built with 400 rooms, including bathroom/shower and lockers, on the basis of one occupant per room. A communal laundry and lounges is provided on each floor in the four buildings. Each building has about 100 rooms spread over four levels. The main corridor between the modules and other buildings will be located on the first floor to minimize the distance from each room using elevators to access the upper floors. The modules will be protected by an intrusion and fire detection alarm system by sending a remote signal to the fire protection system at the security building. It is estimated that an area of 16,622 m^2 will be needed for this facility.

During the construction phase, a temporary cafeteria is used to serve about 1,300 workers per day. At the end of 2014, a permanent cafeteria will also host the permanent workers. It will include storage areas and food preparation, service counters for meals, lunches and pastries. The dining room will seat 300 and includes a lunch area. This facility will require an estimated area of 1,950 m^2.

An area of 2,157 m^2 on two levels will serve as a recreation facility. Seasonal outdoor equipment will also be available, and will include an ice-skating rink. A generator will provide all key services in case of power failure.

18.2.4. Service Buildings

18.2.4.1. Administration Building

The new administration building is located at the industrial area and connected with the warehouse and garage facilities.

This building is designed to incorporate the administration offices and the following: geology; mining and engineering; human resources; environment, health and safety; IT and accounting; permanent dry for 650 employees; nursery; and dispatch. The emergency vehicles, such as the fire truck and the emergency first response vehicle, will be located in the administration building as well. An area of 6,685 m^2 has been estimated for this facility.

18.2.4.2. Warehouse and Garage Facilities

The warehouse and garage are grouped together. The warehouse building will have a total area of 1,150 m^2. It is designed with an inside clearance of 9 m. It will include two offices, a reception desk, a shipping deck and the required storage space. It will be protected by an intrusion and fire detection system. The entire building will be heated.

The garage building will cover an area of 1,250 m^2 including a 250 m^2 mezzanine and will be adjacent to the warehouse. The mezzanine will provide space for offices, a meeting room, toilets, a locker room and the crew room. The ground floor will include a garage with five repair bays, a wash bay, a machine and welding shop, an electrical workshop, a carpentry workshop, a tool storage area and three offices.

The building will be designed to provide an inside clearance of 9 m for the 50 t trucks. A 15/3 t crane will be installed in the garage area. The garage will also be equipped with a lubrication system which will be easily accessible for heavy equipment maintenance. The building will be protected by an intrusion and fire detection system. The entire building will be heated.

18.2.4.3. Assay Laboratory

The laboratory was commissioned for January 2014 and is located close to the process plant. The building area is approximately 800 m^2, on one floor. The main features are offices, a kitchen and dining room, a bathroom and shower area, a reception and sample preparation area, and an assaying area.

The laboratory is equipped with all equipment, instruments and materials necessary for a gold operation. It is designed to process about 500 samples per day. It is protected by an intrusion and fire detection system.

18.2.5. Processing Facilities

The processing facilities comprise four areas: the crushing and screening area, the main process plant, the paste backfill plant, and the tailings filtration plant.

The crushing facilities are located to the northwest of the Gaumond shaft. The fine crushed ore bins will be located north of the shaft. The crushing and ore storage facilities were designed for 8,500 t/d, including 1,500 t/d for daily aggregate production for the paste backfill requirement. The layout includes two buildings. The primary crushing building will crush ore from the Gaumond shaft. It will house the vibrating grizzly, the jaw crusher and screens. The secondary and tertiary crushing building will house the three cone crushers.

All the above buildings will be inter-connected by belt conveyors installed inside a gallery. The fine crushed ore will be stored in two 3,500 t live capacity insulated steel bins. Each bin will be equipped with two reclaim belt feeders that will be installed under the bins in an enclosed heated area.

The processing plant will be located north of the shaft and northeast of the fine ore bins.

The plant building is designed as a rectangular shape measuring 92.5 m wide by 135 long, and has different roof heights depending on the process areas.

The concentrator building will house the grinding area, cyanidation area, the flotation area, the carbon stripping area, the carbon reactivation area, the refining area, the reagents area, the compressors area, the metallurgical laboratory area, an office space area, a dry for 100 employees, and maintenance shops.

Outside the concentrator building, the following components will be installed, bearing in mind any future potential increases in throughput rate: the concentrate leach tanks, the flotation tail thickener, the tails CIP thickener, the flotation tail leach tanks, the cyanide tanks, the lime bin, and the process and fresh water tanks.

Paste backfill and filtration will be grouped in the same building to the west of the concentrator building. The buildings will be connected by an enclosed corridor.

The paste backfill plant is designed for 320 t/d through two parallel lines. Crushed waste material will be transported via a retractable conveyor from the top of the fine ore bin(s). The material will be stored in a 2,000 t live capacity steel bin. The paste backfill and filtration building will house major equipment such as the plate filter press, conveyors, mixer, piston pumps, and thickener and holding tanks. The binder silos will be installed outside of the building. The water tanks and pump box will be housed in a lean-to off the paste backfill plant and will be designed for a maximum 7,000 t/d throughput rate.

An oxygen plant of 48 tpd capacity will be located west of the flotation tails leach tanks to supply oxygen to flotation concentrate cyanidation and cyanide destruction circuits.

18.2.6. Terminal and Airstrip

The air terminal building will be located on the airport site, south of the airstrip. The building will include a waiting room with services for 80 people, offices, and a security zone. The air terminal will be approximately 1.5 km from the plant site. The construction of a garage for storing de-icing equipment will also be included. A generator will provide all the services in case of power failure. This facility will require an estimated area of 442 m^2.

18.3. Electrical Power

It has been assumed for design purposes that all large motors will be equipped with a softstart unit or an AC drive, in order to maintain a good power factor and limit the starting current and voltage variations, while reducing harmonic emissions. Overhead lines have been designed using ACSC (aluminum

conductor steel reinforced) conductors. As required by Goldcorp, all power transformers are considered to be dry-type, except for those inside the main 120(69)/25 kV substation.

The new Éléonore main 120/25 kV substation was designed taking into consideration redundancy, labour and transport costs, as well as the geographic location, and a total substation life expectancy of 15 to 25 years. The total mine complex load during winter is estimated at 48 MW.

The substation consists of one 120 kV overhead incoming with two (2) 120/25 kV 40/53/66.6 MVA oil step-down transformers, for redundancy purposes. Each transformer will be able to support the mine's full load, feeding a 25 kV gas insulated switchgear (GIS). This switchgear will comprise two incomings with a tie breaker. Two 25 kV feeders (fed from the two different bars) are planned for each of the following essential buildings: concentrator and crusher, and the Gaumond hoist/shaft. Other 25 kV loads will not be equipped with a redundant feeder and will be supplied by overhead service lines.

The main 120/25 kV substation is installed near the concentrator and hoist/shaft, to limit the length of conductors feeding the largest loads.

This substation is in operation since 2013 to support further mine site development.
All 25 kV distribution lines will be designed in order to take into account the constraints imposed at this northern latitude such as weather, ice and wind loads in addition to the loads related to the equipment and conductors.

The onsite 25 kV distribution network will consist of overhead lines (ACSR conductors) and power cables (where distances are limited and a rack can be installed above ground). Concentrator and crusher buildings will be fed by a power cable, installed on a rack. Hoists and shaft buildings will be fed using redundant overhead lines (one feeder to each location, plus a tie-in between sites).

The site surface service buildings (laboratory, garage, tailings, campsite, construction site, etc.) will be supplied by two 25 kV overhead radial ACSR lines, mounted on wood poles. The two lines will supply the western and eastern parts of the mine complex. All service buildings requiring less than 500 kVA will be supplied by transformer banks mounted on wooden poles with AC lightning arresters, fuse-disconnect switches (or reclosers), and grounding.

18.4. Fuel Storage

For fuel storage and services, three different installations have been selected. It is important to note that there will be daily fuel tanks at each campsite building with an emergency generator:

- Airstrip;
- Depot (mine site);
- Fuel station.

18.4.1. Airstrip Tank Farm

At the airstrip, fuel storage will consist of:

- Jet "A": two (2) 50,000 L, double-walled, ULC/S601, horizontal, above-ground tanks with cabinet, pumping unit, filtration system, and aircraft refueling;
- Diesel: one (1) 9,000 L, double-wall, ULC/S601, horizontal, above-ground tanks with cabinet and pumping unit.

18.4.2. Depot (Mine Site)

The proposed tank farm will consist of:

- Diesel: six 50,000 L and one 45,000 L, double-walled ULC/S601, horizontal, above-ground tanks;
- Pumping station: two pumps, one for loading of the service truck and one for the unloading of the delivery truck;
- The site will also include equipment for loading and unloading trucks, fences, cement pad for the loading and unloading area, hydrocarbon interceptor and other required safety accessories;
- A computerized fuel management system will be installed to provide detailed reporting of fuel usage and control. The proposed system consists of equipment and software which is specifically designed to automatically control and provide accountability of any metered fuel product;
- The loading and unloading of the fuel trucks will be accomplished through an automated control system which includes a PLC, level probe, automated valves and fuel meter.

18.4.3. Fuel Station

The proposed fuel station will consist of:

- Gasoline: one 50,000 L, double-walled ULC/S601, horizontal, above-ground tanks;
- Diesel: one 50,000 L, double-walled ULC/S601, horizontal, above-ground tanks;
- Submersible fuel pump installed on each tank;
- Dispensing unit for each product (gasoline and diesel);
- A fuelling area will be composed of a concrete pad with equipment protection against vehicle impact.
- A computerized fuel management system is installed to provide detailed reporting of fuel usage and control. The proposed system consists of equipment and software which is specifically designed to automatically control and provide accountability of any metered fuel product.

For fuel delivery around the site, a fuel truck with a storage capacity of 19,100 L will also be supplied.

18.5. Water Management

18.5.1. Potable Water for Campsite and Industrial Area

The drinking water supply for the campsite and industrial area is presently drawn from four existing wells. The wells are located 1.2 km northeast of the project campsite.

The combined capacity of the four wells is greater than the anticipated maximum daily flow (250 m^3/day). The existing treatment system allows for the treatment of non-conforming elements in the water supply. The existing drinking water treatment includes the following (Figure 18.8):

- Sodium hypochlorite injection unit for oxidation before filtration, to remove iron and manganese;
- Catalytic filter media for manganese and iron removal;
- Charcoal filter unit for chlorine removal before the anionic exchange resin bed;
- Anionic exchange resin bed with proper media type for uranium removal;
- UV disinfection units to increase treatment efficiency;
- Chlorine disinfection to protect the distribution network against possible contamination along its course;
- One treated water reservoir with a total volume of 200 m^3;
- Two backwash pumps;
- Two distribution pumps.

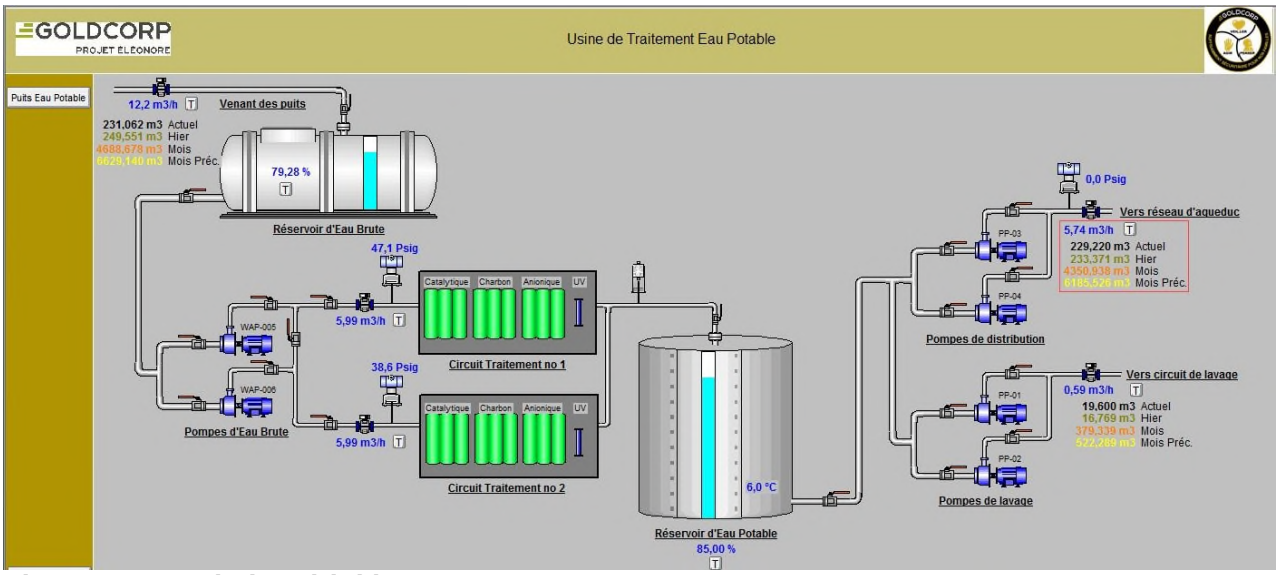


Figure 18-9: Existing drinking water treatment

18.5.2. Wastewater Collection and Treatment

Wastewater is collected by a gravity sewer network. The wastewater will then be pumped to the treatment system via the sewage pumping station. The existing pumping station has a capacity of 2,160 m^3/day.

The wastewater treatment is done by aerated lagoons in series. The treatment system also includes a system of ferric chloride injection for phosphate removal.

The treated water is discharged into the Opinaca Reservoir. The environmental objectives of the discharge as established by MDDEP-DSÉE as follows:

- Domestic wastewater treatment for the camp and industrial area will be executed at the existing treatment plant.
- Disposal of domestic wastewater from the underground mine has not been confirmed, whether it will be discharged into the existing wastewater treatment plant, or resolved by using of propane or electric toilets generating only ash, chemical toilets, etc.
- Wastewater treatment of the airport terminal and gatehouse waters will be done by a septic tank installation at each of these two sites that will be discharged into the aerated lagoons.
- Taking into account the functions of the camp and the industrial area, it is considered that 100% of the flow of drinking water will flow into the wastewater system.
- The design flows are shown in Table 18-1.

Table 18-1: Design flows

	Campsite	Airport Terminal
Average daily flow	300 L/(pers.day)	10 L/(pass.day)
Maximum daily flow	400 L/(pers.day)	20 L/(pass.day)
Peak hour flow	1200 L/(pers.day)	40 L/(pass.day)

The wastewater emitted from the potable water treatment plant (filter backwash for example) will also be evacuated to the wastewater system and is added to the above flows. This wastewater is estimated at 12 m^3/d. Infiltration in the underground network is considered low because the network is new (10% of domestic flow).

Based on a population of 400 people, the flows to consider are as follows:

- Domestic water: 120 m^3/day;
- Potable water system wash water: 12 m^3/day;
- Infiltration: 12 m^3/day;
- Total: 144 m^3/day.

The planned flow of wastewater is therefore less than the design flow of the treatment station authorized by MDDEP.

Wastewater collection will be by underground gravity pipes which will direct all wastewater to the existing main sewer pumping station.

18.5.3. Industrial Effluent Water Collection and Treatment

The design of the plant is based on an operating life of 20 years and allows for a possible expansion considering the uncertainty surrounding the volume of water that will be pumped from the mine. The water treatment plant (see Figure 18-8) includes the water treatment system, pumps and all the collection basins required to store and manage the site runoff water and the mine water.

There are four types of effluents that will require treatment. They are:

- Mine water: water from the underground mine;
- Process water: water purges from the mineral processing plant;
- Tailings area runoff water: water from the tailings management facility.
- Industrial zone runoff: runoff water from the industrial area of the site.

All of the industrial effluents must be collected and treated to meet the following two objectives:

- Produce a treated effluent that meets and exceeds the discharge criteria set out by Directive D019 from the MSDEP (Ministry of Sustainable Development, Environment and Parks) prior to discharge into the Opinaca Reservoir;
- Produce a treated effluent, in sufficient quantity and quality, to provide process water make-up to the Éléonore mineral processing plant.

18.5.3.1. Mine Water

The water treatment plant is designed to deal with an initial flow of 10,000 m^3/day, expandable to 20,000 m^3/day, and will ultimately be able to treat 40,000 m^3/day. Arsenic and iron are present in sampled mine shaft waters to a depth of 300 m; however, these contaminants are primarily present in the form of suspended solids. Additional water sampling will be required from deeper levels of the shaft.

At present, based on the water analysis of the mine water from the shaft, the parameters that will require treatment are the following:

- Total suspended solids (TSS) are very high. A fraction of the TSS will settle in the collection basin ahead of the water treatment plant;
- High pH;
- High concentration of ammonia. This is caused by the dissolution of unconsumed ammonium nitrate explosive in the shaft
- Provision must be made for the treatment of dissolved arsenic and iron.

18.5.3.2. Process Water

Based on the preliminary water balance of process water, the flow of process water sent to the water treatment plant (WTP) is 4.9 m^3/hr (118 m^3/day) when the paste backfill plant is not in service, which will occur intermittently. This flow represents the excess of process water when the process plant is at steady state. The process water purge will be done discontinuously.

When the paste backfill plant is in service, there will be no process effluent sent to the WTP. The paste backfill plant availability is 60%. Therefore, 40% of the time, the backfill plant will be offline and a process water purge will be sent to the WTP.

Excess water from the process plant is sent to the WTP from the process water tank. Four streams feed the process water tank:

- Flotation concentrate thickener overflow;
- Flotation tailings thickener overflow;

- Sulphide tailings thickener overflow (after cyanide removal);
- Non-sulphide tailings thickener overflow (after cyanide removal).

The cyanide destruction unit is designed to achieve 1 mg/L of total cyanide after treatment. Upsets to the cyanide destruction process are considered to be a low risk since the process used is a well known process in the industry (Inco oxidation process by SO_2). Any process upset will be managed in the process plant without released to the WTP.

The main elements of concern in the process water, after cyanide removal, are:

- Arsenic;
- Iron;
- Copper;
- Nickel (potentially, based on the lixiviation test);
- Chemical oxygen demand caused by concentrations of thiosalts and organic reagents used in the flotation circuit, such as xanthate;
- Other dissolved metals present in the ore that may leach during the gold beneficiation process.

Additional tests will be conducted on the ore. As part of the testing, water samples will be taken from the overflows of the flotation concentrate thickener, the flotation tailings thickener, the sulphide tailings thickener and the non-sulphide tailings thickener. The data will then be used to confirm the process water quality.

18.5.3.3. Industrial Zone Runoff

The surface area of the industrial zone is approximately 15 ha. This area includes the process plant, the mine complex, the laydown areas, and the administration buildings.

When considering a rainfall event based on a 1:25 or a 1:50 year return period, the volume of the required storm water runoff basin is estimated to be between 10,000 to 12,000 m^3, assuming a pumping capacity of 1,440 m^3/day (60 m^3/h).

The volume of the collection basin increases significantly when considering a 1:25 year snowfall event melting over 30-day period, and a 1:25 year 24-hour rainfall event. Under the current site layout, there is very limited space in the industrial zone to accommodate a collection basin of this size.

To optimize the volumes to be collected, a study is underway to redefine the industrial zone surface area to reduce the surface area that needs to be collected and treated.

Any runoff water from the sulphide and non-sulphide ore storage and handling facility located in the process plant has to be captured and controlled, then returned either to the process plant or to the WTP.

In addition to the water collection basins at the tailings area and the industrial zone, there will be a collection basin to collect all of the effluents ahead of the WTP and a treated water collection basin.

The effluent collection basin located ahead of the WTP will continuously receive mine water and intermittently the process water purges and the runoffs from the tailings area and industrial zone. Preliminary, the basin will be sized to hold a useful volume of 20,000 m^3, divided into two cells of 10,000 m^3, plus one stand-by cell of 10,000 m^3.

18.6. Workforce

The workforce requirements for the underground operations include supervision, direct mining and mine maintenance. Table 18-2 presents a summary of the mine manpower that will be needed to produce a nominal 3,500 t/d. The workforce will be gradually increased until that milestone is reached.

Table 18-2: Projected workforce requirement for the mine department

			Production rate	
Descripiton			**3500 t/d**	**7000 t/d**
Mine Department				
	Mine Supervision			
		Mine Superintendent	1	1
		Mine ass. Superintendent	2	2
		Mine Captain	4	4
		Supervisor	16	20
	Developement			
		Jumbo Operator	48	48
		Ground Support	52	52
	Production mucking/hauling			
		Scooptram operator	12	20
		Truck operator	12	12
		Truck hauling	8	16
	Long Hole Production			
		Drillers	10	14
		Blasters	6	10
	Backfill u/g		4	8
	Rock breaker/Road maintenance		8	8
	Service / construction			
		Miners	20	24
	Mine Safety & Training		4	4
	Shaft			
		Hoist man	5	5
		Cage / skip tender	8	8
		Shaft man	2	2
Total Mine Department			**222**	**258**

The production schedule will use two shifts, working 11 hours per shift. A delay of one hour between shifts will be put in place for blasting and clearance of blast fumes.

The crews will work on a 7 days on and 7 days off rotation, while the management team will be on a 4 days on and 3 days off rotation.
Tables 18-3 and 18-4 present the proposed workforce for the mine maintenance departments and the workforce needed for the engineering/geology department. Table 18-5 presents a summary of the number of contractor employees needed to reach the proposed late 2014 production rate. Finally, tables 18-6 and 18-7 show the workforce requirement for the process plan and the surface department.

Table 18-3: Estimated workforce requirement for the mine maintenance department

			3500 t/day	7000 t/day
Mine Maintenance Department		Schedule	Total workforce	Total workforce
Maintenance Supervision				
	Superintendant	4/3	1	1
	Superintendant Élec	4/3		
	Mech General foreman	7/7	1	1
	Élec.General foreman	7/7		1
	Fixe equip Supervisor	7/7	1	2
	Mobile equip Supervisor	7/7	2	4
	Supervisor Electrical	7/7	1	2
	Supervisor inst & control	7/7	1	1
	Senior maint planner	7/7		1
	Maint. Planner	4/3	2	3
	Ing/Élect	4/3		1
Mechanical				
	Trainer	7/7		1
	Kitting clerk	7/7	2	4
	Mechanics fixed equipment	7/7	8	6
	Électromechanics fixed equipment	7/7	2	4
	Mechanics mobile equipment	7/7	15	42
	Électomechanics mobile equipment	7/7	2	6
	Welders	7/7	2	4
Electrical				
	Electricians fixed & mobile equip	7/7	10	16
	Senior tech/inst & control	7/7		1
	Instrumentation	4/3	2	3
Total Maintenance Department			**50**	**100**

Table 18-4: Proposed workforce requirement for the engineering and geology departments

			3500 t/day	7000 t/day
Technical Service		Schedule	Total workforce	Total workforce
Engineering				
	Chief Engineer	4/3	1	1
	Sr Mine Engineer	4/3	1	2
	Mine Engineer	7/7	6	8
	Surveyors / Mine Technicians	7/7	11	14
	Mine Technicians	4/3	1	2
	Project Engineer	7/7	1	1
	Geotech Eng.	4/3	1	2
Geology				
	Chief Geologist	4/3	1	1
	Sr Geologist	4/3	1	2
	Production Geologist	7/7	2	4
	Ressources Geologist	7/7	1	1
	Sr Geology Technician	4/3	1	1
	Geology technician	7/7	2	4
	Drilling Supervisor	7/7	2	2
Total Technical Service			**32**	**45**

Table 18-5: Number of contractor employees

Contractors			Total workforce
		Underground drilling	
		Raise Bore Drilling	4
		PST Contractor	6
		Misceallenous	12
		construction	25
		indirect	16
		U/G lateral dev	25
		SHAFT	53
		Underground DDH drilling	26
Total Contractors			167

Table 18-6: Projected workforce requirement for the process department

Departments		Schedule	Total Workforce
Management			9
	Process and Services Surface Manager	4/3	1
	Administrative Assistant	4/3	1
	Chief Metallurgist	4/3	1
	Operation General Foreman	4/3	1
	Maintenance General Foreman	4/3	1
	Services Surface General Foreman	4/3	1
	Health & Safety Coordinator	4/3	1
	Trainer	4/3	1
	Civil Engineer	4/3	1
Metallurgy / Assay Laboratory			23
	Production Metallurgist	14/14	2
	Technician Metallurgist	14/14	2
	Refiner	14/14	2
	Process Control Engineer	14/14	2
	Chief Assayer	4/3	1
	Assayer	14/14	6
	Sample Preparation	14/14	8
Process – Operation			48
	Process Foreman	14/14	4
	Process Operators	14/14	44
Process – Maintenance			27
	Mechanical Foreman	14/14	2
	Maintenance Planner	4/3	1
	Mechanical Engineer	4/3	2
	Maintenance Clerk	4/3	1
	Millwrights	14/14	22

Table 18-7: Projected workforce requirement for the surface department

Departments		Schedule	Total Workforce
Mobile equipment maintenance			11
	Heavy equipment Mechanics	14/14	4
	Small Equipment Mechanics	14/14	4
	Machinists	14/14	2
	Surface Planner	4/3	1
Electric / Instrumentation			19
	Electrical Foreman	14/14	2
	Electrical Engineer	4/3	1
	Electricians	14/14	8
	Instrumentation Technicians	14/14	6
	Electro-Mechanics	14/14	2
Site Maintenance			30
	Site Surface Supervisors	14/14	2
	Site Operators (airport, road maintenance, concrete plant)	14/14	14
	Building Mechanics	14/14	2
	Carpenter	14/14	4
	Water Treatment Technicians	14/14	2
	Surface Laborers	14/14	6

18.7. Comment on Section 18

In the opinion of the QPs, the following interpretations and conclusions are appropriate:

- The project will require construction of significant infrastructure to support the planned production facilities and 600 permanent employees.
- The permanent road access to the Éléonore property was completed in 2013 and is used only for construction and operational material supply.
- Employees are transported by aircraft (Dash-8 / 37 passengers).
- All the water in contact with the process, the tailings, and the industrial area will be treated before any discharge to the environment.

19. MARKET STUDIES AND CONTRACTS

Goldcorp's bullion is sold on the spot market by Goldcorp's in-house marketing experts. The terms contained within the sales contracts are typical of and consistent with standard industry practices, and are similar to supply contracts elsewhere in the world.

19.1. Comment on Section 19

The QPs expect that doré production from the Éléonore Project will be marketed in a similar manner and using similar sales contracts to that of existing Goldcorp operations.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1. Baseline Studies

Goldcorp has completed baseline studies in support of the Strategic Environmental Assessment (SEA; or ESIE in French) and is carrying out continuous monitoring studies to support project permitting and various commitments. Studies, undertaken by third-party consultants, are addressing the following aspects:

- Soil
- Hydrology
- Surface water quality and sediment
- Hydrogeology and groundwater quality
- Air and climate
- Noise and vibration
- Vegetation
- Wetlands
- Wildlife
- Fishes and fish habitat
- Land use and resources
- Archaeology
- Landscape
- Socio-economics

The studies include documentation of the existing baseline for each aspect in the list above, considerations of the impact of the construction phase and of the mining operational phase as separate cases, and considerations relating to reclamation and closure when operations cease.

20.2. Environmental Issues

For the Éléonore Project, the major issues identified include the potential impacts on the environment, the proper management of tailings and waste water, access (roads, airports), social acceptability and management of the post-reclamation site. These areas and the planned mitigation measures are summarized in Table 20-1.

Goldcorp is of the opinion that these issues have been, or can be, addressed and mitigated through a combination of baseline data collection, appropriate engineering and project design studies, and public consultation.

20.3. Closure Plan

A mining reclamation plan was prepared under the requirements of the Mining Act of Québec and approved by the MNR on November 28, 2013.

The closure and reclamation plan incorporates the following components:

- Demolition of buildings, pipelines and power lines (if they cannot be transferred);
- Closure of underground mine openings by capping the ventilation and production shafts, and the ramp access;
- Rehabilitation of accumulation areas, including tailings and settling ponds;
- Dismantling of buildings, infrastructure, equipment and sanitation;
- Removal of machinery, equipment, storage containers and construction trashes;
- Safe removal and disposal of chemical products, petroleum and other hazardous waste;
- Reclamation of sand pits;
- Reforestation of paths and flat surfaces;
- Treatment of contaminated soils;
- Monitoring the physical and chemical stability of the site after closure;
- Preparing a report on the state of the site at the end of the work.

Table 20-1: Key environmental issues identified at a pre-feasibility study level

Identified Issue	Proposed Mitigation Measures
Cyanide in Effluent and Taillings	Incorporation of cyanide detoxification systeme using the SO_2-Air-INCO method; specific cyanide containment areas and monitoring plans.
Taillings Management	The tallings storage area is fully lined with a geo-membrane. All water runoff from the taillings area will be captured and treated before being discharged to the environment. Tallings will be filtered to 85% solids; the result will be a reduction in the tailllings storage area footprint. The mill will produce non-sulphide and sulphide taillings. All the sulphide taillings and half the non-sulphide taillings will be returned underground as backfill.
Waste Rock Management	All waste rock piles are lined with geo-membrane. All water runoff from waste piles will be captured and treated before being discharged to the environment. All waste rock will be reuse in underground backfill during the life of mine. There will be no remaining waste pile at mine closure.
Water and Stormwater Management	The taillings water, the mine water and the water from the waste rock piles will be fully collected and processed in a facility designed and dedicated for this purpose. No infrastructure (except a bridge) and no mining activities will be held in the Opinaca River watershed. Monitoring of surface water and groundwater quality will ongoing throughout the operational life, and is planned to extend to post-closure monitoring.
Transportation	The limit of emission of dust, vehicule speeds will be limited to 60 km/h and water or an approved dust suppressant will be used, To reduce the road footprint, the main road and the secondary road will have the minimum required width.
Flora and Fauna	Consideration of impacts on wetlands will be included in the project permitting plan. The quality of Opinaca River and Opinaca Reservoir lake sturgeon populations will be monitored.

The reclamation work program as envisaged in the plan will take place over a period of about two to three years (excluding on-going monitoring) after completion of mining activities. The estimated cost of reclamation is $38.6 million and is summarized in Table 20-2.

Under the Mining Act of Quebec, a bond, in the form of a bank letter-of-guarantee, must be submitted to the MNR. The amount of guarantee must cover, after a period of three years, 100% of the reclamation costs (Year 1 @ 50% + Year 2 @ 25% + Year 3 @ 25% = 100%). As Goldcorp chose to file the entire amount in Year 1, a bond of $40.1 million was sent to the MNR in February 2014.

Table 20-2: Mine closure cost forecast

Area	Estimated rehabilitation	Comment
Infrastructure	7.5	Included foundation soil recovery/vegetation, dismantling and disposal of infrastructure and security of ramps and wells.
Water ponds	1.1	Reclamation, including water treatment plan if required.
Waste rock facilities	1.2	Membrane removal, site soil recovery, resseding vegetation.
Tailling facility	16.4	Membrane removal, site soil recovery, resseding vegetation.
Soil and waste management	0.7	Contamined waste, hazardous waste and solid waste management.
Indirect cost	3.9	
Post-closure monitoring	1.3	Includes 2 years of post-operating environmental monitoring and 5 years of post reclamantion environmental monitoring.
Contingency	8.0	
TOTAL	40.1	

20.4. Permitting

The Éléonore Project has been removed from the federal review process but is subject to provincial review under Chapter II of the Environmental Quality Act (EQA) for a project north of the 49th parallel. An ESIA has been completed, and has been subject to consultation with the Cree Nation, local communities and the general public. The Global Certificate of Approval under Chapter II of EQA was issued to the Project on November 10, 2011. The Certificate of Approval under Chapter I of the EQA for a second shaft to be sunk at the site and for the construction of related infrastructure and a power plant has also been released. A Certificate of Approval has also been granted for the access road.

In addition, the Project has also received Certificates of Approval for the construction of a production shaft and the construction of the access road.
With the Global Certificate of Approval granted, most of the remaining Certificates of the approval required under Chapter I of the EQA for site construction activities have been received from the Quebec Ministry of Sustainable Development,

Environment and Parks (MSDEP). Other applications for Certificates of Approval are currently being prepared and will be submitted to the appropriate ministries in 2014.

The formal mining lease application has been lodged with the MNR. Goldcorp expects formal granting of the mining lease around the beginning of 2014. Quarry and sandpit licence applications for borrow materials have also been lodged.

Key permits identified to date are summarized in Table 20-3. Where an application has been made for the permits, this is noted in the table.

20.5. Considerations of Social and Community Impacts

20.5.1. First Nations

The Éléonore Project is located on traditional family territories of the Cree Nation of Wemindji, and within the Municipality of James Bay (MBJ). Both are part of the administrative region of Nord-du-Québec (region 10). The Project area is entirely located within the purview of the James Bay and Northern Quebec Agreement (JBNQA).

The JBNQA plays a key role in the organization of the territory and its contemporary use by the Cree. The territory of James Bay is subdivided into three land categories. Under Category I and II lands, the Cree Nation has exclusive hunting, fishing and trapping rights. In Category III lands, Cree peoples have exclusive rights to harvest certain species of wildlife as well as conduct trapping activities. Each hunting area has a tallyman.

The Éléonore Project is located on portions of Cree trapline territories VC 22, VC 28 and VC 29 that collectively constitute the traditional territory of Wemindji.

A collaboration agreement was signed with the Cree Nation of Wemindji in February 2011. Under the agreement:

- The Cree Nation of Québec will share in the profits generated by successful mine operations.
- Goldcorp will provide funding towards a local training centre (Mâyâupiu Training Institute located in Wemindji) to teach skills that translate to the mining industry.
- Goldcorp will fund vocational education, scholarships, business skills and entrepreneurial opportunities.
- Goldcorp will endeavour to ensure the mine workforce is demographically representative of the local male, female and youth population in northern Québec

Table 20-3: Key permits and authorizations required for project construction and operation

Departement of Area	Permit or Approval	Status
Ministry of Natural Resources (MRN)	Quarries and sand pit exploration (sites for the construction of the road and mining infrastructure)	Recieved
	Lands use permits (lease)	Recieved
	Mining exploitation lease	Recieved
	Reclamation plan approuval	Approved
Ministry of Sustainable Developpment and Parks (MSDEP)	**Chapter II of EAQ**	
	Global Certificate of Approval	Recieved
	Certificated of Approval for construction of the road	Recieved
	Chapter I of EAQ	
	Certificated of Approval for: - Production shaft and power	Received
	-Taillings management facility construction	Received
	-Talling management facility operation	To be received
	-Industrial water treatment	Received
	-Process mill construction	Received
	-Process mill operation	To be received
	-Waste rock pad	Received
	-Drinking water pumping	Received
	-Drinking water pumping	Received
	-Sewage water pumping	Received
Municipality	Small sewage water treatment (device installation)	Received
	Small drinking water pumping	Received
	Construction permit	Received

Goldcorp will also encourage the development of a range of offshoot Cree business ventures across the region, adding to those already involved in road building and earth moving.

20.5.2. Community Consultations

During the ESIA process, public consultations were held in the communities of Wemindji and Chibougamau.

20.5.3. Archaeology

Heritage and archaeological studies have been conducted in the proposed mine area, and a total of 30 areas with archaeological potential have been identified. Of these, 11 areas have been subject to archaeological inventory during two separate programs conducted in 2007 and 2009. A number of likely heritage/archaeological sites were identified, and such sites will be protected during Project construction and operation.

20.6. Comment on Section 20

In the opinion of the QPs, the following interpretations and conclusions are applicable:

- There has been a focused effort to collect comprehensive environmental baseline data and lay the groundwork with local and regulatory stakeholders for the successful permitting and development of the Project.
- Goldcorp is of the opinion that environmental issues identified in relation to Project development have been, or can be, addressed and mitigated through a combination of baseline data collection, appropriate engineering and Project design studies, and public consultation.
- Goldcorp has been granted the global Certificate of Approval for the Project.
- Subsequently, the Certificate of Approval under Chapter I, which allows the company to proceed with the sinking a second shaft (production wells) and the construction of related infrastructure and a power plant, has been granted. A Certificate of Approval has also been granted for the access road. Additional Certificates of Approval have been lodged with the appropriate regulatory authority and some others are currently being prepared and will be submitted to the ministries during 2012.
- Key issues for Project development include the proper management of tailings and waste water, access (roads, airports), social acceptability and post-reclamation management.
- A collaboration agreement has been signed with the Grand Council of the Crees (Eeyou Istchee), the Cree Regional Authority and the Cree

Nation of Wemindji; Goldcorp is focusing on sustainable development to benefit the local communities over the long term by providing support and opportunities for direct employment, local procurement and community development projects.

- Closure costs are estimated at $40.1 million, which includes a provision for dismantling and removal of infrastructure, remediation of water ponds, the tailings storage facility and waste rock facility, soil and waste management, indirect cost, post-closure monitoring and contingency.

21. CAPITAL AND OPERATING COSTS

21.1. Capital Costs

An amount of CA$569 M was considered as initial project capital ("sunk" capital), either spent or committed to be spent, and so was not included in the economic evaluation.

Exploration expenditures were not included in the financial analysis. Exploration drilling will be carried out in the future to target mineralization that may lead to an increase in Mineral Resources. Because these future exploration drilling expenditures do not pertain to the current Mineral Reserves, they were not included in the financial model.

Capital costs are based on the latest mine construction data and budgetary figures and quotes provided by suppliers. The 2011-2013 capital costs are based on the actual costs and 2014 capital costs are based on the budget.

All capital costs are in American dollars (US$).

Capital cost estimates include funding for infrastructure, mobile equipment, development and permitting, as well as miscellaneous expenditures required to bring the Éléonore Project into commercial production.

Infrastructure requirements were incorporated into the estimates as needed. Sustaining capital costs reflect current price trends.

Over the life-of-mine (LOM), an additional US$310 M will be required as sustaining capital for underground development. The total capital cost of the Éléonore Project was estimated at US$2.16 B ($1.85 B + $310 M)

The capital cost estimate is summarized in Table 21-1, and the sustaining capital cost estimate is included as Table 21-2

21.2. Operating Costs

Operating costs were estimated by Opinaca personnel, and are based on the 2014 LOM budget. The labour cost estimation is based on Goldcorp's 2013 salary scale and fringe benefits in force. The mining consumables are based on the 2013 costs and contracts and the costs for future operation consumables, such as mill reagents, grinding media, etc, are based on recent supplier quotations.

The Éléonore Project is located at a remote site. The costs for camp accommodation, meals, employee travel and site security were included in the general and administrative (G&A) component of the estimate.

The operating cost estimate over the LOM is presented in Table 21–3 and includes allocations for processing and overhead costs.

An average overall unit cost of CA$111.51/t was estimated, comprising CA$29.89 $/t for processing, including backfill and tailings treatment and transportation, CA$52.41 $/t for mining, CA$8.76/t for services and CA$20.45/t for G&A.

Table 21-1: Initial Capital Cost Estimate

CAPITAL-PROJECT (In thousands $)			2011	2012	2013	2014	TOTAL
Mine :							
	Mine 1	Upper mine	59 597	87 655	114 070	128 625	389 947
	Mine 2	Lower mine	-	12 061	24 940	40 801	77 802
	Exploration shaft		2 402	53 285	1 417	-	57 104
	Production shaft		14 475	4 343	1 778	95	20 691
		TOTAL	**76 475**	**157 344**	**142 205**	**169 521**	**545 545**
							-
Process plant :		***TOTAL***	**1 775**	**48 258**	**232 287**	**194 764**	**477 084**
							-
Tailing :		***TOTAL***	**845**	**20 865**	**8 536**	**571**	**30 817**
On site infrastructure :							
	Site Preparation		9 761	5 945	-	-	15 707
	Sub Station 120 Kv		6 460	6 554	-	-	13 014
	Temporary Facilities & Dismantling		11 576	4 136	-	-	15 712
	Camp Construction		12 261	-	-	-	12 261
	Industriel water services		-	6 798	44 635	18 178	69 611
	Permanent Camp (400)		-	23 298	27 992	22 012	73 302
	ADM Building		-	-	16 425	8 135	24 560
	Warehouse & Garage		-	-	4 854	6 387	11 242
	Ore & Waste Pad Civil Work		-	-	5 705	-	5 705
	Assay Laboratory		-	-	6 888	-	6 888
	Others on site		12 574	23 764	10 269	13 782	60 388
		TOTAL	**52 632**	**70 495**	**116 770**	**68 493**	**308 390**
Off site infrastructure:		**TOTAL**	**20 244**	**54 520**	**11 156**	**18 826**	**104 746**
Owners Costs		***TOTAL***	**38 708**	**63 072**	**89 076**	**90 906**	**281 762**
Indirects		***TOTAL***	**13 773**	**20 379**	**27 707**	**22 873**	**84 732**
Deposits on mining interest		***TOTAL***	**-**	**27 847**	**1 665**	**(20 293)**	**9 219**
Contingency		***TOTAL***	**-**	**-**	**-**	**33 244**	**33 244**
Pre-Operating credits		***TOTAL***	**-**	**-**	**-**	**(22 827)**	**(22 827)**
		TOTAL USD $	**204 451**	**462 780**	**629 402**	**556 078**	**1 852 711**

Table 21-2: Sustaining Capital Cost Estimate

Goldcorp Eleonore Project									
Summary of Sustaining Capital Expenditures									
2014 Budget									
($000)	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**
Development	19 217	19 780	15 083	21 714	18 557	11 377	15 091	18 571	17 829
Diamond Drilling	4 286	4 286	4 286						
Shaft Sinking	21 714								
Infrastructure	9 619	5 761	1 460						
Electrical	1 400	553							
Mobile Equipment	1 241	3 050	152						
Indirect Cost	2 945	2 945	2 945						
Owner's Costs	2 539	2 539	2 539						
Contingency on major construction	6 562	2 870	1 707						
Total Capital Mine	**69 523**	**41 783**	**28 172**	**21 714**	**18 557**	**11 377**	**15 091**	**18 571**	**17 829**
Other Sustaining									
Production Shaft change over	5 714	5 714							
Process Plant - Fix Equipment		95	95	4 286	476	476	238	95	100
Process Plant - Mobile Equipment	2 724	2 724	1 591			476	71		
Process Plant - Oxygen Plant	857	857	857	857	857	857	857	857	900
Tailings				8 032				7 922	
Surface Mobile Equipments		190	143	381		519			
Camp and Infrastructure	476	476	476	476	476	476	476	476	500
Acc. Meals & Site Travel	3 728	2 008	1 291	1 597	1 018	680	855	1 449	1 033
Grand Total	**83 022**	**53 848**	**32 625**	**37 343**	**21 385**	**14 862**	**17 589**	**29 371**	**20 362**

Table 21-3: Operating Cost Estimates

SUMMARY OPERATING COSTS										
	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Tonnes Milled	288 000	1 957 622	1 845 443	2 059 268	2 545 450	2 587 912	2 544 682	2 425 637	1 916 077	1 098 008
Total tonnes milled	**288 000**	**1 957 622**	**1 845 443**	**2 059 268**	**2 545 450**	**2 587 912**	**2 544 682**	**2 425 637**	**1 916 077**	**1 098 008**
	2014 ($/t)	**2015 ($/t)**	**2016 ($/t)**	**2017 ($/t)**	**2018 ($/t)**	**2019 ($/t)**	**2020 ($/t)**	**2021 ($/t)**	**2022 ($/t)**	**2023 ($/t)**
Process Plant	$ 38,78	$ 33,69	$ 32,74	$ 31,61	$ 31,02	$ 28,48	$ 27,62	$ 27,28	$ 26,87	$ 29,66
Mining Operations	$ 88,11	$ 60,45	$ 57,79	$ 53,58	$ 48,08	$ 48,66	$ 51,32	$ 49,92	$ 48,91	$ 50,54
Site Services	$ 14,48	$ 11,29	$ 11,12	$ 9,96	$ 8,51	$ 7,30	$ 7,65	$ 7,58	$ 7,64	$ 7,68
General & Administration	$ 34,41	$ 26,20	$ 25,81	$ 23,75	$ 17,91	$ 18,46	$ 17,91	$ 17,91	$ 17,91	$ 17,91
Total	**$ 175,78**	**$ 131,63**	**$ 127,46**	**$ 118,91**	**$ 105,52**	**$ 102,91**	**$ 104,50**	**$ 102,69**	**$ 101,32**	**$ 105,79**

21.3. Comment on Section 21

In the opinion of the QPs:

- The capital cost estimates are based on a combination of quotes, vendor pricing, and Goldcorp's experience with similar-sized operations.
- The capital cost estimates include direct and indirect costs.
- The total capital cost estimate is US$1.85 B.
- Operating costs were based on estimates from first principals for major items; the costs include allowances or estimates for minor costs.
- The estimated average annual operating cost is US$204.6 M or US$106.20/t milled.

Capital and operating costs have been appropriately estimated and are acceptable within the estimated range of accuracy and can support the financial analysis and mineral reserve estimation

22. ECONOMIC ANALYSIS

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented in this report.

Forward-looking statements in this section include, but are not limited to, statements with respect to the future price of gold, the estimation of Mineral Reserves and Mineral Resources, the realization of Mineral Reserve estimates, the timing and amount of estimated future production, production costs, capital expenditures, costs and timing of the development of new deposits, success of exploration activities, permitting timelines, currency exchange rate fluctuations, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, title disputes or claims, and limitations on insurance coverage.

Additional risk can come from actual results of current exploration activities; actual results of current reclamation activities; conclusions of economic evaluations; changes in the project parameters as plans continue to be refined, possible variations in ore reserves, grade or recovery rates; failure of plant, equipment or processes to operate as anticipated; accidents, labour disputes and other risks of the mining industry; and delays in obtaining governmental approvals.

22.1. Basis of Estimate

The financial analysis of the Project was carried out using a discounted cash flow model prepared in Microsoft Excel. The model was constructed using annual cash flows in 2013 constant dollar terms. No provisions were made for the effects of inflation or de-escalation for the value of tax losses carried forward.

22.2. Taxation Considerations

A tax rate of 40% was used for the financial analysis. Both federal and provincial income taxes were calculated on the mine net income, after calculation of Quebec mining duties.

22.3. Royalty Considerations

A royalty payable on production from Éléonore is set at 2.20% on the first three million ounces of gold (3 Moz Au), and increases by 0.25% per million ounces thereafter, up to a maximum of 3.50%.

The royalty payable in each period is adjusted up or down by an amount ranging

between zero and 10%, depending on the gold price in effect during that period (Table 22-1).

22.4. Cree Payment

An annual payment related to the Cree Collaboration Agreement (refer to Section 4.5) is included in the financial model.

22.5. Capital and Operating Costs

The total capital cost of the Éléonore Project was estimated at US$2.16 B (US$1.85 B in direct capital costs and an additional US$310 M of sustaining capital).

An average overall unit cost of CA$111.51/t was estimated, comprising CA$29.89/t for processing, including backfill and tailings treatment and transportation, CA$52.41/t for mining, CA$8.76/t for services and CA$20.45/t for G&A.

22.6. Other Assumptions

Cash flows were based on the Mineral Reserves outlined in Table 15-1.

The gold price was set at US$1,300/oz Au and the CAD/USD currency exchange rate at 1.05.

The gold refining charge was estimated at US$1.75/oz Au. Bullion delivery charges were estimated at CA$1.67/oz Au.

22.7. Financial Analysis

The internal rate of return (IRR) over the LOM is 3.86%. The Éléonore Project is expected to generate an after-tax IRR 3.15. The projected payback period is eight (8) years. The projected cash flow on an annualized basis for the Project is shown in tables 22-2 and 22-3. Table 22-2 includes three (3) years of pre-production, and five (5) years of mine production. Table 22-3 includes five (5) years of mine production and mine close-out.

Table 22-1: Royalty Adjustment with Gold Price.

Gold Price (US$/oz)	Royalty Adjustment
X ≤ 350	10%
350 < X ≤ 400	-5%
400 < X ≤ 450	0%
450 < X ≤ 500	+5%
X > 500	+10%

Table 22-2: Cashflow Analysis Years 1 to 8 (includes three years of pre-production)

Descripiton	unit	-4 2011	-3 2012	-2 2013	-1 2014	1 2015	2 2016	3 2017	4 2018
Statistics					Pre-Production				
Working days at the mine					61	365	365	365	365
Working days at the mill					61	350	350	350	350
Tonnes per day at the mill					4 721	5 593	5 273	5 884	7 273
Mill Production									
Tonnage		0	0	0	288 000	1 957 622	1 845 443	2 059 268	2 545 450
Grade	1	0,000	0,000	0,000	7,20	5,94	6,14	6,25	6,35
Recovery	93,5%				93,5%	93,5%	93,5%	93,5%	93,5%
Oz produced		0	0	0	62 334	349 501	340 861	386 785	485 594
		0	0	0	62 334	411 836	752 697	1 139 482	1 625 076
					0,00000	0,00000	0,00000	0,00000	0,00000
1 Exchange rate	CDN$ / US$	1,00	1,00	1,00	1,05	1,05	1,05	1,05	1,05
1 Price of Gold	US$				1 300	1 300	1 300	1 300	1 300
Gross Value $US			0	0	81 034 696	454 351 729	443 119 155	502 820 417	631 272 658
Royalty - pre-payment recovery					1 782 763	3 217 237			
Royalty $US	2,20%		0	0	-1 782 763	-9 995 738	-9 748 621	-11 062 049	-13 887 998
Refining charges	1,75 $us/oz		0	0	-109 085	-611 627	-596 507	-676 874	-849 790
Bullion delivery	1,67 $ca/oz		0	0	-99 141	-555 874	-542 131	-615 172	-772 326
Net Revenue $US			0	0	80 826 470	446 405 727	432 231 896	490 466 322	615 762 543
Operating Cost									
Total Operating Cost per Tonn Milled					175,78	130,90	126,68	118,22	104,96
Operating Costs	1				50 624 640	256 258 435	233 782 934	243 442 060	267 169 866
Other Payments						1 432 800	1 432 800	1 432 800	1 432 800
Total Operating Cost - CDN$		0	0	0	50 624 640	257 691 235	235 215 734	244 874 860	268 602 666
Cost / ton milled (CDN$)					175,78	131,63	127,46	118,91	105,52
* Cost / oz produced (CDN$) including Royalty payment					844,09	769,25	722,00	665,04	585,08
Operating earnings (loss) - CDN$					**30 201 830**	**188 714 492**	**197 016 162**	**245 591 462**	**347 159 877**
Total Operating Cost - US$		0	0	0	48 213 943	245 420 223	224 014 985	233 214 153	255 812 063
Cost / ton milled (US$)					167,41	125,37	121,39	113,25	100,50
Cost / oz produced (US$) including Royalty payment					803,89	732,62	687,62	633,38	557,22
Operating earnings (loss) - USD$					**28 763 648**	**179 728 088**	**187 634 440**	**233 896 630**	**330 628 454**
Descripiton	**unit**	**-4 2011**	**-3 2012**	**-2 2013**	**-1 2014**	**1 2015**	**2 2016**	**3 2017**	**4 2018**
Total Capital Expenditure - US$		204 451 000	462 780 000	629 402 000	556 078 000	83 022 201	53 847 932	32 625 082	37 343 340
Cost / ton milled (US$)									14,67
Cost / oz produced (US$)									76,90
Cash Flow									
Revenus $US		0	0	0	0	446 405 727	432 231 896	490 466 322	615 762 543
Operating Costs $US		0	0	0	0	245 420 223	224 014 985	233 214 153	255 812 063
Capital Expenditure $US		204 451 000	462 780 000	629 402 000	556 078 000	83 022 201	53 847 932	32 625 082	37 343 340
Asset Selling									
Reclamation cost / incured (spent)			0	0	0	0	0	0	0
Total Cash Flow before Taxe (US$)		-204 451 000	-462 780 000	-629 402 000	-556 078 000	117 963 302	154 368 979	224 627 087	322 607 140
NPV before tax	5% (110 043 170,44) $								
IRR before taxes	3,86%								
Tax Burden									
Depreciation on current year									
Openning Balance		0	204 451 000	667 231 000	1 296 633 000	1 821 969 735	1 724 775 295	1 597 374 625	1 420 130 949
Addition		204 451 000	462 780 000	629 402 000	556 078 000	83 022 201	53 847 932	32 625 082	37 343 340
Depreciation		0	0	0	-30 741 265	-180 216 641	-181 248 603	-209 868 758	-270 411 015
Closing balance		204 451 000	667 231 000	1 296 633 000	1 821 969 735	1 724 775 295	1 597 374 625	1 420 130 949	1 187 063 274
Operating earnings (loss)		0	0	0	28 763 648	179 728 088	187 634 440	233 896 630	330 628 454
Taxable income after depreciation		**0**	**0**	**0**	**-1 977 617**	**-488 554**	**6 385 837**	**24 027 872**	**60 217 440**
Operating Earnings		**0**	**0**	**0**	**28 763 648**	**179 728 088**	**187 634 440**	**233 896 630**	**330 628 454**
Tax Depreciation per below		**0**	**0**	**0**	**28 763 648**	**179 728 088**	**187 634 440**	**233 896 630**	**330 628 454**
Taxable Income		**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Tax Net Book Value (UCC)									
Beginning Balance	$ 364 933 000	364 933 000,0	467 158 500,0	800 774 000,0	1 346 865 000,0	1 910 841 352,5	2 050 663 365,3	1 931 463 992,0	1 740 803 869,0
Capital Cost Increase	50%	102 225 500,0	333 615 500,0	546 091 000,0	592 740 000,0	319 550 100,5	68 435 066,7	43 236 507,4	34 984 211,3
Tax Net Book Value (UCC)		467 158 500,0	800 774 000,0	1 346 865 000,0	1 939 605 000,0	2 230 391 453,0	2 119 098 432,0	1 974 700 499,3	1 775 788 080,2
Tax Depreciation	30%	0,0	0,0	0,0	28 763 647,5	179 728 087,6	187 634 440,0	233 896 630,4	330 628 454,4
Ending Balance		467 158 500,0	800 774 000,0	1 346 865 000,0	1 910 841 352,5	2 050 663 365,3	1 931 463 992,0	1 740 803 869,0	1 445 159 625,8
Total Taxe payable	40,0%	0	0	0	0	0	0	0	0
Total Cash Flow after taxes (US$)		-204 451 000	-462 780 000	-629 402 000	-556 078 000	117 963 302	154 368 979	224 627 087	322 607 140
NPV after taxes	5% (172 118 144,86) $								
IRR after taxes	3,15%								

Table 22-3: Cash flow analysis of Years 9 to 14 (includes one year of post-production close-out)

Descripiton	unit	5 2019	6 2020	7 2021	8 2022	9 2023	Total PRODUCTION
Statistics							
Working days at the mine		365	365	365	365	365	
Working days at the mill		350	350	350	350	350	
Tonnes per day at the mill		7 394	7 271	6 930	5 475	3 137	
Mill Production							
Tonnage		2 587 912	2 544 682	2 425 637	1 916 077	1 098 008	19 268 101
Grade	1	6,6	6,3	6,0	7,0	8,8	6,486
Recovery	93,5%	93,5%	93,5%	93,5%	93,5%	93,5%	93,5%
Oz produced		515 146	480 821	439 768	404 623	291 327	3 756 761
		2 140 222	2 621 043	3 060 811	3 465 434	3 756 761	
		0,00000	0,00000	0,00275	0,00275	0,00275	
1 Exchange rate	CDN$ / US$	1,05	1,05	1,05	1,05	1,05	
1 Price of Gold	US$	1 300	1 300	1 300	1 300	1 300	
Gross Value $US		669 689 896	625 066 971	571 698 795	526 010 425	378 724 980	4 883 789 724
Royalty - pre-payment recovery							5 000 000
Royalty $US	2,20%	-14 733 178	-13 751 473	-14 149 545	-13 018 758	-9 373 443	-111 503 568
Refining charges	1,75 $us/oz	-901 506	-841 436	-769 595	-708 091	-509 822	-6 574 332
Bullion delivery	1,67 $ca/oz	-819 328	-764 734	-699 441	-643 544	-463 349	-5 975 039
Net Revenue $US		653 235 885	609 709 328	556 080 214	511 640 033	368 378 366	4 764 736 784
Operating Cost							
Total Operating Cost per Tonn Milled		102,36	103,93	102,10	100,57	101,20	
Operating Costs	1	264 886 211	264 481 045	247 659 168	192 708 111	111 113 499	2 132 125 968
Other Payments		1 432 800	1 432 800	1 432 800	1 432 800	5 047 065	16 509 465
Total Operating Cost - CDN$		266 319 011	265 913 845	249 091 968	194 140 911	116 160 564	2 148 635 433
Cost / ton milled (CDN$)		102,91	104,50	102,69	101,32	105,79	111,51
* Cost / oz produced (CDN$) including Royalty payment		548,92	584,98	601,93	515,32	434,24	604,96
Operating earnings (loss) - CDN$		**386 916 874**	**343 795 483**	**306 988 246**	**317 499 121**	**252 217 802**	**2 616 101 350**
Total Operating Cost - US$		253 637 153	253 251 281	237 230 446	184 896 106	110 629 109	2 046 319 460
Cost / ton milled (US$)		98,01	99,52	97,80	96,50	100,75	106,20
Cost / oz produced (US$) including Royalty payment		522,78	557,13	573,27	490,78	413,57	
Operating earnings (loss) - USD$		**368 492 261**	**327 424 270**	**292 369 759**	**302 380 116**	**240 207 430**	**2 491 525 096**
Descripiton	**unit**	**5 2019**	**6 2020**	**7 2021**	**8 2022**	**9 2023**	**Total PRODUCTION**
Total Capital Expenditure - US$		21 384 896	14 861 649	17 589 132	29 370 701	20 241 026	2 162 423 959
Cost / ton milled (US$)		8,26	5,84	7,25	15,33	18,43	112,23
Cost / oz produced (US$)		41,51	30,91	40,00	72,59	69,48	575,61
Cash Flow							
Revenus $US		653 235 885	609 709 328	556 080 214	511 640 033	368 378 366	4 683 910 314
Operating Costs $US		253 637 153	253 251 281	237 230 446	184 896 106	110 629 109	1 998 105 518
Capital Expenditure $US		21 384 896	14 861 649	17 589 132	29 370 701	20 241 026	2 162 996 959
Asset Selling							
Reclamation cost / incured (spent)		0	0	0	0	2 247 600	2 247 600
Total Cash Flow before Taxe (US$)		378 213 836	341 596 399	301 260 637	297 373 226	235 260 632	520 560 238
NPV before tax 5%	(110 043 170,44) $						
IRR before taxes	3,86%						
Tax Burden							
Depreciation on current year							
Openning Balance		1 187 063 274	916 412 902	654 277 815	411 709 384	184 637 459	
Addition		21 384 896	14 861 649	17 589 132	29 370 701	20 241 026	
Depreciation		-292 035 269	-276 996 735	-260 157 563	-256 442 626	-204 878 484	
Closing balance		916 412 902	654 277 815	411 709 384	184 637 459	0	
Operating earnings (loss)		368 492 261	327 424 270	292 369 759	302 380 116	240 207 430	
Taxable income after depreciation		**76 456 992**	**50 427 535**	**32 212 195**	**45 937 490**	**35 328 946**	**328 528 137**
Operating Earnings		**368 492 261**	**327 424 270**	**292 369 759**	**302 380 116**	**240 207 430**	
Tax Depreciation per below		**368 492 261**	**327 424 270**	**243 886 763**	**177 764 709**	**131 877 055**	
Taxable Income		**0**	**0**	**48 482 996**	**124 615 407**	**108 330 375**	
Tax Net Book Value (UCC)							
Beginning Balance	$ 364 933 000	1 445 159 625,8	1 106 031 482,7	796 730 485,4	569 069 112,9	414 784 320,3	0
Capital Cost Increase	50%	29 364 118,2	18 123 272,4	16 225 390,1	23 479 916,2	24 805 863,1	0
Tax Net Book Value (UCC)		1 474 523 744,0	1 124 154 755,1	812 955 875,5	592 549 029,0	439 590 183,4	0
Tax Depreciation	30%	368 492 261,3	327 424 269,7	243 886 762,7	177 764 708,7	131 877 055,0	0
Ending Balance		1 106 031 482,7	796 730 485,4	569 069 112,9	414 784 320,3	307 713 128,4	0
Total Taxe payable	40,0%	0	0	19 393 198	49 846 163	43 332 150	**112 571 512**
Total Cash Flow after taxes (US$)		378 213 836	341 596 399	281 867 439	247 527 063	191 928 482	**407 988 727**
NPV after taxes 5%	(172 118 144,86) $						
IRR after taxes	3,15%						

Goldcorp have performed a conceptual review of the likely impact of production rate increases on the mine plan prior to commencement of major plant and infrastructure construction activities. In this concept study, mining activities were assumed to occur on four mining fronts which would result in a doubling of the production rate from the current 3,500 t/d from two fronts to 7,000 t/d. Goldcorp have stated that the company's objective will now be to focus on drilling activities from underground, when the shaft reaches the appropriate depth, to delineate additional mineralization that can be used to achieve the increased production rate. The company has determined that construction activities will assume that the 7,000 t/d rate can be achieved, and that the plant and infrastructure will be sized and built accordingly.

Cash flow fluctuations during the LOM primarily result from fluctuations in the sustaining capital and mill head grade.

Inferred Mineral Resources above the cut-off grade were treated as "waste" in this evaluation. This mineralization represents upside potential for the Éléonore Project if some or all of the Inferred Mineral Resources identified within the LOM production plan can be upgraded to higher-confidence mineral resource categories, and eventually to Mineral Reserves.

22.8. Sensitivity Analysis

A sensitivity analysis was carried out for the base case scenario described above to test the sensitivity of the Éléonore Project's after-tax internal rate of return (IRR) to a 15% and 30% change in gold price, operating costs and sustaining capital (Figure 22-1).

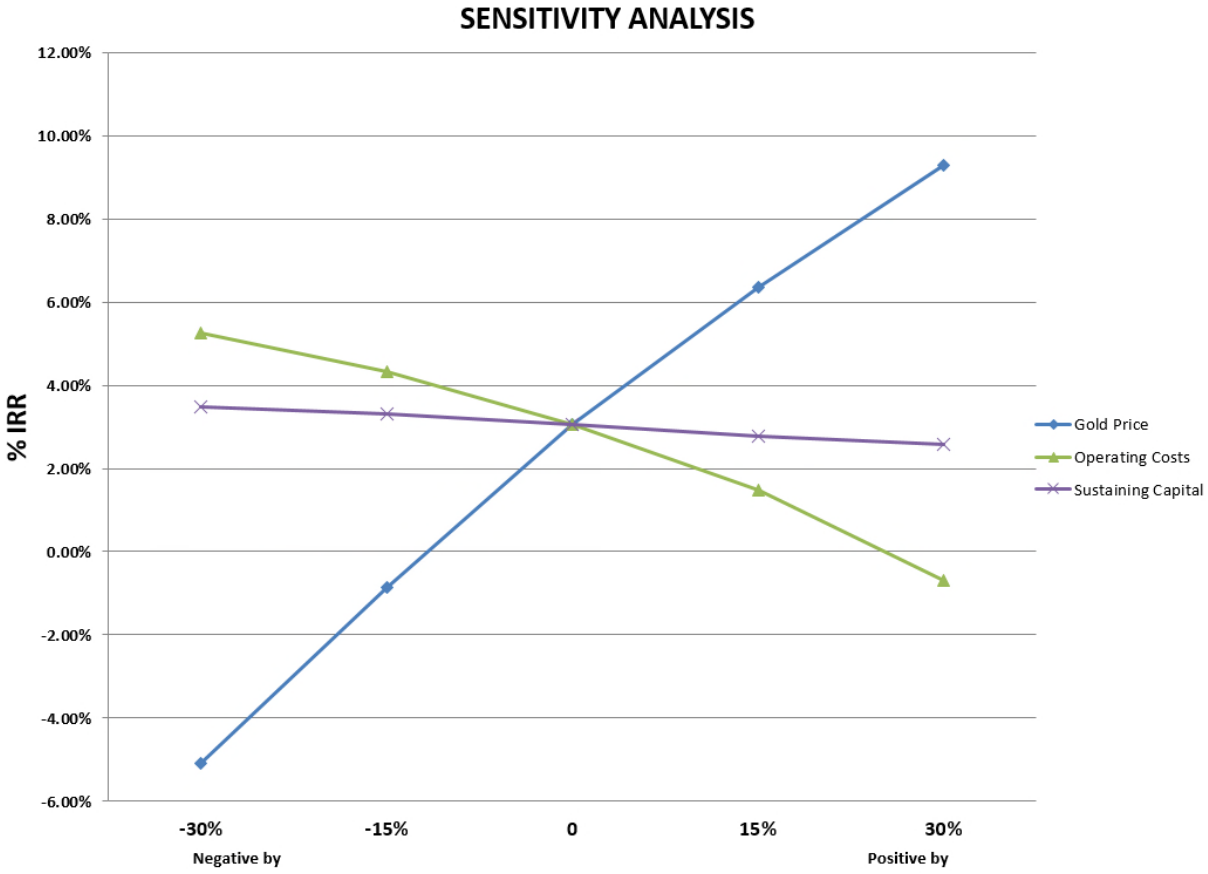
The Project is most sensitive to the gold price. It is less sensitive to changes in capital costs and operating costs.

22.9. Comment on Section 22

In the opinion of the QPs, the following interpretations and conclusions are appropriate:

- Based on the assumptions detailed herein, the Éléonore Project is expected to yield an after-tax IRR of 3.15%.
- The projected payback period is eight (8) years.
- Cash flow fluctuations during the LOM primarily result from fluctuations in the sustaining capital and mill head grade.
- The Project is most sensitive to the gold price. It is less sensitive to changes in capital costs and operating costs.

Figure 22-1: Sensitivity analysis



23. ADJACENT PROPERTIES

23.1. Opinaca A and B Properties

The following description of the Opinaca A and B properties has been summarized from the Management's Discussion and Analysis (2013 Annual Report) of Azimut Exploration Inc ("Azimut"), published on the SEDAR website.

Azimut acquired extensive holdings both before and after the 2004 Éléonore discovery. The Opinaca A property is adjacent to Goldcorp's Éléonore property, and Opinaca B comprises two claim blocks to the east. In April 2010, Azimut confirmed that its partner Everton Resources Inc. ("Everton") had earned its 50% interest on the Opinaca A and Opinaca B properties. In September 2010, the properties became subject to a three-way agreement between Azimut, Everton and Aurizon Mines Ltd ("Aurizon"; now Hecla Quebec Inc.: "Hecla") whereby Hecla has the option to acquire a 50% interest in each of the two properties.

Exploration on the Opinaca A property began in 2005 when surface prospecting revealed a 1.7-km trend of gold showings with values up to 50.9 g/t Au, named the Inex Zone. During the summer of 2007, follow-up prospecting work led to the discovery of the Charles Target in the central part of Opinaca A and results yielded up to 35.9 g/t Au. That same year, a major gold-bearing system was identified at the Claude Target on the Opinaca B property, which appeared to be part of a much larger NE–SW to E–W gold trend extending from the Manuel showing on Everton's adjacent Wildcat property (12.0 g/t Au over 4.6 m in a channel sample) to Azimut's Eleonore South property west of Opinaca B. In the winter of 2007, Everton, as operator, completed two drilling programs totalling 3,390 m on both properties to cover all major targets. Assay results included 0.22 g/t Au over 187 m, including 1.0 g/t Au over 21.5 m at the Claude Target. During the winter programs of 2008, Everton Resources Inc. carried out ground geophysics (160 line-km of IP and magnetics) and 1,600 m of follow-up diamond drilling (Charles, Smiley and Lola targets at Opinaca A; Dominic Target at Opinaca B). Drilling results included 4.2 g/t Au over 1 m and 0.4 g/t Au over 1 m on the Smiley Target, and 0.6 g/t Au over 0.3 m and 0.6 g/t Au over 1.2 m on the Dominic Target.

In 2010, Aurizon became project operator according to the terms of the three-way option agreement. In 2011, Aurizon carried out a $1.0 M exploration program consisting of surface sampling, geophysical surveys and 2,000 m of drilling. In 2012, Aurizon followed up with an extensive program comprising 622 line-km of helicopter-borne magnetic-EM surveying, 684 soil samples, 243 rock grab samples, 290 rock channel samples from 258.35 m of channels, and 93 till samples (press release of November 19, 2012). The main results were the discovery of three new outcropping targets on the Opinaca B property. These

targets, located along an E–W structure at the boundary between the Opinaca and La Grande geological subprovinces, share geological similarities with Goldcorp's Éléonore Project.

The D8 Trench displays a 20-m-wide sheared and altered sedimentary unit with amphibolite and quartz-tourmaline veinlets. Best channel sampling results include 2.3 g/t Au over 1.0 m and 0.55 g/t Au over 4.0 m. This area presents gold anomalies in soil and till. The Eric Prospect, less than 1 km north of D8, yielded eight (8) bedrock samples with values above 0.1 g/t Au, including two above 0.5 g/t Au. The gold-bearing samples were collected within a kilometre-scale arsenic-gold soil geochemistry target. Mineralization is related to calc-silicate-altered sediments and arsenopyrite-tourmaline-bearing pegmatites. The Penelope Prospect,

about 1 km west of the Eric and D8 targets, yielded ten (10) bedrock samples with values above 0.1 g/t Au, including four (4) with values above 0.5 g/t Au and up to 4.26 g/t Au. Mineralization is associated with quartz-tourmaline veins and veinlets.

23.2. Opinaca D Property

The following description of Opinaca D property has been summarized from the Management's Discussion and Analysis (2013 Annual Report) of Azimut Exploration Inc. ("Azimut"), published on the SEDAR website.

Azimut's wholly-owned Opinaca D Project is about 8 km northwest of Goldcorp's Éléonore property. Exploration on the Opinaca D property began in 2005 and has included reconnaissance geological mapping and prospecting over a number of exploration targets defined by VTEM and/or soil geochemistry anomalies. The soil geochemistry surveys confirmed a broad trend of gold, arsenic and antimony anomalies, with respective maximum values of 7.32 g/t Au, 447 ppm As, and 2.3 ppm Sb. The strong gold-arsenic-antimony soil anomalies have not yet been tested by drilling. Several drill targets have been defined on the project.

23.3. Eleonore South Property

The following description of Eleonore South property has been summarized from the Management's Discussion and Analysis (2013 Annual Report) of Azimut Exploration Inc. ("Azimut"), published on the SEDAR website.

The Eleonore South property is covered by a three-party agreement between Azimut, Les Mines Opinaca Ltée (a wholly-owned subsidiary of Goldcorp) and Eastmain Resources Inc. ("Eastmain"). Eastmain is the project operator.

Major exploration programs (prospecting, geophysics, trenching and drilling, funded by Azimut's partners) have mainly focused on the JT Gold Zone, which is characterized by altered, sulphide-bearing metasedimentary rocks comparable to those hosting the Roberto gold deposit 12 km to the northwest. Drilling and trenching on the JT Zone have defined a 1.2 km x 100 m auriferous halo, also comparable in nature to the geochemical halo surrounding the Roberto gold deposit.

From 2006 to 2010, thirty-five (35) trenches were excavated on the Eleonore South property and 5,063 one-metre (1-m) channel samples were collected. The most significant channel result was 5.3 g/t Au over 8 m on the JT Zone. Diamond drilling programs in 2008, 2009 and 2010 tested several high-priority sediment-hosted gold targets. The most significant result from the 2008 program (16 holes; 3,129 m) was 1.5 g/t Au over 5.7 m in the JT Zone. During the $1.6-million program in 2009 (14 holes; 3,697 m), nine (9) of the twelve (12) holes in the JT area intersected wide intervals of gold-bearing sedimentary rocks along a 1-km-long corridor. The most significant result was 1.40 g/t Au over 10.0 m. The $1.6 million exploration program in 2010-2011 focused on drill-testing the extensions of the JT Zone with the aim of determining ore grade thicknesses, as well as testing other priority targets elsewhere on the property. A number of attractive areas remain untested by drilling.

As at Q1 2014, Goldcorp and Eastmain have disbursed $3,600,000 in cumulative work expenditures as part of the joint venture, and ownership of the property is currently as follows: Azimut 26.0%, Goldcorp 37.0% and Eastmain 37.0%. Eastmain, as operator, is currently completing a $250,000 program funded by Goldcorp and Eastmain.

23.4. Opinaca Property

The following description of the Opinaca property was summarized from the Management's Discussion and Analysis (2013 Annual Report) and various press releases of Beaufield Resources Inc. ("Beaufield"), published on the SEDAR website.

Beaufield's wholly-owned Opinaca property forms part of the western boundary with Goldcorp's Éléonore Project. Most of the anomalous gold values obtained from the Opinaca property were from the Vortex Zone discovered in 2005.

In January 2006, Goldcorp invested in Beaufield through a brokered private placement to have a long-term relationship with Beaufield as owner of a geologically important property adjoining the Éléonore property. The subscription agreement provides that Beaufield and Goldcorp cooperate on technical, financial and permitting issues relating to Beaufield's Opinaca property. This agreement

allows both companies to pool their exploration efforts in the camp. Beaufield will continue to control and explore its Opinaca property. Beaufield completed its 2013 summer geology sampling and geochemical program. A total of 1,008 rock samples were collected and assayed, with areas of significantly anomalous gold and arsenic being detected. The best samples were approximately 2 g/t Au. A soil geophysical grid covered the Snoopy area where 972 B-horizon soil samples were collected.

In 2007, Beaufield completed a 12-hole (3,252 m) diamond drilling program. Three (3) holes were drilled to test the Vortex area, which previously returned 9.9 g/t Au and 9.5 g/t Au in outcrop and 1.1 g/t Au over 20 m in DDH OP-06-01. Drill hole OP-07-19 intersected three (3) mineralized and altered zones. The mineralization and alteration context (pyrrhotite and arsenopyrite in amphibolitized and tourmalinized rocks) is similar to that of the Roberto East Zone on Goldcorp's Éléonore Project. Drill Hole 07-20 intersected a pegmatitic dyke returning values of 5.0 g/t Au, 3.1 g/t Ag and 0.11% Mo over 0.3 m, and mineralized conglomerates between 264 to 265 m that graded 5.8 g/t Au over 1 m.

A 2008 summer program of geology and geochemistry succeeded in extending the 300-m-long surface discovery exposure of the Vortex Zone by another 4,700 m for a total strike length of 5 km. Samples assaying up to 5 g/t Au were collected. A geochemical survey consisting of 971 soil samples were collected on a grid covering the Kessel and Ylesia gold-copper-molybdenum zone. In June 2009, Beaufield carried out a field geological and prospecting program.

In June 2010, an Induced Polarization (IP) geophysical survey totaling 30.5 km was carried out over the Vortex Zone. Beaufield's geologists collected 296 grab samples over the area covered by the survey: 56 samples returned values greater than 100 ppb Au, 22 greater than 500 ppb Au, and 17 above 1 g/t Au. The best results were: 19.00 g/t Au, 2.50 g/t Au, 2.40 g/t Au and 2.27 g/t gold.

In July 2013, Beaufield conducted a drilling program consisting of 13 holes totalling 2,321 m and targeted a series of geochemical, geophysical, structural and geological anomalies. Hole OP-13-22 intersected several gold bearing tourmaline, quartz and arsenopyrite veins hosted in mafic volcanic rocks. Several thick, semi-massive to massive pyrrhotite horizons occurs at the contact with a diorite intrusion with associated silver anomalies. Following this program, the

technical team concluded that the northern portion of the property shows features of porphyry-type mineralization. Such features include sericite alteration, locally very-fine veins in stockwork, silver and copper anomalies. Previous work has also shown significant molybdenum mineralization (up to 12% Mo) in the same area.

This porphyry intrusion (monzonite) is in contact with Éléonore-type sedimentary rocks.

23.5. Wildcat Property

The following description of the Wildcat property was summarized from the Management's Discussion and Analysis of Everton Resources Inc. ("Everton") and various press releases, published on the SEDAR website.

Everton's wholly-owned Wildcat property comprises six different blocks of mining claims. Exploration work conducted to date by Everton led to the discovery of the Inex Extension gold zone, which is located on trend with the Inex gold zone on the Opinaca A property (Azimut-Everton) along a 2.8 km long gold corridor, and the Manuel gold prospect which returned 12.0 g/t Au over 4.6 m in a channel sampling. Hecla Quebec Inc. ("Hecla") has the option to earn up to a 65% interest in the Wildcat Project from Everton.

During 2011, an extensive exploration program was conducted on the Wildcat property, including 2,000 m of drilling. During the summer of 2012, the exploration program consisted of surface sampling and geochemical analyses, and geochemical surveys. A total of 728 line-km of helicopter-borne magnetic-EM survey were carried out by Hecla, and a total of 303 soil samples, 763 rock grab samples, 15 channel samples and 146 till samples were also collected.

23.6. Cheechoo B West Property

The following description of the Cheechoo Project was summarized from the Management's Discussion and Analysis (Q1 2013) of Sirios Resources Inc. ("Sirios"), published on the SEDAR website.

Sirios acquired claims forming the Cheechoo and Shark properties (collectively the Cheechoo Project) by map designation in 2004. In December of the same year, an option agreement was signed with Golden Valley Ltd ("Gold Valley") who in 2009, after executing fieldwork totalling almost $4 M, acquired a 60% interest in the Cheechoo Project. In June 2012, Sirios concluded a new agreement with Golden Valley to re-acquire the project. The agreement allowed Sirios to increase its participation from 40% to 45% by executing fieldwork. Today, the Cheechoo B West property, adjacent to the east boundary of Goldcorp's Éléonore Project, is jointly held by Sirios (45%) and Golden Valley Mines (55%). The claims are adjacent to the east boundary of Goldcorp's Éléonore Project.

The work carried out between June and October 2012 included line cutting and geophysical surveying (IP and ground Mag) were performed on an area roughly 12 km^2. The region is located around 10 km southeast of the Roberto deposit. A total of eight (8) diamond drill holes of NQ caliber were completed for a total of

938 m during the first diamond drilling program in November 2012. The results from the first three (3) drill holes confirmed a significant gold discovery. The best result was 0.65 g/t Au over 128.6 m in hole CH-919-12-03. A NI 43-101 technical report was filed by Sirios on June 14, 2013 on SEDAR website.

During fall 2013, for the first time, a trail that allows land access to the property was completed as well as a camp that enabled the second drilling campaign. Also, in September 2013, following a new visual exam of some drilling samples from the 2012-2013 campaign, the presence of visible gold had been detected in three witness half drill core samples.

The second drilling program of Sirios was finished at the end of October after the execution of four holes of NQ caliber, totalizing 750 m. More than eight new gold zones were intersected by three drill holes that were aiming and that intersected tonalitic intrusion. These new holes confirmed a low grade gold envelope with a minimum area of 250 m by 525 m and a depth of up to 200 m vertically. The best result was 0.89 g/t Au over 57.0 m in hole CH-919-13-09.

23.7. Éléonore Régional Property

The following description of Éléonore Régional property was summarized from the website of Virginia Mines Inc. ("Virginia Mines"). Owned 100% by Virginia Mines, the Éléonore Régional property is located west of Goldcorp's Éléonore Project. The geological environment is similar to the Roberto deposit. The two main mineralized zones on the Éléonore Régional property were discovery by Virginia Mines. The first zone consists of quartz-tourmaline veins that yielded 1.85 g/t, 2.09 g/t and 2.95 g/t Au in grab samples from a dioritic intrusion. The second zone consists of a 500-km^2 mineralized boulder field in the central portion of the property. These boulders of graywacke containing 2-3% finely disseminated pyrite, returned values of up to 7 g/t Au. Drill holes testing this auriferous boulder field intercepted lithologies similar to that of the auriferous blocks, but no significant values were reported. Several areas have seen very little exploration.

23.8. Comment on Section 23

In the opinion of the QPs, the following interpretations and conclusions are appropriate:

- The QPs have been unable to verify the above information on the adjacent properties.
- The presence of significant mineralization on these adjacent properties is not necessarily indicative of similar mineralization on the Éléonore Project.

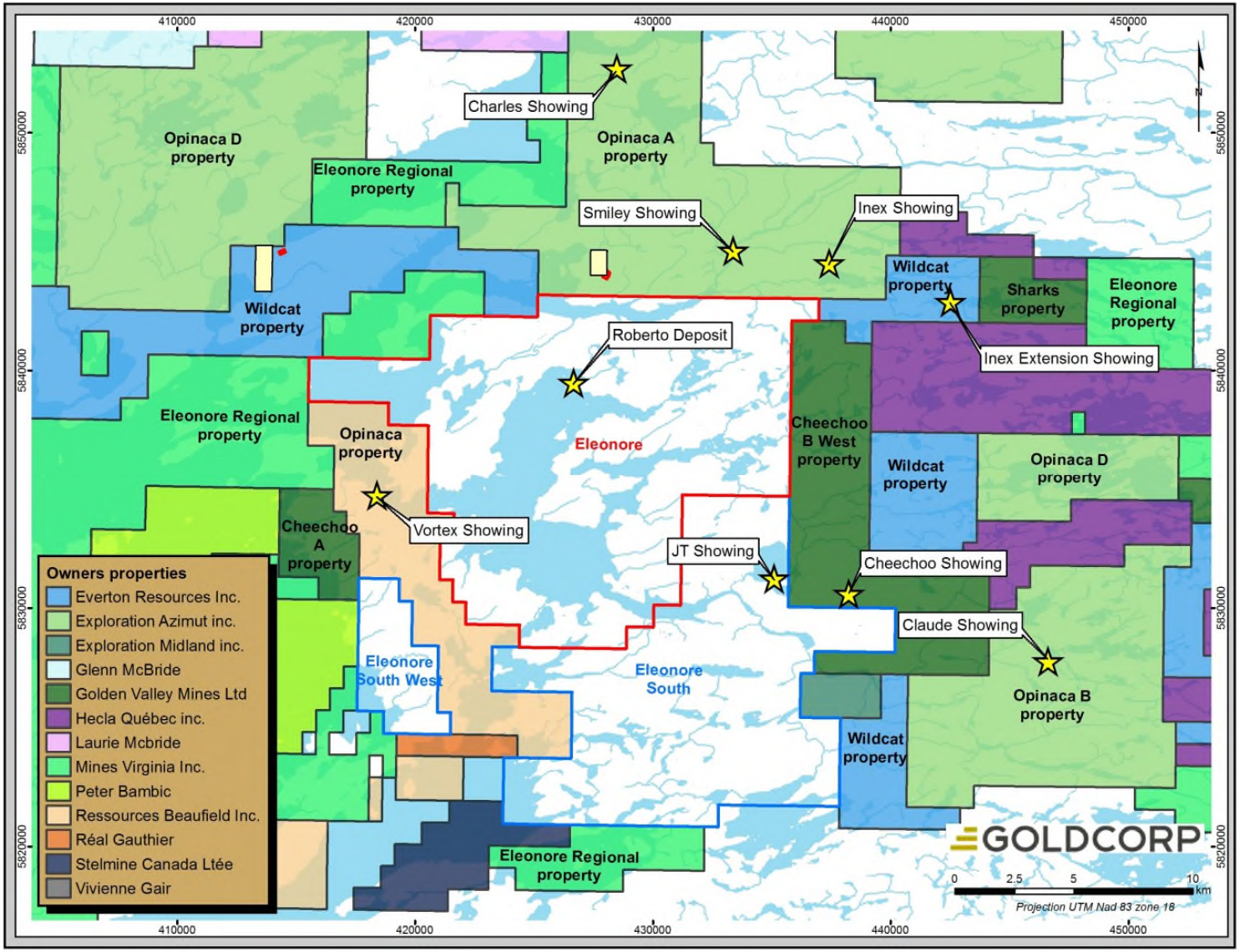


Figure 23-1: Éléonore Project and adjacent properties

24. OTHER RELEVANT DATA AND INFORMATION

Because the most favourable extensions of the Roberto Deposit are at depth, surface drilling is becoming increasingly difficult. Since the end of 2012, all drilling has been completed from underground infrastructure, namely the 650 level and the ramp. The sinking of a 715 m deep exploration shaft (the Gaumond shaft) started in 2010 and was completed in 2012, the change-over to production shaft was made in Q1 2013. A deep drilling program from the 650 shaft level started in June 2013 and is still ongoing. This allows zones to be drilled from a depth of 700 m to approximately 1,200 m.

Ramp excavation commenced in early 2011 to support additional exploration activities in the central portion of the deposit. The purpose of the ramp is to provide drilling access to the proposed mine area for definition purposes. At 2013 year-end, ramp excavation had reached level 620.

24.1. Preliminary Development Schedule

The mine is currently scheduled to start production in late 2014.

24.2. Future Development Strategy

The Gaumond shaft has a nominal 3,500 t/d ore-hoisting capacity, and a maximum hoisting capacity of 7,000 t/d (20 hours/day). Goldcorp are excavating a second shaft at Éléonore, which will have a nominal 8500 t/d hoisting capacity (17 hours/day). The objective is to transfer all the hoisting in the production shaft with the ore production in lower mine.

The current plant is designed for a throughput of 3,500 t/d with a ramp up period from 2015 to 2018 to reach 7,000 t/day in 2018, which is commensurate with the current Mineral Reserves. However, Goldcorp has designed the plant to be able to expand to 7,000 t/d earlier in the mine life production if Inferred Mineral Resources can be upgraded to Mineral Reserves.

The corporate strategy is therefore to initiate underground drilling from the Gaumond exploration shaft and exploration ramp, with the intent of delineating sufficient additional mineralization that can support Mineral Resource estimation and eventual disclosure of Mineral Reserves so that the hoisting and plant capacity can be fully utilized, and the current mine life can extended.

25. INTERPRETATION AND CONCLUSIONS

In the opinion of the QPs, the following interpretations and conclusions are appropriate.

25.1. Mineral Tenure, Surface Rights, Agreements, and Royalties

- Information from legal experts and Goldcorp's in-house experts support that the mining tenure held is valid and sufficient to support a disclosure of Mineral Resources and Mineral Reserves. Tenures are not surveyed; this is in accordance with appropriate regulatory requirements. Annual claim-holding fees have been paid to the relevant regulatory authority.
- The surface rights are held by Les Mines Opinaca Ltée.
- The Roberto deposit is located under the Opinaca Reservoir; water levels within the reservoir are controlled by Hydro-Québec.
- A sliding-scale royalty is payable to Virginia Mines Inc., and is capped at 3.5%Advance royalty payments commenced in April 2009.
- An annual payment is due to the Cree Nation under the collaborative agreement.
- Permits obtained by the company to explore and undertake project development are sufficient to ensure that activities are conducted within the regulatory framework required by the local, provincial and federal governments.

25.2. Geology and Mineralization

- Knowledge of the deposit settings and lithologies, as well as the structural and alteration controls on mineralization and the mineralization style and setting, is sufficient to support Mineral Resource and Mineral Reserve estimation;
- The Roberto deposit of the Project area are considered to be examples of clastic sediment-hosted stockwork-disseminated gold deposits in an orogenic setting.

25.3. Exploration, Drilling and Data Analysis

- The exploration programs completed to date are appropriate for the style of the deposits and prospects on the Project. The exploration and research work supports the interpretations of genesis and affinity.
- The drilling pattern provides adequate sampling of the gold mineralization for the purpose of estimating Mineral Resources and Mineral Reserves. Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit

area. Additional definition drilling is performed from underground via the existing ramp access, and exploration drilling is ongoing at depth from the 650 level access.

- The quantity and quality of the lithological, geotechnical, collar and down-hole survey data collected during the exploration and delineation drilling programs are sufficient to support Mineral Resource and Mineral Reserve estimation. The collected sample data adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits. Sampling is representative of the gold grades in the deposits, reflecting areas of higher and lower grades.
- Sampling methods are acceptable. The methods meet industry-standard practices for a clastic sediment-hosted stockwork-disseminated gold deposit in an orogenic setting and are acceptable for Mineral Resource and Mineral Reserve estimation.
- The quality of the analytical data for gold is sufficiently reliable to support Mineral Resource and Mineral Reserve estimation.
- Sample preparation, analysis and security are generally performed in accordance with exploration best practices and industry standards. Exploration and infill core samples were analyzed by independent laboratories using industry-standard methods for gold.
- The QA/QC programs adequately address issues of precision, accuracy and contamination. Drilling programs typically included blanks, duplicates and SRM samples. QA/QC submission rates meet industry-accepted standards. The QA/QC programs did not detect any material sample biases.
- The data verification programs concluded that the data collected from the Project adequately support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource and Mineral Reserve estimation. The collected data were subject to validation using built-in program that triggersautomatically checked data on upload to the Project database.
- Sample security has relied upon the fact that the samples were always attended or locked in appropriate storage facilities. Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory along with sample shipments to make certain that all samples are received by the laboratory.
- Current sample storage procedures and storage areas meet industry standards.
- Additional exploration potential remains within the Roberto deposit at depth. Other targets have been identified around the Roberto deposit, including the hanging wall veins (HWV), the North Zone (NZ), and the 494 area. Little exploration has been undertaken outside the immediate area of the Roberto deposit, and the greater concession area retains

good exploration potential.

25.4. Metallurgical Testwork

- Metallurgical testwork and associated analytical procedures were performed by recognized testing facilities, and the tests performed were appropriate to the type of mineralization.
- Samples selected for testing were representative of the various types and styles of mineralization on the Roberto deposit. Samples were selected from a range of depths within the Roberto deposit. Sufficient samples were taken to ensure that tests were performed on sufficient sample mass.
- Testwork has established the most appropriate grind size for plant design. The proposed grind size is 65 µm.
- LOM gold recovery assumptions are based on appropriate testwork, and are expected to average 93.0–93.5%.
- Deleterious elements are present in the deposit and must be accommodated in the process design. About 97% of the arsenic will be removed from the iron sulphide flotation and sent back to the underground workings through the paste backfill plant. All the water in contact with process residues in the mine and tailings management facility will be pumped to and treated at the water treatment plant.

25.5. Mineral Resource Estimation

- The Mineral Resource estimation for the Project conforms to industry best practices and meets the requirements of CIM (2010).
- Drill data are typically verified prior to the Mineral Resource (and Mineral Reserve) estimation by running a software program check.
- Inferred Mineral Resources total 13.25 Mt at an average of 9.63 g/t Au. A cut-off grade of 3 g/t Au was used, based on a gold price of $1,300/oz.
- Several factors may affect the Mineral Resource Estimate, including metal price, mining method and geotechnical factors.

25.6. Mineral Reserve Estimation

- The Mineral Resource estimation for the Project conforms to industry best practices and meets the requirements of NI 43-101.
- Mineral Resources were converted to Mineral Reserves using appropriate modifying factors. These included the consideration of dilution, mining widths, ore losses, mining extraction losses, appropriate underground mining methods, metallurgical recoveries, permitting and infrastructure requirements.
- Mineral Reserves are estimated using a gold price of $1,300/oz, and an

economic function that includes variable operating costs and metallurgical recoveries.

- A complete study (hydrogeological and geotechnical) on the potential recovery of the surface crown pillar will be necessary to convert Mineral Resources into Proven Mineral Reserves in this area. Any future mining method needs to be carefully evaluated because the orebody is directly beneath the Opinaca Reservoir.
- Probable Mineral Reserves total 19.3 Mt at an average grade of 6.5 g/t Au. A cut-off grade of 3 g/t Au was used.
- Factors that might affect the Mineral Reserve estimate are:

  - ✓ Lower recovery at the mill because of a possible change in the hardness of the rock: The grinding could take longer if the rock becomes harder deeper in the mine. No laboratory tests on rock resistance have been performed on drill core from an elevation lower than 500 m.
  - ✓ More water infiltration from the surface than expected: Water infiltration could come from geological structures and diamond drill holes. Some of the old diamond drill holes were not correctly grouted or simply not grouted, and therefore even if all current holes are correctly grouted, there is a likelihood that some of the old non-grouted holes could be intercepted during stoping. In this situation, if water infiltration is very high, Goldcorp may have to leave the broken rock in the stope and immediately commence stope backfilling operations.
  - ✓ In situ stress in the rock: Currently no in situ stress measurements were performed in the Éléonore area. These measurements should be done in 2014 when development headings will be available in the 650 area. Changes in the mining sequence and on pillar dimensions may be necessary if the program indicates higher in-situ stresses.
  - ✓ Rock bursts: It is difficult to predict the future behaviour of the rock mass using the available information. Ramp and shaft station excavation has triggered minor seismic events. These seismic events appear to be associated with changes in the geology. Seismic events may become bigger and dangerous if the planned mining sequence is not followed, and could compromise the Mineral Reserve Estimate.
  - ✓ Drill hole deviations: If production drill holes deviate significantly, this may cause more dilution. The hole diameter will be only 102 mm (4″) but the length will be 27 m (90 ft). Controlling the deviation will be an important factor in reducing unplanned dilution. If additional dilution results, the gold cut-off grade may need to be increased.
  - ✓ Paste backfill strength: Laboratory tests have indicated good results,

but in cases where a stope has water infiltration issues or where the stope is in an area of cold temperature (i.e. in the upper mine levels), then paste backfill strengths can be lower than expected. This could result in unplanned dilution and therefore require an increase in the gold cut-off grade.

- ✓ Changes in commodity price and exchange rate assumptions.

25.7. Mine Plan

- It is expected that any future mining operations can be conducted year-round.
- The proposed mine plan was developed by Goldcorp personnel. Two different methods are planned. Long-hole stoping (down-hole drilling) on longitudinal retreat with consolidated backfill (paste backfill mixed with crushed waste rock) and transverse stoping approach where the mineralized lens is wider.
- Operations will be accessed using a surface ramp and shaft. The production rate will be approximately 3,500 t/d with two mining fronts;
- The production rate will reach 7,000 t/d for 4 years with lower mine production;
- The proposed mine life is 10 years
- No mining is planned above 55 m in order to mitigate risks associated with potential water inflow from the Opinaca Reservoir and to respect preliminary recommendations for the surface crown pillar.
- Due to the presence of open subhorizontal decompression joints encountered mainly within the first 150 m below surface couple with the proximity of the reservoir, the management of ground water infiltration is considered paramount for successful project implementation.

25.8. Proposed Process Plant

- The process design is based on a conventional gold plant flowsheet consisting of three stages of crushing, grinding, gravity concentration, sulphide flotation, leaching flotation tails and concentrate, CIP and electrowinning circuit. The mill was designed to operate 365 days/year with a design capacity of 2.55 Mt of ore per year (7,000 t/d).
- Tailings will be filtered at the mill, and will be trucked to the tailings area. The tailings facilities will be completely lined, and all water coming into contact with the tailings will be collected and treated. The surface area of the exposed tailings will be kept to a minimum through the use of filtered tailings, which allows for progressive reclamation. The tailings design envisions a storage capacity of 26 Mt. This is sufficient for the current LOM.
- The majority of non-sulphide tailings and sulphides concentrate and all

waste rocks will be returned to the underground operation in the form of backfill.

25.9. Infrastructure Considerations

- The Project will require the construction of significant infrastructure to support the planned producing facilities.
- The existing and planned infrastructure; staff availability; existing power, water, and communications facilities; the methods whereby goods will be transported to the mine; and any planned modifications or supporting studies are all well- established, or the requirements to establish such are well understood by Goldcorp and can support the disclosure of Mineral Resources and Mineral Reserves.
- Éléonore Project has been accessible by road since 2012. Road access is for material supplies only as employees will be transported by aircraft.
- The Éléonore Project mine and ore processing plant will be fed through a 120 kV overhead electrical power line supplied and installed by Hydro-Québec from the existing distribution point at Eastmain power generation substation. Since 2012, Hydro-Quebec has changed the operating voltage to 120 kV to supply Opinaca's two permanent 120/25 kV substations. All 25 kV distribution lines were designed in order to take into account the constraints imposed at this northern latitude, such as weather, ice and wind loads.
- For fuel storage and services, three different installations have been selected. There is a daily fuel tanks for the light and heavy mobile equipment. A jet-A fuel tank to supply the aircraft at the airport and several emergency generators to support the critical activities and infrastructures at site;
- The drinking water supply for the campsite will be drawn from three wells. The combined well capacity will be of the order of 220 m^3/day, which is considered adequate to supply a crew of 600 workers given a unit consumption of 330 litres/person/day (198 m^3/d). All of the water from the mine, the tailings facility, the collected drainage water (waste storage and industrial areas), and water from the process plant will be treated for suspend solid arsenic and heavy metals in the water treatment plant. The treated water will be pumped to a 8,000 m^3 capacity storage basin before being pumped as required to the process plant as fresh water or being discharged to the environment.
- Underground infrastructure will comprise: a circular shaft that will have a final depth of 715 m; a 4.5-km-long access ramp; a shaft loading station; ore and waste passes; ore and waste storage bins; a rock breaker/grizzly arrangement; a transfer drift; an exhaust raise; and a mine dewatering system.
- Process plant infrastructure will consist of: a jaw crusher; a secondary

cone crusher; a tertiary crusher; a single-stage ball mill, cyclones; a Knelson concentrator; an Acacia reactor; flotation and leach tanks; carousel CIP circuits; a Zadra stripping circuit; and a filtration and paste backfill plant.

- Surface infrastructure will include: the main access road and gate house building; a camp; an airstrip; main mine substation (120/25 kV); an administration and emergency vehicle building; warehouse facilities; garages facilities; the Gaumond shaft; potable and wastewater treatment plants; an industrial water treatment plant; an assay laboratory; fuel and propane storage; and oxygen plant.
- Workforce requirements are estimated at 600 persons to support the proposed mine nominal throughput rate of 7,000 t/d.

25.10. Markets and Contracts

- Goldcorp's bullion is sold on the spot market by Goldcorp's in-house marketing experts. It is expected that the same process will be used for the gold produced at Éléonore.
- The terms contained within the existing sales contracts are typical and consistent with standard industry practices, and are similar to contracts for the supply of doré elsewhere in the world. There is an expectation that the same style of contracts will apply to gold produced from Éléonore.

25.11. Environmental, Social Issues and Permitting

25.11.1. Environment

- Goldcorp's team of on-site environmental experts will need to be continuously monitoring regulatory compliance in terms of approvals, permits, and observance of directives and requirements.
- There has been a focused effort to collect comprehensive environmental baseline data and lay the groundwork with local and regulatory stakeholders for the successful permitting and development of the Project.
- Closure costs are estimated at $40.1 M, which includes provisions for dismantling and removal of infrastructure; the remediation of water ponds, the tailings storage facility and the waste rock facility; soil and waste management; indirect costs; and post- closure monitoring and contingency

25.11.2. Community

- Goldcorp has undertaken community consultation with the communities of Wemindji and Chibougamau.

- A Collaboration Agreement has been signed with the Grand Council of the Crees (Eeyou Istchee), the Cree Regional Authority and the Cree Nation of Wemindji, and an economic partnership agreement has been concluded with the local communities of the James Bay region.

25.11.3. Permitting

- Goldcorp has been granted a Global Certificate of Approval for the Project. Subsequently, many other Certificates of Approval were also granted under Chapter I of the Environemental Quality Act., which allows the company to proceed with the construction of various infrastructure elements. Additional Certificates of Approval have been filed with the appropriate regulatory authorities and others are currently being prepared and will be submitted to the appropriate ministries.
- Key issues for project development include the proper management of tailings and waste water, access (roads, airports), social acceptability and post-reclamation management.

25.12. Capital and Operating Cost Estimates

- The capital cost estimates are based on a combination of quotes, vendor pricing, and Goldcorp experience with similar-sized operations
- Capital costs are based on the latest mine construction data and budgetary numbers/quotes provided by suppliers. The 2011-2013 capital costs are based on the actual costs, and the 2014 capital costs are based on the budget.
- Project capital costs, including direct and indirect costs, are estimated at US$1.85 B. During the life-of-mine, an additional US$310 M will be required as sustaining capital for underground development. The total capital cost of the Éléonore project was estimated at US$2.16 B.
- An average unit operating cost of CAD$111.51/t was estimated, which comprises CAD$29.89/t for processing costs, including backfill and tailings treatment and transportation, CAD$52.41/t for mining costs, CAD$8.76/t for infrastructure costs, and CAD$20.45/t for G&A costs.

25.13. Financial Analysis

- The Éléonore Project is expected to yield an after-tax internal rate of return (IRR) of 3.15% over a 10-year mine life. Under the assumptions in the financial analysis, payback will be achieved in Year 8.
- In this concept study, mining activities were assumed to occur on four mining fronts which would result in a doubling of the production rate from the current 3,500 t/d from two fronts to 7,000 t/d.
- Cash flow fluctuations during the LOM primarily result from fluctuations

in the sustaining capital and mill head grade

25.14. Conclusions

There is sufficient support from the updated results for Goldcorp to move to a production phase starting in 2014.

26. RECOMMENDATIONS

In order to continue with project development and prepare for the production phase, intensive exploration and delineation drilling must be carried out over the next 3 years. The first objective of the drilling will be to convert resources to reserves. Drilling at depth and laterally is also required to add new resources.

The suggested program involves a rate of approximately 25,000 m per year of exploration drilling to test the deposit's extensions and add new resources, another 50,000 m per year of infill drilling (at 25 m spacing) to convert Indicated Resources to Reserves, and 40,000 m per year of final delineation or definition drilling (production drilling) for stope delineation. This amounts to a total of 115,000 metres of drilling per year, representing an estimated annual budget of CA$17.25 M.

26.1. Exploration Drilling

Drilling has identified additional exploration potential within the Roberto area.

In the HWV area, 8,000 m of surface drilling is proposed for a total projected cost of CA$1.4 M. The data from this program will be used to produce a mineral resource estimate for the mineralization contained in these shear and alteration zones found near the shaft and exploration ramp; the work is necessary because these zones may become accessible during mine development activities.

In the NZ area, mineralization crops out at surface and may represent a potential open pit target. Additional drilling is warranted, totalling CA$1.5 M for 10,000 m.

In the 494 area, 8,000 m of underground drilling is proposed at a total cost of CA$1.2 M to drill-test a potential mineralized corridor located between the 494 area and a surface showing (Trench #10) that displays similar mineralized features.

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Appendix A
Claims lists

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
36780	Eleonore	33B12	24	1	52,21	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13789	Eleonore	33B12	21	1	52,24	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
36784	Eleonore	33B12	26	1	52,19	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13126	Eleonore	33B12	23	1	52,22	Active	2004-02-13	2016-02-12	5	1270,14	1800	Les Mines Opinaca ltée	100
13124	Eleonore	33B12	22	1	52,23	Active	2004-02-13	2016-02-12	5	873,93	1800	Les Mines Opinaca ltée	100
13787	Eleonore	33B12	20	1	52,25	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
36786	Eleonore	33B12	27	1	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36788	Eleonore	33B12	28	1	52,17	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36782	Eleonore	33B12	25	1	52,20	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
13790	Eleonore	33B12	21	2	52,24	Active	2004-02-26	2016-02-25	5	873,94	1800	Les Mines Opinaca ltée	100
36781	Eleonore	33B12	24	2	52,21	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36783	Eleonore	33B12	25	2	52,20	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13127	Eleonore	33B12	23	2	52,22	Active	2004-02-13	2016-02-12	5	1270,14	1800	Les Mines Opinaca ltée	100
36789	Eleonore	33B12	28	2	52,17	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13125	Eleonore	33B12	22	2	52,23	Active	2004-02-13	2016-02-12	5	2144,08	1800	Les Mines Opinaca ltée	100
36787	Eleonore	33B12	27	2	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36785	Eleonore	33B12	26	2	52,19	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13788	Eleonore	33B12	20	2	52,25	Active	2004-02-26	2016-02-25	5	873,94	1800	Les Mines Opinaca ltée	100
38864	Eleonore	33B12	20	3	52,25	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38866	Eleonore	33B12	21	3	52,24	Active	2004-09-21	2014-09-20	4	1319,06	1800	Les Mines Opinaca ltée	100
38874	Eleonore	33B12	25	3	52,20	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38876	Eleonore	33B12	26	3	52,19	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38880	Eleonore	33B12	28	3	52,17	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38872	Eleonore	33B12	24	3	52,21	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38868	Eleonore	33B12	22	3	52,23	Active	2004-09-21	2014-09-20	4	1319,06	1800	Les Mines Opinaca ltée	100
38878	Eleonore	33B12	27	3	52,18	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
38870	Eleonore	33B12	23	3	52,22	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38869	Eleonore	33B12	22	4	52,23	Active	2004-09-21	2014-09-20	4	1319,06	1800	Les Mines Opinaca ltée	100
38867	Eleonore	33B12	21	4	52,24	Active	2004-09-21	2014-09-20	4	1319,05	1800	Les Mines Opinaca ltée	100
38873	Eleonore	33B12	24	4	52,21	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38871	Eleonore	33B12	23	4	52,22	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38879	Eleonore	33B12	27	4	52,18	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38865	Eleonore	33B12	20	4	52,25	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38875	Eleonore	33B12	25	4	52,20	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38881	Eleonore	33B12	28	4	52,17	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
38877	Eleonore	33B12	26	4	52,19	Active	2004-09-21	2014-09-20	4	445,12	1800	Les Mines Opinaca ltée	100
42308	Eleonore	33B12	20	5	52,25	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42314	Eleonore	33B12	23	5	52,22	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42322	Eleonore	33B12	27	5	52,18	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42320	Eleonore	33B12	26	5	52,19	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42324	Eleonore	33B12	28	5	52,17	Active	2004-10-04	2014-10-03	4	79,45	1800	Les Mines Opinaca ltée	100
42318	Eleonore	33B12	25	5	52,20	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42312	Eleonore	33B12	22	5	52,23	Active	2004-10-04	2014-10-03	4	1920,6	1800	Les Mines Opinaca ltée	100
42310	Eleonore	33B12	21	5	52,24	Active	2004-10-04	2014-10-03	4	1920,61	1800	Les Mines Opinaca ltée	100
42316	Eleonore	33B12	24	5	52,21	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42313	Eleonore	33B12	22	6	52,23	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42321	Eleonore	33B12	26	6	52,19	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42323	Eleonore	33B12	27	6	52,18	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42311	Eleonore	33B12	21	6	52,24	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42317	Eleonore	33B12	24	6	52,21	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42319	Eleonore	33B12	25	6	52,20	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42315	Eleonore	33B12	23	6	52,22	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100
42325	Eleonore	33B12	28	6	52,17	Active	2004-10-04	2014-10-03	4	79,45	1800	Les Mines Opinaca ltée	100
42309	Eleonore	33B12	20	6	52,25	Active	2004-10-04	2014-10-03	4	1046,67	1800	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
43678	Eleonore	33B12	28	7	52,17	Active	2004-10-12	2014-10-11	4	0	1800	Les Mines Opinaca ltée	100
43679	Eleonore	33B12	28	8	52,17	Active	2004-10-12	2014-10-11	4	0	1800	Les Mines Opinaca ltée	100
2234530	Road	33C09	29	8	52,17	Active	2010-05-19	2014-05-18	1	923,26	450	Les Mines Opinaca ltée	100
2185700	Road	33C09	29	9	52,17	Active	2009-07-27	2015-07-26	2	0	900	Les Mines Opinaca ltée	100
45820	Road	33C09	30	27	52,15	Active	2004-11-17	2014-11-16	4	1331,94	1800	Les Mines Opinaca ltée	100
45814	Road	33C09	29	27	52,16	Active	2004-11-17	2014-11-16	4	516,28	1800	Les Mines Opinaca ltée	100
39452	Eleonore	33C09	24	31	52,21	Active	2004-09-21	2014-09-20	4	0	1800	Les Mines Opinaca ltée	100
42189	Eleonore	33C09	25	31	52,20	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
42190	Eleonore	33C09	25	32	52,20	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
39453	Eleonore	33C09	24	32	52,21	Active	2004-09-21	2014-09-20	4	0	1800	Les Mines Opinaca ltée	100
42191	Eleonore	33C09	25	33	52,20	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
39454	Eleonore	33C09	24	33	52,21	Active	2004-09-21	2014-09-20	4	0	1800	Les Mines Opinaca ltée	100
42183	Eleonore	33C09	24	34	52,21	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
42192	Eleonore	33C09	25	34	52,20	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
42193	Eleonore	33C09	25	35	52,20	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
42184	Eleonore	33C09	24	35	52,21	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
42194	Eleonore	33C09	25	36	52,20	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
42185	Eleonore	33C09	24	36	52,21	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
42195	Eleonore	33C09	25	37	52,20	Active	2004-10-04	2014-10-03	4	751,11	1800	Les Mines Opinaca ltée	100
42186	Eleonore	33C09	24	37	52,21	Active	2004-10-04	2014-10-03	4	2,51	1800	Les Mines Opinaca ltée	100
42180	Eleonore	33C09	23	37	52,22	Active	2004-10-04	2014-10-03	4	0	1800	Les Mines Opinaca ltée	100
42187	Eleonore	33C09	24	38	52,21	Active	2004-10-04	2014-10-03	4	2,51	1800	Les Mines Opinaca ltée	100
42181	Eleonore	33C09	23	38	52,22	Active	2004-10-04	2014-10-03	4	261,86	1800	Les Mines Opinaca ltée	100
42196	Eleonore	33C09	25	38	52,20	Active	2004-10-04	2014-10-03	4	491,76	1800	Les Mines Opinaca ltée	100
42182	Eleonore	33C09	23	39	52,22	Active	2004-10-04	2014-10-03	4	1042,41	1800	Les Mines Opinaca ltée	100
42197	Eleonore	33C09	25	39	52,20	Active	2004-10-04	2014-10-03	4	491,76	1800	Les Mines Opinaca ltée	100
42188	Eleonore	33C09	24	39	52,21	Active	2004-10-04	2014-10-03	4	896,76	1800	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
1105812	Eleonore	33C09	24	40	52,21	Active	2002-11-28	2014-11-27	5	0	1800	Les Mines Opinaca ltée	100
1028884	Eleonore	33C09	19	40	52,26	Active	2001-09-22	2015-09-21	6	480,42	2500	Les Mines Opinaca ltée	100
36753	Eleonore	33C09	27	40	52,18	Active	2004-09-14	2014-09-13	4	141,58	1800	Les Mines Opinaca ltée	100
1042271	Eleonore	33C09	22	40	52,23	Active	2001-12-12	2015-12-11	6	0	2500	Les Mines Opinaca ltée	100
36746	Eleonore	33C09	26	40	52,19	Active	2004-09-14	2014-09-13	4	0	1800	Les Mines Opinaca ltée	100
1028892	Eleonore	33C09	20	40	52,25	Active	2001-09-22	2015-09-21	6	0	2500	Les Mines Opinaca ltée	100
1103421	Eleonore	33C09	23	40	52,22	Active	2002-10-25	2014-10-24	5	44,56	1800	Les Mines Opinaca ltée	100
36739	Eleonore	33C09	25	40	52,20	Active	2004-09-14	2014-09-13	4	608,89	1800	Les Mines Opinaca ltée	100
1028907	Eleonore	33C09	21	40	52,24	Active	2001-09-22	2015-09-21	6	0	2500	Les Mines Opinaca ltée	100
36747	Eleonore	33C09	26	41	52,19	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36754	Eleonore	33C09	27	41	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13191	Eleonore	33C09	17	41	52,28	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1028885	Eleonore	33C09	19	41	52,26	Active	2001-09-22	2015-09-21	6	2581,01	2500	Les Mines Opinaca ltée	100
1105813	Eleonore	33C09	24	41	52,21	Active	2002-11-28	2014-11-27	5	72,46	1800	Les Mines Opinaca ltée	100
1042272	Eleonore	33C09	22	41	52,23	Active	2001-12-12	2015-12-11	6	280,47	2500	Les Mines Opinaca ltée	100
1028893	Eleonore	33C09	20	41	52,25	Active	2001-09-22	2015-09-21	6	0	2500	Les Mines Opinaca ltée	100
36740	Eleonore	33C09	25	41	52,20	Active	2004-09-14	2014-09-13	4	0	1800	Les Mines Opinaca ltée	100
13187	Eleonore	33C09	16	41	52,29	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1028908	Eleonore	33C09	21	41	52,24	Active	2001-09-22	2015-09-21	6	0	2500	Les Mines Opinaca ltée	100
1103422	Eleonore	33C09	23	41	52,22	Active	2002-10-25	2014-10-24	5	745,35	1800	Les Mines Opinaca ltée	100
1028894	Eleonore	33C09	20	42	52,25	Active	2001-09-22	2015-09-21	6	1919,85	2500	Les Mines Opinaca ltée	100
13188	Eleonore	33C09	16	42	52,29	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36741	Eleonore	33C09	25	42	52,20	Active	2004-09-14	2014-09-13	4	0	1800	Les Mines Opinaca ltée	100
1105814	Eleonore	33C09	24	42	52,21	Active	2002-11-28	2014-11-27	5	72,46	1800	Les Mines Opinaca ltée	100
36748	Eleonore	33C09	26	42	52,19	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13182	Eleonore	33C09	15	42	52,30	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13192	Eleonore	33C09	17	42	52,28	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
1103423	Eleonore	33C09	23	42	52,22	Active	2002-10-25	2014-10-24	5	1949,65	1800	Les Mines Opinaca ltée	100
1028886	Eleonore	33C09	19	42	52,26	Active	2001-09-22	2015-09-21	6	321,32	2500	Les Mines Opinaca ltée	100
1028909	Eleonore	33C09	21	42	52,24	Active	2001-09-22	2015-09-21	6	1786,37	2500	Les Mines Opinaca ltée	100
36755	Eleonore	33C09	27	42	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
1042273	Eleonore	33C09	22	42	52,23	Active	2001-12-12	2015-12-11	6	886,36	2500	Les Mines Opinaca ltée	100
13195	Eleonore	33C09	18	42	52,27	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13164	Eleonore	33C09	14	43	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1028910	Eleonore	33C09	21	43	52,24	Active	2001-09-22	2015-09-21	6	2119	2500	Les Mines Opinaca ltée	100
13193	Eleonore	33C09	17	43	52,28	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13183	Eleonore	33C09	15	43	52,30	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36756	Eleonore	33C09	27	43	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36742	Eleonore	33C09	25	43	52,20	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
1103424	Eleonore	33C09	23	43	52,22	Active	2002-10-25	2014-10-24	5	1286,78	1800	Les Mines Opinaca ltée	100
1105815	Eleonore	33C09	24	43	52,21	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
13196	Eleonore	33C09	18	43	52,27	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1028887	Eleonore	33C09	19	43	52,26	Active	2001-09-22	2015-09-21	6	6615,56	2500	Les Mines Opinaca ltée	100
1042274	Eleonore	33C09	22	43	52,23	Active	2001-12-12	2015-12-11	6	2118,99	2500	Les Mines Opinaca ltée	100
1028895	Eleonore	33C09	20	43	52,25	Active	2001-09-22	2015-09-21	6	5985,18	2500	Les Mines Opinaca ltée	100
13189	Eleonore	33C09	16	43	52,29	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36749	Eleonore	33C09	26	43	52,19	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13190	Eleonore	33C09	16	44	52,29	Active	2004-02-12	2016-02-11	5	873,93	1800	Les Mines Opinaca ltée	100
13194	Eleonore	33C09	17	44	52,28	Active	2004-02-12	2016-02-11	5	873,93	1800	Les Mines Opinaca ltée	100
1028896	Eleonore	33C09	20	44	52,25	Active	2001-09-22	2015-09-21	6	8709,73	2500	Les Mines Opinaca ltée	100
36757	Eleonore	33C09	27	44	52,18	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13197	Eleonore	33C09	18	44	52,27	Active	2004-02-12	2016-02-11	5	1805,05	1800	Les Mines Opinaca ltée	100
1028888	Eleonore	33C09	19	44	52,26	Active	2001-09-22	2015-09-21	6	8159,41	2500	Les Mines Opinaca ltée	100
1028911	Eleonore	33C09	21	44	52,24	Active	2001-09-22	2015-09-21	6	13269,12	2500	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
1105828	Eleonore	33C09	25	44	52,20	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
1103425	Eleonore	33C09	23	44	52,22	Active	2002-10-25	2018-10-24	7	0	2500	Les Mines Opinaca ltée	100
1105816	Eleonore	33C09	24	44	52,21	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
1042275	Eleonore	33C09	22	44	52,23	Active	2001-12-12	2015-12-11	6	3280,84	2500	Les Mines Opinaca ltée	100
13165	Eleonore	33C09	14	44	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13184	Eleonore	33C09	15	44	52,30	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
6652	Eleonore	33C09	26	44	52,19	Active	2003-11-12	2017-11-11	6	0	2500	Les Mines Opinaca ltée	100
13726	Eleonore	33C09	18	45	52,27	Active	2004-02-26	2016-02-25	5	3685,21	1800	Les Mines Opinaca ltée	100
1105829	Eleonore	33C09	25	45	52,20	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
13694	Eleonore	33C09	16	45	52,29	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
1028889	Eleonore	33C09	19	45	52,26	Active	2001-09-22	2015-09-21	6	4601,53	2500	Les Mines Opinaca ltée	100
13166	Eleonore	33C09	14	45	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1042276	Eleonore	33C09	22	45	52,23	Active	2001-12-12	2015-12-11	6	5895,62	2500	Les Mines Opinaca ltée	100
36758	Eleonore	33C09	27	45	52,18	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
1028912	Eleonore	33C09	21	45	52,24	Active	2001-09-22	2015-09-21	6	14106,34	2500	Les Mines Opinaca ltée	100
13710	Eleonore	33C09	17	45	52,28	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
1105817	Eleonore	33C09	24	45	52,21	Active	2002-11-28	2014-11-27	5	4850,27	1800	Les Mines Opinaca ltée	100
1103426	Eleonore	33C09	23	45	52,22	Active	2002-10-25	2018-10-24	7	0	2500	Les Mines Opinaca ltée	100
1028897	Eleonore	33C09	20	45	52,25	Active	2001-09-22	2015-09-21	6	10704,25	2500	Les Mines Opinaca ltée	100
13185	Eleonore	33C09	15	45	52,30	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
6653	Eleonore	33C09	26	45	52,19	Active	2003-11-12	2015-11-11	5	3450,74	1800	Les Mines Opinaca ltée	100
1105818	Eleonore	33C09	24	46	52,21	Active	2002-11-28	2014-11-27	5	5564,5	1800	Les Mines Opinaca ltée	100
6654	Eleonore	33C09	26	46	52,19	Active	2003-11-12	2017-11-11	6	0	2500	Les Mines Opinaca ltée	100
1103427	Eleonore	33C09	23	46	52,22	Active	2002-10-25	2018-10-24	7	0	2500	Les Mines Opinaca ltée	100
13711	Eleonore	33C09	17	46	52,28	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
1028913	Eleonore	33C09	21	46	52,24	Active	2001-09-22	2015-09-21	6	7969,09	2500	Les Mines Opinaca ltée	100
1028898	Eleonore	33C09	20	46	52,25	Active	2001-09-22	2015-09-21	6	5638,79	2500	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
13167	Eleonore	33C09	14	46	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1042277	Eleonore	33C09	22	46	52,23	Active	2001-12-12	2015-12-11	6	7082,72	2500	Les Mines Opinaca ltée	100
13186	Eleonore	33C09	15	46	52,30	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1042266	Eleonore	33C09	19	46	52,26	Active	2001-12-12	2015-12-11	6	2622,25	2500	Les Mines Opinaca ltée	100
1105830	Eleonore	33C09	25	46	52,20	Active	2002-11-28	2014-11-27	5	4492,93	1800	Les Mines Opinaca ltée	100
13695	Eleonore	33C09	16	46	52,29	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
13727	Eleonore	33C09	18	46	52,27	Active	2004-02-26	2016-02-25	5	3283,15	1800	Les Mines Opinaca ltée	100
36759	Eleonore	33C09	27	46	52,18	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13168	Eleonore	33C09	14	47	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13696	Eleonore	33C09	16	47	52,29	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1028914	Eleonore	33C09	21	47	52,24	Active	2001-09-22	2015-09-21	6	5638,9	2500	Les Mines Opinaca ltée	100
1103428	Eleonore	33C09	23	47	52,22	Active	2002-10-25	2014-10-24	5	4919,33	1800	Les Mines Opinaca ltée	100
1028899	Eleonore	33C09	20	47	52,25	Active	2001-09-22	2015-09-21	6	5638,9	2500	Les Mines Opinaca ltée	100
6655	Eleonore	33C09	26	47	52,19	Active	2003-11-12	2017-11-11	6	0	2500	Les Mines Opinaca ltée	100
13680	Eleonore	33C09	15	47	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1105831	Eleonore	33C09	25	47	52,20	Active	2002-11-28	2014-11-27	5	5387,5	1800	Les Mines Opinaca ltée	100
13712	Eleonore	33C09	17	47	52,28	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
13728	Eleonore	33C09	18	47	52,27	Active	2004-02-26	2016-02-25	5	3226,8	1800	Les Mines Opinaca ltée	100
13150	Eleonore	33C09	13	47	52,32	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1042267	Eleonore	33C09	19	47	52,26	Active	2001-12-12	2015-12-11	6	2565,9	2500	Les Mines Opinaca ltée	100
36760	Eleonore	33C09	27	47	52,18	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
1105819	Eleonore	33C09	24	47	52,21	Active	2002-11-28	2014-11-27	5	6054,61	1800	Les Mines Opinaca ltée	100
1042278	Eleonore	33C09	22	47	52,23	Active	2001-12-12	2015-12-11	6	2622,25	2500	Les Mines Opinaca ltée	100
13151	Eleonore	33C09	13	48	52,32	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36767	Eleonore	33C09	28	48	52,17	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
1103429	Eleonore	33C09	23	48	52,22	Active	2002-10-25	2014-10-24	5	4919,33	1800	Les Mines Opinaca ltée	100
1129597	Eleonore	33C09	27	48	52,18	Active	2004-02-12	2018-02-11	6	0	2500	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
1105832	Eleonore	33C09	25	48	52,20	Suspended	2002-11-28	2014-11-27	5	6189,61	1800	Les Mines Opinaca ltée	100
1042279	Eleonore	33C09	22	48	52,23	Active	2001-12-12	2017-12-11	7	0	2500	Les Mines Opinaca ltée	100
13729	Eleonore	33C09	18	48	52,27	Active	2004-02-26	2016-02-25	5	658,41	1800	Les Mines Opinaca ltée	100
13713	Eleonore	33C09	17	48	52,28	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
6656	Eleonore	33C09	26	48	52,19	Suspended	2003-11-12	2017-11-11	6	0	2500	Les Mines Opinaca ltée	100
13697	Eleonore	33C09	16	48	52,29	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
1028900	Eleonore	33C09	20	48	52,25	Active	2001-09-22	2015-09-21	6	4112,22	2500	Les Mines Opinaca ltée	100
13169	Eleonore	33C09	14	48	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13681	Eleonore	33C09	15	48	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1028915	Eleonore	33C09	21	48	52,24	Active	2001-09-22	2015-09-21	6	5638,9	2500	Les Mines Opinaca ltée	100
1042268	Eleonore	33C09	19	48	52,26	Active	2001-12-12	2015-12-11	6	1144,68	2500	Les Mines Opinaca ltée	100
1105820	Eleonore	33C09	24	48	52,21	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
13152	Eleonore	33C09	13	49	52,32	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1105833	Eleonore	33C09	25	49	52,20	Suspended	2002-11-28	2014-11-27	5	324478	1800	Les Mines Opinaca ltée	100
1016890	Eleonore	33C09	20	49	52,25	Active	2001-05-09	2015-05-08	6	5402,67	2500	Les Mines Opinaca ltée	100
13714	Eleonore	33C09	17	49	52,28	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
1042280	Eleonore	33C09	22	49	52,23	Active	2001-12-12	2017-12-11	7	0	2500	Les Mines Opinaca ltée	100
13698	Eleonore	33C09	16	49	52,29	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
6657	Eleonore	33C09	26	49	52,19	Suspended	2003-11-12	2015-11-11	5	2172,62	1800	Les Mines Opinaca ltée	100
1042269	Eleonore	33C09	19	49	52,26	Active	2001-12-12	2015-12-11	6	1144,68	2500	Les Mines Opinaca ltée	100
1103430	Eleonore	33C09	23	49	52,22	Active	2002-10-25	2014-10-24	5	5602,59	1800	Les Mines Opinaca ltée	100
36768	Eleonore	33C09	28	49	52,17	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
1016892	Eleonore	33C09	21	49	52,24	Active	2001-05-09	2015-05-08	6	5675,49	2500	Les Mines Opinaca ltée	100
13730	Eleonore	33C09	18	49	52,27	Active	2004-02-26	2016-02-25	5	1532,34	1800	Les Mines Opinaca ltée	100
1129598	Eleonore	33C09	27	49	52,18	Active	2004-02-12	2018-02-11	6	0	2500	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
13682	Eleonore	33C09	15	49	52,30	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
13170	Eleonore	33C09	14	49	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36769	Eleonore	33C09	28	50	52,17	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13731	Eleonore	33C09	18	50	52,27	Active	2004-02-26	2016-02-25	5	1315,46	1800	Les Mines Opinaca ltée	100
13153	Eleonore	33C09	13	50	52,32	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13171	Eleonore	33C09	14	50	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13699	Eleonore	33C09	16	50	52,29	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
6658	Eleonore	33C09	26	50	52,19	Suspended	2003-11-12	2015-11-11	5	110569,98	1800	Les Mines Opinaca ltée	100
1105822	Eleonore	33C09	24	50	52,21	Suspended	2002-11-28	2014-11-27	5	5503572,11	1800	Les Mines Opinaca ltée	100
1105834	Eleonore	33C09	25	50	52,20	Suspended	2002-11-28	2014-11-27	5	4772009,01	1800	Les Mines Opinaca ltée	100
13683	Eleonore	33C09	15	50	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1016891	Eleonore	33C09	20	50	52,25	Active	2001-05-09	2015-05-08	6	4149,22	2500	Les Mines Opinaca ltée	100
1016893	Eleonore	33C09	21	50	52,24	Active	2001-05-09	2015-05-08	6	5675,9	2500	Les Mines Opinaca ltée	100
1042281	Eleonore	33C09	23	50	52,22	Active	2001-12-12	2015-12-11	6	3426,71	2500	Les Mines Opinaca ltée	100
1028927	Eleonore	33C09	22	50	52,23	Active	2001-09-22	2015-09-21	6	3447	2500	Les Mines Opinaca ltée	100
1129599	Eleonore	33C09	27	50	52,18	Active	2004-02-12	2018-02-11	6	0	2500	Les Mines Opinaca ltée	100
1042270	Eleonore	33C09	19	50	52,26	Active	2001-12-12	2015-12-11	6	332,52	2500	Les Mines Opinaca ltée	100
13715	Eleonore	33C09	17	50	52,28	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
13752	Eleonore	33C09	20	51	52,25	Active	2004-02-26	2016-02-25	5	318,82	1800	Les Mines Opinaca ltée	100
6659	Eleonore	33C09	26	51	52,19	Suspended	2003-11-12	2015-11-11	5	4206,1	1800	Les Mines Opinaca ltée	100
13716	Eleonore	33C09	17	51	52,28	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
1129600	Eleonore	33C09	27	51	52,18	Active	2004-02-12	2018-02-11	6	0	2500	Les Mines Opinaca ltée	100
1028936	Eleonore	33C09	23	51	52,22	Active	2001-09-22	2015-09-21	6	3112,36	2500	Les Mines Opinaca ltée	100
1105823	Eleonore	33C09	24	51	52,21	Suspended	2002-11-28	2014-11-27	5	2429677,82	1800	Les Mines Opinaca ltée	100
1028928	Eleonore	33C09	22	51	52,23	Active	2001-09-22	2017-09-21	7	0	2500	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
13732	Eleonore	33C09	18	51	52,27	Active	2004-02-26	2016-02-25	5	2078,8	1800	Les Mines Opinaca ltée	100
13172	Eleonore	33C09	14	51	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36770	Eleonore	33C09	28	51	52,17	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13684	Eleonore	33C09	15	51	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13742	Eleonore	33C09	19	51	52,26	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13700	Eleonore	33C09	16	51	52,29	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
1028916	Eleonore	33C09	21	51	52,24	Active	2001-09-22	2017-09-21	7	873,93	2500	Les Mines Opinaca ltée	100
1105835	Eleonore	33C09	25	51	52,20	Suspended	2002-11-28	2014-11-27	5	5488025,89	1800	Les Mines Opinaca ltée	100
13154	Eleonore	33C09	13	51	52,32	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36771	Eleonore	33C09	28	52	52,17	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13155	Eleonore	33C09	13	52	52,32	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
1129601	Eleonore	33C09	27	52	52,18	Active	2004-02-12	2018-02-11	6	0	2500	Les Mines Opinaca ltée	100
6660	Eleonore	33C09	26	52	52,19	Active	2003-11-12	2015-11-11	5	2172,62	1800	Les Mines Opinaca ltée	100
13753	Eleonore	33C09	20	52	52,25	Active	2004-02-26	2016-02-25	5	168,3	1800	Les Mines Opinaca ltée	100
13685	Eleonore	33C09	15	52	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13173	Eleonore	33C09	14	52	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13733	Eleonore	33C09	18	52	52,27	Active	2004-02-26	2016-02-25	5	88,02	1800	Les Mines Opinaca ltée	100
1028917	Eleonore	33C09	21	52	52,24	Active	2001-09-22	2017-09-21	7	873,93	2500	Les Mines Opinaca ltée	100
42666	Road	33C09	30	52	52,15	Active	2004-10-13	2014-10-12	4	1461,86	1800	Les Mines Opinaca ltée	100
1028929	Eleonore	33C09	22	52	52,23	Active	2001-09-22	2017-09-21	7	0	2500	Les Mines Opinaca ltée	100
1028937	Eleonore	33C09	23	52	52,22	Active	2001-09-22	2015-09-21	6	3420,97	2500	Les Mines Opinaca ltée	100
13743	Eleonore	33C09	19	52	52,26	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1105824	Eleonore	33C09	24	52	52,21	Suspended	2002-11-28	2014-11-27	5	114049,74	1800	Les Mines Opinaca ltée	100
13701	Eleonore	33C09	16	52	52,29	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
2268091	Road	33C15	2	52	52,14	Active	2011-01-18	2015-01-17	1	0	450	Les Mines Opinaca ltée	100
1105836	Eleonore	33C09	25	52	52,20	Suspended	2002-11-28	2014-11-27	5	112980,4	1800	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
13717	Eleonore	33C09	17	52	52,28	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13156	Eleonore	33C09	13	53	52,32	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13718	Eleonore	33C09	17	53	52,28	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1028918	Eleonore	33C09	21	53	52,24	Active	2001-09-22	2015-09-21	6	873,93	2500	Les Mines Opinaca ltée	100
13744	Eleonore	33C09	19	53	52,26	Active	2004-02-26	2016-02-25	5	1529,96	1800	Les Mines Opinaca ltée	100
1028938	Eleonore	33C09	23	53	52,22	Active	2001-09-22	2015-09-21	6	2930,86	2500	Les Mines Opinaca ltée	100
13734	Eleonore	33C09	18	53	52,27	Active	2004-02-26	2016-02-25	5	168,3	1800	Les Mines Opinaca ltée	100
1105837	Eleonore	33C09	25	53	52,20	Active	2002-11-28	2014-11-27	5	112980,3	1800	Les Mines Opinaca ltée	100
13754	Eleonore	33C09	20	53	52,25	Active	2004-02-26	2016-02-25	5	2078,8	1800	Les Mines Opinaca ltée	100
2268092	Road	33C15	2	53	52,14	Active	2011-01-18	2015-01-17	1	0	450	Les Mines Opinaca ltée	100
1028930	Eleonore	33C09	22	53	52,23	Active	2001-09-22	2017-09-21	7	0	2500	Les Mines Opinaca ltée	100
13702	Eleonore	33C09	16	53	52,29	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1129602	Eleonore	33C09	27	53	52,18	Active	2004-02-12	2018-02-11	6	0	2500	Les Mines Opinaca ltée	100
13174	Eleonore	33C09	14	53	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13686	Eleonore	33C09	15	53	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
6661	Eleonore	33C09	26	53	52,19	Active	2003-11-12	2015-11-11	5	3715,99	1800	Les Mines Opinaca ltée	100
1105825	Eleonore	33C09	24	53	52,21	Active	2002-11-28	2014-11-27	5	111759,11	1800	Les Mines Opinaca ltée	100
36772	Eleonore	33C09	28	53	52,17	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13157	Eleonore	33C09	13	54	52,32	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13687	Eleonore	33C09	15	54	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13762	Eleonore	33C09	21	54	52,24	Active	2004-02-26	2016-02-25	5	1683,47	1800	Les Mines Opinaca ltée	100
1129603	Eleonore	33C09	27	54	52,18	Active	2004-02-12	2018-02-11	6	0	2500	Les Mines Opinaca ltée	100
1105826	Eleonore	33C09	24	54	52,21	Active	2002-11-28	2014-11-27	5	7057,52	1800	Les Mines Opinaca ltée	100
13719	Eleonore	33C09	17	54	52,28	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
36773	Eleonore	33C09	28	54	52,17	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13735	Eleonore	33C09	18	54	52,27	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1105802	Eleonore	33C09	22	54	52,23	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
13755	Eleonore	33C09	20	54	52,25	Active	2004-02-26	2016-02-25	5	441,53	1800	Les Mines Opinaca ltée	100
13745	Eleonore	33C09	19	54	52,26	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
13175	Eleonore	33C09	14	54	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13703	Eleonore	33C09	16	54	52,29	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
6662	Eleonore	33C09	26	54	52,19	Active	2003-11-12	2015-11-11	5	5974,31	1800	Les Mines Opinaca ltée	100
1105838	Eleonore	33C09	25	54	52,20	Active	2002-11-28	2014-11-27	5	6616,52	1800	Les Mines Opinaca ltée	100
1105807	Eleonore	33C09	23	54	52,22	Active	2002-11-28	2014-11-27	5	4079,2	1800	Les Mines Opinaca ltée	100
13736	Eleonore	33C09	18	55	52,27	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
36761	Eleonore	33C09	27	55	52,18	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13763	Eleonore	33C09	21	55	52,24	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
6663	Eleonore	33C09	26	55	52,19	Active	2003-11-12	2017-11-11	6	0	2500	Les Mines Opinaca ltée	100
13688	Eleonore	33C09	15	55	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13176	Eleonore	33C09	14	55	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36774	Eleonore	33C09	28	55	52,17	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
1105839	Eleonore	33C09	25	55	52,20	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
13704	Eleonore	33C09	16	55	52,29	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1105803	Eleonore	33C09	22	55	52,23	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
13756	Eleonore	33C09	20	55	52,25	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
13746	Eleonore	33C09	19	55	52,26	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13720	Eleonore	33C09	17	55	52,28	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1105827	Eleonore	33C09	24	55	52,21	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
1105808	Eleonore	33C09	23	55	52,22	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
36775	Eleonore	33C09	28	56	52,17	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36762	Eleonore	33C09	27	56	52,18	Active	2004-09-14	2018-09-13	6	0	2500	Les Mines Opinaca ltée	100
13689	Eleonore	33C09	15	56	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
6664	Eleonore	33C09	26	56	52,19	Active	2003-11-12	2015-11-11	5	2056,39	1800	Les Mines Opinaca ltée	100
1105804	Eleonore	33C09	22	56	52,23	Active	2002-11-28	2014-11-27	5	207,46	1800	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
6648	Eleonore	33C09	24	56	52,21	Active	2003-11-12	2015-11-11	5	2056,63	1800	Les Mines Opinaca ltée	100
13764	Eleonore	33C09	21	56	52,24	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
13737	Eleonore	33C09	18	56	52,27	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1105809	Eleonore	33C09	23	56	52,22	Active	2002-11-28	2018-11-27	7	0	2500	Les Mines Opinaca ltée	100
13177	Eleonore	33C09	14	56	52,31	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
13757	Eleonore	33C09	20	56	52,25	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13705	Eleonore	33C09	16	56	52,29	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
6650	Eleonore	33C09	25	56	52,20	Active	2003-11-12	2015-11-11	5	2056,63	1800	Les Mines Opinaca ltée	100
13747	Eleonore	33C09	19	56	52,26	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13721	Eleonore	33C09	17	56	52,28	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13765	Eleonore	33C09	21	57	52,24	Active	2004-02-26	2016-02-25	5	2144,07	1800	Les Mines Opinaca ltée	100
13706	Eleonore	33C09	16	57	52,29	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
1105805	Eleonore	33C09	22	57	52,23	Active	2002-11-28	2014-11-27	5	1477,6	1800	Les Mines Opinaca ltée	100
36776	Eleonore	33C09	28	57	52,17	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13690	Eleonore	33C09	15	57	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
6651	Eleonore	33C09	25	57	52,20	Active	2003-11-12	2017-11-11	6	0	2500	Les Mines Opinaca ltée	100
1105810	Eleonore	33C09	23	57	52,22	Active	2002-11-28	2014-11-27	5	1342,6	1800	Les Mines Opinaca ltée	100
13758	Eleonore	33C09	20	57	52,25	Active	2004-02-26	2016-02-25	5	62,89	1800	Les Mines Opinaca ltée	100
13748	Eleonore	33C09	19	57	52,26	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
6665	Eleonore	33C09	26	57	52,19	Active	2003-11-12	2017-11-11	6	0	2500	Les Mines Opinaca ltée	100
13738	Eleonore	33C09	18	57	52,27	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
6649	Eleonore	33C09	24	57	52,21	Active	2003-11-12	2017-11-11	6	0	2500	Les Mines Opinaca ltée	100
36763	Eleonore	33C09	27	57	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13722	Eleonore	33C09	17	57	52,28	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13749	Eleonore	33C09	19	58	52,26	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13759	Eleonore	33C09	20	58	52,25	Active	2004-02-26	2016-02-25	5	1148,04	1800	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
36736	Eleonore	33C09	24	58	52,21	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
1105806	Eleonore	33C09	22	58	52,23	Active	2002-11-28	2014-11-27	5	1557,78	1800	Les Mines Opinaca ltée	100
13739	Eleonore	33C09	18	58	52,27	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13766	Eleonore	33C09	21	58	52,24	Active	2004-02-26	2016-02-25	5	2144,07	1800	Les Mines Opinaca ltée	100
36750	Eleonore	33C09	26	58	52,19	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13723	Eleonore	33C09	17	58	52,28	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13691	Eleonore	33C09	15	58	52,30	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
13707	Eleonore	33C09	16	58	52,29	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
36764	Eleonore	33C09	27	58	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36777	Eleonore	33C09	28	58	52,17	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36743	Eleonore	33C09	25	58	52,20	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
1105811	Eleonore	33C09	23	58	52,22	Active	2002-11-28	2014-11-27	5	2105,94	1800	Les Mines Opinaca ltée	100
36765	Eleonore	33C09	27	59	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
13767	Eleonore	33C09	21	59	52,24	Active	2004-02-26	2016-02-25	5	873,93	1800	Les Mines Opinaca ltée	100
1129593	Eleonore	33C09	22	59	52,23	Active	2004-02-12	2016-02-11	5	1380,73	1800	Les Mines Opinaca ltée	100
13760	Eleonore	33C09	20	59	52,25	Active	2004-02-26	2016-02-25	5	1148,04	1800	Les Mines Opinaca ltée	100
36778	Eleonore	33C09	28	59	52,17	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100
36737	Eleonore	33C09	24	59	52,21	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
36751	Eleonore	33C09	26	59	52,19	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
36744	Eleonore	33C09	25	59	52,20	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
1129595	Eleonore	33C09	23	59	52,22	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36752	Eleonore	33C09	26	60	52,19	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
13768	Eleonore	33C09	21	60	52,24	Active	2004-02-26	2016-02-25	5	0	1800	Les Mines Opinaca ltée	100
36738	Eleonore	33C09	24	60	52,21	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
1129594	Eleonore	33C09	22	60	52,23	Active	2004-02-12	2016-02-11	5	1380,73	1800	Les Mines Opinaca ltée	100
36745	Eleonore	33C09	25	60	52,20	Active	2004-09-14	2014-09-13	4	1324,12	1800	Les Mines Opinaca ltée	100
36779	Eleonore	33C09	28	60	52,17	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100

Claim Name	PROPERTY NAME	NTS	ROW	COL.	AREA (ha)	STATUS	Date of Grant	Expiry Date	Nb of Renewal	Credits ($)	Work Req. ($)	Owner	%
13761	Eleonore	33C09	20	60	52,25	Active	2004-02-26	2016-02-25	5	1147,77	1800	Les Mines Opinaca ltée	100
1129596	Eleonore	33C09	23	60	52,22	Active	2004-02-12	2016-02-11	5	0	1800	Les Mines Opinaca ltée	100
36766	Eleonore	33C09	27	60	52,18	Active	2004-09-14	2014-09-13	4	450,19	1800	Les Mines Opinaca ltée	100