TECHNICAL REPORT

UPDATED RESERVES AND RESOURCES ON DECEMBER 31, 2009, THE SLEEPING GIANT MINE, NORTHWESTERN QUEBEC

North American Palladium Ltd.
2116 – 130 Adelaide Street West
Toronto, Ontario
Canada, M5H 3P5

Vincent Jourdain, Eng. PhD

March 31, 2010

TABLE OF CONTENTS

List of Figures

List of Tables

List of Appendices

SUMMARY (ITEM 3)

Vincent Jourdain, Eng. PhD has been mandated by Cadiscor Resources Inc. (Cadiscor), a subsidiary of North American Palladium Ltd., to prepare a technical report consistent with National Instrument 43-101, Companion Policy NI 43-101CP and form 43-101F1, discussing Cadiscor's Sleeping Giant Mine with an effective date of December 31, 2009. This report is being prepared for company corporate purposes. Mr. Jourdain is an employee of Cadiscor Resources Inc.

The Sleeping Giant Property is located at 80 km north of the city of Amos, Quebec and at the junction of the Maizeret, Glandelet, Soissons and Chaste townships. The property covers an area of 3,141 hectares and is composed of four mining leases and 69 mining claims surrounding the mining infrastructures. Cadiscor holds a 100% interest in the property.

The Sleeping Giant Property is easily accessible via Highway 109, connecting Amos to Matagami, which passes less than 1 km from the mine site

The landscape is relatively flat and lightly timbered. It is limited to the west and south by the Harricana and Coigny Rivers.

Exploration work in the area began in 1957. Several aerial and ground geophysical surveys, as well as some drilling, targeting base metals were carried out. These were followed with an exploration program which was carried out from 1976 to 1982 by Matagami Lake Exploration. With subsequent diamond drilling campaigns (12,900 m), gold mineralisation in Zone A was discovered.

The Sleeping Giant property is located in the first volcanic cycle of the North Volcanic Zone of the Abitibi sub-province. The location of the Sleeping Giant Mine matches a disturbance in the regional tectonic grain which forms a triple junction emphasized by the three tonalitic polyphase and synvolcanic plutons arrangement. This area is affected by major deformation zones E-W and NW-SE. The Joutel mining camp is located at 50 km NW, and the Matagami mining camp is located at 80 km from the Sleeping Giant Mine.

At the deposit scale, the orebody geometry increases in complexity towards the south which corresponds to the paleo-surface. No other Abitibi deposit presents a geological setting similar to the Sleeping Giant Mine. Its origin is thus different in at least some respects from other synorogenic vein type gold mineralization.

Based on historical data and new drilling from underground stations, a calculation of current resources and reserves has been completed for the Sleeping Giant gold mine.

Calculations were carried out on cross-sections and inclined longitudinal sections generated by Cadiscor. Intersection grade was calculated using orthogonal thickness of the drill hole intersections as calculated using the program Promine and a plane fit to the vein geometry. This corresponds to the method historically used at the Sleeping Giant Mine. The resource calculation was done by the polygon method.

Exploration drilling in 2009 was oriented at defining down-dip extensions of known and suspected zones, while detailed definition drilling was oriented at providing a more detailed understanding of reserves identified in the Genivar report of 2008. The results of these drill programs have been compiled into new estimates of Mineral Resources and Mineral Reserves, which have been categorised and are disclosed here:

Table 1 Statement of Mineral Resources

RESOURCES*	2009-12-31		
	Measured	90,800 tonnes	at 8.9 g/t Au
	Indicated:	309,000 tonnes	at 8.8 g/t Au
	Inferred:	243,500 tonnes	at 12.9 g/t Au

* tonnages and grades are rounded to reflect precision of calculations

Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

Within this geological resource, engineering studies have identified Mineral Reserves accessible from current and planned workings of:

Table 2 Statement of Reserves

*RESERVES**	2009-12-31		
	Proven:	*90,800 tonnes*	*at 8.9 g/t*
	Probable:	*96,100 tonnes*	*at 9.8 g/t*
TOTAL: 186,900 tonnes at 9.4 g/t for 56,244 ounces recovered			

* tonnages and grades are rounded to reflect precision of calculations

Mineral Reserves as stated can be produced at or below a production cost of $CDN 850 per troy ounce. With an exchange rate of 1$US = 1.07 $CDN, this corresponds to a gold price of 794 $US per troy ounce.

Reserve estimates are based on historical mine operating costs and gold recoveries at the mine. The estimated cost of each stope has included development costs based on current mine costs and per-shift production.

If all converted to reserves, current resources would represent 43 additional months of production at the mine production rate of 15 000 tonnes per month.

INTRODUCTION AND TERMS OF REFERENCE (ITEM 4)

Vincent Jourdain, Eng. PhD has been mandated by Cadiscor Resources Inc. (Cadiscor) to prepare an independent technical report consistent with National Instrument 43-101, Companion Policy NI 43-101CP and Form 43-101F1, discussing the company's Sleeping Giant Mine. This report is being prepared for company corporate purposes to disclose estimates of Mineral reserves and Mineral Resources with an effective date of December 31, 2009.

The writer is an employee of Cadiscor Inc. and has supervised and managed a number of geological reports and studies. Reports of previous work at the Sleeping Giant Mine have been made available to the author and geological reports and maps prepared by the Ministère de l'Énergie et des Ressources, Québec have also been used in preparing the current report.

This report is based on and follows from a report prepared in 2008 by Genivar LP (Birkett *et al*., 2008) which is available on SEDAR through Cadiscor Resources Ltd.

Metric units and Canadian dollars ($CDN) are used throughout this report, unless other units are stipulated.

The effective date of this report is December 31, 2009 with the Sleeping Giant Mine reserve and resource estimates being completed in March 2010.

RELIANCE ON OTHER EXPERTS (ITEM 5)

This report has been prepared by Vincent Jourdain Eng. PhD for Cadiscor. The information, conclusions, opinions and estimates contained herein are based on:

- All information available to the author at the time of preparation of this report.
- Assumptions, conditions and qualifications as set forth in this report.
- Data, reports and opinions supplied by Cadiscor Inc. and from public sources.

The author has relied on reports and opinions from third party sources for the following information:

- Historical property information provided by IAMGOLD to Cadiscor.
- Environmental compliance data and requirements supplied by former mine operator IAMGOLD to Cadiscor.
- Current and historical costs, productivities and mill recoveries provided by IAMGOLD and by Cadiscor.

Information and opinions expressed in this report are based on the ongoing experience of an operating mine, including real costs, mill recoveries and geological interpretations versus mining experience.

PROPERTY DESCRIPTION AND LOCATION (ITEM 6)

The Sleeping Giant Property (Figure 1) is located 80 km north of the town of Amos, Quebec and at the junction of Maizeret, Glandelet, Soissons and Chaste Townships. Provincial highway 109, connecting Amos and Matagami is located less than 1 km east of the mine site. The landscape is relatively flat and lightly timbered. It is limited to the west and south by the Harricana and Coigny Rivers. Overburden thickness varies between 15 and 60 m with an average of 30 meters.

The Sleeping Giant Mine property is composed of four mining leases and 69 mining claims surrounding the mining infrastructures (Appendix 1).

IAMGOLD Corporation held 100 % of mineral rights, claims and interest of the Sleeping Giant Mine after acquiring it from Cambior Inc. in November of 2006. In October 2007, Cadiscor Resources Inc. signed an agreement with IAMGOLD in order to acquire mineral rights at the end of the commercial production (originally estimated March 31, 2009) with delivery to Cadiscor at the end of October, 2008.

This property is subject to two royalties. The first one, in favour of Central Asia Goldfield Corporation, is constituted of 2 % on the operational gross margin. The second one of 15 % of incomes is held by Matagami Lake Exploration Ltd. To this day, no royalties have been paid. The Company previously exercised its right to buy back a 1% net smelter return royalty on the Sleeping Giant mine held by IAMGOLD Corporation for $1 million.

ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURES AND PHYSIOGRAPHY (ITEM 7)

The Sleeping Giant Mine is accessed via provincial highway 109, which connects Amos to Matagami. There are no services or infrastructure in the immediate vicinity of the property. The nearest significant urban centre is Amos, about 80 km south of the property.

The landscape is relatively flat and lightly timbered. It is limited to the west and south by the Harricana and Coigny Rivers. The major forest vegetation consists of Black Spruce.

The climate is typical of north-western Quebec. Weather data for Amos, the nearest reporting centre, show that January is the coldest month with an average maximum of -12 ˚C and an average minimum of -23 ˚C, while July is the warmest month with an average maximum of 22 ˚C and an average minimum of 10 ˚C. Rainfall is highest in July with 115 mm and snowfall is highest in December with 57cm.

HISTORY (ITEM 8)

In 1957, following the discovery of the Lac Matagami Zn-Cu deposit located approximately 65 km north of the Sleeping Giant Mine, work started in the Sleeping Giant area. Several aerial and ground geophysical surveys, as well as some drilling, searching for base metals were carried out. These were followed with an exploration program which was carried out from 1976 to 1982 by Matagami Lake Exploration. With subsequent regional input surveys that were carried out in the boundaries of the property, anomalies that were detected were systematically verified with ground line cutting, electromagnetic and magnetic surveys and on some occasions induced polarization. With subsequent diamond drilling campaigns (12 900 m), the Zone A was discovered.

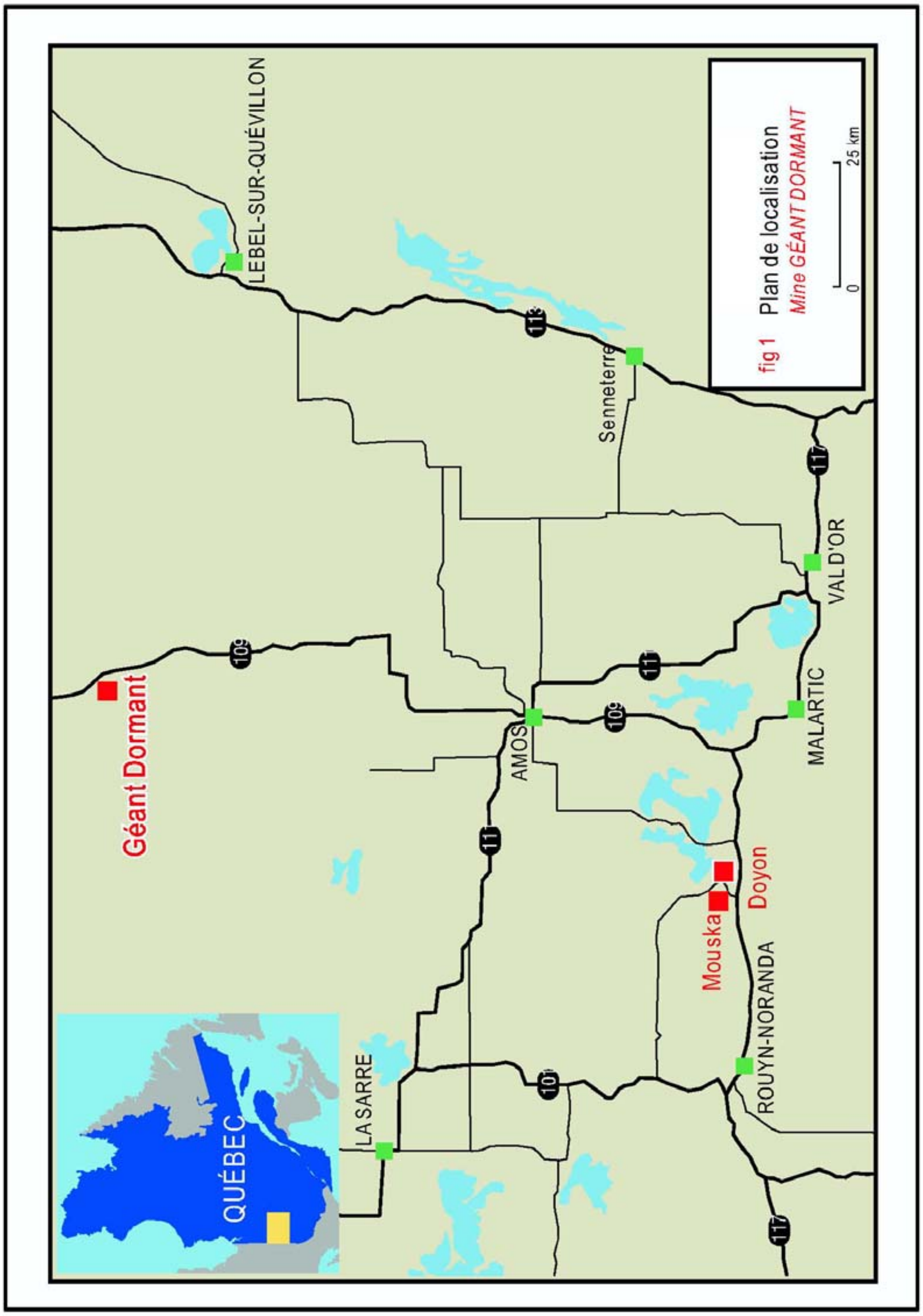


Figure 1 Location Map, the Sleeping Giant property

In 1983, Perron Gold Mines (now named Aurizon Mines Ltd) acquire 50% interest in the property by carrying out ground geochemistry and geophysical surveys (magnetic and very-low-frequency studies), drilling, as well as the beginning of underground exploration. Between 1984 and 1987, two shafts were sunk and sufficient reserves were delimited to begin development work. The first phase of commercial production occurred between 1988 and 1991, during which 494,000 tonnes at 6.4 g Au/t were extracted from levels 55 to 415. By the end of 1990, Aurizon Mines, then sole owner of the Sleeping Giant Mine, stopped work due to the depletion of reserves.

In 1991, an agreement between Aurizon and Cambior allowed Cambior to acquire 50% interest in the property by investing in drilling and in underground work. With this Cambior became the project manager. Some 13,354 meters of drilling completed between 1991 and 1993 lead to the discovery of four new mineralised veins (20, 30, 40 et JD) as well as the second phase of commercial production which started in 1993 and is still in progress at this date. Major significant facts of this period are: the discovery of lens 2, 3, 4, 5, 6, 7, 8, 9, 16, 18 and 50 as well as the sinking of the shaft in two phases, that is to say level 485 to 785 in 1995 and level 785 to 975 in 2003. By the end of 2007, the second commercial production phase had seen a total of 868,000 ounces of gold extracted from 2 476 100 tonnes of ore at an average grade of 11.2 Au g/t.

From November 2006, IAMGOLD was the sole owner of the Sleeping Giant property following the acquisition of all CAMBIOR’s assets. In October, 2008, the mine and its mineral rights were purchased from IAMGOLD by Cadiscor Inc.

GEOLOGICAL SETTING (ITEM 9)

REGIONAL GEOLOGY

The Sleeping Giant property is located in the first volcanic cycle of the North Volcanic Zone of the Abitibi sub-province. The location of the Sleeping Giant Mine matches a disturbance of the regional tectonic grain which forms a triple junction emphasized by the three tonalitic polyphase and synvolcanic plutons arrangement (Figure 2). This area is affected by major deformation zones E-W and NW-SE. The Joutel mining camp is located at 50 km NW, and the Matagami mining camp is located at 65 km from the Sleeping Giant.

LOCAL GEOLOGY

The mine geology is composed of a volcanic and sedimentary sequence intruded by a felsic complex and post-mineralization dykes. The volcano-sedimentary rocks from a homoclinal sequence striking East-West with a steep southern dip (Figure 3).

As for the deposit geometry, the economic gold zones are restricted to the volcano-sedimentary sequence located north and south of the central dacitic intrusion.

The Sleeping Giant Mine gold ore is contained in sulphide bearing quartz veins. At the mine scale, the mineralized zones are spatially distributed inside 1 sq km surface to the north; the veins strike east-west with a steep southern dip of between 65 and 75 degrees. They are characterized by a vertical continuity of over 700 meters and a lateral continuity between 100 and 200 meters. To the south, a complex system made of four family of veins show a gradual change of the strike and connections with other veins at different attitudes. These veins are less continuous and extensive than those at the north. Their sizes vary between 50 to 100 meters laterally and less than 200 meters vertically.

PROPERTY GEOLOGY

All data related to drilling which were compiled since 2002 allowed increasing several aspects of the knowledge related to the Sleeping Giant Mine geological context.

The new characterizations of volcanic rocks of this sector identify two local-scale volcanic cycles (the North Cycle and the mine Cycle) in relation with an important intrusive complex.

At the base of the stratigraphic sequence is the North Cycle (north western part of the property), which contains mostly high-iron tholeiitic basalts and comagmatic gabbro sills. These tholeiites are easily distinguished from the tholeiites from the Mine Cycle due to their high TiO_2 content (>1.2%).

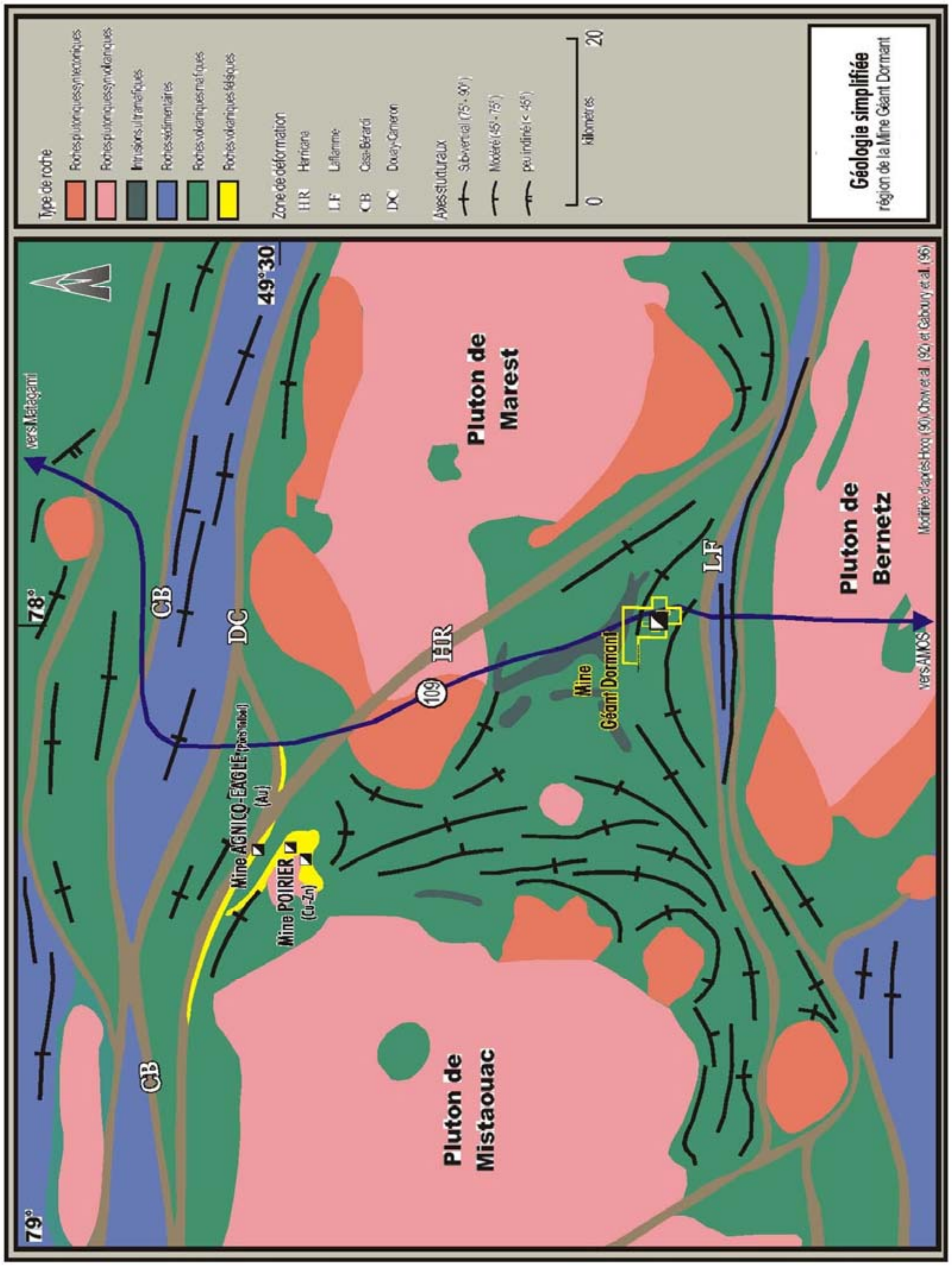


Figure 2 Sleeping Giant property setting in terms of regional geology

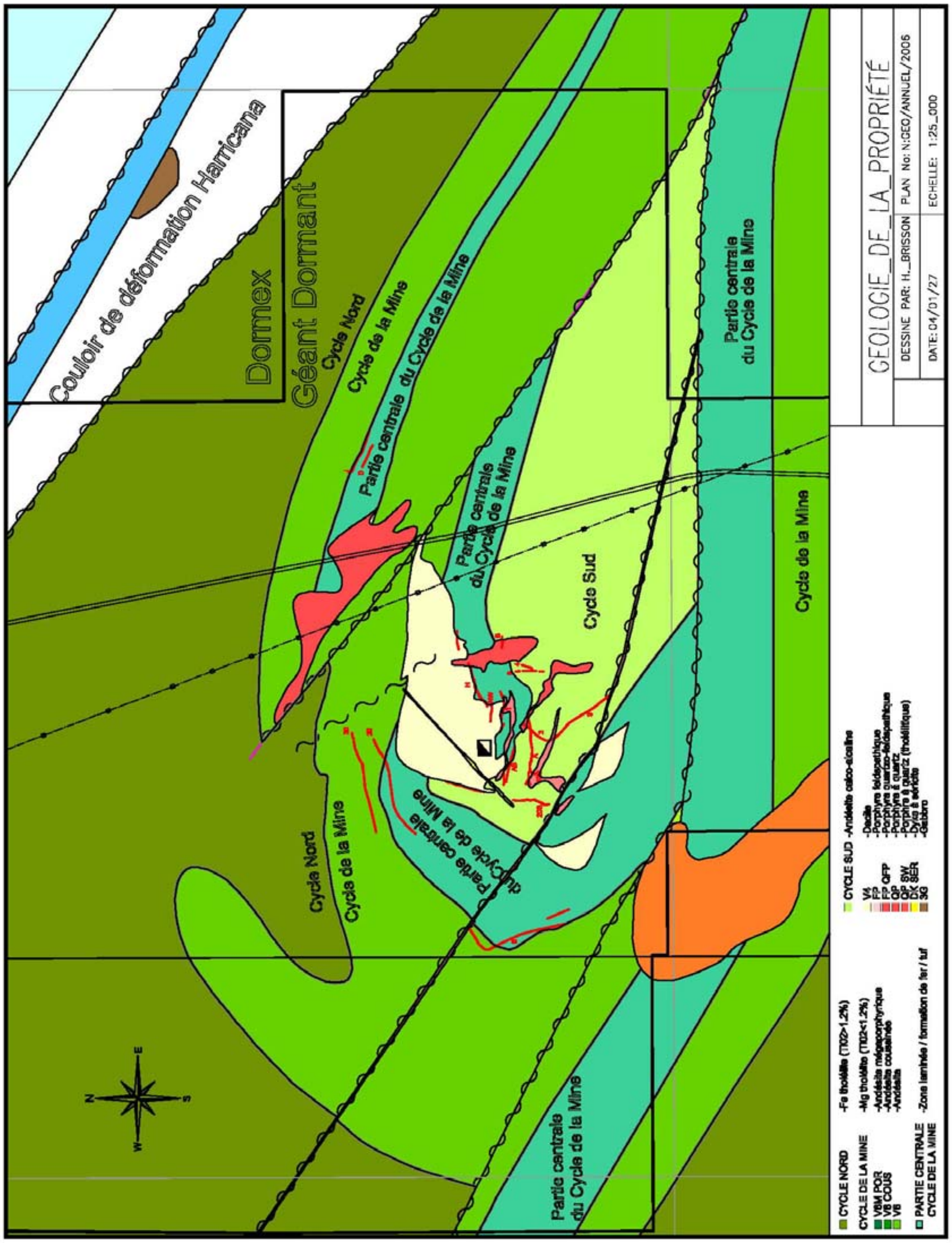


Figure 3 Local geology of the Sleeping Giant property

Stratigraphically above and lying in the same pattern as the North Cycle, the Mine Cycle represents the dominant host unit of the Sleeping Giant Mine. This Cycle mainly contains high-magnesium tholeiitic basalts and comagmatic gabbro sills. Some bedded deposits composed of fine clastic sediments; tuffs and iron-formation (with magnetite) are interbedded in the sequence. These sedimentary and volcaniclastic rocks define important units in the central part of the Mine Cycle.

The Mine Cycle stratigraphic sequence is cut by an important group of intrusive rocks of felsic to intermediate composition of calc-aclkailne afffinity which constitute the Sleeping Giant Complex. This intrusive complex is contemporary with the volcanic rocks. It includes a dominant dacitic mass, several smaller dacitic masses and a multitude of porphyric felsic dykes. Four major phases are recognized in the magmatic evolution of this complex:

1. dacite with mafic phenocrysts (chlorite mottles);
2. dacite with feldspar and feldspar + mafic phenocrysts ;
3. porphyry with quartz + feldspar phenocrysts (locally with granitic texture);
4. quartz porphyry.

The main dacitic mass occupies the central part of the mine and may reach up to 400 meters of thickness. Later intrusive phases (present as dykes) cut at a high angle (NW-SE to WNW-ESE) all the volcano-sedimentary sequence as well as the main dacitic mass. These dykes have various thicknesses (cm to m) and the largest examples are locally polyphase.

The most recent surveys show a more important volume of dacitic rock through the southwest (base of the Mine Cycle) in the lowest levels. This suggests that the dacitic mass is following the stratigraphy and therefore the center of the intrusive system could be lowered as the stratigraphic sequence going deeper.

Some post-mineralization tholeiitic dykes are observed in the mine. These dykes predate the main deformation. Most of them run along gold veins and might constitute markers in order to find the extensional gold bearing structures.

A quartz porphyry (sector SW) and a sericite dyke (oriented NE-SW) represent late felsic intrusions according to the main deformation. These intrusions are geochemically similar and are distinguished from those of the Sleeping Giant Complex by their low concentrations of MgO and TiO_2 and their ratio of Zr/Y (<5).

Finally, a lamprophyre dyke with hornblende phenocrysts (the mine gabbro) 5 to 25 meters thick, running NW-SE with a shallow dip going NE crosses the entire mine. This is a late dyke later than the main deformation. Several small dykes of the same type are observed at several locations in the supporting structure.

Recent data show that the mine area is the site of a tight fold dipping east with its axial surface sub-vertical and oriented ENE-WSW. Beds which are oriented ESE-WSW with a steep slope going south in the north area of the mine, pass N-S with a moderate slope going east in the south sector and come back at ESE to WSW in the south sector. Due to polarities, this is a syncline structure. More precise information shows this is a coffer style hinge line. Overall, the fold's dip is moderate going east, but information suggests that it has a steeper slope in the deepest levels.

Some faults oriented NW-SE show another important structural aspect of the mine sector. Two categories of fault NW-SE are distinguished in the mine environment: ductile dextral structures (sector SW) and brittle sinistral faults (sector NE). Both fault categories are late according to the upthrust and they displace the gold zones. In the SW sector of the mine, recent drilling showed the presence of an important NW-SE ductile, dextral fault (with a large zone of schistose rocks). This fault would have a dip of about 70° NE direction and a dextral horizontal throw of about 2 km. Characteristics suggest that this fault belongs to the NW-SE right-slip fault family which is recognized at the scale of the entire Abitibi sub-province. In the south fault wall of this fault, favourable lithologies of the Mine Cycle can be found in which no economic mineralization was known until recently. In the NE sector of the mine, an important NW-SE brittle, sinistral fault has been identified. The fault has a dip of about 65° going NE and a left horizontal throw in the order of 500 to 1 000 meters. In the same family, some NW-SE faults can be found and which support the main dacitic mass and a NW-SE fault associated to the main lamprophyre dyke. Running along the side of the dyke, this one is slightly inclined NE and has a net slip of about 100 meters.

DEPOSIT TYPES (ITEM 10)

The Sleeping Giant deposit is a member of the type of gold deposits formed by groups of veins with gold associated with sulphide minerals and whose geometry was controlled by the stress field in the rocks at the time of vein formation.

MINERALIZATION (ITEM 11)

The Sleeping Giant is a quartz-sulphide vein type gold deposit. The best-mineralised veins typically contain four sulphide minerals: pyrite, pyrrhotite, chalcopyrite and sphalerite, which form 5 to 60% of the veins. The typical vein thickness is between 20 and 80 cm with average grade between 35 and 85 Au g/t (uncut channel sample analyses). Besides gold, the veins contain silver and a small proportion of copper and zinc. The ratio Au : Ag is about 1 : 2. Zones 20 and 30 have a lateral/vertical continuity of 300 / 670 meters, that is to say a much more important vertical continuity than a lateral one. In zone 8, the lateral/vertical continuities are over 600 / 500 meters.

In new extensions of the multi-vein gold system, no change was observed in the nature of veins, i.e. no improvement related to tonnes and grades. Therefore, it is considered that future exploration in the extensions of the mineralized system is likely to show veins of the same type, tonnes and grades than those found up until now.

The economic veins are grouped in the Mine Cycle rocks and in North Cycle rocks surrounding the main intrusive mass of the Sleeping Giant complex (Figure 4). Lithologies and stratigraphic units affect the style and geometric characteristics of the ore structures, therefore on ore quality.

Controls of gold-bearing structures correspond to permeability zones in the supporting structure such as: faults, lithological contacts, joints, specific lithologies. Gold veins are usually oblique compared to bedding. Most of the veins are found in faults. Geological markers show that movements caused by these faults are limited, in the order of meters.

A zone's structural type and characteristics may change according to the lithologic environment. For example, zone 8 passes through a mixed environment including laminar units in its upper part to a more homogeneous environment in its lower part. This style change comes with a dip change in the ore structure.

Important veins seem associated with swarms of porphyry dykes. For example, zones 20, 30 and 8 are transversal structures to a series of sericitized quartz porphyry dykes.

Gold vein emplacement occurred before the regional deformation and the stratigraphic orientation change in the west sector affects the ore zones orientations. Veins bend in connection with stratigraphy. ENE - ESE veins usually have a steep dip to the south even though NW NE veins have a moderate dip to the east.

Ore veins occur mainly to the north, south and west of the main dacitic mass. The image is then an ore crescent on the west perimeter of the Sleeping Giant intrusive system. Veins in this crescent show a certain periodicity. Therefore, moving away from the intrusive center, a recurring spacing between ore structures is shown.

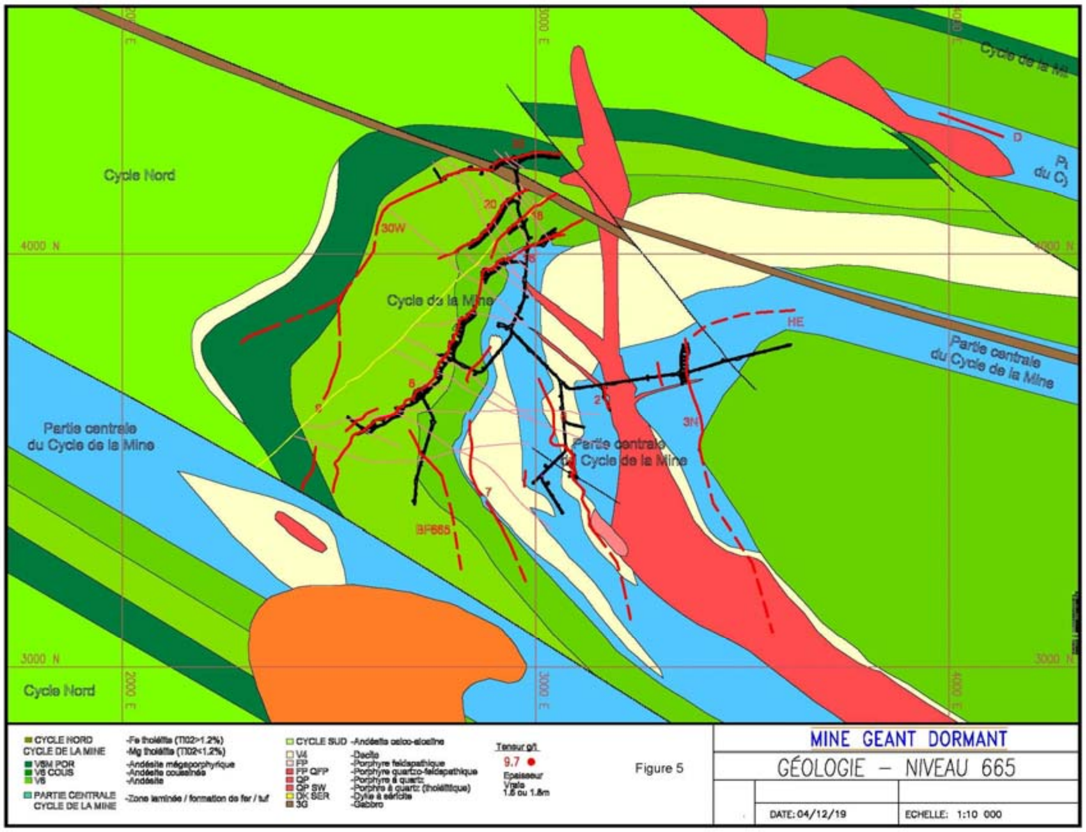


Figure 4 Sleeping Giant mine geology, plan view, level 655

With regard to spatial distribution of this vein-type system, several veins occur at the level of the fold hinge line. Since veins were in place before the folding, it is considered this abundance shows the fact that this hinge line is a site of favourable preservation, in contrast to the limb where veins might have been boudinaged. The hinge line then appears as a significant target where well-preserved ore veins can be found.

EXPLORATION (ITEM 12)

In 2007 and 2008, Cadiscor completed 90 underground drill holes for a total of 18 669 meters. These drill holes were completed with the objective of verifying the economic potential of veins below the current mine workings and of increasing knowledge of selected areas which had not been mined in the past even though gold-bearing veins had been defined there.

In 2009, an additional 24,718 m of definition drilling were completed for purposes of stope definition, and an additional 11,017 m of exploration drilling were completed to investigate the extension of known zones above and below the last working level of the mine.

DRILLLING (ITEM 13)

Drilling for purposes of the present report includes drill holes completed by the mine operations in stopes or areas which were subsequently not mined, as well as new drilling by Cadiscor in their exploration program. In all cases, drilling was from underground stations by standard methods with drill core of BQ size for exploration drilling and AQTK for definition drilling. Core boxes were closed at the drill station and transported to the core logging facility on surface for core description and sampling.

SAMPLING METHOD AND APPROACH (ITEM 14)

Drill Core Sampling

The core samples chosen for the analysis must be at least 50 cm long even if the ore zone is shorter. The maximum length of a sample is limited to 1 meter. Sampling of core is defined with the possibility that the observed mineralized zone (typically a vein in this situation) contains gold. During the operational phase of the mine, the entire core was sent to the laboratory. This practice, although unusual from an exploration point of view, is justified in a production setting where mineralised zones are recognised and followed over periods of months or years and the professional personnel control the drilling and geological programs over extended periods of time.

Drill core samples taken from holes are split and one-half of the core is retained. Samples are split from drill core using a hydraulic splitter which is standard in the industry.

SAMPLE PREPARATION, ANALYSES AND SECURITY (ITEM 15)

Laboratory Procedures

All stope definition samples were analyzed at the laboratory located at the mine site. The analytical method was fire assay with an atomic absorption finish. This method has a lower detection limit of 0.03 g/t Au. Samples returning a high gold concentration are reanalysed following dilution. To simplify calculations, results are typically reported to one decimal place.

All exploration samples were analysed by ALS-Chemex at Val-d'Or Quebec. ALS-Chemex is a certified laboratory with a robust internal system of standards, duplicate analyses and blanks.

Sample reception and preparation at the mine laboratory follow industry standards. The objective of the drying, crushing, quartering and pulverisation steps is to produce a rock sample of approximately 500 grams with 70 % passing 200 mesh. This sample preparation allows adequate homogeneity for reproducible results. A powdered sample of 15 grams (approximately ½ assay-tonne) is used for the gold analysis. This amount of sample is less than typical in exploration programs, but adequate when a larger number of samples will be used to define a stope for eventual mining.

Considering the number of potential sources of errors in any sampling and laboratory program, the Sleeping Giant geology department and laboratory established a QA/QC program. This program consisted of 1) the use of a check laboratory in order to verify the precision of the results (splits of the pulps), 2) insertion of blanks in order to control contamination errors, 3) continuous insertion of drill core pulps and tailings (re-numbered) in order to evaluate the reproducibility and finally 4) insertion of certified reference material samples.

Analysis laboratories must give reliable analytical results. It is important they show they have the required expertise to manage and execute analyses consistently. The Sleeping Giant laboratory has a control system and quality control program that has continuously demonstrated acceptable results. The Sleeping Giant laboratory QA/QC program includes:

- Insertion of RockLabs certified reference materials with each batch of samples. Results are presented in a series of graphs, below (Figures 5 to 11 and 13 to 15).
- Standards must return values within the Sleeping Giant laboratory average value ± two standard deviations for the included certified reference material for a given batch to be accepted.
- Series of samples which do not comply with the standards are reanalysed
- Blanks are inserted by the mine geology department in a regular but random manner in order to verify potential contamination and sample handling errors. Blanks are also inserted by the laboratory on a regular basis.
- A randomly chosen sample is reanalyzed (1 for approximately every 23 analyses).

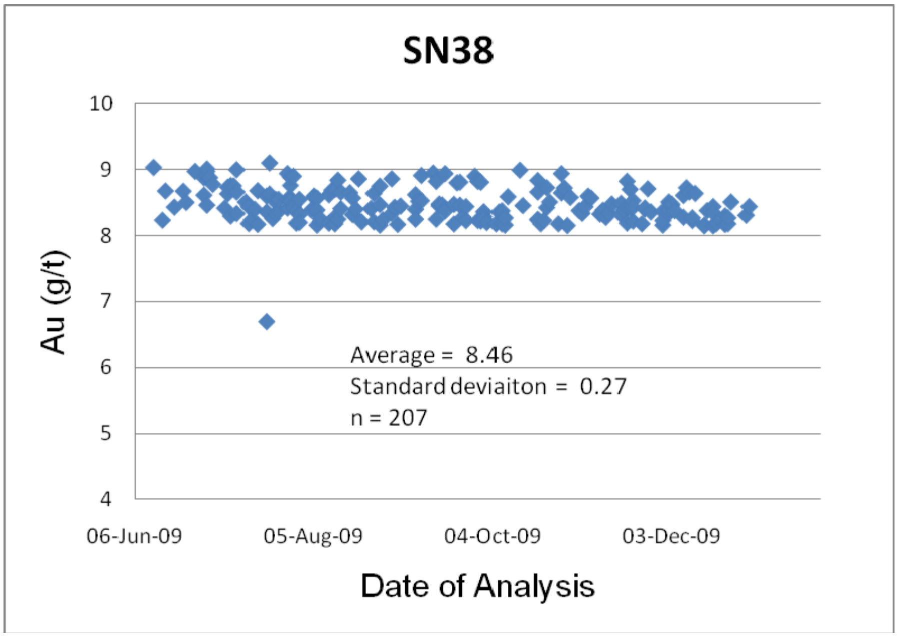


Figure 5 Graph of analyses of RockLabs product SN38 through the operating period of 2009.

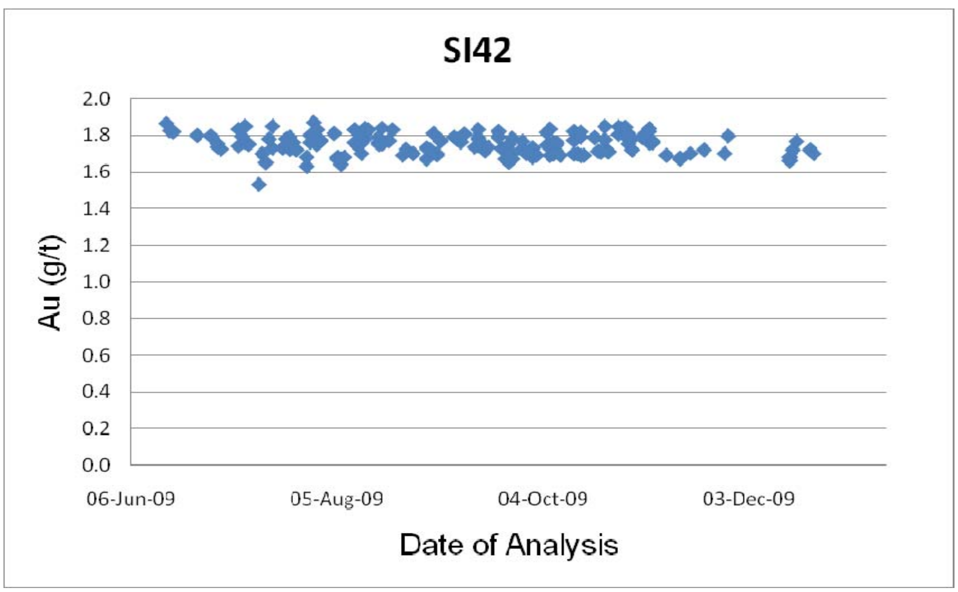


Figure 6 Graph of analyses of RockLabs product SI42 through the operating period of 2009.

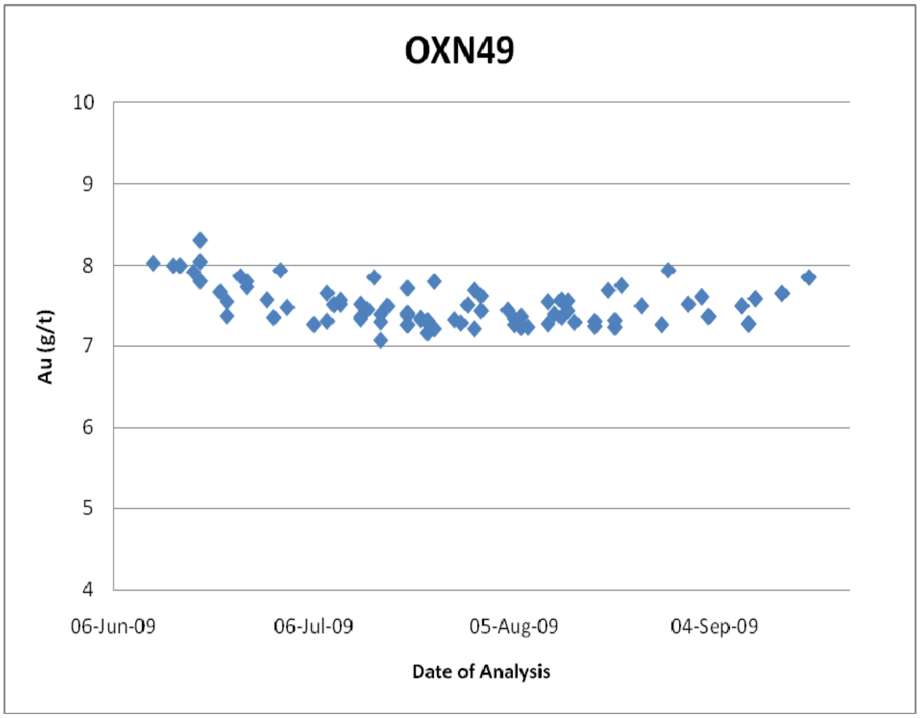


Figure 7 Graph of analyses of RockLabs product OXN49 through the operating period of 2009.

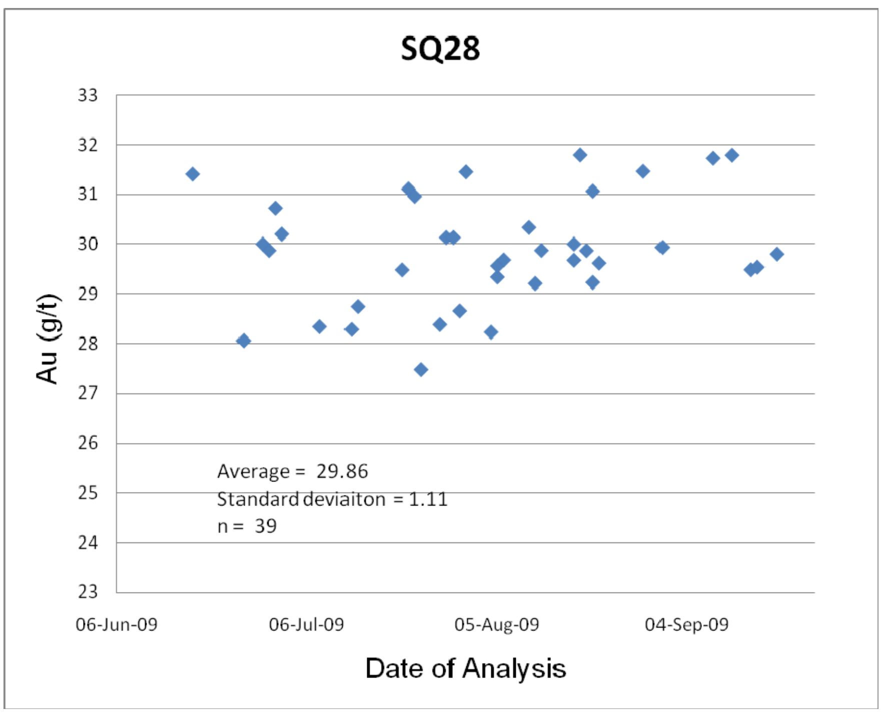


Figure 8 Graph of analyses of RockLabs product SQ28 through the operating period of 2009.

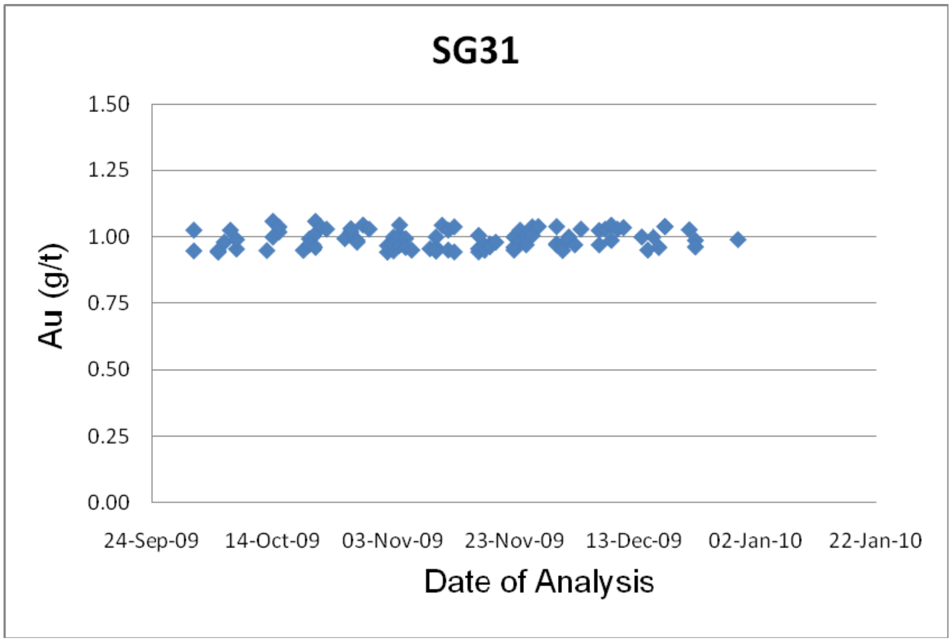


Figure 9 Graph of analyses of RockLabs product SG31 through the operating period of 2009.

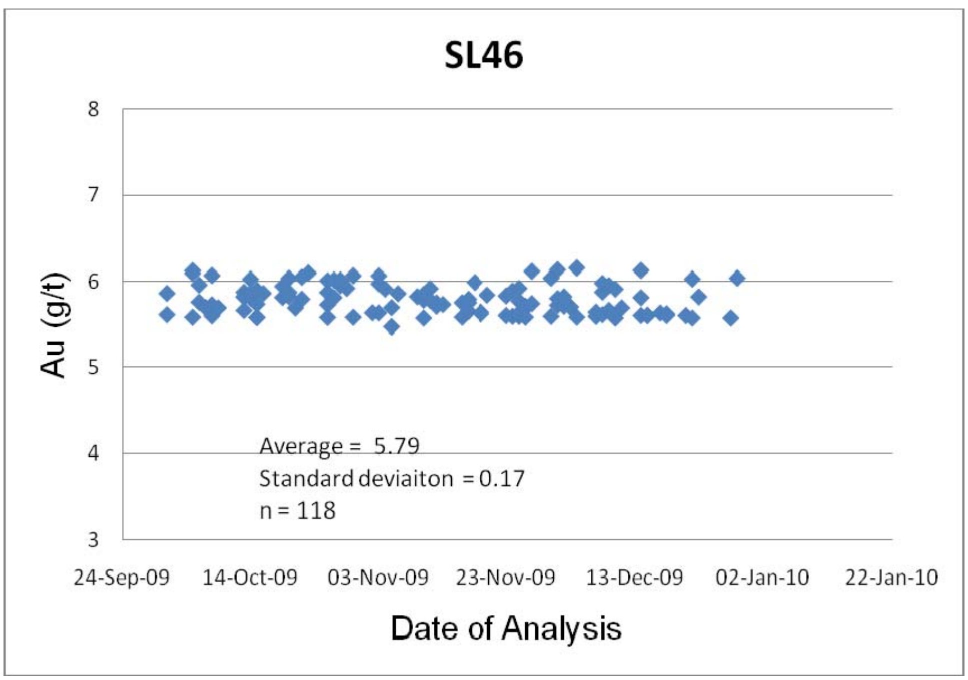


Figure 10 Graph of analyses of RockLabs product SL46 through the operating period of 2009.

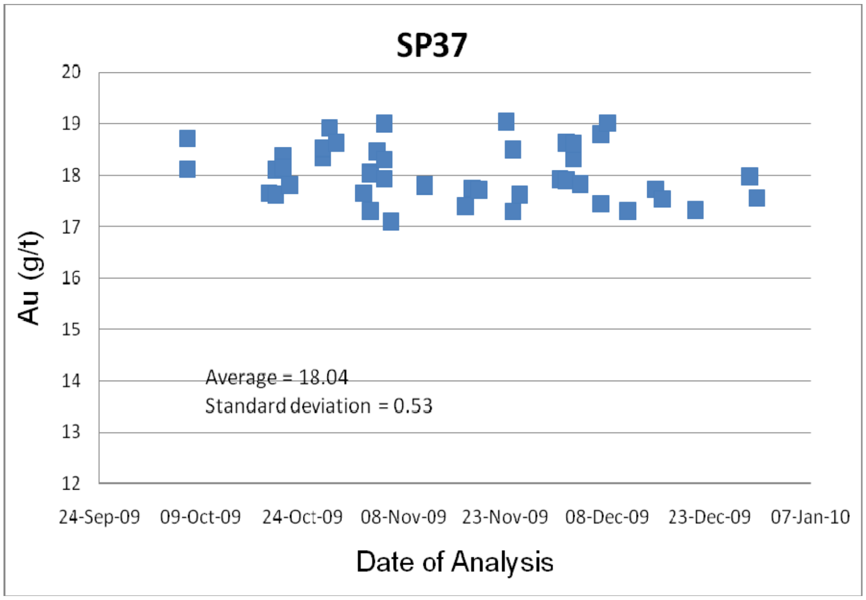


Figure 11 Graph of analyses of RockLabs product SP11 through the operating period of 2009.

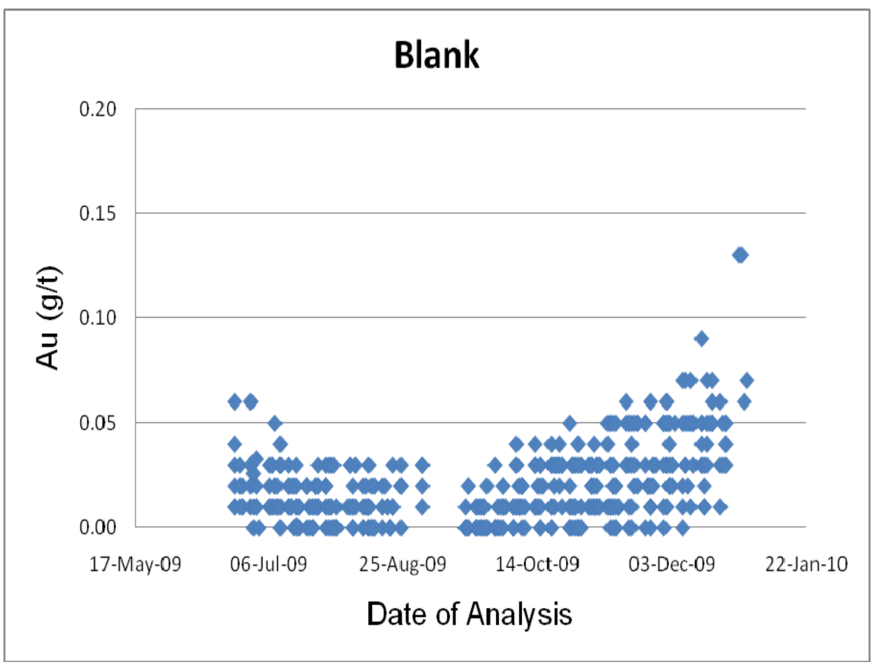


Figure 12 Results of Sleeping Giant analysis of laboratory blank samples throughout the operating period of 2009.

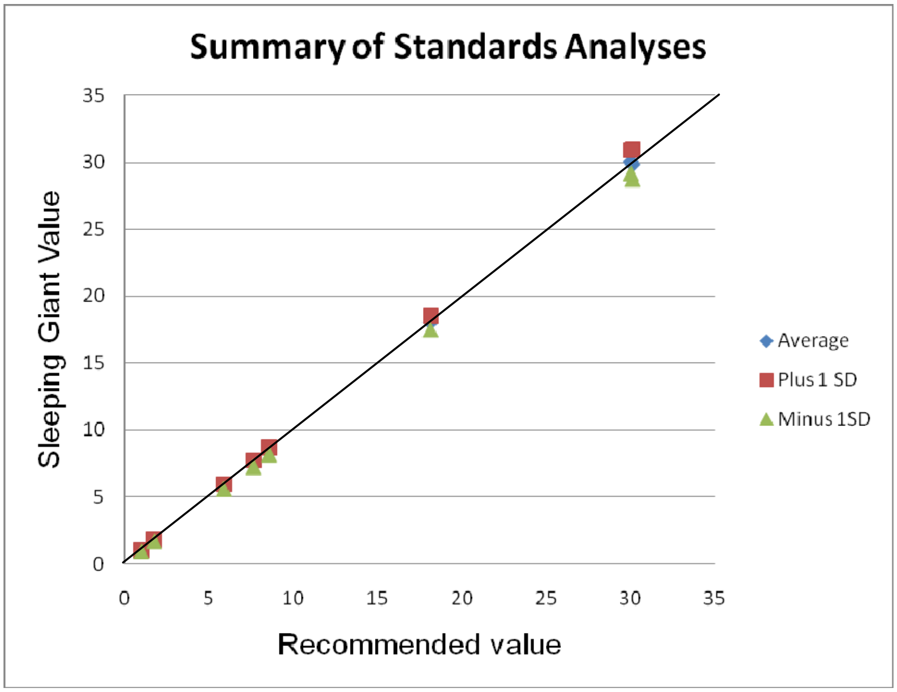


Figure 13 Summary of analyses of RockLabs products during the operating period of 2009. The Sleeping Giant results are plotted as the average values with plus and minus one standard deviation of the results populations.

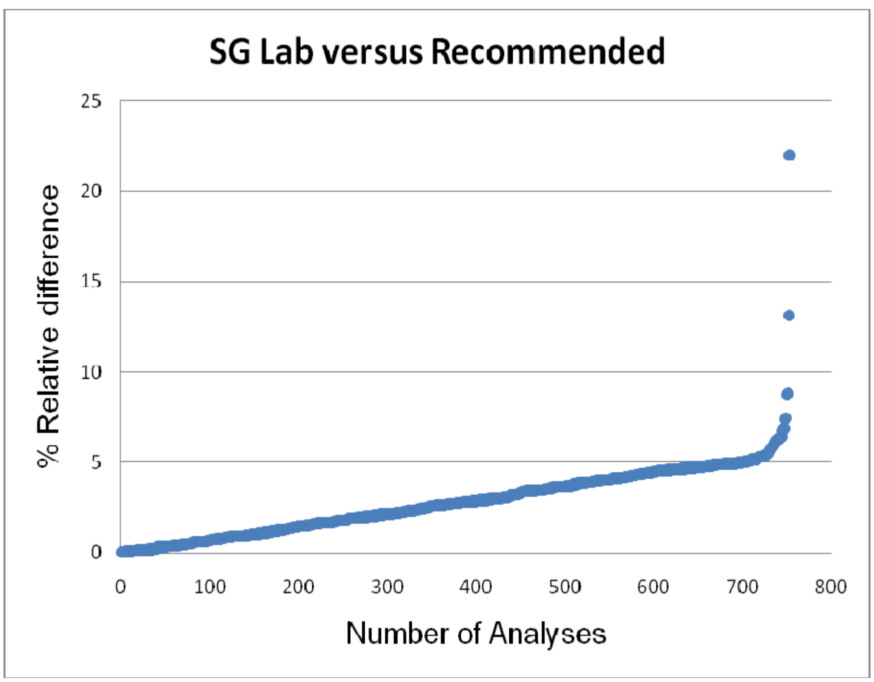


Figure 14 Percent relative difference between the RockLabs recommended value for their products versus the Sleeping Giant laboratory results during the operating period of 2009.

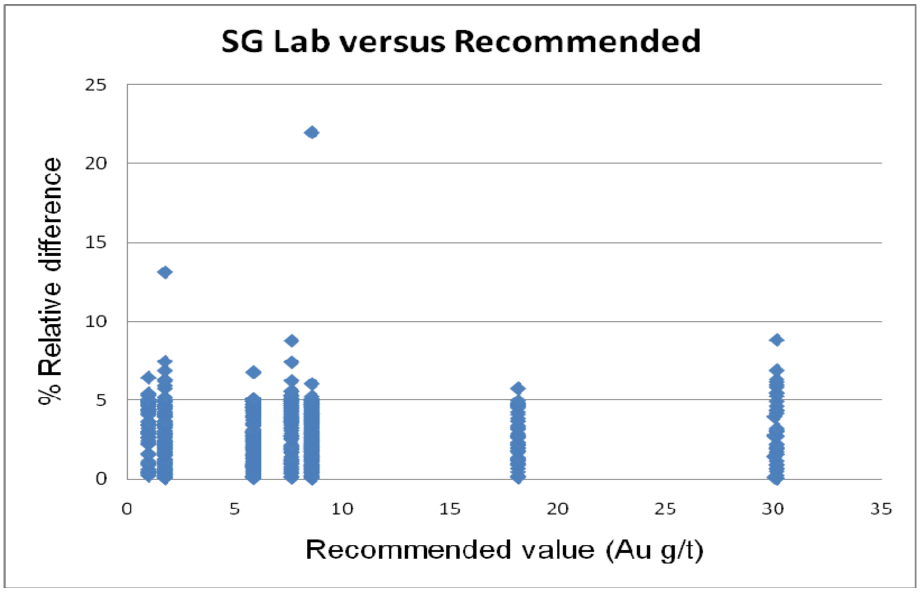


Figure 15 Percent relative difference between the RockLabs recommended value for their products versus the Sleeping Giant laboratory results during the operating period of 2009 presented as the relative difference versus the recommended value for the product. There is no apparent trend of relative precision versus grade.

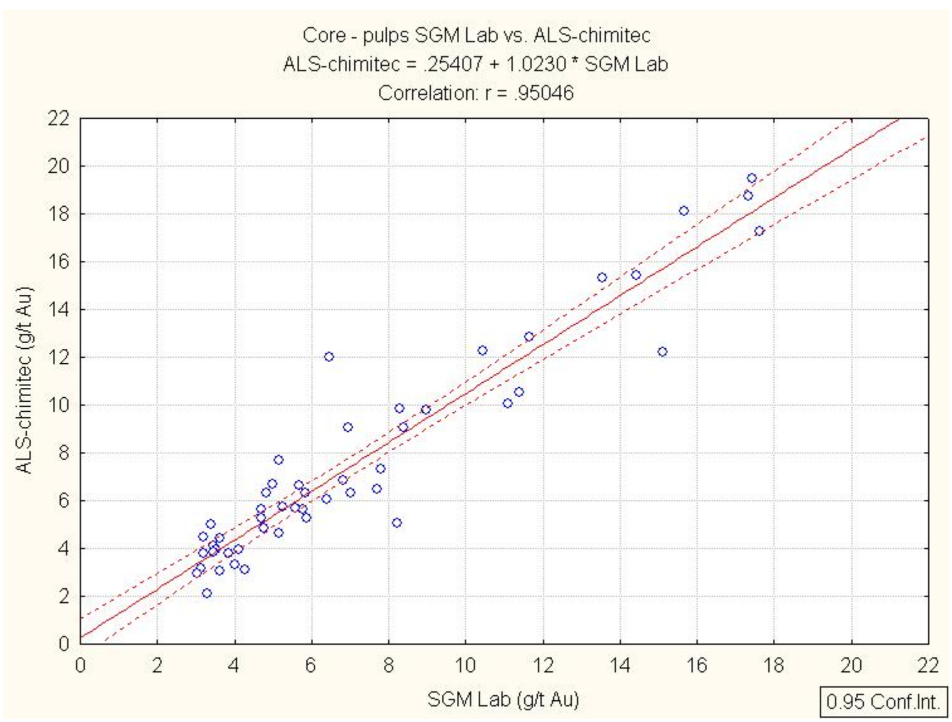


Figure 16 Check analyses by ALS-Chemex of Sleeping Giant laboratory results.

Core Handling

Core handling, logging and sampling are generally to industry standards. Core logging is carried out in a dedicated facility with logging tables, water for wetting the core and adequate lighting. Core is delivered from the underground drills in closed boxes transported on pallets and opened at the core logging facility. Logging is carried out using a computerised system which captures data directly (no transcription). Core boxes are measured and marked with embossed aluminum tags.

Samples for analysis are marked directly on the core with a wax marker and a sample tag placed at the beginning of each sample interval. No sample tag is fixed by a staple to the core box, so no physical record is available of where samples start. Core is split in a separate room in the core logging facility, material placed directly from the tray in the core splitter into a sample bag with the sample tag the geologist placed at the beginning of the interval.

Opinion on the Sleeping Giant Sampling and Laboratory Protocols and Results

After a review of methods and internal checks and series of check analyses in an external laboratory, it is the opinion of the author, Vincent Jourdain, Eng. PhD that core handling, sampling, sample security and analysis at the Sleeping Giant Mine meet current industry standards and are adequate to support estimates of Mineral Resources and Mineral Reserves.

DATA VERIFICATION (ITEM 16)

The technical information which forms the basis of this report was acquired by personnel of the Sleeping Giant mine in their capacity as employees at the mine. There has been continuity in personnel and in accumulated knowledge of the mine through changes in ownership which has benefited the current study. Since most of the new resources are extensions of existing veins and existing stopes, the geometries of the mineralised zones are well-constrained and detailed verification of such data has been minimal.

Date verification is routine in core logging and analytical operations, using the computer program GeoticLog v5.2.20 with checks against inconsistent sample intervals and sample numbers.

ADJACENT PROPERTIES (ITEM 17)

This report is limited to the Sleeping Giant mine and no relationship with adjacent properties is considered herein.

MINERAL PROCESSING AND METALLURGICAL TESTING (ITEM 18)

In 1993, the Sleeping Giant's milling facility was restarted using the Merill-Crowe process. The recovery rate slightly increased in the following years, while costs decreased. In 1998, the material used for the Merill-Crowe process was so deteriorated that the milling process was questioned. Once a study was completed, it was decided to modify the milling facility in order to use the CIL process (carbon in leach). This process allowed, in the first months of its use, a recovery increase with the reduction of the liquid tailings and in a short period of time reduction global milling and processing costs. With this system, fresh water demand and water quantity which needs to be treated are reduced. Mill costs and recoveries are presented in Table 3.

Table 3: Mill costs and recoveries at the Sleeping Giant Mine from 1995 to 2007 by calendar year. Costs are in $CDN

Year	Recovery	Cost/tonne	Cost/ounce
1995	96,4%	25,53	66,18
1996	96,4%	25,18	65,12
1997	96,6%	22,80	68,12
1998	96,4%	18,85	50,65
1999	94,4%	19,44	54,37
2000	98,4%	17,15	48,67
2001	96,7%	18,65	62,51
2002	97,0%	19,60	60,12
2003	97,1%	21,35	56,58
2004	97,1%	19,82	57,05
2005	96,9%	21,76	65,73
2006	97,2%	22,15	64,37
2007	97,4%	19,06	48,61

The Sleeping Giant mill has a nameplate capacity of 900 tonnes per day and has typically operated at approximately 500 to 600 tonnes per day. It has operated for periods of some months at 800 tonnes per day

MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES (ITEM 19)

Mineral Resources

Mineral resources are volumes of rock of economic potential, defined by geological or minimum-mining-width parameters, to which an estimated grade is attached. Mineral resources as defined by current CIM criteria are assigned to one of three classes, Measured, Indicated, or Inferred. The level of geological and engineering information combined with observations and assumptions of geometrical continuity serve to assign the class to each rock volume.

In the present study, Measured Resources are defined as those where an underground opening in the mine provides access and sampling to the volume under question, in addition to drill information. Indicated Resources are those defined by drilling. The distances over which drill hole data have been projected, combined with knowledge of the mine and its mineralised zones, allow the classification of these volumes of rock as Indicated Resources. Inferred Resources considered in this report are defined by drilling where drill hole data is projected farther than for Indicated Resources.

Mineral Resource – Definitions (from CIM)

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

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

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and *economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions, might become economically extractable. The assumptions must be presented explicitly in Reports.*

Inferred Mineral Resource

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

Due to the uncertainty which may attach to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic 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.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration 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.

Mineralization of other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variations from this estimated would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Mineral Resource Calculation Methodology

Mineral resources were calculated using the polygon method on inclined longitudinal sections. Orthogonal thickness of drill hole intersections was calculated using the computer program Promine and a plane which approximates the geometry of the zone being estimated. From the calculated orthogonal intersection, diluted grades were calculated. This methodology has been used historically in the Sleeping Giant Mine and has been shown to yield reliable results through conciliation of estimates with production for stopes throughout the mine operations. Geological interpretations were based on core logging using the same criteria and know-how that allowed the mine to operate successfully for some 20 years.

Capping of Analyses

The philosophy of capping (or cutting) analyses to some maximum upper value comes from historical observations in mining operations. It has been observed that unusually elevated analyses for gold (or other elements present in trace or in minor amounts) in some situations cannot be reliably repeated and are not reflected in mined grades from the sampled volume of rock. These analyses are considered anomalous outliers in the data set for the situation under study.

Anomalous outliers in a population of analyses can result from inadequate sampling, from the effects of sampling statistics, or from lack of accuracy in laboratory measurements – often the exact cause cannot be determined. Without further consideration of the cause of outliers, the remedy is to reduce the analysis values, for computational purposes, to a value which is considered likely to represent the rock unit or volume in question. In many cases a convenient number is chosen (*e.g.* 20 g/t), but more sophisticated methods can be applied, such as the mean plus 2 standard deviations for the geological unit sampled, or a limit defined by a break-in-slope on probability diagrams.

Because estimates of mineral resources are expected to be conservative, a parallel system for increasing the grade of anomalously low analyses is not used.

Capping analyses in mineral deposit resource evaluation is an important subject. First, the precision of the estimate is affected by outlying values, and second the actual estimated value for the overall deposit can be unduly inflated by a relatively small number of high-grade analytical results.

Some estimation methods, through their mathematical approach, naturally reduce the effect of isolated anomalously elevated values (*e.g*. kriging).

While there seems to be no hard-and-fast rule for capping analyses, the decision to apply an upper limit is generally based on two types of considerations. First is the question of whether the analysis in question is a part of a continuous population or an anomalous outlier which does not accurately reflect an underlying population. Second is the geographic distribution of the analysis in question – is it isolated or is it part of a higher-grade zone within a deposit.

The most common method of justifying a decision to cap analyses in a deposit, and a technique to establish the capping value, is based on a cumulative frequency diagram of analyses, either as raw data or as composites at a scale appropriate to the zones under study. Various portions of the deposit are represented by sub-populations on such diagrams – low-grade materials surrounding the mineralized zones on the one hand, and the mineralised zones themselves on the other. These populations are typically separated by changes in slope on a cumulative frequency diagram. A change-in-slope at the upper extremity of the mineralised population on this type of diagram is generally taken to indicate the presence of anomalous samples where capping may be required. Other, similar, approaches include using a diagram of the cumulative coefficient of variation or a diagram of the cumulative mean. In all cases, due consideration of the natural zones within a mineral deposit must be integrated into the analysis.

Grade capping at the Sleeping Giant Mine was carried out on a vein-by-vein basis using factors developed by the mine operations and shown to produce useful grade estimates. In terms of a statistical approach, values are cut at 85 to 90 % of the cumulative frequency population.

Capping values used in the mine planning operations and adopted here are reported in Table 4.

Table 4: Capping values by Zone, Sleeping Giant Mine

Zone	Capping Value	
	Drilling	Channel samples
3	60	180
8	70 to 90	70 to 100
20		120
30	250	250
50	100	55
18	60	250

Using the parameters described above, Mineral Resources have been calculated for the Sleeping Giant mine and are disclosed in Table 5. Details of calculations are presented in Appendix 2 with illustrations of selected longitudinal sections.

Statement of Mineral Resources

Table 5 Detailed Mineral Resource Statement for the Sleeping Giant Mine

Measured Resources Dec 31st 2009

Stope	Tonnes	Grade (g/t Au)	Ounces
85-50-025 Inf.	3100	5.9	588
72-7-600	3100	10.5	1047
72-7-625	7900	7.9	2007
72-7-630	2100	9.1	614
66-2-3660	4300	8.3	1147
54-8-370	3100	8.6	857
54-8-280	2400	11.8	911
85-8-350	3900	8.2	1028
85-8-350 H	14200	10.7	4885
85-8-025	3400	9.4	1028
85-8-050	3200	11.4	1173
85-8-100	3800	6.7	819
78-50-000	5600	12.4	2233
97-8-350	8000	8.5	2186
85-30-2930	6900	8.5	1886
85-30-2930S	15800	7.1	3607
Total	90800	8.9	26014

Indicated Resources Dec 31st 2009

Stope	Tonnes	Grade (g/t Au)	Ounces
78-50-100	3900	12.5	1567
54-8-370H	1900	13.2	806
78-50-075	1000	6.9	222
85-50-SH-075	9100	10.7	3131
78-8-400	8300	10.0	2669
97-8-100	1900	15.1	922
72-8-325	3300	21.1	2239
97-9-3480	2500	7.7	619
97-9-100-3550	4700	8.0	1209
97-30-2970	20700	6.0	3993
85-50-025 Sup	9300	9.7	2900
78-30-2930S	6100	7.0	1373
66-7-628	11200	14.0	5041

54-8-300	7100	9.4	2146
97-30-2950	5100	8.5	1394
85-50-50	8300	6.7	1779
91-09-3520	7200	13.4	3102
Zone 4	7000	6.7	1503
Zone 5	7800	6.5	1641
30 S3	8900	13.0	3720
78-H	54500	8.1	14193
I 2-3	8000	13.5	3472
16 S2	8000	7.3	1878
16 S4	13600	7.0	3061
16 S1	14300	6.6	3034
20 S1	8500	7.2	1968
15 S1	14100	10.5	4760
15 S2	19100	6.0	3684
15 S3	23600	8.4	6374
30W over 975	10100	8.7	2825
Totals	309100	8.8	87223
Measured and Indicated	399900	8.8	113237

Inferred Resources Dec 31, 2009

Zone	Tonnes	Grade (g/t Au)	Ounces
785N	100300	10.0	32247
8N-18 S1	9200	13.1	3875
8N-18 S2	14300	23.5	10784
30 W deep	119700	14.0	53878
Totals	243500	12.9	100784

Notes

The Qualified Person for the Mineral Resource estimates as defined by Regulation 43-101 is Vincent Jourdain, ing., Ph.D.

The effective date of the estimate is December 31st , 2009.

Grade capping was carried out using the lowermost historical values of 60 g/t for all the zones

Resources were evaluated from drill hole results using the polygonal method on inclined longitudinal sections.

The maximum distance was fixed at 30 metres for indicated resources and 60 metres for the inferred resources

A specific gravity of 2.85 t/m3 was used

For Shrinkage stopes the intersections are internally diluted (at zero grade) to minimum true thickness of 1.6 metres

For Long Hole or Room and Pillar stopes the intersections are internally diluted (at zero grade) to minimum true thickness of 1.8 metres

An external dilution of 15% (at zero grade) and a mining recovery of 95% are applied to the Shrinkage stopes.

An external dilution of 25% (at zero grade) and a mining recovery of 95% are applied to the Long Hole stopes.

An external dilution of 15% (at zero grade) and a mining recovery of 85% are applied to the Room and Pillar stopes.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability

Mineral Reserves

Mineral Reserve- Definitions (from CIM)

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

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

Mineral Reserves are those parts of the Mineral Resources which, after the application of all mining factors result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking into account of all relevant processing, metallurgical, economic, marketing, legal, environment, socio-economic and government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Probable Mineral Reserve

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

Proven Mineral Reserve

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

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

Factors Used in Estimation of Mineral Reserves

The conversion of Mineral Resources to Mineral Reserves has been based on a stope-by-stope analysis of cost and revenue. The convention was adopted that Measured Resources, where they are demonstrated to be economic with the data and assumptions of this report, are converted to Proven Reserves, and Indicated Resources to Probable Reserves.

All information including grades, tonnages and costs which are used in an ongoing basis in mine operations has been used in assigning the resource or reserve status of any given volume of rock. Most of this information was originally provided to Cadiscor by IAMGOLD, and came from actual mine operational performances. This information has been updated based on recent operating experience.

Cadiscor has evaluated each stope to determine the most appropriate mining method within the economic and technological framework of the mine. The development work required to access each stope has been evaluated from plans and sections. The methodology to establish the economic profile of each stope is as follows.

Development requirements for each stope have been estimated from plans and sections, and are based on using existing mine infrastructure to the fullest.. Development has been considered in three categories, drifts, sub-levels and raises. An extra 20 % has been added to the estimated costs of sublevels for longhole stopes to account for the space requirements of equipment for this mining method.

The estimated costs for each stope include mining, mucking, development, services and others. All costs are estimated in current Canadian dollars.

Revenues:

Estimated ounces produced from each proposed stope include recovery as a function of mining method, as well as dilution. Mill recovery of 97 % has been used in calculating a net value for each stope.

To evaluate the economic profile of each stope, the cost-revenue balance was calculated. Thus a potential net return for each stope was available to justify its inclusion in the reserve base.

Alternate mining methods were evaluated for a number of proposed stopes to assure the most efficient methods of extraction possible. Only the method retained is presented in the appendices.

A number of factors enter into the estimation of Mineral Reserves. These are listed here:

Mining Width Based on the mining history, the minimum width in reserve calculations is related of the dip of the lens in the estimated area. Each drill intersection is recalculated with a true thickness which varies according to the presumed dip, between 1.6 meter (dip over 50º) and 1.8 meter (dip less than 50º).

Based on the details of each proposed stope, a mining method has been selected. Criteria used to select a mining method include the thickness and dip of the zone to be mined, and any supplementary information such a faults or offsets. Minimum thickness mined for each mining method is presented in Table 6.

Table 6: Minimum thickness mined as a function of mining method

Method	Thickness
Shrinkage stoping	1.6m
Longhole	1.8m
Room and pillar	1.8m

Mining Dilution In addition to the planned stopes, some extra material is typically mined. A nominal dilution of 15 % at zero grade is applied to all shrinkage stopes. Some stopes might have a higher dilution rate depending on ground conditions. In some cases, the long-hole method is used to recover pillars and waste. Dilution for this technique is between 25 and 50% or even more according to ground conditions. These dilution factors are based on historical data from the Sleeping Giant mine. Dilution for room-and-pillar mining is 25 %.

Mining Recovery estimated mining recovery varies between 75% and 100% depending on the chosen mining method. With shrinkage stope mining, recovery is 95 to 100%, while for the room-and-pillar method it is 85 % and for long-hole 95 %. In the case of these methods, the recovery rate might be higher if some pillars are recovered at the end of the project. At that time, ground conditions will limit the recovery of pillars. Reserves which are shown in this report include pillars which are planned to be recovered.

Based on the mining method selected, rates of recovery (the percentage of the mineralised volume actually mined) and dilution (excess material mined) are applied to each stope through the mine (Table 7).

Table 7: Mining recovery and dilution

Method	Mining recovery	Dilution
Shrinkage stoping	95%	15%
Longhole	95%	25%
Room and pillar	85%	15%

The calculated tonnage of each stope (including recovery and dilution) is used to estimate the cost for the stope. Costs including mining and development are based on actual average costs over the last five years of operations at the mine.

Rock Density ore density at the mine was verified by three samples taken every month in 2001, with further measurements in 2002. Results varied between 2.8 g/cm^3 and 2.9 g/cm^3, with an average of 2.86 g/cm^3. A density of 2.85 g/cm^3 was used from December 2002 onward, both for reserves estimation and for engineering. Historically, this density has allowed acceptable reconciliation between planned and produced tonnes.

Mill Recovery The historical recovery of 97 % has been retained

Exchange Rate 1.07 $CDN = 1.00 $US. This factor has been used to calculate gold prices in Canadian dollars for the purposes of this report.

Price of Gold A gold price of 850 per troy ounce (expressed in $US) has been used to estimate reserves.

Fixed Costs Historical mine costs have been provided by the Sleeping Giant mine and retained for this report (Table 8).

Operating Costs Historical mine costs have been provided by the Sleeping Giant mine and retained for this report (Table 8).

Total costs per tonne on a basis of mining method have been estimated and are presented in Table 9.

Table 8: Fixed and Variable Costs, Sleeping Giant Mine

Historical costs of mine services used as a basis for calculations of overall costs.

Year	2003	2004	2005	2006	2007
Production (oz)	33,304 oz	33,509 oz	39,967 oz	45,716 oz	66,826 oz
Tonnes milled	88,248 t	96,475 t	121,249 t	132,965 t	170,392 t
Tonnes hoisted	88,275 t	96,550 t	120,748 t	133,300 t	170,467 t
Grade	12.09 g/t	11.12 g/t	10.63 g/t	11.01 g/t	15.52 g/t
Mill recoveries	97.08%	97.11%	96.88%	97.17%	97.40%
Mine services	19.67 $/t hoisted	18.99 $/t hoisted	20.78 $/t hoisted	21.95 $/t hoisted	17.72 $/t hoisted
Mechanical services	5.53 $/t hoisted	5.61 $/t hoisted	7.78 $/t hoisted	8.62 $/t hoisted	5.50 $/t hoisted
Electrical services	3.34 $/t hoisted	3.31 $/t hoisted	3.74 $/t hoisted	4.33 $/t hoisted	2.42 $/t hoisted
Surface services	8.27 $/t hoisted	8.32 $/t hoisted	9.84 $/t hoisted	9.91 $/t hoisted	8.12 $/t hoisted
Engineering	3.67 $/t hoisted	3.47 $/t hoisted	4.83 $/t hoisted	5.74 $/t hoisted	3.68 $/t hoisted
Geology	2.37 $/t hoisted	2.18 $/t hoisted	2.11 $/t hoisted	2.50 $/t hoisted	2.19 $/t hoisted
Total services	**42.87 $/t hoisted**	**41.89 $/t hoisted**	**49.09 $/t hoisted**	**53.05 $/t hoisted**	**39.63 $/t hoisted**
Environment	2.44 $/t milled	2.06 $/t milled	2.78 $/t milled	2.48 $/t milled	1.28 $/t milled
Milling	18.92 $/t milled	17.76 $/t milled	19.59 $/t milled	19.52 $/t milled	17.78 $/t milled
Environment & Milling	**21.35 $/t milled**	**19.82 $/t milled**	**22.36 $/t milled**	**22.00 $/t milled**	**19.06 $/t milled**
Site administration	**11.74 $/t hoisted**	**11.40 $/t hoisted**	**14.07 $/t hoisted**	**15.40 $/t hoisted**	**11.84 $/t hoisted**

Table 9: Costs per tonne

Type	Cost
Services and others	79 $/t
Room and pillar	84 $/t
Shrinkage stoping	51 $/t
Longhole	35 $/t
Mucking	12 $/t

Services and others include:	Environment and milling:	22.07 $/t
	Administration:	14.20 $/t
	Services:	42.81 $/t
	Definition drilling	13.65 $/t

Development costs Costs of providing mine infrastructure to each stope have been estimated based on historical cost data and per-shift productivity (Table 10).

Table 10: Development costs

Type of excavation	Cost
Drift	1225 $/m
Sub level (shrinkage stoping and room and pillar)	1475 $/m
Sub level (longhole)	1770 $/m
Raise	1925 $/m

Costs were calculated for each stope for access (drifts, raises, crosscuts) as well as mining, mucking and transport. Costs for milling and all other burdens were added for the estimated tonnes in each stope to arrive at a final stope-by-stope decision on probable profitability.

Statement of Mineral Reserves

In addition to the Mineral Resources disclosed above in Table 5, Mineral Reserves have been identified and are disclosed in Table 11. The cost per ounce of gold has been calculated for each stope using the mining method, development and mill recovery data of this report. Thus each stope can be assigned a net value based on gold price. For the purposes of this report, a cost of $CDN 850 per ounce has been taken as the cut-off for Mineral Reserves.

Table 11

	Proven Reserves Dec 31st 2009			Probable Reserves Dec 31st 2009		
Stope	Tonnes	Grade (g/t Au)	Ounces	Tonnes	Grade (g/t Au)	Ounces
85-50-025 Inf.	3100	5.9	588			
72-7-600	3100	10.5	1047			
72-7-625	7900	7.9	2007			
72-7-630	2100	9.1	614			
66-2-3660	4300	8.3	1147			
54-8-370	3100	8.6	857			
54-8-280	2400	11.8	911			
85-8-350	3900	8.2	1028			
85-8-350 H	14200	10.7	4885			
85-8-025	3400	9.4	1028			
85-8-050	3200	11.4	1173			
85-8-100	3800	6.7	819			
78-50-000	5600	12.4	2233			
97-8-350	8000	8.5	2186			
85-30-2930	6900	8.5	1886			
85-30-2930S	15800	7.1	3607			
78-50-100				3900	12.5	1567
54-8-370H				1900	13.2	806
78-50-075				1000	6.9	222
85-50-SH-075				9100	10.7	3131
78-8-400				8300	10.0	2669
97-8-100				1900	15.1	922
72-8-325				3300	21.1	2239
97-9-3480				2500	7.7	619
97-9-100-3550				4700	8.0	1209
97-30-2970				20700	6.0	3993
85-50-025 Sup				9300	9.7	2900
78-30-2930S				6100	7.0	1373
66-7-628				11200	14.0	5041
54-8-300				7100	9.4	2146
97-30-2950				5100	8.5	1394
TOTAL	90,800	8.9	26,014	96,100	9.8	30,223

Notes

The Independent and Qualified Persons for the Mineral Resource estimates as defined by Regulation 43-101 is Vincent Jourdain, ing., Ph.D.

The effective date of the estimate is December 31st , 2009.

Grade capping was carried out on a vein-by-vein basis using the historical values of 60 g/t in zones 2, 7 and 9; 90 g/t in Zone 8 ; 100 g/t in Zone 50
Reserves were evaluated from drill hole results using the polygonal method on inclined longitudinal sections.

A specific gravity of 2.85 t/m^3 was used
For Shrinkage stopes the intersections are internally diluted (at zero grade) to minimum true thickness of 1.6 metre

For Long Hole or Room and Pillar stopes the intersections are internally diluted (at zero grade) to minimum true thickness of 1.8 metre

An external dilution of 15% (at zero grade) and a mining recovery of 95% are applied to the Shrinkage stopes.
An external dilution of 25% (at zero grade) and a mining recovery of 95% are applied to the Long Hole stopes.
An external dilution of 15% (at zero grade) and a mining recovery of 85% are applied to the Room and Pillar stopes.

Table 11: Detailed Mineral Reserves Statement for the Sleeping Giant Mine

Table 12: Summary Statement of Mineral Reserves

*RESERVES**

Proven:	*90,800 tonnes*	*at 8.9 g/t*
Probable:	*96,100 tonnes*	*at 9.8 g/t*

TOTAL: 186,900 tonnes at 9.4 g/t for 56,237 ounces recovered

OTHER RELEVANT DATA AND INFORMATION (ITEM 20)

The exploration potential of the Sleeping Giant mine remains excellent. Exploration drilling at depth has intersected the mineralised zones as deep as 445 m below the current workings with significant gold grades comparable to those in the current levels.

With the current context of increasing gold prices, new interpretation of the existing geological interpretation of the mine and drilling to extend known zones to depth seems justified. Zones recently drilled such as 30W remain open both down and up dip.

INTERPRETATIONS AND CONCLUSIONS (ITEM 21)

Since the report by Genivar (2008), Sleeping Giant has produced an estimated 32,822 tonnes of ore. The reserves (Proven plus Probable) calculated by Genivar were 235,300 tonnes. Thus the difference between the estimated reserves in 2008 and those of December 31, 2009, when production is taken into account, is 15,578 tonnes no longer considered to be reserves, a difference of -6.6%. This change is a consequence of more detailed knowledge of the veins because of extensive drilling. Estimated Mineral Resources have changed from 489,200 tonnes of Measured and Indicated (from which 32,822 tonnes were extracted) to 399,900 tonnes of Measured and Indicated plus 243,500 tonnes of Inferred resources. These changes are also attributable to the increased information available following exploration and development drilling in 2009. The net change in Measured plus Indicated Resources is -12%.

Although the estimated resources at the Sleeping Giant mine have decreased in the Measured and Indicated categories, the increased drilling has improved confidence in the estimates. In addition, substantial new resources in the Inferred category have been identified.

RECOMMENDATIONS (ITEM 22)

Definition drilling of reserves to bring them to the production stage, with production to follow.

Drilling of Indicated Resources to bring them to the stage of Reserves.

Drilling of Inferred resources to bring them to the stage of Measured and Indicated Resources.

Continued exploration and compilation of available information proximal to the mine, with exploration drilling on extensions of known zones both laterally and at depth.

Develop lower zones through shaft sinking and drifting, in conjunction with definition drilling.

ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES (ITEM 25)

Mine Infrastructure

The mine is accessed by a four-compartment production shaft with a total depth of 1053 m. Levels are spaced at 45 m from surface to 235 m, and from there to 975 m are spaced at 60 m. The exploration shaft and various raises allow all portions of the mine to be ventilated with fresh air. An ore pass and a waste pass allow material to be handled and raised to the surface. The deepest working level of the mine is 975 m. The mine is worked using electric locomotives and cars of 3 and 5 tonnes.

Mining Operations

The mine and the mill currently operate with all required permits in place.

Mining Methods

Three methods are in current or recent use to extract ore. The long-hole method was discontinued during 2002 because of the shallow dip in zone 8 as well as the high dilution which was obtained. Since then, the use of this method is limited to the pillar recovery and to complete stopes in which mining with other methods is not appropriate. The type of mining method is determined according to the studied zone's dip:

<u>Slope over 65º</u>: Long-hole and shrinkage stope extraction. When used, the long-hole method consisted in excavating a raise to a maximum length of 65 meters between two sub-levels. Following this step, three levels are excavated, with maximum length 70 meters and they are vertically spaced between 15 to 17 meters, according to a "dice five" pattern. When drilling is completed, blasting of the three benches can be carried out. The shrinkage method is described below.

<u>Slope between 65º and 45º</u>: Shrinkage stope mining with some stopes by long-hole methods.. Shrinkage stope mining consists in excavating a raise to a maximum length of 85 meters between two levels. The length of the stope is usually between 20 and 100 meters, that is to say blocs of 7 000 to 35 000 tonnes. The ore is broken in horizontal slices of approximately 2.6 m thickness, working from the base to the top of the stope. For each slice, 30% of the blasted ore is removed therefore allowing employees to move along the stope on the broken ore. When the ore breaking is completed, the remaining ore can be extracted from the stope. With this method, the recovery rate is between 95% and 100%.

Slope below 45º: Room and pillar extraction. During room and pillar extraction, the ore is blasted in slices, but contrary to shrinkage stope mining methods the broken ore is removed immediately in order to allow workers to circulate in the stope. Therefore, no access raise is required. The size of the rooms and pillars is determined according to the rock mass stability. Usually, rooms are 6,5 meters by 6,5 meters and are separated by pillars which can be recovered in part once the stope is completed. The mining recovery is typically at least 85 % when using this method.

Recoverability

The ongoing mine operations provide recovery data for gold from the mine. The historical recovery within recent operating experience has been 97.2 %. The estimated mineral reserves come from the same areas of the mine as current and recent production, and the same mill recovery has been assumed for the new production.

Markets

Gold is sold in a liquid market which can accommodate the planned production from the mine.

Contracts

Planned production is based on selling gold into the spot market. No hedging is planned.

Environmental Considerations

The mine currently operates with all required government permits in place. Tailings at the close of operations by IAMGOLD remain the responsibility of IAMGOLD. Site decommissioning will be the responsibility of Cadiscor. It is estimated that the break-up value of the mine infrastructure will pay for closure and restoration at the site.

Taxes

The mine is owned and operated by Cadiscor, and with accumulated tax credits due to past losses and development costs to be incurred, it is projected that no taxes will be paid during the operating period of the current report.

Capital and Operating Cost Estimate

The estimates of operating costs have been incorporated in the estimation of Mineral Reserves.

Mine Life

The current Mineral Reserves will support operations for 12 months. The current Mineral Resources, if all were converted to reserves, would support mine operations for an additional 31 months.

Signature Page

UPDATED RESERVES AND RESOURCES ON DECEMBER 31, 2009, THE SLEEPING GIANT MINE, NORTHWESTERN QUEBEC

for

North American Palladium Ltd.

2116 – 130 Adelaide Street West
Toronto, Ontario
Canada, M5H 3P5

(Signed & Sealed) Signed at Val-d'Or, Quebec, March 31, 2010

Vincent Jourdain, Eng. PhD
Cadiscor Resources Ltd.
1495 4th Street
Val-d'Or (Quebec)
J9P 6X1

CERTIFICATE OF QUALIFIED PERSON

Certificate of Qualifications

Vincent Jourdain

I, Vincent Jourdain, Eng. PhD, as author of this report entitled UPDATED RESERVES AND RESOURCES ON DECEMBER 31, 2009, THE SLEEPING GIANT MINE, NORTHWESTERN QUEBEC prepared for Cadiscor Resources Ltd. and dated March 31, 2010 do hereby certify that:

1. I am an Engineer employed by Cadiscor Inc. with offices at 1495 4th Street, Val-d'Or, Quebec, Canada J9P 6X1.

2. I am a graduate of Laval University, Quebec, with a Bachelor in Geological Engineering obtained in 1984. I am a graduate of Université du Québec à Chicoutimi with a Master in Earth Sciences obtained in 1987. I am a graduate of Université du Québec à Montreal with a PhD in Mineral Resources obtained in 1993.

3. I am registered as an Engineer in the Province of Quebec (Membership Number 040485).

4. I have worked in the fields of mineral exploration, mineral deposits evaluation, and structural geology for a total of 26 years since my graduation. My relevant experience for the purpose of this technical report is:

    a. 25 years of active experience in mineral exploration, mineral deposit evaluation, and structural geology throughout Quebec
    b. experience in a wide variety of gold deposit types and geological settings within the Superior, Grenville and Appalachian geological provinces.
    c. continuing education through seminars, short courses and field trips concerning a variety of mineral deposit types and geological environments.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I visited the Sleeping Giant Property on numerous occasions since 2007.

7. I am responsible for all items in this report.

8. I am not independent of the Issuer applying the test set out in Section 1.4 of National Instrument 43-101.

9. I have had prior involvement with the mining property which is the subject of this report.

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

11. To the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 31st day of March, 2010

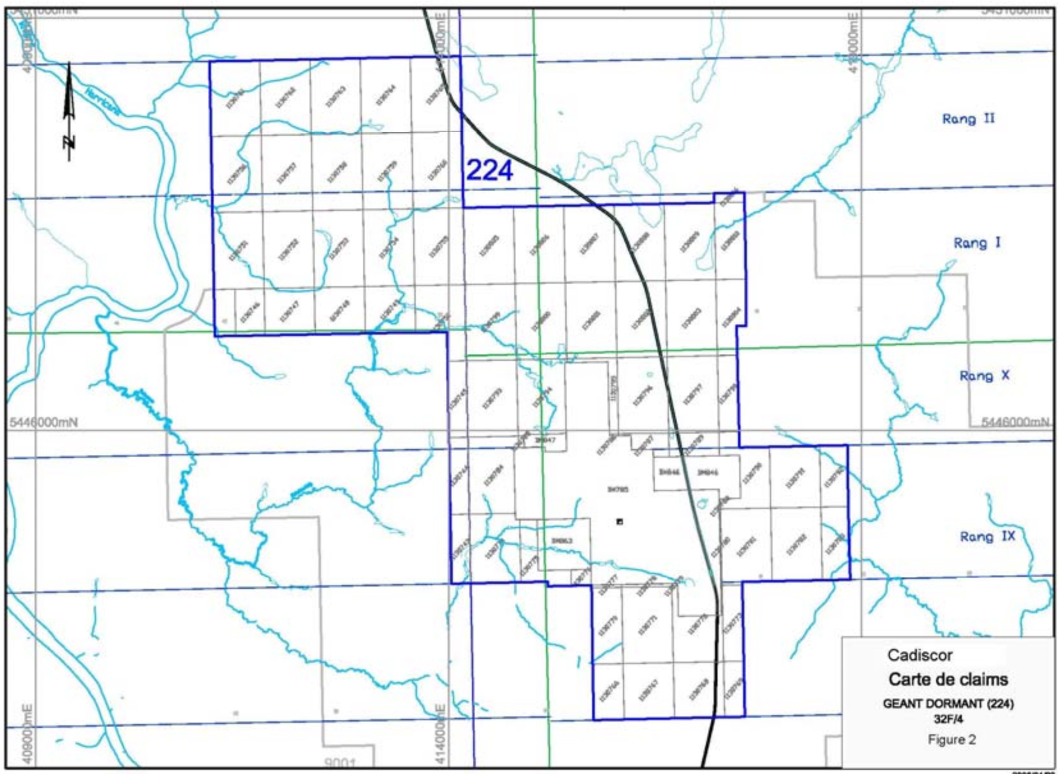
(Signed & Sealed)

Vincent Jourdain, Eng. PhD

Appendices

Appendix 1

Claims and licenses forming the Sleeping Giant property



Appendix 2

Sections of stopes for mine planning and resource estimates and spreadsheet calculations for each stope

This section presents spreadsheets used in the calculation of grades and tonnages for each stope or zone reported here, and a series of typical inclined longitudinal sections for zones and stopes.

16-S1, 785N niv 855

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
48-012	45.2	0.27	960	7.7	486	1104	6.7	7404	226	0.27	210.4
48-783	1.0	0.91	2341	0.6	77	2692	0.5	1325	40	0.91	513.4
54-125	8.2	1.4	2360	7.2	14	2714	6.3	16970	518	1.4	517.5
54-126	29.3	0.99	1670	18.1	62	1921	15.7	30150	921	0.99	366.3
54-196	9.5	1.08	2971	6.4	48	3416	5.6	18986	580	1.08	651.5
54-209	10.0	1.47	1072	9.2	9	1233	8.0	9869	301	1.47	235.2
54-210	17.2	0.82	1690	8.9	94	1944	7.7	14984	458	0.82	370.7
Total diluted recovery			13064	7.6		14272	6.6	99687	3028		

16_S2 785N level 855

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
72-208	35.5	0.38	3075	8.3	327	3536	7.2	25585	781	0.38	674
78-276	8.8	1.90	960	8.8	0	1104	7.7	8447	258	1.90	177
78-277	13.3	0.60	1448	5.0	166	1665	4.4	7249	221	0.60	318
78-286	23.5	0.95	1032	14.0	68	1187	12.1	14413	440	0.95	226
78-436	22.7	0.49	795	7.0	225	914	6.1	5549	169	0.49	174
Total diluted recovery			7310	8.4		7986	7.3	61244	1874		

16_S3 785N level 855

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
78-208	24.2	0.99	2295	15.0	62	2640	13.0	34312	1048	0.99	503.3
78-281	13.2	0.80	866	6.6	100	995	5.7	5717	175	0.8	189.8
78-411	6.4	1.95	9267	6.4	0	10657	5.6	59310	1811	1.95	1670.2
Total diluted recovery			12428	8.0		13577	7.0	99340	3055		

20_S1

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-029	45.1	0.36	819	10.2	344	941	8.8	8305	254	0.36	179.5
78-074	20.4	1.20	14	15.3	34	16	13.3	206	6	1.2	3.0
78-076	8.9	0.75	66	4.2	113	76	3.6	274	8	0.75	14.4
78-077	14.0	0.71	187	6.2	127	215	5.4	1153	35	0.71	41.0
78-078	14.0	0.32	704	2.8	404	809	2.4	1955	60	0.32	154.3
78-079	3.6	0.71	49	1.6	127	56	1.4	77	2	0.71	10.6
78-106	12.2	0.92	33	7.0	75	38	6.1	232	7	0.92	7.3
78-107	9.2	0.84	21	4.9	89	25	4.2	103	3	0.84	4.7
78-108	13.2	0.77	72	6.3	109	82	5.5	451	14	0.77	15.7
78-109	30.6	0.38	21	7.2	323	24	6.3	154	5	0.38	4.7
78-110	0.2	0.31	21	0.0	410	24	0.0	1	0	0.31	4.6
78-120	6.7	0.97	24	4.1	64	28	3.6	98	3	0.97	5.3
78-121	10.2	0.93	31	5.9	72	35	5.1	180	6	0.93	6.7
78-127	60.0	0.44	37	16.6	261	43	14.5	614	19	0.44	8.1
78-128	40.7	0.42	23	10.7	281	26	9.3	245	7	0.42	5.0
78-143	41.9	0.42	1776	11.1	279	2042	9.6	19646	600	0.42	389.4
78-144	27.3	0.65	1797	11.0	148	2067	9.6	19822	605	0.65	394.1
78-221	19.9	0.75	124	9.3	115	143	8.1	1152	35	0.75	27.2
78-224	12.7	0.79	404	6.2	104	465	5.4	2521	77	0.79	88.6

78-238	10.3	0.69	552	4.4	132	634	3.9	2449	75	0.69	121.0
78-242	24.9	1.08	41	16.8	48	47	14.6	687	21	1.08	9.0
78-244	12.2	1.12	140	8.5	43	161	7.4	1190	36	1.12	30.7
78-346	13.9	0.46	33	4.0	247	38	3.5	131	4	0.46	7.2
78-430	7.4	0.92	130	4.3	73	149	3.7	555	17	0.92	28.5
78-431	4.5	0.35	533	1.0	359	613	0.9	522	16	0.35	116.8
78-443	40.5	0.6	108	15.1	168	124	13.2	1634	50	0.6	23.7
78-445	17.2	0.96	23	10.3	67	26	9.0	233	7	0.96	4.94
Total diluted recovery			7778	8.3		8497	7.2	64588	1967		

30_S3

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-140	37.5	0.64	491	15	150	565	13.1	7365	225	0.64	107.7
66-143	2.9	1.67	516	2.88	0	593	2.5	1486	45	1.67	108.3
66-145	60.0	0.42	677	15.74	281	779	13.7	10654	325	0.42	148.5
66-898	25.1	1.65	690	25.1	0	793	21.8	17309	529	1.65	146.6
66-899	16.9	2.18	1142	16.9	0	1313	14.7	19296	589	2.18	183.4
66-900	47.9	1.66	424	47.9	0	487	41.7	20296	620	1.66	89.6
66-901	13.2	2.19	699	13.2	0	804	11.5	9223	282	2.19	112.1
66-902	10.5	1.35	360	8.88	18	414	7.7	3198	98	1.35	79.0
66-903	11.2	1.39	357	9.73	15	411	8.5	3476	106	1.39	78.3
66-904	8.4	3.38	1019	8.4	0	1172	7.3	8560	261	3.38	105.9
66-905	8.6	1.47	465	7.92	9	534	6.9	3682	112	1.47	101.9
66-907	13.1	2.19	1325	13.1	0	1524	11.4	17355	530	2.19	212.5
Total diluted recovery			8163	14.9		8918	13.0	121900	3727		

30W

Zone: Chantier: 30W 975

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Teneur	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
78-010	31.9	0.70	5799	12.3	159	6668	10.7	71533	1955	0.7	1130.3
97-110	12.1	1.03	4553	6.9	75	5236	6.0	31450	859	1.03	887.5
Total diluted recovery			10352	9.9		10119	8.7	102984	2830		

Zone: Chantier: 30Wdeep

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
78-524	17.5	1.68	34456	16.4	7	39624	14.3	565647	15458	1.68	6716.5
85-146-09	13.9	1.12	14207	8.7	60	16338	7.5	123094	3364	1.12	2769.3
97-027	22.4	1.30	26010	16.2	38	29912	14.1	422100	11535	1.3	5070.2
97-108	20.6	2.40	29735	20.6	0	34195	17.9	611649	16715	2.4	4352.3
97-109	30.5	0.82	18022	13.9	119	20726	12.1	250922	6857	0.82	3513.1
Total diluted recovery			122430	16.1		119675	14.0	1973413	53864		

Zone: 78 Chantier: 78-30-2930S

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
72-065	13.7	1.03	1735	7.8	75	1995	6.8	13575	436	1.03	338.2
72-069	30.0	0.32	1872	5.3	468	2153	4.6	9886	318	0.32	364.9
78-104	18.9	0.84	1670	8.8	114	1921	7.7	14726	473	0.84	325.6
78-142	13.1	1.23	403	8.9	47	463	7.8	3594	116	1.23	78.5
78-179	19.0	1.42	522	15.0	27	600	13.0	7804	251	1.42	101.8
Total diluted recovery			6202	8.0		6062	7.0	49585	1594		

85-50-50

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-844	21.6	0.58	2300	7.0	208	2645	6.1	16140	441	0.58	448.4
78-502	16.7	0.86	2732	7.9	110	3142	6.9	21674	592	0.86	532.5
78-503	34.2	0.42	2190	7.9	333	2519	6.9	17296	473	0.42	426.9
85-100	11.3	1.29	180	8.1	39	207	7.1	1460	40	1.29	35.0
85-100	11.3	1.29	1120	8.1	39	1288	7.1	9097	249	1.29	218.3
Total diluted recovery			8522	7.7		8330	6.7	65667	1785		

Zone: Chantier: 85-50-025inf

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
72-132	29.9	0.47	514	7.8	285	591	6.8	3986	109	0.47	100.1
72-280	14.3	1.08	393	8.6	66	452	7.5	3384	92	1.08	76.6
72-293	8.5	1.37	704	6.5	32	809	5.6	4546	124	1.37	137.2
72-302	2.3	0.44	9	0.6	305	10	0.5	5	0	0.44	1.7
78-549A	9.2	1.87	115	9.2	0	132	8.0	1056	29	1.87	21.6
78-550	6.2	2.84	960	6.2	0	1105	5.4	5955	163	2.84	118.5
85-111	10.1	0.91	466	5.1	97	536	4.5	2396	65	0.91	90.9
Total diluted recovery			3160	6.7		3089	5.9	21326	583		

Zone: Chantier: 91-09-3520

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
72-156	25.4	1.57	1698	22.1	15	1952	19.2	37510	1025	1.57	330.9
72-217	16.0	0.57	2264	5.1	215	2604	4.4	11497	314	0.57	441.4
72-223	9.3	1.9	1989	9.3	0	2287	8.1	18516	506	1.9	366.7
85-122	90.0	0.65	1421	32.6	176	1634	28.3	46318	1266	0.65	277.0
Total diluted recovery			7372	15.4		7206	13.4	113841	3104		

Zone: Chantier: LT85-8-025

Minimum width	1.8
External Dilution	25
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
72-228	51.8	0.72	950	20.7	150	1187	16.6	19692	601	0.72	185.2
72-237	0.8	0.46	375.7	0.2	293	470	0.2	77	2	0.46	73.2
78-564	9.0	0.21	2.5	1.1	750	3	0.9	3	0	0.21	0.5
85-074	6.9	0.69	0	2.6	162	0	2.1	0	0	0.69	0.0
85-075	0.3	0.77	1.6	0.1	133	2	0.1	0	0	0.77	0.3
85-077	38.5	0.82	528.9	17.5	120	661	14.0	9239	282	0.82	103.1
85-078	6.4	0.53	355.5	1.9	243	444	1.5	664	20	0.53	69.3
85-081	6.5	1.63	641.4	5.9	10	802	4.7	3774	115	1.63	125.0
Total diluted recovery			2856	11.7		3391	9.4	33448	1022		

Zone: Chantier: 85-8-050

Minimum width	1.8
External Dilution	25
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-352	17.9	1.81	903	17.9	0	1128	14.3	16158	494	1.81	174.6
78-262	17.4	1.23	256	11.9	46	320	9.5	3040	93	1.23	49.9
78-377A	28.2	0.62	269	9.6	193	337	7.7	2595	79	0.62	52.5
78-529	2.8	0.99	2	1.5	83	2	1.2	3	0	0.99	0.4
78-534	24.4	0.96	1255	13.1	87	1569	10.4	16384	500	0.96	244.7
Total diluted recovery			2685	14.2		3188	11.4	38180	1166		

LT97-08-350

Zone: Chantier:

Minimum width	1.8
External Dilution	25
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
78-313	35.2	0.41	390	8.0	340	488	6.4	3124	95	0.41	76.0
97-063	55.6	0.89	670.2	27.4	103	838	21.9	18343	560	0.89	130.6
97-077	27.5	0.93	821.5	14.2	94	1027	11.4	11666	356	0.93	160.1
97-122	14.6	1.27	1177.3	10.3	42	1472	8.2	12082	369	1.27	229.5
97-123	9.1	2.07	1930.1	9.1	0	2413	7.3	17564	536	2.07	327.4
97-125	7.2	1.37	1068	5.5	31	1335	4.4	5862	179	1.37	208.2
97-127	9.6	0.97	721.4	5.2	85	902	4.2	3741	114	0.97	140.6
Total diluted recovery			6778	10.7		8049	8.5	72381	2211		

LT97-30-2950

Minimum width	1.8
External Dilution	25
Mining recovery	95

Drill Hole	Grade	Length	Internal Dilution Tonnes	Grade	External Dilution Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
78-054	18.6	1.08	1125.3	11.2	66	1406.6	8.96	12602.1	385	1.08	219.36
78-220	59.9	0.48	348.4	15.87	277	435.5	12.7	5529.3	169	0.48	67.92
78-226	13.6	1.14	1639.4	8.63	58	2049.2	6.91	14150.5	432	1.14	319.56
78-242	48.2	0.48	785.3	12.85	275	981.7	10.28	10092	308	0.48	153.08
91-082	34.8	0.39	370.6	7.48	365	463.2	5.99	2773.7	85	0.39	72.24
Total diluted recovery			4269	10.6		5069	8.5	45148	1379		

CM85-08-350

Minimum width	1.6
External Dilution	15
Mining recovery	95

Drill Hole	Grade	Length	Internal Dilution Tonnes	Grade	External Dilution Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-164	41.9	0.50	1274	13.1	220	1465	11.4	16664	509	0.5	279.3
78-202	3.5	0.80	17	1.8	101	20	1.5	30	1	0.8	3.7
78-205	18.0	0.58	381	6.6	174	438	5.7	2505	77	0.58	83.5
78-548	6.1	0.71	0	2.7	126	0	2.4	0	0	0.71	0.0
78-571	10.5	0.63	802	4.1	153	923	3.6	3324	102	0.63	176.0
78-572	11.4	1.43	1057	10.2	12	1215	8.8	10732	328	1.43	231.7
78-573	6.4	0.58	0	2.3	174	0	2.0	0	0	0.58	0.0
Total diluted recovery			3530	9.4		3857	8.2	33255	1016		

CPL54-08-280

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
48-310	23.2	1.00	527	13.0	79	606	11.3	6826	187	1	102.7
48-346	88.1	0.39	95	19.0	364	110	16.5	1812	50	0.39	18.6
48-354	11.7	1.10	668	7.2	63	768	6.2	4787	131	1.1	130.2
48-798	23.9	2.69	593	23.9	0	682	20.8	14175	387	2.69	77.3
48-799	71.9	0.71	111	28.3	154	127	24.6	3129	86	0.71	21.6
48-801	12.7	2.04	128	12.7	0	147	11.0	1625	44	2.04	22.1
48-802	4.8	1.34	358	3.6	34	411	3.1	1279	35	1.34	69.7
Total diluted recovery			2480	13.6		2852	11.8	33634	919		

CPL54-08-370

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
48-371	2.7	1.70	498	2.6	6	572	2.2	1270	35	1.7	97.0
48-374	15.5	1.64	655	14.1	10	753	12.3	9236	252	1.64	127.7
48-490	8.8	1.57	584	7.7	15	671	6.7	4472	122	1.57	113.8
48-792	8.9	1.67	1094	8.3	8	1258	7.2	9042	247	1.67	213.2
48-794	28.8	1.29	387	20.6	40	445	17.9	7963	218	1.29	75.5
Total diluted recovery			3218	9.9		3144	8.6	31981	874		

CPL 72-7-600

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-317	5.5	1.85	774	5.5	0	891	4.8	4236	116	1.85	146.8
66-323	2.7	1.55	9	2.3	16	10	2.0	21	1	1.55	1.7
66-324	18.9	2.09	178	18.9	0	205	16.4	3363	92	2.09	29.9
66-352	5.2	0.46	0	1.3	294	0	1.2	0	0	0.46	0.0
66-354	4.6	1.73	0	4.4	4	0	3.8	1	0	1.73	0.0
66-356	20.1	0.81	0	9.1	121	0	7.9	0	0	0.81	0.0
66-379	0.5	0.60	0	0.2	200	0	0.1	0	0	0.6	0.0
66-380	6.9	1.89	501	6.9	0	576	6.0	3436	94	1.89	93.0
66-381	17.0	1.21	518	11.4	49	596	9.9	5918	162	1.21	101.0
66-408	23.8	2.27	110	23.8	0	127	20.7	2623	72	2.27	17.0
66-427	10.0	0.54	0	3.0	234	0	2.6	0	0	0.54	0.0
66.-469	18.5	0.91	140	9.3	98	160	8.1	1301	36	0.91	27.2
66-574	34.4	1.71	302	2.0	32.59	348	28.3	9849	269	1.71	58.9
72-202	60.0	0.35	621	11.5	421	714	10.0	7154	195	0.35	121.0
72-321	2.6	0.77	0	1.1	132	0	1.0	0	0	0.77	0
Total diluted recovery			3153	12.0		3082	10.5	37901	1036		

CPL- 72-7-625

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-355	9.1	2.14	773	9.1	0	888	7.9	6999	191	2.14	126.6
66-356	20.1	0.81	181	9.1	121	208	7.9	1646	45	0.81	35.3
66-356	20.1	0.81	80	9.1	121	92	7.9	726	20	0.81	15.6
66-370	6.6	1.37	1152	5.0	32	1325	4.4	5768	158	1.37	224.5
66-398	60.0	0.45	1079	15.2	296	1241	13.2	16361	447	0.45	210.3
66-399	32.8	1.09	1195	19.9	65	1375	17.3	23790	650	1.09	233.0
66-412	24.7	0.43	1426	5.8	323	1640	5.1	8333	228	0.43	278.0
66-414	2.5	1.19	150	1.6	51	172	1.4	245	7	1.19	29.2
66-467	3.7	0.59	717	1.2	203	824	1.1	875	24	0.59	139.7
66-962	8.0	1.55	278	6.9	16	320	6.0	1920	52	1.55	54.2
66-962	8.0	1.55	835	6.9	16	961	6.0	5771	158	1.55	162.8
72-320	7.1	1.68	235	6.6	7	270	5.8	1558	43	1.68	45.8
Total diluted recovery			8100	9.1		7918	7.9	73991	2022		

CPL-72-7-630

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-320	2.5	1.71	388	2.3	5	447	2.0	903	25	1.71	75.7
66-325	21.9	0.71	423	8.6	154	487	7.5	3650	100	0.71	82.5
66-326	5.7	1.13	0	3.6	60	0	3.1	1	0	1.13	0.1
66-351	20.2	1.14	529	12.8	58	608	11.1	6740	184	1.14	103.0
66-409	15.9	2.00	621	15.9	0	714	13.8	9851	269	2	108.8
66-415	9.9	1.14	150	6.2	58	173	5.4	939	26	1.14	29.3
66-423	4.1	2.01	0	4.1	0	0	3.5	0	0	2.01	0.0
Total diluted recovery			2112	10.5		2064	9.1	22084	604		

8N-18 secteur 1

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
97-90	31.3	0.71	4770	13.9	125	5486	12.1	66435	2029	0.71	1046.1
97-91	36.5	0.77	2179	17.6	107	2506	15.3	38370	1172	0.77	477.9
97-93	15.2	1.57	1440	14.9	2	1657	13.0	21504	657	1.57	315.9
Total diluted recovery			8390	15.1		9166	13.1	126308	3860		

8N-18 secteur 2 niv 975-1035

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-939	60.0	0.98	5967	36.6	64	6862	31.8	218322	6668	0.98	1308.6
66-924	38.5	0.59	5628	14.1	172	6472	12.3	79596	2431	0.59	1234.2
66-939	60.0	0.98	1529	36.6	64	1759	31.8	55952	1709	0.98	335.4
Total diluted recovery			13125	27.0		14338	23.5	218322	10812		

8N-18 secteur 1

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
97-90	31.3	0.71	4770	13.9	125	5486	12.1	66435	2029	0.71	1046.1
97-91	36.5	0.77	2179	17.6	107	2506	15.3	38370	1172	0.77	477.9
97-93	15.2	1.57	1440	14.9	2	1657	13.0	21504	657	1.57	315.9
Total diluted recovery			8390	15.1		9166	13.1	126308	3860		

78-H

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-791	44.5	0.47	7518	13.1	239	8645	11.4	98622	3012	0.47	1648.6
66-924	3.1	0.88	2685	1.7	81	3087	1.5	4587	140	0.88	588.7
66-929	32.6	0.77	1875	15.8	107	2156	13.7	29541	902	0.77	411.2
66-930	18.3	0.68	1460	7.8	136	1679	6.7	11315	346	0.68	320.1
66-937	35.9	0.48	6424	10.9	230	7387	9.5	69858	2134	0.48	1408.7
66-939	38.5	0.49	1971	11.8	225	2267	10.3	23335	713	0.49	432.3
66-940	8.8	1.13	4843	6.2	42	5569	5.4	29856	912	1.13	1062.0
78-499	60.0	0.22	11448	8.2	635	13165	7.1	93503	2856	0.22	2510.5
78-501	23.3	1.23	5072	17.9	30	5833	15.6	90807	2773	1.23	1112.2
78-509	10.1	0.31	6567	2.0	414	7552	1.7	12901	394	0.31	1440.1
Total diluted recovery			49861	9.3		54473	8.1	464325	14185		

785N

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
DDH	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-791	8.9	0.91	19645	5.0	76	22592	4.4	98856	3019	0.91	4308.2
66-924	36.2	0.46	7333	10.4	249	8433	9.0	76092	2324	0.46	1608.0
66-939	25.0	0.74	14928	11.6	116	17167	10.1	172917	5281	0.74	3273.6
66-940	42.5	1.47	11125	39.1	9	12794	34.0	435027	13287	1.47	2439.8
85-001	21.2	0.66	23206	8.7	144	26687	7.6	201871	6166	0.66	5089.1
91-022	8.8	0.84	15560	4.6	90	17894	4.0	72094	2202	0.84	3412.4
Total diluted recovery			91797	11.5		100289	10	1056855	32242		

I-23

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
48-172	49.7	0.50	1349	15.6	219	1552	13.5	21006	642	0.5	295.9
48-174	24.7	0.80	1449	12.4	99	1666	10.8	17990	549	0.8	317.7
48-709	28.0	1.57	902	27.4	2	1038	23.8	24738	756	1.57	197.9
48-711	59.7	0.46	1763	17.3	245	2028	15.0	30482	931	0.46	386.6
48-712	35.8	0.47	1867	10.6	237	2147	9.2	19806	605	0.47	409.4
Total diluted recovery			7330	15.6		8008	13.5	114022	3476		

66-2-3660

Minimum width	1.8
External Dilution	15
Mining recovery	100

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-211	32.6	0.47	581	8.6	281	668	7.4	4966	160	0.47	113.2
66-717	8.3	0.37	50	1.7	386	57	1.5	85	3	0.37	9.7
66-768	10.5	0.37	332	2.2	381	382	1.9	725	23	0.37	64.7
66-788	14.9	2.30	1599	14.9	0	1839	13.0	23839	766	2.3	243.8
66-933	16.7	0.69	865	6.4	159	995	5.6	5569	179	0.69	168.6
66-934	5.9	0.69	339	2.3	163	390	2.0	761	24	0.69	66.0
Total diluted recovery			3765	9.5		4329	8.3	35945	1156		

Zone 4 Gauche

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
14-039	21.9	0.43	616	5.2	321	709	4.5	3206	88	0.43	120.1
14-041	60.0	0.47	844	15.7	281	970	13.7	13276	363	0.47	164.5
14-057	8.9	0.51	80	2.5	252	92	2.2	202	6	0.51	15.6
14-063	23.3	0.48	828	6.2	277	952	5.4	5120	140	0.48	161.4
14-100	26.3	0.41	724	5.9	344	833	5.2	4290	117	0.41	141.1
14-107	5.9	1.80	579	5.9	0	666	5.1	3399	93	1.8	112.8
14-108	22.1	0.38	497	4.7	370	571	4.1	2335	64	0.38	96.9
14-97	16.8	0.49	157	4.5	271	180	3.9	709	19	0.49	30.5
14-98	8.7	0.59	144	2.9	203	165	2.5	413	11	0.59	28.0
14-99	21.0	0.49	215	5.7	269	247	4.9	1217	33	0.49	41.8
14-044	43.8	0.37	786	9.1	382	904	7.9	7153	195	0.37	153.3
14-060	27.1	0.90	642	13.5	100	738	11.8	8685	237	0.9	125.1
14-065	26.0	0.34	1034	4.9	430	1189	4.3	5074	139	0.34	201.6
Total diluted recovery			7144	7.7		6984	6.7	55078	1505		

Zone 5 Gauche

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
14-039	13.7	0.85	2026	6.4	113	2330	5.6	12987	355	0.85	394.9
14-057	31.9	0.40	276	7.0	355	317	6.1	1931	53	0.4	53.7
14-062	9.1	0.87	896	4.4	106	1031	3.8	3928	107	0.87	174.7
14-096	22.3	0.66	422	8.2	173	485	7.1	3442	94	0.66	82.2
14-107	34.1	0.86	676	16.3	109	778	14.2	11010	301	0.86	131.8
14-108	30.7	0.39	675	6.7	356	777	5.9	4543	124	0.39	131.6
14-109	12.2	1.08	728	7.3	67	837	6.3	5302	145	1.08	141.8
14-058	24.6	0.42	1548	5.8	324	1781	5.1	8986	246	0.42	301.8
14-110	40.4	0.47	735	10.5	286	845	9.1	7696	210	0.47	143.2
Total diluted recovery			7981	7.5		7802	6.5	59824	1641		

Zone: Chantier: 15 S1

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
41-035	2.4	0.73	1133	1.1	119	1303	1.0	1241	38	0.73	248.4
41-037	4.7	1.85	1659	4.7	0	1907	4.1	7796	238	1.85	313.8
GD-91-119	19.3	1.21	1181	14.6	32	1359	12.7	17293	528	1.21	259.1
GD-92-126	35.7	1.50	2929	33.4	7	3368	29.0	97769	2986	1.5	642.3
GD-92-127	2.7	0.66	3155	1.1	142	3628	1.0	3513	107	0.66	691.8
GD-92-130	22.0	0.74	2837	10.1	117	3263	8.8	28741	878	0.74	622.2
Total diluted recovery			12893	12.1		14086	10.5	156353	4775		

		Zone:		Chantier:	15 S2						
Minimum width		1.6									
External Dilution		15									
Mining recovery		95									
			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
35-085	18.3	0.94	601	10.8	70	691	9.4	6465	197	0.94	131.8
35-086	3.1	3.75	1957	3.1	0	2250	2.7	6065	185	3.75	183.2
35-089	16.1	1.98	1091	16.1	0	1255	14.0	17565	536	1.98	193.7
35-225	6.9	1.33	1009	5.8	20	1160	5.0	5798	177	1.33	221.2
35-226	10.7	1.46	498	9.8	9	572	8.5	4871	149	1.46	109.1
407-47	3.8	0.79	68	1.9	102	79	1.6	128	4	0.79	15.0
407-48	0.9	0.59	84	0.3	170	97	0.3	28	1	0.59	18.5
407-49	0.2	0.36	261	0.1	343	301	0.0	12	0	0.36	57.3
408-51	0.9	1.22	504	0.7	31	579	0.6	345	11	1.22	110.5
408-52	4.5	1.19	676	3.4	34	778	2.9	2269	69	1.19	148.3
408-53	0.2	1.07	799	0.1	49	919	0.1	107	3	1.07	175.2
408-54	13.2	1.90	780	13.2	0	897	11.5	10293	314	1.9	144.2
408-55	4.1	1.59	1159	4.1	1	1333	3.5	4710	144	1.59	254.1
408-57	59.9	0.56	1351	21.0	185	1553	18.3	28380	867	0.56	296.2
408-58	6.3	1.03	725	4.1	55	834	3.5	2951	90	1.03	159.0
41-038	2.3	0.80	842	1.2	100	968	1.0	968	30	0.8	184.7
41-039	8.7	1.41	1313	7.7	13	1510	6.7	10083	308	1.41	287.9
41-200	3.0	0.98	881	1.8	63	1013	1.6	1619	49	0.98	193.3
GD-91-050	18.9	0.65	1089	7.7	147	1253	6.7	8348	255	0.65	238.9
GD-91-114	17.4	1.93	195	17.4	0	225	15.1	3398	104	1.93	35.6
GD-91-118	9.0	0.62	869	3.5	159	999	3.0	3019	92	0.62	190.5
GD-92-121	5.3	1.02	699	3.4	58	804	2.9	2353	72	1.02	153.3
Total diluted recovery			17451	6.9		19065	6.0	119773	3678		

Zone: Chantier: 15 S3

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
41-041	12.3	2.21	1434	12.3	0	1649	10.7	17632	539	2.21	227.6
41-086	14.0	3.14	2477	14.0	0	2848	12.2	34675	1059	3.14	277.1
41-089	53.2	0.49	1244	16.4	224	1431	14.3	20455	625	0.49	272.9
41-092	12.1	0.93	2335	7.1	71	2686	6.2	16507	504	0.93	512.2
41-093	4.2	5.94	6789	4.2	0	7807	3.7	28512	871	5.94	401.3
41-203	30.2	0.91	1060	17.2	76	1219	15.0	18216	556	0.91	232.4
41-204	17.0	2.48	1916	17.0	0	2204	14.8	32574	995	2.48	271.5
41-206	11.9	0.83	3176	6.2	93	3652	5.4	19580	598	0.83	696.4
GD-91-052	60.0	0.5	1135	18.7	221	1305	16.3	21219	648	0.5	248.9
Total diluted recovery			21565	9.7		23560	8.4	209369	6362		

Zone: Chantier: CM85-08-350-350H

Minimum width	1.6
External Dilution	15
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-164	41.9	0.5	1273.5	13.09	220	1464.5	11.38	16663.6	509	0.5	279.27
78-202	3.5	0.8	16.9	1.75	101	19.5	1.52	29.6	1	0.8	3.72
78-205	18	0.58	380.9	6.58	174	438	5.72	2505.3	77	0.58	83.53
78-548	6.1	0.71	0	2.7	126	0	2.35	0	0	0.71	0
78-571	10.5	0.63	802.3	4.14	153	922.7	3.6	3324.1	102	0.63	175.95
78-572	11.4	1.43	1056.5	10.16	12	1215	8.83	10732	328	1.43	231.69
78-573	6.4	0.58	0	2.33	174	0	2.03	0	0	0.58	0
Total diluted recovery			3530	9.4		3857	8.2	33255	1016		

Zone: Chantier: LT85-08-350-350H sn3-niv

Minimum width	1.8
External Dilution	25
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-194	64.6	1.56	311.8	55.86	16	389.8	44.69	17419.7	532	1.56	60.78
66-196	24.6	0.79	943.9	10.73	129	1179.9	8.59	10131.6	309	0.79	183.99
78-213	30.7	0.59	323.7	10.01	207	404.6	8	3238.4	4.99	0.59	63.1
78-291	57.3	1.07	394.7	34.04	68	493.4	27.24	13438.1	410	1.07	76.94
78-416	30.2	1.5	217.9	25.14	20	272.4	20.11	5476.6	167	1.5	42.47
78-535	18.2	0.7	49.9	7.06	158	62.3	5.65	352.2	11	0.7	9.72

78-536	33.6	0.65	279.8	12.06	179	349.7	9.65	3375.1	103	0.65	54.54
78-541	16.2	0.64	396.9	5.79	180	496.1	4.63	2298.3	70	0.64	77.37
78-567	39.3	1.12	577.2	24.4	61	721.4	19.52	14084.4	430	1.12	112.51
Total diluted recovery			3496	20		4151	16	69814	2132		
					S.N. #3	2616	10.9	28514	917		
					Total	6767	14	94738	3046		

Zone: Chantier: LT85-08-350-350H sn3- sn2 est

Minimum width 1.8
External Dilution 25
Mining recovery 95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
78-294	5.6	0.42	256.6	1.29	333	320.7	1.04	332	10	0.42	50.01
78-541	16.2	0.64	220.9	5.79	180	276.1	4.63	1279.2	39	0.64	43.06
78-542	24.6	0.37	173.1	5.07	385	216.3	4.06	877.7	27	0.37	33.74
78-545	1.1	1.82	19.7	1.1	0	24.6	0.88	21.7	1	1.82	3.79
78-546	5.3	0.66	106.8	1.94	173	133.5	1.55	207.4	6	0.66	20.81
Total diluted recovery			777	3.5		923	2.8	2718	83		
					sn #3	930	11.66	10844	349		
					sn #2 (1)	259	9.75	2525	82		
					sn #2 (2)	356	8	2848	92		
					Total	2468	7.62	18806	605		

Zone: Chantier: LT85-08-350-350H sn3- sn2 ouest

Minimum width 1.8
External Dilution 25
Mining recovery 95

Internal Dilution External Dilution

Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
78-537	11.2	0.82	597.2	5.11	119	746.5	4.09	3053.8	93	0.82	116.42
Total diluted recovery			597	5.1		709	4.1	3054	93		
					sn #3	1308	13.59	17776	572		
					Total	2017	10.25	20674	665		

Zone: Chantier: LT85-08-350-350H sn2- sn1 est

Minimum width 1.8
External Dilution 25
Mining recovery 95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
78-204	1.2	1.81	0	1.2	0	0	0.96	0	0	1.81	0
78-542	24.6	0.37	98.9	5.07	385	123.7	4.06	501.7	15		
78-569	12.7	0.88	1013.8	6.2	105	1267.3	4.96	6289.7	192	0.88	197.63
Total diluted recovery			1113	6.1		1321	4.9	6791	207		
					SN #2	259	9.75	2525	82		
					Total	1580	5.69	8998	289		

Zone: Chantier: LT85-08-350-350H sn2- sn1 ouest

Minimum width 1.8
External Dilution 25
Mining recovery 95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-602	7	1.18	281.4	4.58	53	351.7	3.66	1288.9	39	1.18	54.85
78-568	8.7	1.97	847.1	8.7	0	1058.8	6.96	7369.6	225	1.97	150.98
Total diluted recovery			1128	7.7		1340	6.1	8659	264		

TOTAL LT 85-8-350H

	Tonnes	Grade	Grams	Onces
LT85-08-350-350H sn3-niv	6767	14	94738	3046
LT85-08-350-350H sn3- sn2 est	2468	7.62	18806	605
LT85-08-350-350H sn3- sn2 ouest	2017	10.25	20674	665
LT85-08-350-350H sn2- sn1 est	1580	5.69	8998	289
LT85-08-350-350H sn2- sn1 ouest	1340	6.1	8659	264
	14172	10.7	151383	4867

Zone: Chantier: 66-07-628

Minimum width	1.8
External Dilution	15
Mining recovery	85

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-358	8.6	1.4	1505.4	6.65	29	1731.2	5.78	10011.4	274	1.4	293.44
66-397	13.6	1.36	700.1	10.21	33	805.1	8.88	7150.9	195	1.36	136.47
66-443	14	1.66	2291	12.95	8	2634.7	11.26	29677.6	811	1.66	446.59
66-947	49.5	0.74	1932.9	20.21	145	2222.8	17.58	39068.6	1068	0.74	376.77
66-949	10.6	1.98	1786.8	10.63	0	2054.9	9.24	18994.1	519	1.98	316.45
66-950	40.9	1.05	1336.1	23.74	72	1536.5	20.65	31721.6	867	1.05	260.45
66-951	31.3	1.48	1871.9	25.71	22	2152.7	22.36	48128.1	1315	1.48	364.9
Total diluted recovery			11424	16.2		11167	14.1	184752	5049		

Zone: Chantier: LT85-8-100

Minimum width	1.8
External Dilution	25
Mining recovery	95

			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
72-127	22	0.98	231.1	11.94	84	288.9	9.55	2760.5	84	0.98	45.06
72-158	7.8	0.78	0	3.36	132	0	2.69	0	0	0.78	0
78-124	7.9	0.76	0	3.36	135	0	2.68	0	0	0.76	0
78-175	10.6	1.47	349.2	8.65	23	436.5	6.92	3019.2	92	1.47	68.07
78-195	4.5	1.38	171.2	3.46	30	214	2.77	592.1	18	1.38	33.37
78-216	32.4	2.18	75.7	32.4	0	94.7	25.92	2454.1	75	2.18	12.17
78-217	2.9	0.26	0	0.42	591	0	0.34	0	0	0.26	0
78-326	7.3	0.59	0	2.41	203	0	1.93	0	0	0.59	0
78-376	11.4	1.26	566.2	8	42	707.8	6.4	4532.1	138	1.26	110.37
78-530	6.4	2.76	459.9	6.4	0	574.9	5.12	2943.3	90	2.76	58.4
78-531	8.1	2.28	113.2	8.1	0	141.5	6.48	917.2	28	2.28	17.45
Total diluted recovery			1967	8.8		2335	7	17219	526		

Zone: Chantier: 85-8-100 (Bas)

Minimum width	1.6
External Dilution	15
Mining recovery	85

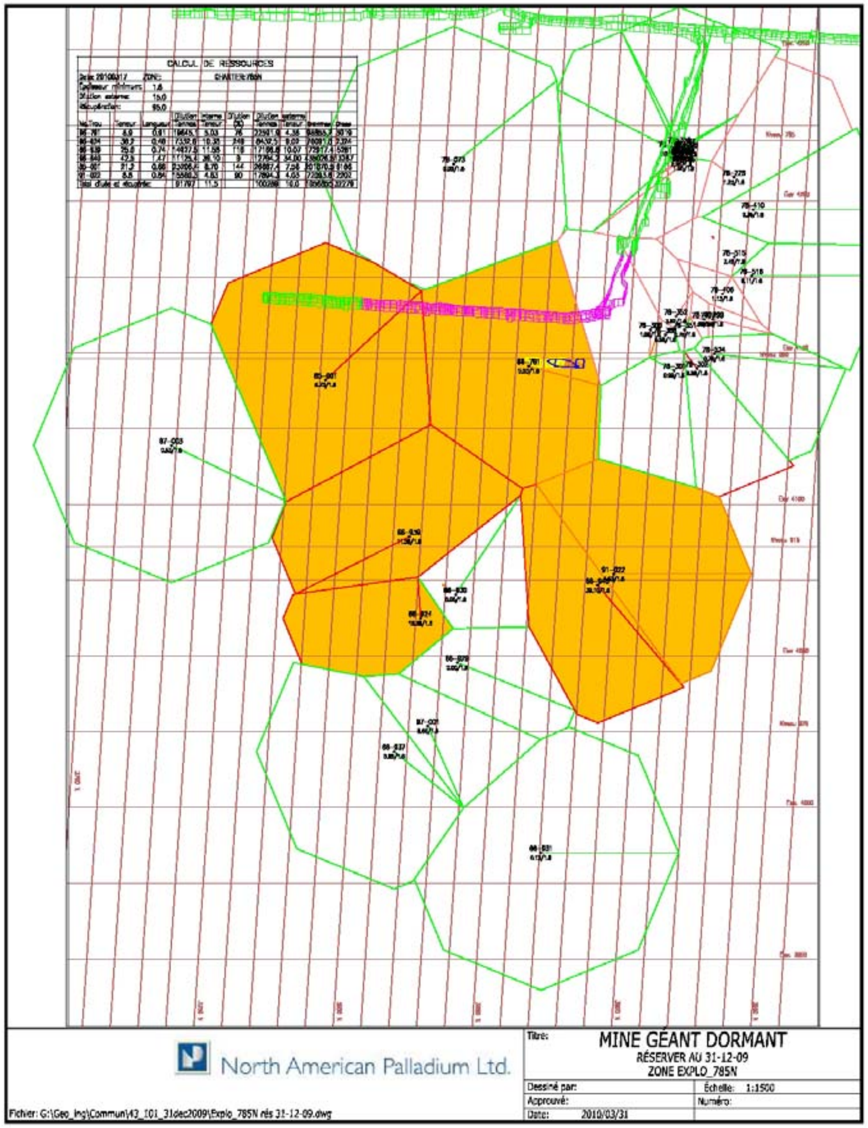
			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
72-145	0.5	0.59	172.8	0.18	171	198.7	0.16	31.9	1	0.59	37.89

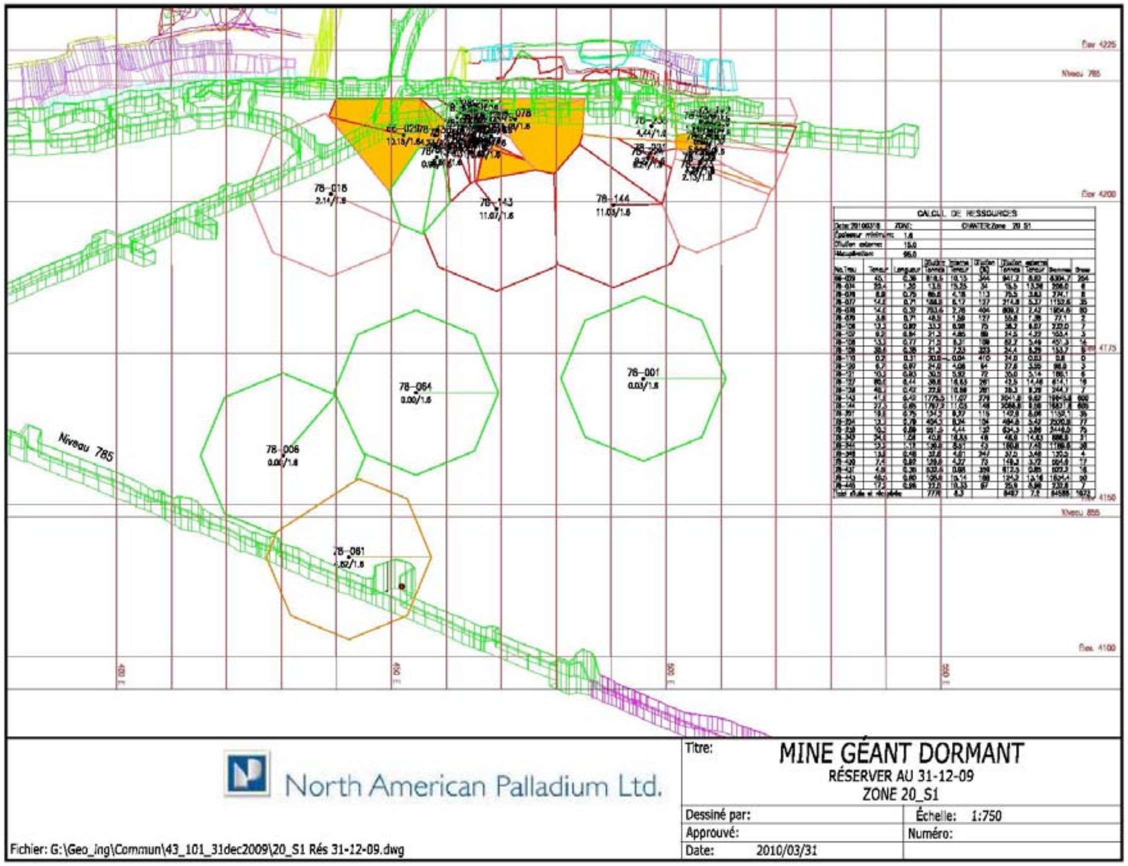
72-221	0.7	0.78	17.4	0.34	106	20	0.3	5.9	0	0.78	3.82
72-225	3.2	1.8	64.2	3.2	0	73.8	2.78	205.4	6	1.8	12.52
72-235	16.2	1.2	225	12.14	33	258.7	10.56	2732.5	75	1.2	49.34
85-050	3	1.52	0	2.85	5	0	2.48	0.1	0	1.52	0.01
85-051	11	0.63	8.2	4.31	155	9.5	3.75	35.4	1	0.63	1.8
85-053	1.4	0.41	117.9	0.36	292	135.5	0.31	42.1	1	0.41	25.85
85-054M	1.2	0.66	0.2	0.24	391	0.3	0.21	0.1	0	0.33	0.05
85-055	14	1.38	509.5	12.06	16	585.9	10.49	6145.7	168	1.38	111.73
85-061	4.7	1.29	335.7	3.78	24	386	3.29	1270.3	35	1.29	73.61
85-073	14.7	0.41	1.3	3.74	293	1.5	3.25	0	0	0.41	0.29
Total diluted recovery			1452	7.2		1420	6.3	10474	286		

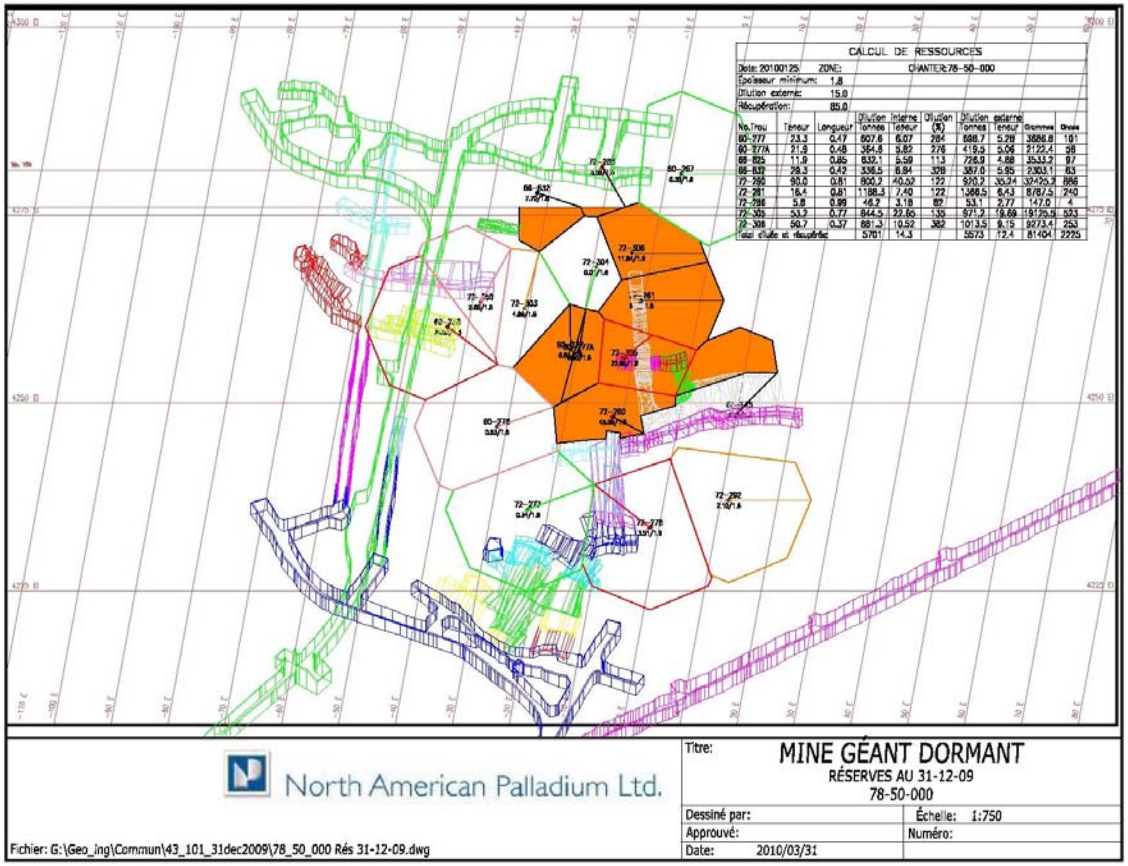
	Zone:	Chantier:	Resevers 66-07-628

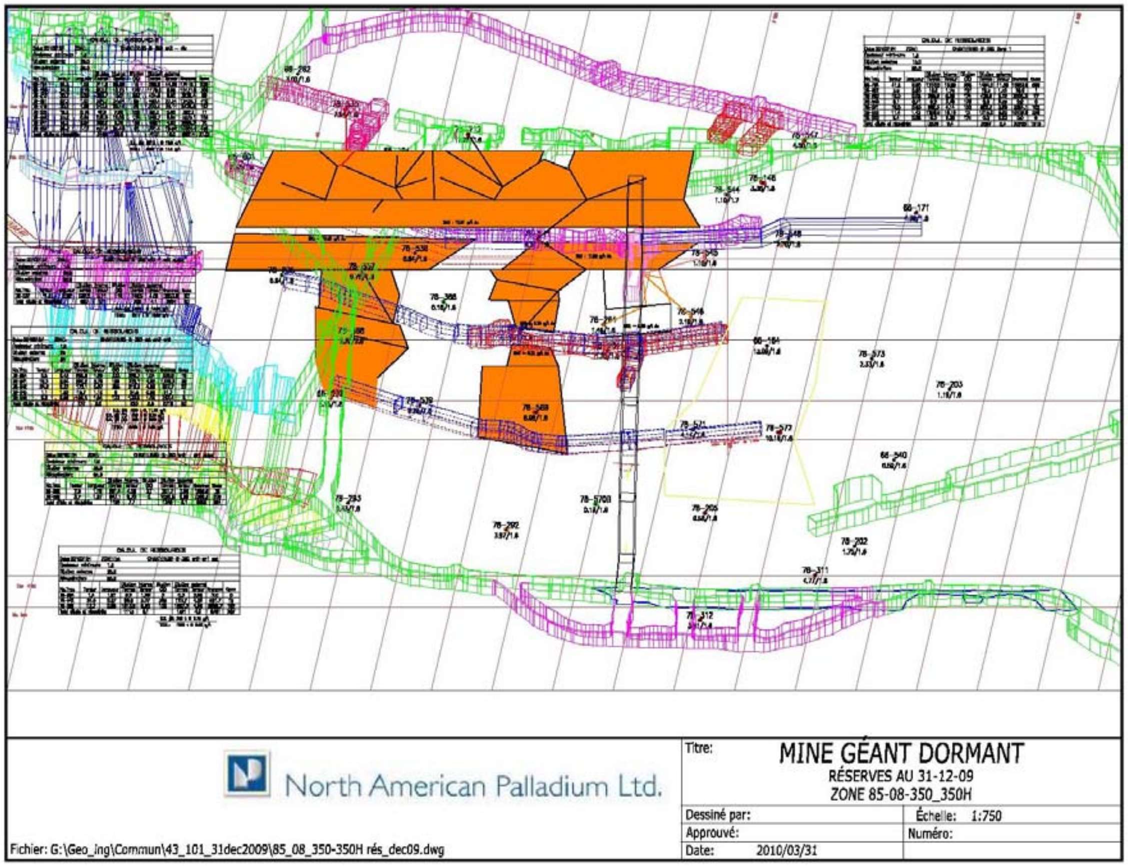
Minimum width	1.8
External Dilution	15
Mining recovery	85

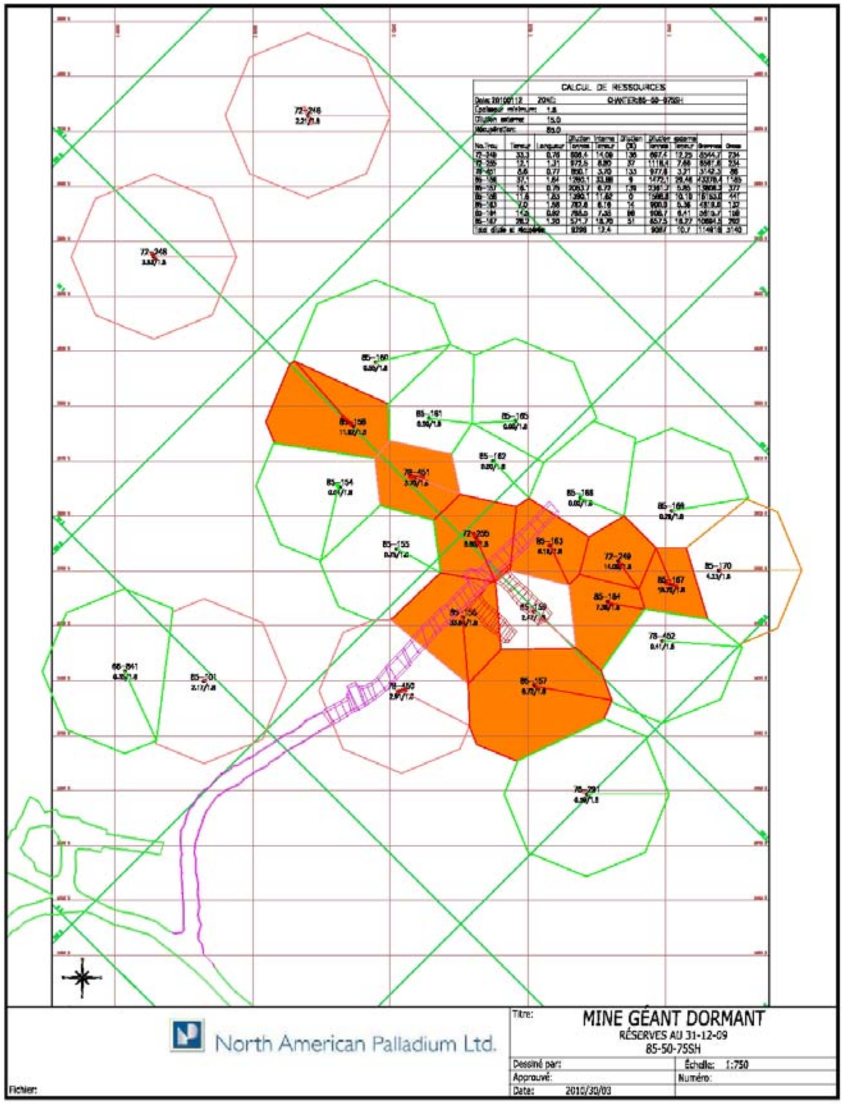
			Internal Dilution		External Dilution						
Drill Hole	Grade	Length	Tonnes	Grade	Dilution	Tonnes	Grade	Grams	Ounces	Thickness	Area
66-358	8.6	1.4	1505.4	6.65	29	1731.2	5.78	10011.4	274	1.4	293.44
66-397	13.6	1.36	700.1	10.21	33	805.1	8.88	7150.9	195	1.36	136.47
66-443	14	1.66	2291	12.95	8	2634.7	11.26	29677.6	811	1.66	446.59
66-947	49.5	0.74	1932.9	20.21	145	2222.8	17.58	39068.6	1068	0.74	376.77
66-949	10.6	1.98	1786.8	10.63	0	2054.9	9.24	18994.1	519	1.98	316.45
66-950	40.9	1.05	1336.1	23.74	72	1536.5	20.65	31721.6	867	1.05	260.45
66-951	31.3	1.48	1871.9	25.71	22	2152.7	22.36	48128.1	1315	1.48	364.9
Total diluted recovery			11424	16.2		11167	14.1	184752	5049		


CALCUL DE RESSOURCES
North American Palladium Ltd.
Titre:
MINE GÉANT DORMANT
RÉSERVER AU 31-12-09
ZONE EXPLO_785N
Dessiné par:
Approuvé:
Date: 2010/03/31
Échelle: 1:1500
Numéro:
Fichier: G:\Geo_Ing\Commun\43_101_31dec2009\Explo_785N rés 31-12-09.dwg


Niveau 785
CALCUL DE RESSOURCES
North American Palladium Ltd.
Titre:
MINE GÉANT DORMANT
RÉSERVER AU 31-12-09
ZONE 20_S1
Dessiné par:
Échelle: 1:750
Approuvé:
Numéro:
Date: 2010/03/31
Fichier: G:\Geo_ing\Commun\43_101_31dec2009\20_S1 Rés 31-12-09.dwg


CALCUL DE RESSOURCES
Date: 20100125 ZONE: CHANTER:78-50-000
Épaisseur minimum: 1,8
Dilution externe: 15.0
Récupération: 85.0
No.Trou | Teneur | Longueur | Dilution interne Tonnes | Teneur | Dilution (%) | Dilution externe Tonnes | Teneur | Grammes | Onces
80-777 | 23.3 | 0.47 | 607.6 | 6.07 | 284 | 698.7 | 5.28 | 3686.8 | 101
80-777A | 21.9 | 0.48 | 364.8 | 5.82 | 276 | 419.5 | 5.06 | 2122.4 | 68
88-825 | 11.9 | 0.85 | 632.1 | 5.59 | 113 | 726.9 | 4.88 | 3533.2 | 97
88-832 | 28.3 | 0.42 | 336.5 | 6.84 | 328 | 387.0 | 5.95 | 2303.1 | 63
72-280 | 90.0 | 0.81 | 800.2 | 40.52 | 122 | 920.2 | 35.24 | 32425.2 | 886
72-281 | 16.4 | 0.81 | 1188.3 | 7.40 | 122 | 1366.5 | 6.43 | 8787.5 | 240
72-286 | 5.8 | 0.99 | 46.2 | 3.18 | 82 | 53.1 | 2.77 | 147.0 | 4
72-305 | 53.2 | 0.77 | 844.5 | 22.65 | 135 | 971.2 | 19.69 | 19126.6 | 523
72-308 | 50.7 | 0.37 | 881.3 | 10.52 | 382 | 1013.5 | 9.15 | 9273.4 | 253
Total dilué et récupérée | | | 5701 | 14.3 | | 5573 | 12.4 | 81404 | 2225
North American Palladium Ltd.
Titre:
MINE GÉANT DORMANT
RÉSERVES AU 31-12-09
78-50-000
Dessiné par:
Échelle: 1:750
Approuvé:
Numéro:
Date: 2010/03/31
Fichier: G:\Geo_ing\Commun\43_101_31dec2009\78_50_000 Rés 31-12-09.dwg


North American Palladium Ltd.
Titre:
MINE GÉANT DORMANT
RÉSERVES AU 31-12-09
ZONE 85-08-350_350H
Dessiné par:
Échelle: 1:750
Approuvé:
Numéro:
Date: 2010/03/31
Fichier: G:\Geo_ing\Commun\43_101_31dec2009\85_08_350-350H rés_dec09.dwg


CALCUL DE RESSOURCES
North American Palladium Ltd.
Titre:
MINE GÉANT DORMANT
RÉSERVES AU 31-12-09
85-50-75SH
Dessiné par:
Échelle: 1:750
Approuvé:
Numéro:
Date: 2010/30/03
Fichier: