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TECHNICAL REPORT ON THE DINGMAN GOLD PROPERTY MADOC, ONTARIO, CANADA

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1.0 SUMMARY

1.1 Scope of Work

Shaft & Tunnel Engineering Services Ltd. (Shaft & Tunnel) was commissioned by Opawica Explorations Inc. (Opawica) to provide an independent updated mineral resource estimate in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in NI 43-101, Standards of Disclosure for Mineral Projects for the Dingman Property. Shaft & Tunnel is also involved in the preparation of a Technical Report as defined in NI 43-101 and in compliance with Form 43-101F1 (the Technical Report). The Dingman Au Property is located in southeastern Ontario within the Grenville Province, in the Marmora and Madoc Townships, approximately 55 km north of Belleville, Ontario.

The July 16, 2009 Dingman Updated Mineral Resource Estimate and Technical Report were completed by Terry Link (independent consultant) and Robert Laakso, P.Eng., of Shaft & Tunnel. The aggregate resource in Section 17.6 was also completed by Robert Laakso, P.Eng., of Shaft & Tunnel. Robert Laakso is the supervising Qualified Person (QP) for the Updated Technical Report and Resource Estimate.

1.2 Location and Ownership

The Dingman Property is located in southeastern Ontario within the Grenville Province, in the Marmora and Madoc Townships, approximately 55 km north of Belleville, Ontario.

The mining claims covering the Dingman Property were staked and recorded between November 2004 and September 2006, by Douglas Baird (Baird) of Toronto, Ontario, and Edward Neczkar (Neczkar) of Etobicoke, Ontario. In August 2006, Opawica entered into an option agreement with Baird and Neczkar, granting Opawica the right to earn a 100% interest in the mineral rights of the Dingman Property.

1.3 History of Property

The Dingman Property was staked in 1985 by Mark Dingman following the discovery of anomalous gold values in granite. In 1986, Noranda Exploration Company Limited (Noranda) optioned the Dingman Property and carried out extensive exploration work between 1986 and 1988, including channel sampling, diamond drilling and metallurgical test work. A wide zone of gold mineralization was outlined to a depth of approximately 150 m, hosted within the western portion of the Dingman granite stock. The property was subsequently transferred to Hemlo Gold Mines Inc., which was acquired in 1996 by Battle Mountain Canada Ltd. (Battle Mountain). In 1997, Deloro Minerals Ltd. (Deloro) purchased the Dingman Property from Battle Mountain and completed additional diamond drilling on the property.

Historical estimates of mineral resources have been completed for the Dingman Property. They are mentioned here for completeness, but they should be considered for information purposes only, as they are not compliant with NI 43-101 requirements. Deloro [formerly 698366 Alberta Ltd., and Rajong Resources Ltd. (Rajong)] completed an historical resource in 1997, prepared by Roscoe Postle Associates Inc. (RPA) for Rajong, which defined an estimated “resource” of 5.64 million short tons grading 0.034 oz/ton Au based on the concept of mining the bulk of the deposit by open pit and processing by heap leaching (Roscoe, 1997).

Barnes in 1998 estimated the Dingman resource as combined "Measured + Indicated Mineable Resource" of 6.5 million tonnes grading 0.99 g/t Au at a US$400/oz gold price with a cut-off grade of 0.5 g/t. **This tonnage and grade estimate are considered by Shaft & Tunnel to be historical estimates as a Qualified Person has not done sufficient work to classify the historical estimates as current mineral resources and Opawica is not treating any of the historical estimates as current mineral resources and the historical estimates should not be relied upon.**

1.4 Geology and Mineralization

The Dingman Property is located within low grade metamorphic rocks of the Grenville Supergroup of Proterozoic age. Supracrustal rocks comprise mafic to felsic metavolcanic rocks, marbles derived from both limestones and dolomites, and clastic metasedimentary rocks. These are intruded by mafic to felsic plutons, sills and dykes.

The property lies between the Deloro Granite pluton to the south and the Gawley Creek Syenite pluton to the north. The property is underlain by carbonate and intercalated clastic sedimentary rocks and an elongated granite intrusion striking ENE, and measuring about 800 m long by 150 m wide, and known as the Dingman Granite which forms a wooded, rocky ridge with maximum relief of 20 m above the adjacent fields. The granite strikes ENE and dips at 55° to 60° to the NNW, with marble adjacent which is mostly grey, fine grained impure calcitic marble.

Gold mineralization on the Dingman Property occurs as the hydrothermal quartz-carbonate vein gold subtype consisting of simple to complex quartz-carbonate vein systems associated with shear zones and folds in deformed and metamorphosed volcanic, sedimentary and granitoid rocks. In these deposits, gold occurs in veins or as disseminations in adjacent altered wall rocks.

The bulk of the gold mineralization identified on the property occurs within the Dingman granite mineralization and is associated with moderate to strong sericite alteration with increased foliation development or shearing and variable but generally increased amounts of quartz veining, with elevated sulphide contents typically greater than 2-3%.

The major mineralization outcrop is located in the western portion of the Dingman granite between Noranda grid coordinates 150E and 250W, generally oriented in an east-northeast, west-southwest direction at an azimuth of 060°, parallel to the trend of the granite and the dominant foliation, but is also seen locally oriented in a northeast direction at an azimuth of 025°, parallel to the fracture-shear foliation and cross faults, and dips to the north-northwest at approximately 50-60°, sub-parallel to the dip of the granite body. Local variations in the dip of the zones appear to be controlled by changes in the dip of the granite contact and by the cross faults.

Diamond drilling has defined mineralization to a vertical depth of approximately 700 m. One drill hole intersected mineralization at a vertical depth of between 500 m and 700 m and appears to remain open at depth.

The bulk of the gold mineralization (sericite-quartz-sulphide) occurs in the western portion of the Dingman granite, west of the East cross fault. Sericite-quartz-sulphide mineralization is also located in the eastern part of the Dingman granite; however, it appears less well mineralized and no significant gold mineralization has been identified from the widely spaced drilling in this area.

1.5 Exploration Programs

Prior to Opawica's involvement, exploration on the Dingman Property was carried out by three different owners: in 1985 by Mark Dingman (Dingman); in 1986-1988 by Noranda; and in 1997 by Deloro.

The Dingman 1985 program consisted of initial prospecting, geological mapping and sampling. Samples taken from the western portion of the granite stock returned values ranging between 0.19 g/t Au and 0.75 g/t Au, while samples from the eastern portion of the granite returned low gold values.

Noranda established a grid system with a baseline at 060° azimuth. A 2007 survey of the property applied a correction to the baseline, which is now 061.42° azimuth. Noranda drilling was carried out in two phases during 1987-1988: the first phase included 13 holes drilled at a dip of -45° which tested the altered Dingman granite and intersected cross faults and second fracture-shear foliation; and the second phase of Noranda drilling included 25 holes, oriented along Noranda grid south, drilled at 50 m spacing on 100 m sections across the Dingman granite to a depth of 150 m. Two holes were drilled at -75° and the remaining 23 holes were drilled at a dip of -45°.

Deloro drilled 14 holes in 1997 to test grade and continuity of mineralization within the Dingman granite. The holes were spaced at 25 m on 25 m sections to a vertical depth of 100 m, oriented along Noranda grid south at a dip of -45°.

The Noranda 1986-1988 exploration work consisted of geological mapping, ground magnetometer and VLF-EM geophysical surveys, soil geochemical surveys, minor stripping and trenching, and channel sampling. A metric grid was established with a 950 m long baseline oriented at 060° azimuth centered along the trend of the Dingman granite stock. Grid lines were at 50 m spacing and stations at 25 m intervals, extending up to 275 m north and south of the baseline.

Soil samples indicating strong gold-arsenic-zinc-lead anomalies occur in four areas of the grid, coinciding with sericite-quartz-sulphide zones and possibly related to the weak gold mineralization encountered in channel sampling and diamond drilling. The absence of soil anomalies in the central portion of the Dingman granite is confirmed by the generally low gold values encountered in the diamond drilling carried out in this area.

A total of 563 m of channel samples, primarily on the western portion of the Dingman Property were oriented to transect the east-northeast, west-southwest trend of the Dingman granite and dominant foliation, as well as the second fracture-shear foliation trending north-easterly at 025° azimuth.

The Noranda exploration results were considered sufficient to warrant the subsequent diamond drilling programs in 1987 and 1988.

In 1997, Deloro re-logged and conducted a check-sampling program on the Noranda drill core, as well as some reconnaissance prospecting and collection of grab samples which were sent for analyses.

Opawica began exploration on the property in 2007 and completed 20 diamond drill holes designed to verify the results of the Noranda and Deloro drilling programs and to better delineate the gold mineralization.

In 2009, Opawica continued with the program and drilled 16 diamond drill holes for a total of 3,926 m. Of the 16 holes, 8 holes were for gold exploration and 8 holes were for delineation of marble and granite for quarry products.

1.6 QA/QC Programs

QA/QC validation of data collected prior to 2009 was completed in the previous Technical Report (Palmer et al., 2009); therefore, a review of this data was not completed during the updated mineral resource estimate.

The majority of the original historical drilling data was provided as paper logs and sections with assay certificates provide for the Noranda and Deloro drilling data. No assay certificates were available for the channel sample data. All the drilling data was used in the mineral resource with 52 drill holes from historical drilling and 20 drill holes from Opawica. In addition, Deloro re-logged and assayed a selection of Noranda's drill hole data as part of their data validation and this included a selection of QA/QC procedures.

During the 2009 Opawica drill program, external QA/QC procedures were provided by standards and blanks inserted by Opawica into the sample stream during sample packing at a rate of one standard and one blank for every twenty samples prior to shipment to the laboratory. In addition, a duplicate was inserted into the sample stream for every twenty samples.

Opawica's 2009 quality control program consisted of the following:

- Gold standard assays varying greater than 15% of the accepted value were classified as a failed batch and re-assayed.
- Blank sample assays varying at greater than five times the detection limit of the assay method and the batch was classified as failed and re-assayed.
- Opawica submitted a total of 84 reject duplicates for routine re-analysis by Swastika Laboratories Ltd. (Swastika) during the 2009 drill program.

In addition, internal quality control procedures by Swastika consisted of standards, blanks and duplicate samples. One standard and blank was inserted per batch of 3 to 74 samples, and 15% of the samples were re-assayed on the original pulp. Swastika re-assayed a total of 240 pulps during the 2009 drill program.

1.7 Data Validation

A selection of data verification checks were completed for the previous mineral resource estimate by Golder (Palmer et al., 2009), dated January 29, 2009, on the historical data and the Opawica drill hole sample data the 2007 exploration program; therefore, only verification work was completed on the 2009 exploration data since sufficient verification checks were previously completed. A summary of the verification checks previously completed by Golder have been included and the new verification checks on the 2009 data are also included.

Shaft & Tunnel received the drill hole database containing the Opawica 2009 drill holes, from Opawica in four digital files (collar, surveys, lithology and assays) using a Comma Separated Value (CSV) text format. A total of 1,490 assay entries are contained in the 2009 Dingman Assay CSV file.

Validation checks were run to determine if errors were present in the collar location data and the downhole survey, lithology and assay interval data.

Shaft & Tunnel compared approximately 10% (150) of the 2009 assays supplied in the CSV files to the lab assay certificates. Samples were selected randomly. All checked values in the assay file matched the values in the individual assay certificates.

Shaft & Tunnel has reviewed the sources of the data provided by Opawica through validation checks against Opawica's 2009 database and against a selection of the original source data provided by Opawica. Shaft & Tunnel has found that Opawica's 2009 database is sufficiently free of errors to be used in the 2009 updated mineral resource estimate.

Robert Laakso, P.Eng., has conducted Dingman Property site visits previous to 2009 and during the 2009 exploration program.

1.8 Metallurgy

Gekko has completed additional processing testing including: Bond Work Index that shows that the ore is an average hardness at 15.3 kwh/tonne, progressive grind tabling tests followed by conventional flotation that resulted in concentration of 96.1% of the gold to a mass of 5.3%, and Gravity-Flotation-Intensive Leach on different grinds with an overall gold recovery of 93.9% expected.

1.9 Mineral Resource

The July 16, 2009 Dingman Property Updated Mineral Resource Estimate was calculated under the direction of Robert Laakso, P.Eng., QP. This is the first revision (August 2009) to the original independent NI 43-101 mineral resource estimate completed for the Dingman Property (March 2009).

The revised mineral resource estimate is based on an additional 16 drill holes from the 2009 Opawica drilling program, with a total length of 3,926 m. Additional mineralized zones were modelled, within the Dingman granite, using an approximate cut-off grade of 0.40 g/t Au. 291 samples, with a total length of 289.6 m, lay inside the additional mineralized zones and resulted in additional inferred resources of 5,628,000 tonnes at 1.2 g/t Au.

Overall, additional mineralized zone limits are 112.5 m West to 112.5 m East, 30 m North to 300 m North and 280 m to 870 m elevation (local grid coordinates). The Inverse Distance Squared interpolation method was used for additional resource estimation.

The Indicated Resource for the Dingman Property includes a total of 8,801,000 tonnes at 0.97 g/t Au, which did not change from the previous estimate. The Inferred Resource has increased from 5,673,000 tonnes at 0.76 g/t Au to 11,301,000 tonnes at 0.98 g/t Au. These resources are based on a 0.40 g/t Au cut-off grade and a capping Au strategy of 30 g/t.

The Au cut-off grade is based on cost information provided by Robert Laakso, P.Eng., QP, assuming the deposit will be mined by open pit methods and proposed ramp access using bulk tonnage underground mining methods. No recoveries (mining or metal) or dilution factors have been considered in these estimates, and the results should be considered strictly in situ.

TABLE 1-1
JULY 16, 2009 DINGMAN PROPERTY UPDATED MINERAL RESOURCE ESTIMATE SAMPLES CAPPED TO 30 AU G/T

Classification	Cut-off Grade Au (g/t)	Tonnes	Au (g/t)	Au (oz)
Indicated	0.40	8,801,000	0.97	275,000
Inferred	0.40	11,301,000	0.98	355,000

Note: Tonnes and ounces are rounded to the nearest 1000.

1.10 Conclusions

Based on recommendations from the previous Technical Report (Palmer et al., 2009), Opawica has completed 16 additional drill holes (in 2009) on the Dingman Property which have increased the mineral resource. The Inverse Distance interpolation method was used for additional resource estimation of zones outlined by the 2009 drill holes.

The core logging and QA/QC sampling program implemented by Opawica for the 2009 drilling campaign was reviewed and included property and core logging facility site visits by the Shaft & Tunnel QP. Opawica's drill hole database was verified against logs, surveys, and assay certificates. The results of these reviews indicated that Opawica's database and QA/QC sampling program are to industry standards and acceptable for mineral resource evaluation.

Gekko has completed additional processing testing including Bond Work Index, progressive grind tabling tests, and Gravity-Flotation-Intensive Leach on different grinds with an overall gold recovery of 93.9% expected.

As a result of the 2009 drilling, the aggregate resource estimate has increased. It has been estimated by QP Robert Laakso, P.Eng., that now 75,000,000 tonnes of granite and limestone aggregate exists on the property which may have a positive effect on the overall property development.

1.11 Recommendations

Opawica has completed revised reporting of the mineral resource estimate for the Dingman Property. As the project advances, the following steps are recommended:

- Exploration drilling:
  - Infill drilling of the Indicated Resource "higher" grade core between 200 West and 100 East with the purpose of potentially increasing the confidence in these areas to a higher resource classification.
  - Infill drilling of the Inferred Resource east of the Indicated Resource with the purpose of potentially increasing the confidence in these areas to a higher resource classification.
  - Deeper drilling to confirm the down-dip potential of the resource.
- Development of a robust drilling database system. Currently, all data is stored in a Microsoft Excel file which does not include built-in data verification.
- Bulk sample of approximately 40,000 tonnes.
- Duplicate samples should be collected in all samples above 5 g/t to determine variability in these samples based on review of QA/QC data.
- Future mineral resource estimation should review Au capping strategy currently used.
- A more detailed assessment for all categories of aggregate resources located on the property should be conducted once the current drilling program has been completed east of 150 m East.
- Costs associated with maintaining and/or acquiring surface rights.
- Baseline studies and monitoring for permitting and environmental work.
- Collection of geotechnical data to support a preliminary open pit mining assessment.
- Preliminary open pit mining assessment of the Au resource and the aggregate resource.

1.12 Estimate Cost of 2010 Exploration Programs

PHASE I	
Drilling	
At least 4 deep holes at up to 700 m per hole and 10 shallow holes at 150 m per hole, say 4,500 m – at $100/m all in cost	$ 450,000
Contingency in-fill drilling on strike and to depth, 25 holes averaging 300 m per hole, say 7,500 m – at $100/m all in cost	$ 750,000
Personnel	
Project Manager – 9 months at $10,000 / month	$ 90,000
Geologist – 6 months at $7,000 / month	$ 42,000
Field Technician-assistants – 2 at 6 months at $5,000 / month	$ 60,000
Transportation	
Vehicle rentals and core transport – 6 months	$ 15,000
Room and Board	
Lodging and Meals – 6 months	$ 30,000
Drill Core and Assay Costs	
Shipping – 12 months	$ 3,000
Assays – up to 4,000; 1 m interval samples	$ 80,000
Core Splitting/Storage – 12 months	$ 40,000
Reports/Supplies and Other	
Reports	$ 70,000
Supplies	$ 25,000
Communications	$ 15,000
SUBTOTAL PHASE-I WITH CONTINGENCY DRILLING	**$ 1,670,000**
Contingency (10%)	$ 167,000
PROPOSED TOTAL	**$ 1,837,000**
PHASE II	
Preliminary Scoping and Baseline Studies (2009-2010)	
Contingent upon positive results from above programs: Preliminary Scoping Studies with final in-fill drilling/Baseline/Environmental Study Costs (2009-2010)	**$ 1,250,000**

2.0 INTRODUCTION

Shaft & Tunnel was commissioned by Opawica to provide an independent revised mineral resource estimate and technical report for the Dingman Gold Property (Dingman Property).

Shaft & Tunnel has completed an independent Updated Mineral Resource Estimate in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in NI 43-101, Standards of Disclosure for Mineral Projects for the Dingman Property. It also involved the preparation of a Technical Report as defined in NI 43-101 and in compliance with Form 43-101F1 (the Technical Report). The Dingman Au Property is located in southeastern Ontario within the Grenville Province, in the Marmora and Madoc Townships, approximately 55 km north of Belleville, Ontario.

The July 16, 2009 Dingman updated Mineral Resource Estimate and Technical Report were completed by Terry Link (independent consultant) and Robert Laakso, P.Eng., of Shaft & Tunnel. The aggregate resource in Section 17.6 was also completed by Robert Laakso, P.Eng., of Shaft & Tunnel. Robert Laakso is the supervising QP for the Updated Technical Report and Resource estimate.

The Shaft & Tunnel QP has conducted site visits at both Dingman and Opawica's core logging facility in Matachewan, Ontario. Site visits were conducted by Robert Laakso during the 2009 drilling program, between February 24 and May 5, 2009. During the site visits at Dingman, observations were made regarding drill hole locations, core drilling, core handling and shipping. During the site visits to the Matachewan core logging facility, observations were made regarding core logging, the sample selection procedure, core sawing, sample storage and delivery to the analytical laboratory. Opawica's sample QA/QC programs were also reviewed during the site visit. Mr. Laakso has also completed site visits and assisted in directing exploration programs during the 2007 exploration program.

2.1 Terms of Reference

The principle sources of information for this Technical Report are:

- Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada by Paul Palmer, Greg Greenough and Robert Laakso, dated March 25, 2009.
- Report on Results of an Exploration Program on the Dingman Option, August – November 1986 for Noranda by P.S. LeBaron (including geological, geophysical and geochemical survey maps, and channel sampling maps), dated November 25, 1986.
- Summary Report on the Dingman Option Property for Noranda by B.R. King (including diamond drill plan), dated March 1988.
- Noranda drill logs, assay record sheets, assay certificates including check assay data, and electronic data files of the drill holes in Borsurv format.
- Historical resource estimates undertaken by Noranda
- Memorandum on the Mineralogical Evaluation of Mineralized Dingman Samples for Noranda by E. Clemson, dated September 16, 1988.

- Metallurgical studies undertaken for Noranda.
- Historical resource estimates of Rajong.
- Deloro re-logging and check sampling of Noranda core notes, drill logs and assay certificates; Deloro downhole surveys notes, drill logs, assay certificates and core photographs.
- An electronic drill hole and channel sample database and accompanying report on the Dingman project resource model for Deloro completed by Barnes Engineering Services, Inc. (Barnes), dated September 1998.
- Opawica drill logs, downhole survey tests, assay certificates, specific gravity data, QA/QC data and check assay data.
- Government maps and reports.

All units of measure (see Figure 2-1) used in this report are in the metric system, unless stated otherwise. For the contained metal quantities shown in the mineral resource estimate, gold is expressed in grams/tonne (g/t) and troy ounces (oz). Currencies outlined in the report are in US dollars unless otherwise stated.

FIGURE 2-1: UNITS OF MEASURE AND ABBREVIATIONS

Centimetre	cm
Cubic centimetre	cm^3
Cubic metre	m^3
Degree	°
Degrees Celsius	°C
Gram	g
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometre per hour	km/h
Less than	<
Metre	m
Metres above sea level	masl
Millimetre	mm
Million	M
Million tonnes	Mt
Ounce (troy ounce - 31.1035 gms)	oz
Percent	%
Pound(s)	lb
Parts per million	ppm
Parts per billion	ppb
Square km	km^2
Square metre	m^2
Short Tons (907 kgs)	tons
Tonnes (1000 kgs)	t

2.2 Conversions

The following conversions are used in the preparation of this report.

- 1 troy ounce = 31.1035 g
- 1 troy ounce / ton (short) = 34.28 g / tonne
- 1 pound = 14.5833 oz
- 1 short ton = 2,000 lb
- 1 tonne = 2,204.627 lb
- 1 foot = 0.3048 m

Where specific gravity (SG) has been used throughout this report, it refers to the bulk density as g/cm^3.

3.0 RELIANCE ON OTHER EXPERTS

There is no reliance on other experts for the Technical Report.

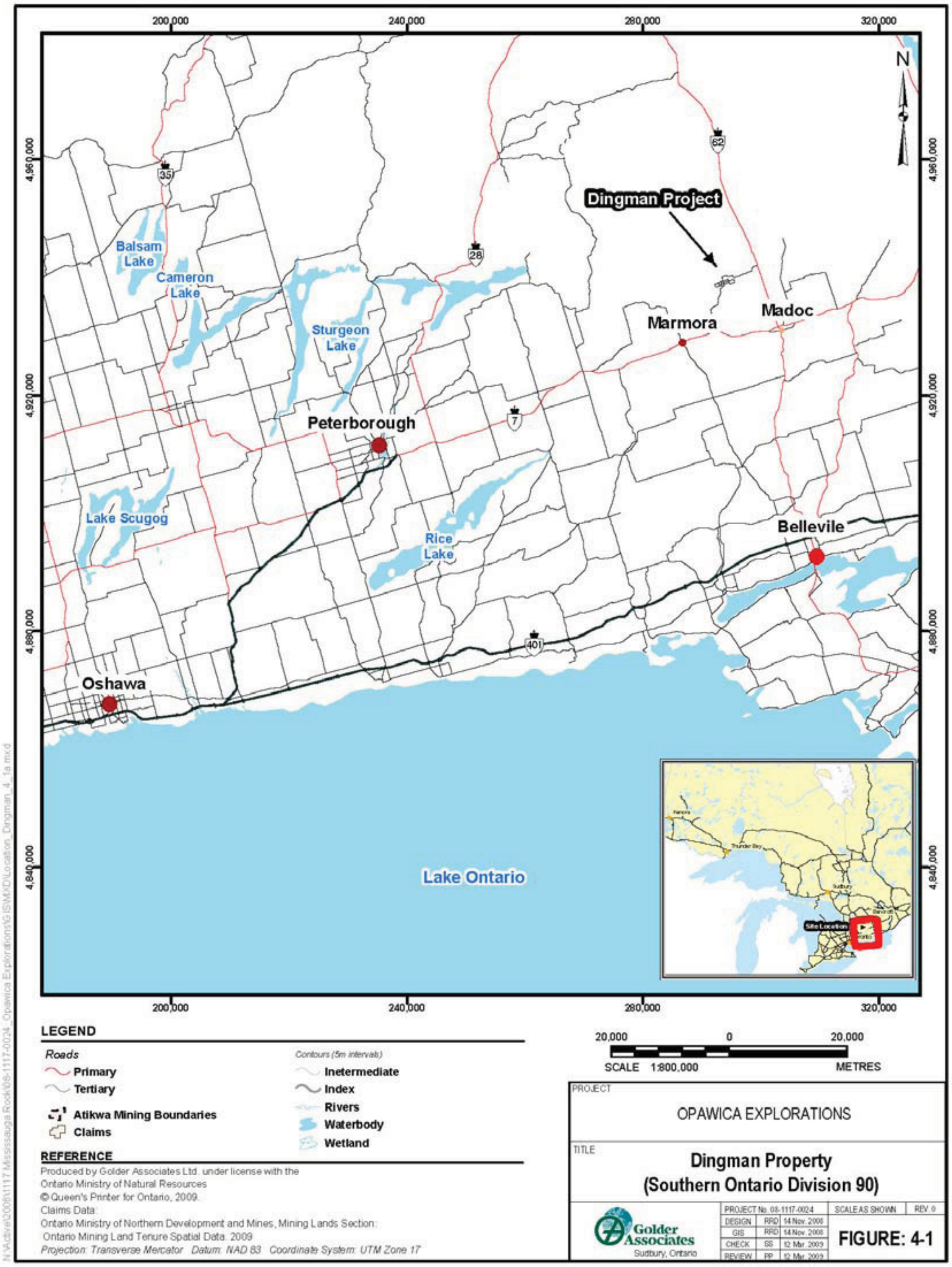
4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

This section is an excerpt from the Dingman Technical Report titled “Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada”, by Palmer et al. dated March 25, 2009.

The Dingman Property is located on the boundary between Madoc and Marmora Townships in Hastings County, southeastern Ontario, approximately 175 km northeast of Toronto, Ontario and 55 km north of Belleville, Ontario, shown on Figure 4-1. The centre of the property is at latitude 44º 34' 30" North and longitude 77º 35' 47" West and UTM Zone 18 NAD 83 coordinates 293859E and 4939029N. The property is situated on claim maps G-1269 (Madoc Township) and G-1270 (Marmora Township) and on NTS map sheet 31 C/12.

FIGURE 4-1: LOCATION MAP (PALMER ET AL., 2009)



4.2 Property Description

The Dingman Property consists of eight staked mining claims which together encompass an area of 200.6 ha. The mining claims are contiguous and form a rectangular block that is 3.30 km in an east-northeast, west-southwest direction by 0.62 km in a north-northwest, south-southeast direction (Figure 4-2). Madoc and Marmora Townships have been surveyed into lots; the staked mining claims consist of one quarter or one half of a lot. The Dingman Property has not been surveyed by Opawica. A detailed description of the property with claim numbers, specific claim location, claim size, claim recording dates, claim expiry dates, work in reserve, and work required is included in Table 4-1.

Six of the eight mining claims were recorded with the MNDM on November 26, 2004, and sufficient assessment work has been filed to keep these claims in good standing until November 26, 2012, or November 26, 2013. One of the mining claims (consisting of 2 claim units) was recorded with the MNDM on April 12, 2005, and sufficient assessment work has been filed to keep this claim in good standing until April 12, 2013. One of the mining claims (consisting of 2 claim units) was recorded with the MNDM on September 27, 2006, and sufficient assessment work has been filed to keep this claim in good standing until September 27, 2013. The mining claims may be renewed for a further year with the MNDM by either assigning the assessment work credits in reserve, or by filing sufficient additional exploration work expenditures totalling $400 per claim unit per year.

4.3 Property Agreements

4.3.1 Mineral Rights

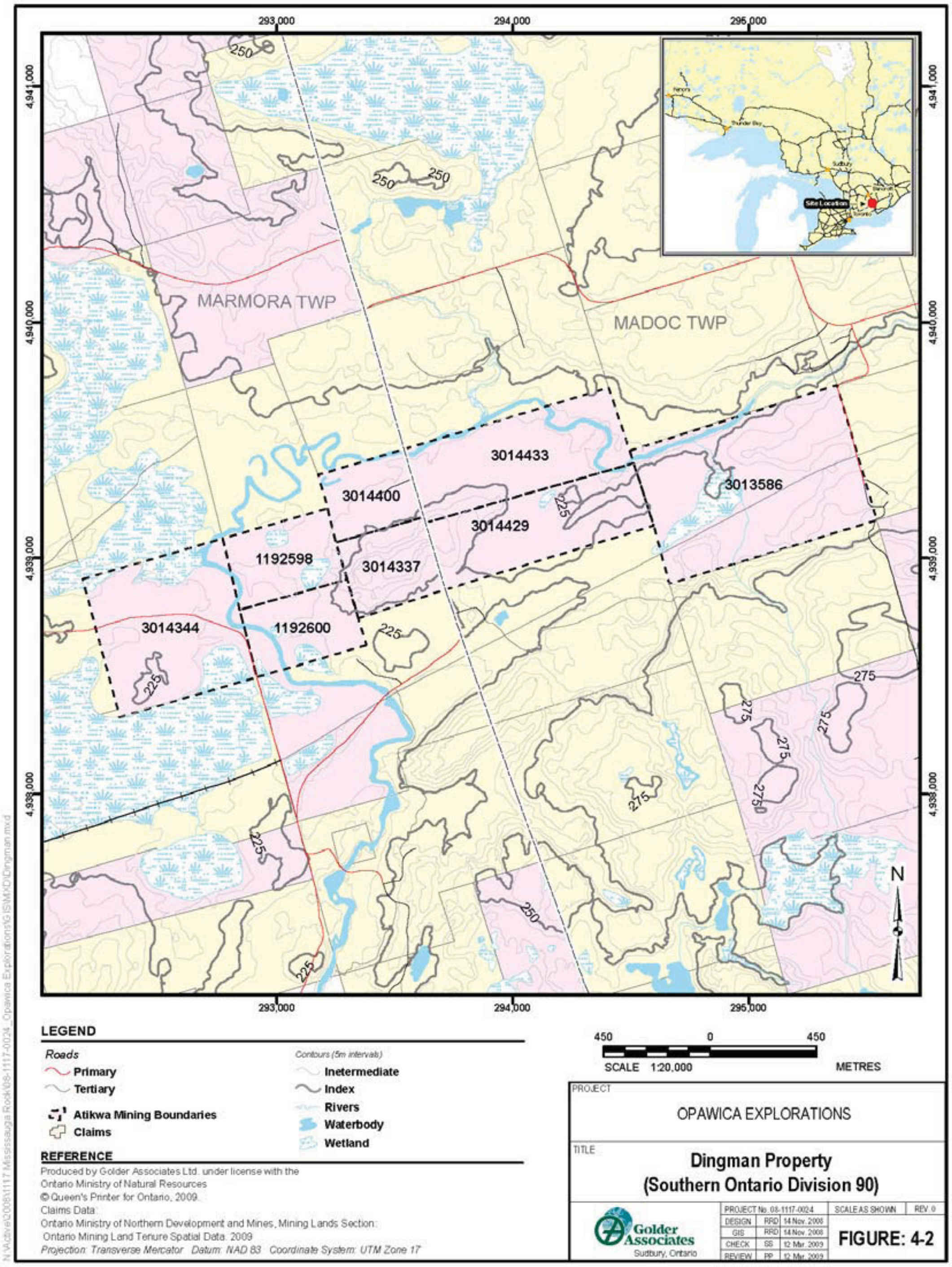
The Dingman Property consists of eight unpatented staked claims in Marmora and Madoc Townships, Ontario.

The Dingman Property was optioned by Opawica pursuant to an option agreement with Baird and Neczkar, the Vendors dated August 31, 2006.

On or about December 10, 2007, Opawica paid all final consideration due to the Vendors and, as a result, had earned its 100% interest in the mineral rights to the staked Dingman claims. As at January 2, 2008, the claims were transferred into Opawica's name and the claims are in good standing.

The Dingman Property is subject to a 2% Net Smelter Return royalty of which half of this royalty may be purchased by Opawica at any time for $250,000.

FIGURE 4-2: DINGMAN PROPERTY (SOUTHERN ONTARIO DIVISION 90) (PALMER ET AL., 2009)



Upon the completion of a fully funded positive bankable feasibility, Opawica must pay $250,000 in cash or, solely at Opawica's option, pay $125,000 in cash and $125,000 in common shares of Opawica using the trading price of the Opawica's common shares on the TSX at the time the payment is due.

In 2008, work credits were filed on the claims with the MNDM (Ontario) which will extend the expiry date of the eight claims to between November 26, 2012 and November 26, 2013. A total of approximately $706,000 in work credits have been filed in Reserve which, when applied, could extend the above expiry dates by five years.

4.3.2 Surface Rights - Dingman

The surface rights that cover the Dingman claims are used or optioned by Opawica pursuant to two separate surface rights or access agreements referred to herein as the Eastern Surface Rights agreement and the Western Surface Rights Access agreement.

Eastern Surface Rights: Opawica has optioned approximately 169 acres which covers the eastern half of the Dingman granitic stock as well as covers surface rights north, east and south of the Dingman granitic stock.

Opawica entered into an option agreement on these surface rights on September 5, 2006. The remaining payments on this option consist of a payment of $16,000 due on September 5, 2009 and $32,000 on or before September 5, 2010 and Opawica has agreed to pay $500 per drill hole drilled on the property to the Vendor. Thereafter, Opawica may purchase the surface rights for three times the appraised surface rights land value but, in any event, not to exceed $1,500,000 on or before September 5, 2011.

Western Surface Access Rights: The surface rights that cover the western half of the Dingman claims and granitic stock is owned by a separate local Vendor. On September 20, 2007, Opawica entered into an agreement with this local Vendor whereby Opawica has been granted access to conduct drilling activities on the Dingman Property. This agreement has now been replaced with a new agreement effective February 20, 2009, whereby Opawica may drill up to 10 holes on or before December 31, 2009 and Opawica may conduct baseline studies and environmental permitting studies on the Dingman Property on or before August 21, 2010. As at February 26, 2009, drilling has commenced under this new agreement.

Under the new agreement, Opawica must pay $10,000 cash on signing, $1,500 per hole drilled and pay 250,000 shares of Opawica. The parties have now entered into negotiations for Opawica to option approximately 100 acres of the Vendor's land for purchase over the next 2 years.

4.4 Environmental Requirements and Permits

Opawica has stated to Shaft & Tunnel that, to the best of their knowledge, there are no known environmental issues or liabilities on the Dingman Property and all the proper permits required to conduct exploration activities on the property in 2009 and currently have been obtained.

The claims listed in the following table were historically represented as claims 950277, 950276, 950274, 950275 (Marmora Twp/Area G-1270) and 950290, 950293, 950291, 950292 (Madoc Twp/Area G-1269). All claims are in Southern Ontario Division 90 and listed under Opawica on the MNDM mining claim database.

TABLE 4-1: LIST OF MINING CLAIMS, DINGMAN GOLD PROPERTY

Description	Number	Area (Ha)	Staked	Recorded	Expires	Grouping
E1/2 Lot 19 Con 10 (1)	3014344	37	12/04/2005	17/03/2005	12/04/2013	G-1270
NW1/4 Lot 19 Con 10 (2)	1192598	14.5	19/11/2004	26/11/2004	26/11/2012	G-1270
S1/2 Lot 19 Con 11 (1)	1192600	14	19/11/2004	26/11/2004	26/11/2012	G-1270
N1/2 of E1/2 Lot 19 Con 11 (1)	3014400	11.4	18/11/2004	26/11/2004	26/11/2013	G-1270
SE1/4 Lot 19 Con 11 (1)	3014337	12.8	18/11/2004	26/11/2004	26/11/2013	G-1270
NW1/4 Lot 19 Con 1 (1)	3014433	30.2	20/11/2004	26/11/2004	26/11/2013	G-1269
SW1/4 Lot 19 Con 1 (1)	3014429	26.4	18/11/2004	26/11/2004	26/11/2012	G-1269
E1/2 Lot 19 Con 1 (2)	3013586	54.3	16/09/2006	27/09/2006	27/09/2013	G-1269

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

This section is an excerpt from the Dingman Technical Report titled "Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada", by Palmer et al., dated March 25, 2009.

5.1 Accessibility

The Dingman Property is located 55 km north of the city of Belleville, Ontario, 13 km northwest of the village of Madoc, Ontario and 12 km northeast of the village of Marmora, Ontario (Figure 4-1). The property can be accessed via Madoc by taking Highway 62 north and then paved County Road 11 west and south, or alternatively via Marmora by taking Highway 7 east and then paved County Road 11 north through the village of Deloro.

5.2 Climate

The climate of the area is continental, with warm moderately humid summers and moderately cold winters. Summer (June to August) temperatures average approximately 18ºC and range between 10ºC and 30ºC. Winter (December to March) temperatures average approximately -5 ºC and range between 0ºC and -15ºC. Annual precipitation is approximately 830 mm with 80% being from rainfall and 20% being from snowfall. Approximately 1.6 m of snow falls in the winter. The ground is snow covered from generally early December to the middle of March.

5.3 Local Resources and Infrastructure

Agriculture, forestry and mining are the main industries in the area. The Madoc-Marmora area has a long mining history and a number of quarries are presently active. Until recently, Canada Talc Ltd., a division of Dynatec Mineral Products, produced talc, dolomite and barite from ore mined at the Henderson Mine near Madoc.

The villages of Marmora, with a population of approximately 1,600, and Madoc, with a population of approximately 1,400, are the nearest population centres in the area. Both communities offer a variety of accommodation, supplies and services to the surrounding area. A major hydroelectric power line is located approximately 1 km north of the Dingman Property.

5.4 Physiography

The Dingman Property is located near the southern edge of the Canadian Shield, approximately 15 km north of the St. Lawrence Lowlands. The elevation of the property is between 215 and 245 m above sea level (masl). The property is situated within an area of beef cattle and dairy farms and is characterized by low to moderate relief.

The topography is characterized by an east-northeast, west-southwest trending granite outcrop ridge located in the central part of the property. This feature has a length of approximately 800 m, a width of 150 m, and up to 25 m of relief. Bedrock surrounding the ridge is Precambrian marble which supports a clay soil up to 15 m in thickness. The Moira River runs along the north boundary of the property and then turns south to flow across the western part of the property (Figure 4-2). The granite ridge hosts a sparse mixed forest consisting of maple, oak, spruce and cedar.

6.0 HISTORY

This section is an excerpt from the Dingman Technical Report titled “Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada”, by Palmer et al. dated March 25, 2009.

6.1 General

Southeastern Ontario has a mining history beginning in 1866 with the discovery of gold at the Richardson Farm near Eldorado, located 6 km east of the Dingman Property. Gold exploration and development continued in southeastern Ontario into the next century, with a total of 38,592 ounces of gold produced from 14 small mines operating primarily during the period of 1892 to 1908 (Table 15 in Sangster et al., 2007).

There is no record of work conducted on the Dingman Property prior to 1985, although several small pits or trenches sunk on quartz veins in granite likely date from the late 1800s. In 1985, Mark Dingman carried out a gold exploration program in the area, consisting of prospecting, geological mapping and grab sampling. The property was staked in 1985 by Dingman following the discovery of anomalous gold values in a small granite stock (Dingman granite).

In 1986, Noranda optioned the Dingman Property and carried out exploration work between 1986 and 1988, consisting of geological mapping, ground geophysical surveys, soil geochemical surveys, minor stripping and trenching, channel sampling, diamond drilling (38 holes totalling 5024.94 m), thin section work, metallurgical test work and baseline environmental studies. A wide zone of gold mineralization was outlined to a depth of approximately 150 m, hosted within the western portion of the Dingman granite. The property was subsequently transferred to Hemlo Gold Mines Inc. which was acquired in 1996 by Battle Mountain.

In 1997, Deloro, formerly 698366 Alberta Ltd. and Rajong, purchased the Dingman Property from Battle Mountain. In 1997, Deloro carried out a program of re-logging and check sampling of Noranda drill core and then completed 2,061 m of diamond drilling in 14 holes on the property. The drill holes were designed to test the grade, continuity and lateral extent of the gold mineralization reported by Noranda. The Deloro drill holes encountered gold mineralization within the Dingman granite with similar grades and over similar widths as reported by Noranda.

The mining claims covering the Dingman prospect were staked and recorded between November 2004 and September 2006 by Baird of Toronto, Ontario and Neczkar of Etobicoke, Ontario. In 2005, Neczkar submitted 17 reject samples of Deloro drill core to SGS Lakefield Research Limited for metallurgical test work. In 2006, Baird and Neczkar carried out a soil geochemical survey and VLF-EM geophysical survey in the eastern part of the property.

Opawica optioned the property in 2006 and, in 2007, completed 4,726 m of diamond drilling in 20 holes. The Opawica drill program was designed to verify the Noranda and Deloro results and delineate the gold mineralization within the Dingman granite to a level sufficient to undertake a mineral resource estimate in accordance with the requirements of NI 43-101 and the CIM Definition Standards on Mineral Reserves and Mineral Resources.

6.2 Historical Resource Estimates

There are several historical estimates of mineral resources for the Dingman Property completed by Noranda in 1988 and 1989, one historical estimate completed by Rajong in 1997, and one historical estimate completed for Deloro in 1998 by Barnes. **These mineral resource estimates were considered by Shaft & Tunnel to be historical estimates as a qualified person has not done sufficient work to classify the historical estimates as current mineral resources and Opawica is not treating the historical estimates as current mineral resources and the historical estimates should not be relied upon and have been provided solely for historical background on the property.**

A preliminary "reserve" calculation by Noranda, dated February 23, 1988, outlined "probable geological reserves" of 4.21 million short tons grading 0.043 oz/ton Au, including a higher-grade section of 1.62 million short tons grading 0.073 oz/ton Au (Noranda memo in Appendix II of King, 1988). The estimate was based on the results of the diamond drilling completed by Noranda between sections 225W and 125E, to a maximum vertical depth of 150 m. A cross-sectional method of calculation was used, assuming the area of influence of an averaged assay interval in any one hole is a function of the distance to any other adjacent hole (mid-point between the holes). An average specific gravity value of 2.73 g/cm^3, from 19 altered granite samples, was used to calculate a factor of 11.74 cubic feet per ton. A factor of 12.0 cubic feet per ton was used for the "reserve calculations".

No cut-off grade or cutting of high grade assays are stated in the February 23, 1988 Noranda memo. The resource estimate uses a category different than those set out by the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by the CIM council on November 14, 2004. The 1988 estimate is not compliant with NI 43-101 and should be considered a "historical estimate" in accordance with NI 43-101.

The latest Noranda estimate, completed in 1989, resulted in a "mineral inventory" of an "in situ Mineral Resource" for the entire Dingman granite stock of 4.05 million short tons grading 0.048 oz/ton Au, based on the concept of mining the deposit by open pit (Huska, 1989). The estimate is based on 5 m bench composites and a cut-off grade of 0.02 oz/ton Au. A cross-sectional method of calculation was used, with most of the resource located between sections 100E and 200W, and from surface to a maximum vertical depth of 100 m. This estimate should be considered a "historical estimate" in accordance with NI 43-101. The estimate is presented in this report only to complete the historical background on the project.

Sometime prior to acquiring the Dingman Property from Battle Mountain, presumably in 1996 or 1997, Rajong estimated a "resource" of 5.64 million short tons grading 0.034 oz/ton Au based on the concept of mining the bulk of the deposit by open pit and processing by heap leaching (Roscoe, 1997). The Rajong resource is located within a preliminary pit design, called the base case pit, and the base case resources do not extend west of section 125W (Roscoe, 1997). The estimation method used by Rajong is not stated in Roscoe (1997). High grade assays were not cut. A cut-off grade of 0.01 oz/ton Au was used by Rajong based on open pit mining the entire mineralized zone in the Dingman granite stock using a quarry style approach, rather than selectively mining higher grade portions of the mineralized zone (Roscoe, 1997). The stripping ratio is 0.64:1 and the pit bottom is a maximum of 120 m below surface (Roscoe, 1997). RPA considered that the Rajong resource within the base case pit could be classified as an Indicated Resource (Roscoe, 1997).

Rajong estimated “additional resources” of 1.42 million short tons grading 0.035 oz/ton Au within modified pit designs from the base case pit, including extensions further west to section 200W and 20 m below the base case pit (Roscoe, 1997). RPA considered that the additional material could be classified as Inferred Resources, with excellent potential for increasing resources at depth (Roscoe, 1997). Rajong’s resource estimates should be considered a “historical estimate” in accordance with NI 43-101. The estimate is presented in this report only to complete the historical background on the project.

In 1998, Deloro commissioned Barnes to prepare a “resource model” of the Dingman deposit and estimate the potentially mineable portion of that resource (Barnes, 1998). The data used for the resource estimate included 36 diamond drill holes with a total length of approximately 4,947 m and 82 surface channels with a total length of 536.1 m. The diamond drill holes include 22 holes completed by Noranda and 14 holes completed by Deloro at approximately 25 m section spacing, between sections 200W and 100E. The drilling data was received from Deloro as an electronic file and was not audited by Barnes. The channel data was digitized by Barnes from plan maps provided by Deloro.

The drill hole data consisted of a number of quantitative variables, including interval length and gold values, with one rock type for each interval. The drill hole data also consisted of qualitative variables for each interval, including texture, grain size, sericite alteration index, percentage quartz veining, pink feldspar alteration index and total sulphide percentage. The channel sample data consisted of only gold assays. Barnes calculated statistics for the raw drill hole data, including gold statistics by rock type. Barnes decided to estimate the resource only in the intrusive, based on the sparse mineralization observed in the sedimentary rocks. A 3D model of the intrusive domain was created based on a cross-sectional interpretation and checked on plan.

The drill hole and channel data was composited into 6 m intervals and then declustered to provide more representative composite statistics. All the variables for each composite, along with the gold data, were examined to define any relationships that could allow a better definition of the gold mineralization (Barnes, 1998). Based on examination of the histograms, Barnes concluded that the distribution of the gold mineralization was bimodal, with one population that is generally less than 2 g/t Au, and a second population that is greater than 2.5 g/t Au. Barnes observed a moderate correlation between gold and sericite and total sulphide, and a weak correlation between gold and quartz veining and pink feldspar.

The declustered composite data was used to set indicator thresholds to be used in calculating variograms and indicator kriging was used as the method in estimating the resource. The block size selected for the model was 10 m by 10 m in plan and 6 m vertically. The model dimensions were 350W to 250E, 200S to 200N, and from surface to a vertical depth of 250 m. A specific gravity of 2.73 g/cm^3 was used for all material within the model. The “Global Measured and Indicated Resource” estimated by Barnes for various cut-off grades is shown in Table 6-1.

TABLE 6-1: HISTORICAL ESTIMATE OF "GLOBAL MEASURED AND INDICATED RESOURCE" FROM BARNES (1998)

Cut-off Grade Au (g/t)	Million Tonnes	Au (g/t)
0.5	5.62	1.30
1.0	2.51	2.05
1.5	1.35	2.74
2.0	0.73	3.58
2.5	0.41	4.65

Barnes estimated the "potentially economic portion" of the "Measured + Indicated Resource" based on the concept of mining the deposit by open pit and processing by heap leaching. At a US $400/oz gold price, the breakeven cut-off grade is 0.44 g/t Au and, at a US $350/oz gold price, the breakeven cut-off grade is 0.50 g/t Au. The results were a "Measured + Indicated Mineable Resource" of 6.5 million tonnes grading 0.99 g/t Au at a US $400/oz gold price and 5.6 million tonnes grading 1.05 g/t Au at a US $350/oz gold price.

The Deloro resource estimate completed by Barnes uses categories different than those set out by the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by the CIM council on November 14, 2004. The estimates are not compliant with NI 43-101 and should be considered "historical estimates" in accordance with NI 43-101. The results of the statistical analysis of the assay data and the correlation between gold and sericite, sulphides and quartz veining are considered relevant to the geological interpretation of the gold mineralization on the Dingman Property, discussed in Section 9 of this report. Verification of the digital Dingman diamond drill hole and channel sample database received from Barnes is described in Section 14.

Again, all the mineral resource estimates described in Section 6.2 were considered by Shaft & Tunnel to be historical estimates as a qualified person has not done sufficient work to classify the historical estimates as current mineral resources and Opawica is not treating the historical estimates as current mineral resources and the historical estimates should not be relied upon and have been provided solely for historical background on the property.

7.0 GEOLOGICAL SETTING

This section is an excerpt from the Dingman Technical Report titled "Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada", by Palmer et al., dated March 25, 2009.

7.1 Regional and Local Geology

The Dingman Property is located within the southern portion of the Central Metasedimentary Belt in the Grenville Province of the Canadian Shield (Figure 7-1). The Grenville Province is a complex northeast-southwest trending, orogenic belt of circa 1.1 billion years in age that truncates several older geologic provinces (Easton, 1992). The Grenville Province is subdivided in Ontario, from northwest to southeast, into the Grenville Front Tectonic Zone, the Central Gneiss Belt, the Central Metasedimentary Belt Boundary Zone and the Central Metasedimentary Belt (Figure 7-1).

The Central Metasedimentary Belt in which the Dingman Property is situated is a major middle Proterozoic accumulation of volcanic and sedimentary rocks that has been intruded by compositionally diverse plutonic rocks and the entire succession has been metamorphosed at grades varying from greenschist to granulite facies. The Central Metasedimentary Belt has been subdivided into several lithotectonic terranes based on differences in rock type, geologic and structural history, and ages of plutonism and metamorphism (Figure 7-2).

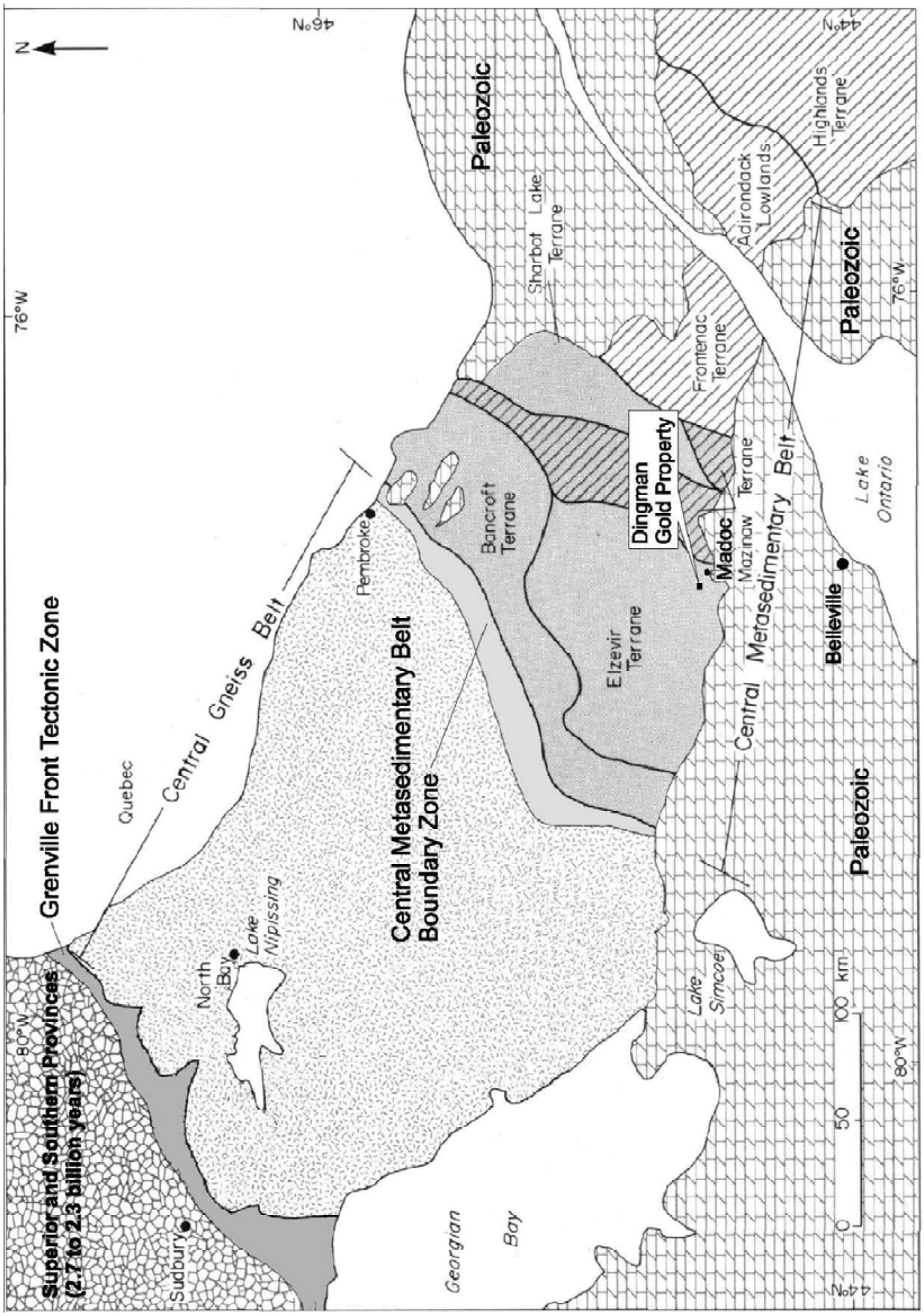
The Elzevir Terrane in which the Dingman Property is situated includes the classical Grenville Supergroup (a term dating back to Logan, 1863), and is characterized by volcanism and sedimentation between 1.3 and 1.25 billion years ago, followed by plutonism, metamorphism and deformation at 1.25 to 1.23 billion years ago and at 1.13 to 1.07 billion years ago (Easton, 1992). Large areas of the south-central and eastern Elzevir Terrane, including the Dingman Property, are preserved at greenschist facies; this area has been historically referred to as the "Hastings Metamorphic Low".

The Elzevir Terrane has been further divided into three domains: the Harvey-Cardiff Arch; and the Belmont and Grimsthorpe domains (Easton, 1992) (Figure 7-2). The Dingman Property is located within the Belmont domain. The Belmont domain consists of carbonate and clastic sedimentary rocks of the Mayo Group and mafic to felsic volcanic rocks of the Hermon Group.

A diverse suite of mafic to felsic intrusive rocks is present in the Elzevir Terrane, based primarily on the work of Lumbers et al. (1990) and summarized in Easton (1992). The Dingman Property is situated between the Deloro granite stock (Deloro granite) to the south and the Gawley Creek syenite stock to the north (Figure 7-3).

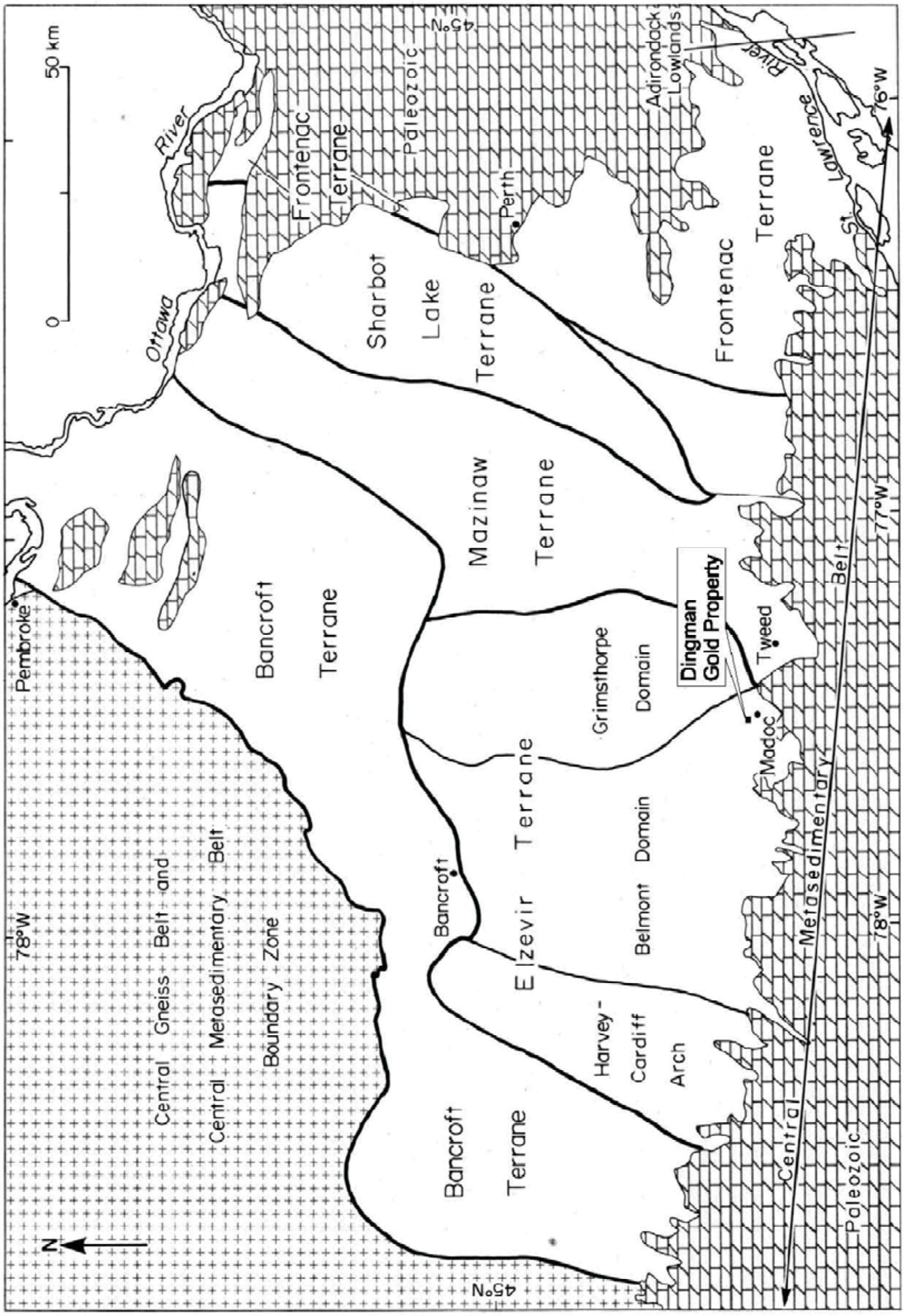
The Deloro granite, located just south of the property, is a pink monzogranite stock which occupies approximately 35 square kilometres in south-central Marmora and Madoc Townships. A heterogeneous zone of gabbro, diorite, syenite and granite phases occurs along the west margin of the intrusion Easton (1992). The Deloro granite has been assigned to the circa 1.24 to 1.25 billion year old alaskitic "Methuen" suite and is the most widely distributed and the second most voluminous plutonic suite within the Central Metasedimentary Belt (Lumbers et al., 1990).

FIGURE 7-1: DIVISIONS OF THE GRENVILLE PROVINCE IN ONTARIO



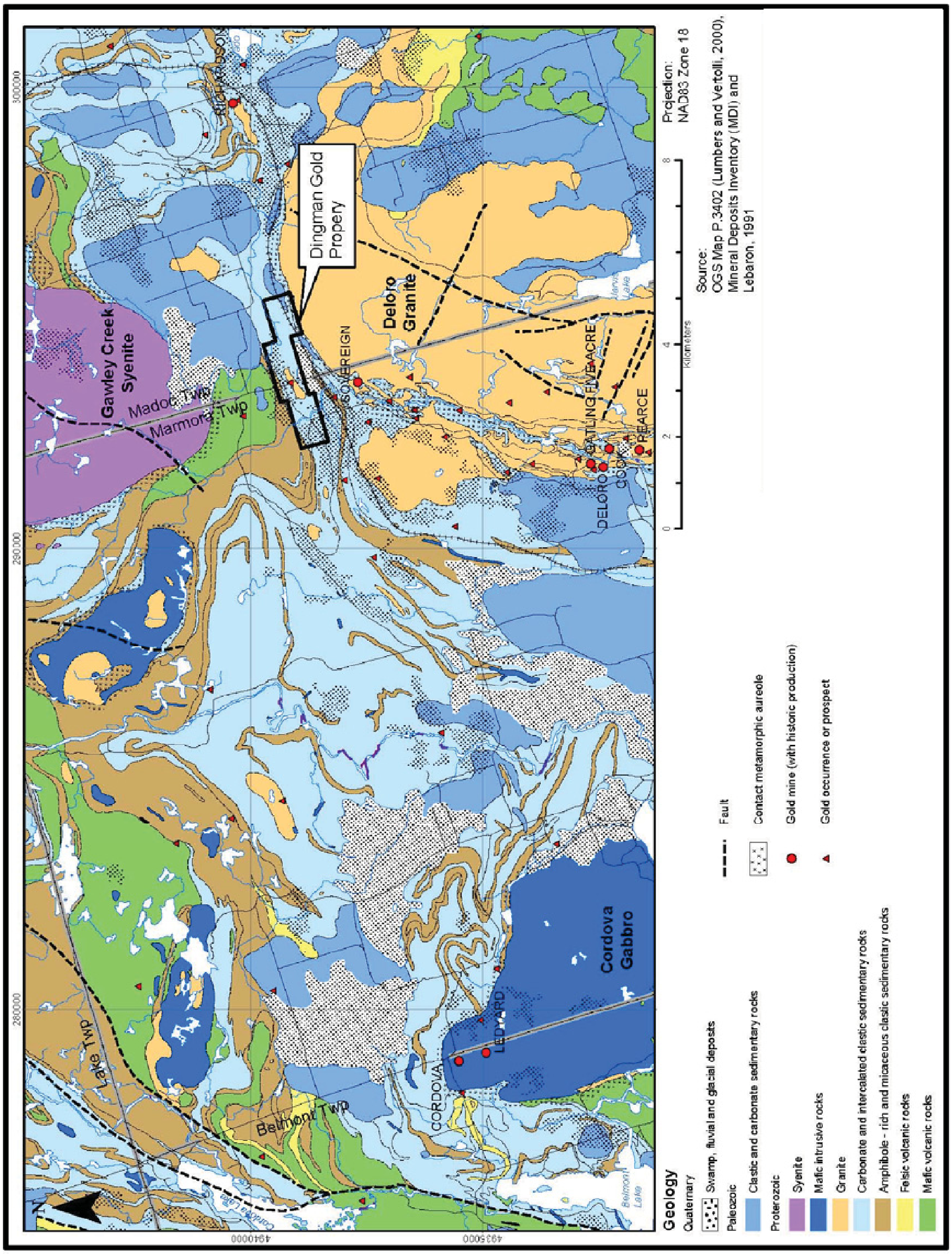
Map Source: Fig. 19.2, Geology of Ontario (Easton 1992)

FIGURE 7-2: LITHOTECTONIC TERRANES AND DOMAINS OF THE METASEDIMENTARY BELT IN ONTARIO



Map Source: Fig. 19.51a, Geology of Ontario (Easton 1992)

FIGURE 7-3: REGIONAL GEOLOGY



Map Source: OGS Map 3402 (Lumbers and Vertolli, 2000), Mineral Deposits Inventory (MDI) and LeBaron, 1991

7.2 Property Geology

7.2.1 Lithology

The Dingman Property is underlain by carbonate and intercalated clastic sedimentary rocks of the Marmora Formation of the Mayo Group. The sediments are primarily fine-grained banded and well foliated calcitic marbles intercalated with calcareous siltstone to sandstone layers.

The sediments have been intruded by a small elongate granite stock that is informally known as the Dingman granite. The Dingman granite is exposed as an oval-shaped outcrop hill having a surface expression of approximately 800 m by 150 m with a long axis that trends east-northeast, west-southwest at an azimuth of 060° (LeBaron, 1986). The outcrop hill appears to approximate the true dimension of the granite stock, based upon both the observed surface granite-sediment contacts and diamond drill results.

In drill core, the granite-sediment contact is generally sharp; local narrow intervals of granite (typically less than 1 m in width) within the sediments near the contact are interpreted as small apophases of the granite stock. A number of slightly wider intervals of granite (ranging between 1 m and 10 m in width) within the sediments near the contact may also be apophases of the granite stock. Minor syenite occurs as narrow 1-2 m wide dikes in the sediments (LeBaron, 1986). The syenite dikes are fine-grained, consisting of less than 10% quartz and 90% pink to buff-coloured feldspar.

The Dingman granite dips to the north-northwest at approximately 50-60° with some variations in dip. The north contact of the granite appears to have a slightly steeper dip near surface, while the southern contact of the granite is generally less steep near surface. The depth extent and plunge of the Dingman granite is not known; it has been intersected by diamond drilling at a vertical depth of 300 m.

A large portion of the Dingman granite has been variably altered and deformed. Relatively unaltered sections of granite are grey, medium to coarse-grained, consisting of grey quartz and white to buff feldspar phenocrysts up to 5 cm in size in a fine-grained groundmass. Major and trace element geochemical data suggest the Dingman granite is actually a syenogranite with an A-type granite affinity (Easton et al., 2007).

Similarities in composition, texture and alteration suggest that the Dingman granite may be a small satellite body of the larger neighbouring Deloro granite, similar to other small stocks along or close to its margin (Hewitt, 1968). However, a recently acquired U/Pb zircon age of 1.218 billion years by Easton et al. (2007) suggests the Dingman granite is approximately 20 million years younger than the age of the Deloro granite (1.241 billion years: van Breemen and Davidson, 1988). Within the Central Metasedimentary Belt, there are very few circa 1.22 billion U/Pb ages; the Dingman granite intrusion is now interpreted by Easton et al. (2007) to represent a distinct magmatic and/or deformational episode related to a gold mineralizing event.

Bedding and foliation in the sedimentary rocks on the property have an overall east-northeast, west-southwest strike at an azimuth of 060° and dip north at approximately 70-80°, although variations in both strike and dip occur in close proximity to the Dingman granite. Within the Dingman granite, the dominant foliation also trends east-northeast, west-southwest at an azimuth of 060° and dips north at approximately 70-80°.

The bedding and foliation in the sedimentary rocks is commonly contorted and crenulated, observed both in outcrop (LeBaron, 1986) and in core close to the granite-sediment contact. The granite-sediment contact is commonly sheared and schistose with locally abundant quartz veining and hematite, alkali feldspar or iron carbonate alteration.

A second strong fracture-shear foliation having a northeasterly strike at an azimuth of 025° and dipping west at approximately 80-90° was observed by LeBaron (1986) to occur as: 1) drag folding and quartz-filled shears in marble; 2) local shears within the granite, with the feldspars altered to sericite schist; 3) a dominant fracture-joint set in the granite; and 4) a series of cross faults with sinistral displacements of the granite on the order of about 20 m at a number of places along its length. In drill core, cross faults occurring within the granite are described as fine-grained, dark green to black, well foliated to sheared or crenulated, consisting of biotite, chlorite and calcite with disseminated pyrite. The cross faults have been logged as a variety of rock types, consisting of mafic, lamprophyre or diorite dykes, and as a variety of fault, shear or contact zones. Where the cross faults transect the granite stock, they are typically between 0.5 and 3 m or an average of 1 m in width. Within the sediments, the cross faults are often described as “contact zones”, consisting of chaotic zones of sheared and alkali feldspar or hematite altered sediments and granite containing disseminated pyrite. Where the cross faults transect the sediments, they are typically between 5 m and 20 m in width.

7.2.2 Alteration

The locations of the alteration zones within the granite mapped by LeBaron (1986) are described below:

- Weakly altered: well preserved coarse quartz and feldspar grains, buff colour with little or no hematite stain (G1 type).
- Weak to moderate alteration: distinct feldspar grains with minor interstitial chlorite-sericite alteration; feldspars pale red (hematite stain) (G1-G2 type).
- Moderate alteration: weak foliation with stretching of feldspar between quartz grains; moderate sericite and hematite alteration of feldspars (G2 type).
- Moderate to strong alteration: banded texture due to extreme stretching and moderate sericite alteration of feldspars; bands are buff to green and locally red (hematitic) (G2-G3 type).
- Strong alteration and foliation with feldspars totally altered to pale green muscovite/sericite with coarse, round blue-grey quartz grains in a green mica schist; locally with strong hematite staining and trace to 3% disseminated pyrite (G3 type).

In drill core, progressive alteration of the granite consists of an increase in sericite as bands and shears, destruction of feldspar phenocrysts, with quartz phenocrysts or grains becoming more prominent. Zones of pink to red alkali feldspar occur sporadically in areas of weak to strong sericite alteration. The alteration scheme of LeBaron (1986) was used in core logging by Noranda, modified by Deloro and used in the Deloro and Opawica drill programs.

In thin section, alteration of the granite has resulted in the destruction of the primary textures, introduction of veinlets, veins and irregular patches of quartz +/- alkali feldspar +/- carbonate +/- fluorite, replacement of feldspar by sericite and deposition of fine to medium-grained pyrite and alkali feldspar (Clemson, 1988). The altered granite commonly contains 1 to 4 mm augen-shaped quartz grains which consist of strained relict quartz phenocrysts. Alteration is described as particularly intense where there is increased veining and shearing.

7.2.3 Veining and Mineralization

In outcrop, quartz veining and disseminated pyrite are more common in the strongly altered sections of granite (LeBaron, 1986). The quartz veins are generally 1 to 3 cm wide, but may vary up to 30 cm, with hematitic vugs and fractures. Several quartz vein sets occur within the granite; the dominant sets are: 1) flat veins trending northwest and dipping 5 to 20° to the northeast; 2) veins parallel to the primary foliation trending at azimuth 060°; and 3) veins parallel to the second fracture-shear foliation trending at azimuth 025° with a near vertical dip (LeBaron, 1986). Several other fracture orientations also locally contain quartz veins. The quartz veins are described by LeBaron (1986) as irregular, discontinuous, or pod-like, broken by shearing or boudinage.

In drill core, the quartz veins vary in orientation from foliation parallel to irregular. The quartz veins consist of narrow, generally less than 3 cm, segregations or stringers of quartz +/- carbonate +/- fluorite +/- pyrite +/- hematite +/- chalcopyrite +/- sphalerite +/- galena +/- rare visible gold. Pyrite is the dominant sulphide and occurs as fine to coarse-grained, disseminated and as stringers both within the granite and along the contacts of the quartz veins and stringers. In general, increased concentrations of quartz veining and/or pyrite mineralization occur within zones of increased sericite alteration and foliation intensity.

In thin section, the veins are composed of quartz, alkali feldspar, carbonate, pyrite, chalcopyrite, sphalerite, galena, plus minor fluorite, muscovite, arsenopyrite, native gold, hematite, ilmenite, molybdenite, Pb-sulfosalt, tetrahedrite, pyrrhotite and covellite (Clemson, 1988). Coarse-grained alkali feldspar commonly occurs along the vein margins. Vein alkali feldspar is commonly intergrown with and cut by later interstitial quartz and carbonate. Both carbonate and fluorite appear to be late in the vein paragenesis. Much of the primary vein fabric is disrupted or destroyed by strong shearing.

In thin section, sulphide mineralization consists of irregular stringers and disseminations of coarse-grained base metal sulphides intergrown with gangue (Clemson, 1988). The vein pyrite is coarse-grained, generally free of inclusions and forms aggregates of partially recrystallized grains. Recrystallization of pyrite has resulted in small grains of galena, chalcopyrite and native gold being trapped along former pyrite grain boundaries. Chalcopyrite, galena and sphalerite are locally remobilized along fractures, resulting in a stringer-like style of mineralization. Arsenopyrite was observed in both veins and altered wall rocks, but is relatively rare (Clemson, 1988).

7.2.4 Gold Mineralogy

A total of 27 of the 51 mineralized core samples contained microscopically visible native gold (Clemson, 1988). A total of 675 grains of native gold were observed, ranging in size from less than 1 micron up to 117 microns, averaging 9 microns. Native gold occurs in the following modes: 1) gold occurring along mainly vein quartz boundaries, but also mica grain boundaries in the altered wall rock (68% by volume); 2) gold occurring along the outside margin of pyrite in contact with vein quartz (14% by volume); 3) gold occurring along fractures within pyrite or along recrystallized pyrite-pyrite contacts (12% by volume); and 4) gold locked within pyrite grains (6% by volume).

Native gold tends to be more abundant when the pyrite content of the vein is high, but native gold was still observed in samples containing only trace sulphides (Clemson, 1988). Native gold occurs frequently in the micaceous wall rock adjacent to veins where it was observed primarily along muscovite grain contacts and within disseminated grains of pyrite.

7.2.5 Geochemical Data

The major oxide, base metal and trace element geochemical data from Noranda (59 samples) and Opawica (11 samples) drill core support the alteration and mineralization observed in outcrop, core and thin section. Gold mineralization in the Dingman granite appears strongly correlated with sodium depletion and potassium-sulphur enrichment in the geochemical data, reflected in the moderate to strong sericite +/- alkali feldspar alteration and increased sulphide mineralization observed in outcrop, core and thin section.

8.0 DEPOSIT TYPE

This section is an excerpt from the Dingman Technical Report titled "Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada", by Palmer et al. dated March 25, 2009.

8.1 Gold

Gold mineralization on the Dingman Property occurs as the hydrothermal quartz-carbonate vein gold subtype 15.2 of Robert (1996). This subtype of gold deposits consists of simple to complex quartz-carbonate vein systems associated with shear zones and folds in deformed and metamorphosed volcanic, sedimentary, and granitoid rocks. In these deposits, gold occurs in veins or as disseminations in adjacent altered wall rocks, and is generally the only or the most significant economic commodity.

In Canada, the significant quartz-carbonate vein gold deposits occur principally in Archean and early Proterozoic greenstone belts of the Superior, Churchill, and Slave provinces, the oceanic terranes of the Canadian Cordillera, and the turbiditic Meguma terrane and the ophiolitic Baie Verte district in the Appalachians (Robert, 1996). Few significant gold deposits of any type have been discovered in the Grenville Province of Ontario, with most of the historic production coming from the Cordova (22,774 oz gold from 120,670 tons grading 0.19 oz/t) and Deloro (10,360 oz gold from 39,143 tons grading 0.26 oz/t) mines (Table 15 in Sangster et al., 2007).

Gold-bearing quartz-carbonate vein occurrences occur throughout the lower metamorphic grade parts of the Central Metasedimentary Belt (Easton and Fyon, 1992). The gold occurrences and deposits occur in clusters or camps (Table 24.6 in Easton and Fyon, 1992), comparable to the clusters or districts within the more productive Archean and early Proterozoic greenstone belts, Canadian Cordillera and Appalachians (Robert, 1996).

Figure 7-3 shows the gold mines with historic production, occurrences and prospects in the southern part of the Central Metasedimentary Belt. Gold mineralization in the Cordova area occurs as quartz-ankerite veins within shear zones in the northern part of the Cordova gabbro. In the Deloro-Malone area, gold mineralization occurs primarily as quartz-arsenopyrite veins within shear zones or fractures in the heterogeneous zone of gabbro-diorite-syenite-granite that occurs along the western margin of the Deloro granite. Gold mineralization also occurs as quartz-arsenopyrite veins within shear zones in the carbonate and clastic sedimentary rocks along the western margin of the Deloro granite. Within the shear zones, the host rocks are typically altered to carbonate-sericite-arsenopyrite schist.

The style of gold mineralization on the Dingman Property is somewhat similar to that of the Deloro-Malone area, with one important difference. The dominant sulphide at Dingman is pyrite, with rare arsenopyrite observed only in thin section, whereas arsenopyrite is generally abundant in the Deloro-Malone gold deposits and occurrences. Arsenopyrite is generally more common in sediment-hosted deposits (Robert, 1996).

Gold mineralization on the Dingman Property is typical of the significant hydrothermal quartz-carbonate vein deposits hosted by granitoid rocks in Archean and early Proterozoic rocks in Canada. Alteration, quartz veining and gold mineralization on the Dingman Property appear to have been strongly controlled by structures within the Dingman granite. Wall rock hydrothermal alteration consists of large zones of alkali metasomatism primarily in the form of sericite, and sulphidation, primarily pyrite, with lesser amounts of chalcopyrite, galena and sphalerite. The complex style of quartz veining at the Dingman Property is typical of the networks of veins and related host structures characteristic of many vein deposits. An important characteristic of a large number of vein deposits is their significant vertical extent, which exceeds 2 km in several Archean deposits.

8.2 Carbonated Metasediments

Aggregate resources on the property are bedrock and include portions of the Dingman granite and the carbonated metasediments adjacent to the Dingman granite described in Section 7.2.

Marble is the metamorphic equivalent of limestone. Limestones and other carbonate rocks are of common occurrence. The origin of most limestones is not ascertainable because we lack determinate criteria. We can define some limestones as “shell limestones” because of calcareous fossils. On the other hand, there are varieties determined by the presence of some impurity that gives them a distinctive nature, thus we have argillageous (clay), arenaceous (sand), and bituminous (organic) limestones. Marble has also been produced from limestone by the contact metamorphic action of intrusive igneous rocks. This is a likely occurrence at Dingman due to the marble sandwiched between the Dingman granite and the related Deloro Granite. The marble derived from limestone on the Dingman Property lacks impurities appears to be very pure and is supposedly formed by evaporation and precipitation in a lagoon.

The Dingman (Marble) Limestone is a crystalline granular rock composed of calcite grains. The grain size ranges from medium to fine in which some individual grains cannot be distinguished. There is banding of white with grey and greenish grey and some darker marble as well as some mottled appearing rock. Some of the finest grained rock is of statuary quality. Most of the marble is very competent with little core loss. Some fracturing occurs along the near vertical bedding, however this is rare. Nearby quarries of marble and limestones are mined for building stone, stucco and terrazzo, decorative, landscaping and road building material.

9.0 MINERALIZATION

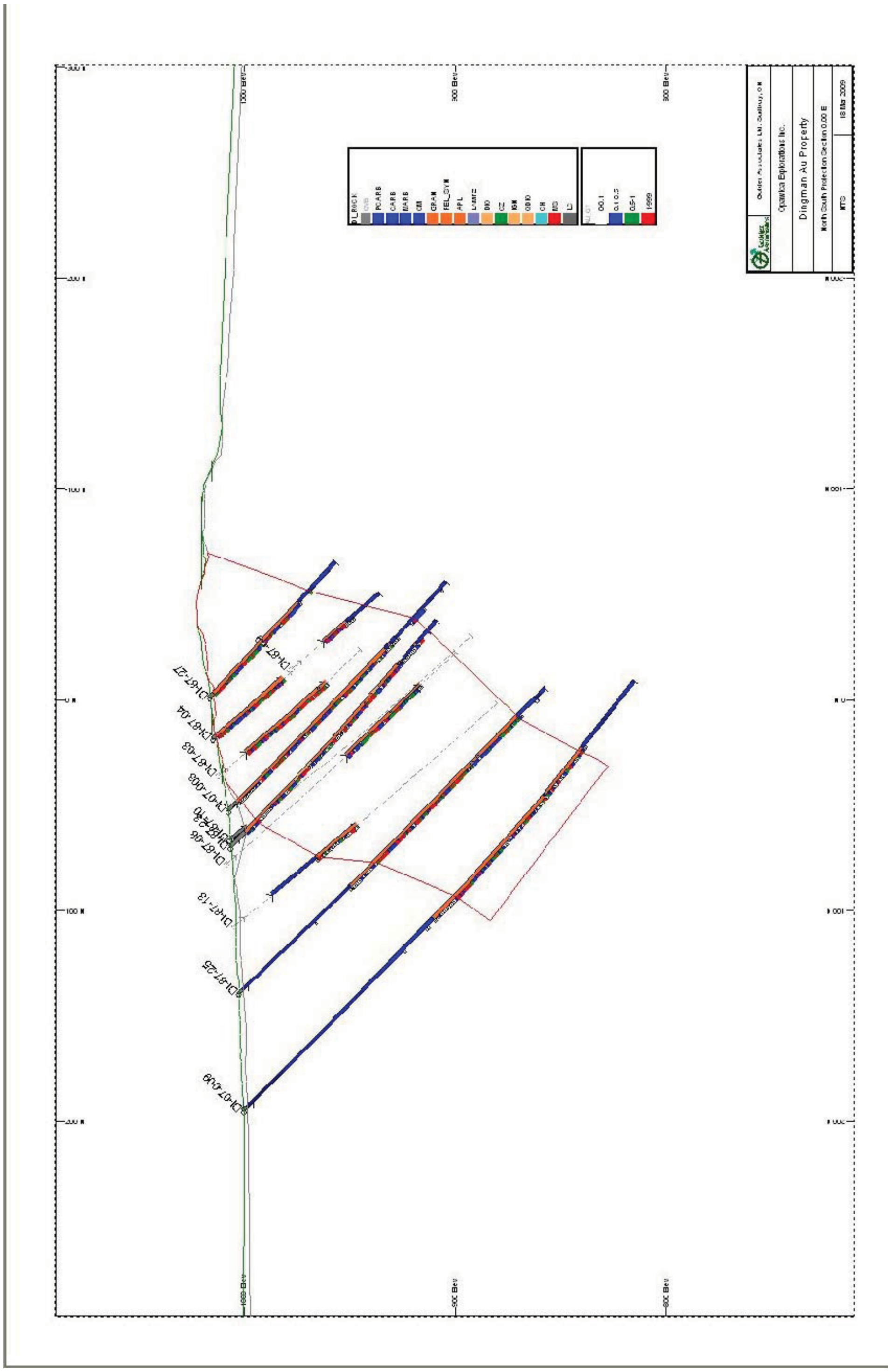
This section is an excerpt from the Dingman Technical Report titled “Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada”, by Palmer et al., dated March 25, 2009.

The bulk of the gold mineralization identified on the property occurs within the Dingman granite and can be grouped into seven major sericite-quartz-sulphide zones A-G but have been combined as a single contiguous zone for mineral resource estimating. The individual zones consist of moderate to strong sericite alteration with increased foliation development or shearing, variable but generally increased amounts of quartz veining, with elevated sulphide contents typically greater than 2-3%.

The bulk of the mineralization outcrops and is located in the western portion of the Dingman granite between Noranda grid coordinates 150E and 250W (sericite-quartz-sulphide). The mineralization is generally oriented in an east-northeast, west-southwest direction at an azimuth of 060°, parallel to the trend of the granite and the dominant foliation, but is also locally oriented in a north-east direction at an azimuth of 025°, parallel to the fracture-shear foliation and cross faults. The mineralization generally dips to the north-northwest at approximately 50-60°, sub-parallel to the dip of the granite body. Local variations in the dip of the mineralization appears to be controlled by changes in the dip of the granite contact and by cross faults.

The sericite-quartz-sulphide zones vary in strike length between 100 and 400 m and generally range between 5 and 30 m in horizontal thickness. Although the zones have reasonably good continuity both along strike and down-dip, they do coalesce or bifurcate, and can locally attain a horizontal thickness of up to 100 m. Diamond drilling has defined the zones to a vertical depth of approximately 175 m, one drill hole intersected the zones at a vertical depth of between 200 m and 300 m; the bulk of the zones appear to remain open at depth.

FIGURE 9-1: CROSS-SECTION OF MINERALIZATION IN THE DINGMAN GRANITE (PALMER ET AL., 2009)



10.0 EXPLORATION

The exploration history and Opawica exploration completed on the Dingman Property is based on information provided from Pope, 2008, from the previous Technical Report titled, Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada, dated March 25, 2009 (Palmer et al., 2009), Shaft & Tunnel and Opawica, and is outlined as follows.

10.1 Dingman - 1985

In 1985, Mark Dingman carried out a gold exploration program in the area, consisting of prospecting, geological mapping and sampling (Dingman, 1985). Dingman interpreted the small granite stock on the property to be the same age and character as the granite stocks known to host gold mineralization elsewhere in the area.

Assay results of the sampling were filed for assessment work credits by Diner (1986) on behalf of Dingman. Accurate sample locations and types of samples analyzed are not documented in the work report; it is most likely they are grab samples of rock. A total of 17 samples were taken by Dingman; all 3 samples from the western portion of the granite stock returned anomalous gold values ranging between 0.19 g/t Au and 0.75 g/t Au, while the 14 samples from the eastern portion of the granite returned generally low gold values, with the highest being 0.20 g/t Au.

10.2 Noranda Exploration Company Limited - 1986 to 1988

Exploration work carried out by Noranda in the summer and fall of 1986 consisted of geological mapping, ground magnetometer and VLF-EM geophysical surveys, soil geochemical surveys, minor stripping and trenching, and channel sampling (LeBaron, 1986).

Noranda established a metric grid with a baseline oriented at an azimuth of 060° centered along the trend of the Dingman granite stock, with lines at 50 m spacing and stations at 25 m intervals. The Noranda baseline was 950 m long with cross lines extending up to 275 m north and south of the baseline to ensure the ground surveys covered the granite-sediment contacts. The grid pickets were removed from the open fields on completion of the surveys at the request of the surface rights owner.

Ground magnetometer and VLF-EM geophysical surveys were carried out over the grid in October 1986. The magnetometer survey results show the granite as a zone of moderate magnetic intensity between a zone of lower intensity in the sediments to the south, and a zone of higher intensity in the sediments to the north. Local magnetic highs situated along both the southern and northern granite-sediment contacts may be caused by narrow magnetite skarn zones (LeBaron, 1986).

The VLF-EM survey results show a number of conductive zones located along the sheared granite-sediment contacts and within the marble up to 180 m from the granite. The conductive zones are interpreted by Noranda to be caused by shear zones or topography, as the granite-sediment contact is locally defined by a steep rock face with up to 5 m of relief (LeBaron, 1986). The conductive zones appear to have been disrupted and offset by the cross faults that trend northeasterly at an azimuth of 025°.

Noranda took a total of 122 B-horizon soil samples at 25 m intervals on 100 m line spacing over the grid (LeBaron, 1986). The soil samples were analyzed for gold, silver, arsenic, zinc, lead and copper. Strong coincident gold-arsenic-zinc-lead anomalies occur in four areas of the grid: 1) along the southern granite-sediment contact from line 0 to 200W, this anomaly coincides with sericite-quartz-sulphide zones C-E; 2) along the northern granite-sediment contact on line 100W, this anomaly coincides with sericite-quartz-sulphide zone A; 3) along the southern granite-sediment contact on line 300E, this anomaly may be the western extent of sericite-quartz-sulphide zone G; and 4) along or near the northern granite-sediment contact from line 300E to 400E, this anomaly is possibly related to the weak gold mineralization encountered in channel sampling and diamond drilling in this area. The absence of soil anomalies in the central portion of the Dingman granite is confirmed by the generally low gold values encountered in the diamond drilling carried out in this area.

Noranda completed a total of 563 m of channel sampling on the Dingman Property in 1986. Most of the channel sampling was carried out in the western or Marmora Township portion of the property. The channel samples were oriented primarily to transect the east-northeast, west-southwest trend of the Dingman granite and dominant foliation, but were also oriented so as to transect the second fracture-shear foliation that trends north-easterly at an azimuth of 025°.

TABLE 10-1: SUMMARY OF NORANDA CHANNEL SAMPLING COMPLETED ON THE DINGMAN PROPERTY

Township	Number of Channels	Length (m)
Marmora	82	536
Madoc	14	27
Total	**96**	**563**

A total of 600 channel samples taken from the property were analyzed for gold (LeBaron, 1986). Of these, 570 samples were taken from the western or Marmora Township portion of the property between 90E and 240W, 90S and 90N. Thirty channel samples were taken from a number of areas in the eastern or Madoc Township portion of the property between 320E and 480E, 55S and 55N.

Results from the 1986 exploration program on the Dingman Property were considered sufficiently interesting by Noranda to warrant the programs of diamond drilling carried out in 1987 and 1988

10.3 Deloro Minerals Ltd. - 1997

In 1997, Deloro carried out a program of re-logging and check sampling of Noranda drill core. The objectives and procedures of the program are discussed in Section 12.3 and the results in Section 13.3.

In September1997 and October 1998, Deloro carried out reconnaissance prospecting and grab sampling in the Dingman, Eldorado and Malone areas. A total of 40 grab samples were submitted to Chemex Labs Ltd. (Chemex). The samples were analyzed for gold by fire assay with atomic absorption spectrometry finish, and multi-element geochemistry performed using ICP-AES. A few anomalous gold values were returned from the sampling, although no detailed sample locations are available.

Deloro established a baseline on the Dingman Property, marked in the field by wood stakes with aluminum tags inscribed with the grid coordinates and grid name (Rajong). The Rajong baseline is at an angle to, and offset from, the original Noranda baseline, discussed in Section 14.2.

In 1997, Deloro completed 2,060.53 m of diamond drilling in 14 holes on the Dingman Property. The objectives, procedures and results of the drill program are discussed in Section 11.3.

10.4 Baird and Neczkar - 2006

In 2006, Baird and Neczkar carried out a soil geochemical survey and ground VLF-EM geophysical survey in the eastern part of the property (Neczkar, 2006). The objective of the program was to explore the eastern extension of the known gold mineralization hosted by the Dingman granite stock.

The Noranda baseline was projected from the last permanent marker at 450E to the Moira River. Cross lines were established at 750E and 850E, with the cross lines extending 120 m north and 210 m south of the baseline to ensure the ground surveys covered the eastern projection of the Dingman granite.

Baird and Neczkar took a total of 26 soil samples at 30 m intervals on the two cross lines (Neczkar, 2006). The soil samples were analyzed by SGS Mineral Services for gold, silver, palladium, cobalt and nickel using the MMI-B5 analytical method. Two samples returning anomalous values occur near the eastern projection of the southern granite-marble contact.

A VLF-EM geophysical survey was carried out over the same cross lines. The VLF-EM survey results show a number of weak conductive zones. The conductive zones may be caused by shear zones or topography, similar to the results of the survey carried out by Noranda.

10.5 Opawica Explorations Inc. - 2007 to 2008

In 2007, Opawica completed 4,726 m of diamond drilling in 20 holes on the Dingman Property. The objectives, procedures and results of the drill program are discussed in Section 11.4. Miller, Ontario Land Surveyor, of Stirling, Ontario, surveyed the Opawica drill hole collars and the existing Rajong baseline on the property.

In May 2008, Miller carried out additional surveying on the Dingman Property for Opawica. The survey program was designed to resolve the elevation discrepancy between the Noranda channel samples and drill hole collars, Deloro drill hole collars, and the Opawica drill hole collars. The program consisted of surveying the surviving Noranda and Deloro drill hole collar locations, Noranda grid wood stakes or pickets, outstanding Opawica drill hole collar locations, Noranda channel samples, and a detailed topographic survey on 25 m line spacing over the western portion of the property. All survey points are UTM Zone 18 NAD 83 with geodetic elevations. The results of the Miller survey are described in Section 14.1.

10.6 Opawica Explorations Inc. - 2009

In 2009, Opawica's exploration program consisted of 3,926 m of diamond drilling in 16 holes on the Dingman Property. The objectives, procedures and results of the drill program are discussed in Section 11.5. The 2009 drill hole collars were surveyed using a Garmin 60cx GPS with coordinates in UTM Zone 18 NAD 83.

11.0 DIAMOND DRILLING

The diamond drilling programs completed on the Dingman Property is based on information provided from Pope, 2008, from the previous Technical Report titled, Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada, dated March 25, 2009 (Palmer et al., 2009), Shaft & Tunnel and Opawica, and is outlined as follows..

11.1 Programs

The drill programs on the Dingman Property completed to the end of 2007 are summarized in Table 11-1 and Figure 11-1 shows the collar locations and horizontal projection of drill holes traces on the property.

TABLE 11-1: SUMMARY OF DRILLING COMPLETED ON THE DINGMAN PROPERTY

Year	Company	Core Size	Hole Series	Number of Holes	Length (m)
1987-1988	Noranda	BQ	DI-87-01 to 28 and DI-88-29 to 38	38	5,025
1997	Deloro	NQ	DI-97-39 to 52	14	2,061
2007	Opawica	NQ	DI-07-001 to 019	20	4,726
			Total	**72**	**11,812**

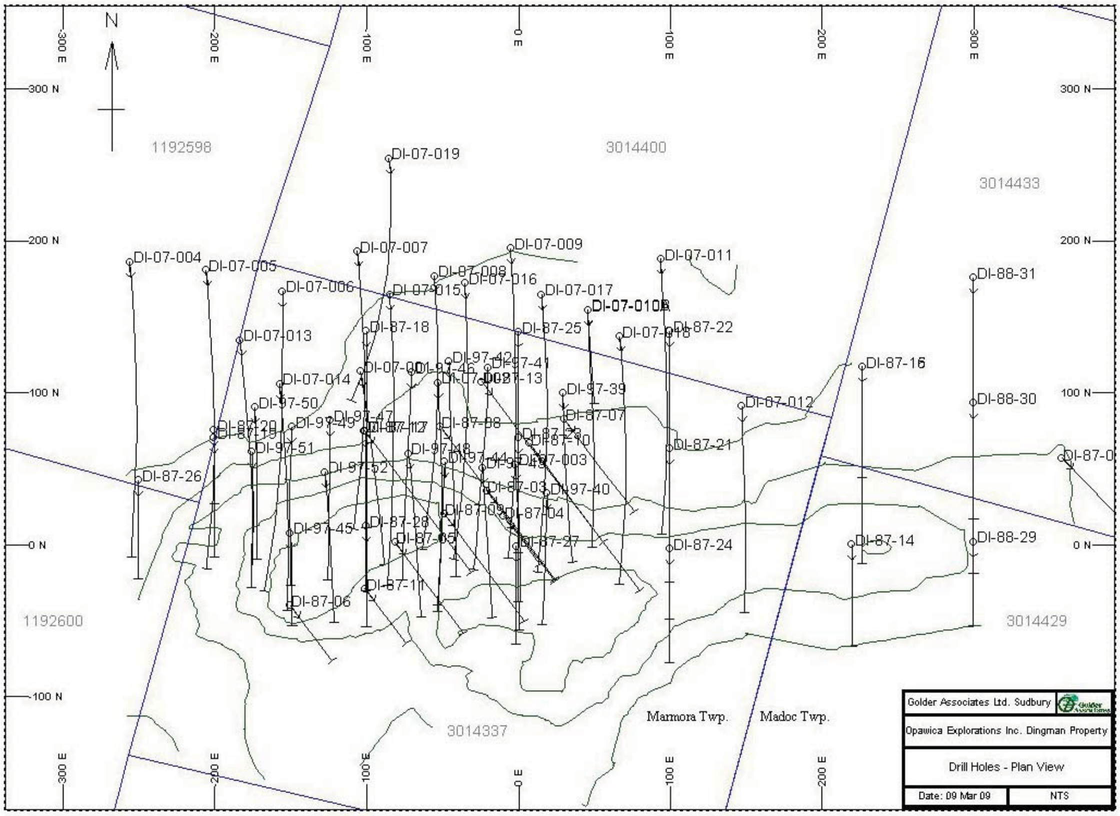
11.2 Noranda Exploration Company Limited Drilling Programs (1987 and 1988)

Noranda carried out a limited diamond drill program on the Dingman Property between July and September of 1987 (King, 1988). The initial program consisted of 13 holes (DI-087-01 to DI-87-13) totalling 1,494 m. The holes were designed to test the altered Dingman granite at an orientation suitable for intersecting the cross faults and second fracture-shear foliation that trend northeasterly at an azimuth 025°, interpreted to be an important control on the gold mineralization. The first phase holes were drilled at a dip of 45°.

Results from the initial program were considered sufficiently interesting to warrant a second phase of drilling (King, 1988). The second phase drill program, carried out between November 1987 and February 1988, consisted of 25 holes (DI-87-14 to DI-88-38) totalling 3531.41 m. The second phase holes were oriented to the Noranda grid south to intersect the east-northeast, west-southwest trend of the altered Dingman granite stock with less emphasis on the northeasterly trending structures. The holes were drilled on approximately 50 m spacing on 100 m spaced sections across the Dingman granite stock between sections 600E and 250W, providing data to a vertical depth of approximately 150 m. Most of the second phase holes were drilled at a dip of 45°; two holes were drilled at a dip of 75°.

The Noranda holes were all BQ with few descriptions of poor core recovery or lost core in the Noranda drill logs. The drill holes were spotted in the field using the picketed grid system (King, pers. com., 2008). Front sites used to line up the drill were spotted using either a compass, or the grid line for holes located on a line.

FIGURE 11-1: PLAN VIEW OF DRILL HOLE TRACES ON THE DINGMAN PROPERTY TO THE END OF 2007 (PALMER ET AL., 2009)



All the drill hole casings from the Noranda drill programs were removed; therefore, no verification of historical collar locations is possible. At the end of the second phase program, all the drill sites were inspected, cleaned and levelled where appropriate (King, 1988). The drill hole collar elevations were initially estimated by Noranda. A number of the collar elevations in the Borsurv database appear to have been adjusted at a later date based on the results of a small elevation survey carried out in the western part of the property.

Drill hole deviation was monitored by dip tests, using the standard acid dip test method. No downhole azimuth tests were taken. Dip tests were taken at a rate of 2 to 3 per hole, or at approximately 60 to 70 m intervals. Dip deviations for the holes drilled by Noranda were estimated between 2° and 4° of flattening per 100 m.

11.3 Deloro Minerals Ltd. Drilling Program (1997)

In September and October 1997, Deloro completed 2,061 m of diamond drilling in 14 holes on the Dingman Property. The holes were designed to test the grade and continuity of the gold mineralization reported by Noranda, with the objective of delineating the area between sections 25E and 200W at a hole spacing of 25 m on 25 m sections to a vertical depth of 100 m. The holes were drilled along Noranda grid south at a dip of 45°.

The drill holes completed by Deloro encountered gold mineralization within the Dingman granite with similar grades and over similar widths as reported by Noranda. The Deloro holes were all NQ and core recovery data was collected by Deloro for each sample interval, with recoveries typically above 98%. Core photographs show generally good recovery within the granite, with local sections of blocky core visible within the altered sediments.

Before the start of the Deloro drill program, Brian King (former Noranda geologist), assisted Nick Nuttycombe of Deloro in locating the Noranda baseline (King, pers. com., 2008). Deloro spotted the holes in the field using the original Noranda grid system and not the "Rajong" baseline, as discussed in Section 14.2. Deloro did not survey the drill hole collars. The casing was pulled from all of the holes drilled by Deloro. The Deloro drill hole collars are marked in the field by a stamped steel plate with an 8-inch long steel rod welded to the bottom and cemented in the hole.

Drill hole deviation was monitored by Tropari downhole dip and azimuth measurements. Tests were taken at a rate of 3 to 5 per hole, with the first test taken at 3 runs or 9 m past the casing, then at approximately 40 to 60 m intervals down the hole. Dip deviations for the holes drilled by Deloro were estimated between 1° and 2° per 100 m with 65% of the holes flattening and 35% of the holes steepening. Azimuth estimated deviations were between 1° and 3° per 100 m with 70% of the holes deviating clockwise and 30% of the holes deviating counter clockwise.

11.4 Opawica Explorations Inc. Drilling Program (2007)

Between October and December 2007, Opawica completed 4,726 m of diamond drilling in 20 holes on the Dingman Property. The diamond drill program was designed to verify the results of the Noranda and Deloro drilling and delineate the gold mineralization to a level sufficient to enable Opawica to undertake a mineral resource estimate in accordance with the requirements of NI 43-101 and the CIM Definition Standards on Mineral Reserves and Mineral Resources.

Three twinned holes were planned and drilling focused on gold mineralization within the Dingman granite between sections 100E and 200W to a vertical depth of approximately 150 m at a hole spacing of between 25 and 50 m on 25 m sections (Figure 11-1). Three drill holes were designed to test for potential extensions of the gold zones along strike and down-plunge. All holes were drilled on the Noranda grid south at a dip of -45°, with the exception of hole DI-07-019, which was drilled at dip of -70° to test the zone down-plunge.

The drill holes completed by Opawica were successful in verifying the Noranda and Deloro results and encountered gold mineralization within the Dingman granite with similar grades and over similar widths. Hole DI-07-012 encountered significant gold mineralization on section 150E, extending the eastern strike length of the main zone of mineralization by a minimum of 50 m and showing a potential for additional mineralization in this area. Hole DI-07-019 encountered a number of wide zones of sericite-quartz-sulphide mineralization between a vertical depth of 200 and 300 m that returned gold values with similar grades and widths as the mineralization defined above 150 m, suggesting that the gold mineralization continues at depth and remains open.

George Downing Estate Drilling Ltd. of Grenville-sur-la-Rouge, Quebec, provided contract drilling for the program. The Opawica holes were all NQ with core recovery as generally excellent and estimated at close to 100%. Sections of blocky core with estimated recoveries of 95-99% occasionally occur within hematite-altered sediments containing vuggy quartz-calcite veining.

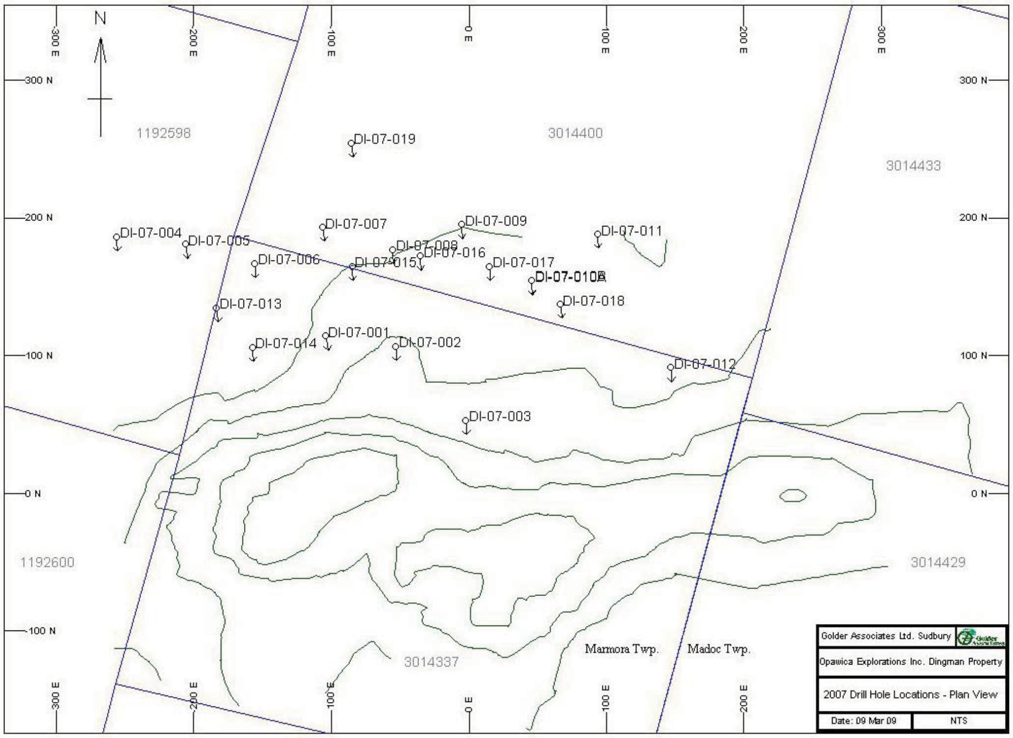
Drill holes DI-07-001 to DI-07-012 and DI-07-019 were spotted in the field using a combination of hand-held GPS and chaining from the re-established Rajong baseline. The collars of completed holes DI-07-001 to DI-07-007 and collar stakes for planned holes DI-07-008 to DI-07-012 and DI-07-013 to DI-07-018 were surveyed in November 2007 by Miller.

Miller used a differential GPS survey method with an accuracy of 2 cm in three directions. The collar coordinates are UTM Zone 18 NAD 83 with geodetic collar elevations, based on horizontal control monument number 00820040015 at 256.873 m. Miller also calculated the grid coordinates for the collars based on the re-established and surveyed the Rajong baseline. A geodetic elevation of 221 m was set at Noranda grid elevation of 1,000 m to establish the level of the Noranda and Deloro channel sample and drill hole data.

Front sites used to line up the drill were spotted using a compass and confirmed by hand-held GPS. The drill holes were abandoned with a cement plug, the casings pulled and the drill sites cleaned. Miller surveyed the collar locations of holes DI-07-010B and DI-07-019 in May 2008.

Drill hole deviation was monitored by a Flexit SmartTool electronic instrument capable of taking downhole dip and azimuth measurements. An azimuth correction was applied to the Flexit readings after the measurements were taken: 12.0° (west) was subtracted from the Flexit reading to give a true azimuth. Tests were taken at a rate of 4-6 per hole, with the first test taken at a downhole depth of between 18 and 21 m, then at 50 m intervals down the hole. Dip deviations for the holes drilled by Opawica are estimated between 1° and 3° of flattening per 100 m. Azimuth deviations are estimated between 3° and 5° per 100 m with 100% of the holes deviating clockwise.

FIGURE 11-2: LOCATION OF DINGMAN 2007 DRILL HOLE COLLARS (PALMER ET AL., 2009)



11.5 Opawica Explorations Inc. Drilling Program (2009)

Between February and May 2009, Opawica completed 3,926 m of diamond drilling in 16 holes on the Dingman Property. The diamond drill program was designed to explore the Dingman granite at depth below all previous drilling to a vertical depth of 700 m, infill drill a section of granite from 175 E to 250 E, and delineate additional aggregate resources from 350 E to 700 E within and adjacent to the Dingman granite.

Five drill holes tested the Dingman granite gold zones between grid 100 m W and 80 m E below previous drilling to a maximum vertical depth of 700 m. Three drill holes tested the Dingman granite gold zones between 175 m E and 250 m E and above 150 m vertical. Eight drill holes tested the granite and marble between 350 m E and 700 m E and above 150 m vertical depth to delineate the aggregate resource. All holes were drilled on the Noranda grid southward at dips of -45° for shorter holes to maximum of -83° for the longest hole.

The drill holes completed by Opawica were successful in extending the gold zones to 700 m vertical depth and indicating a widening of the Dingman granite at depth, and expanding the aggregate resource eastward.

George Downing Estate Drilling Ltd. of Grenville-sur-la-Rouge, Quebec, provided contract drilling for the program. The Opawica holes were all NQ with core recovery as generally excellent and estimated at close to 100%. Sections of blocky core with estimated recoveries of 95-99% occasionally occurred within hematite-altered sediments containing vuggy quartz-calcite veining.

Drill holes were spotted using a Garmin 60cx GPS. The collar coordinates are UTM Zone 18 NAD 83.

Front sites used to line up the drill were spotted using a compass and confirmed by hand-held GPS. The drill holes were abandoned with a cement plug set in overburden and into bedrock. The casing was left in drill hole DI-09-01 and capped with an aluminum cap inscribed with the drill hole ID. The casings for DI-09-02 to 16 were pulled and each collar location was marked by a steel bolt and aluminum tag assembly cemented into the top of each drill hole at a depth of 15 cm to 35 cm below ground. Relocating any of holes DI-09-02 to 16 is possible by using a metal detector to detect the steel bolt/aluminum tag assembly, digging down 15 cm to the assembly top and reading the hole ID inscribed on the aluminum tag.

Drill hole deviation was monitored by a Flexit SmartTool electronic instrument capable of taking downhole dip and azimuth measurements. An azimuth correction was applied to the Flexit readings after the measurements were taken: 12.1° (west) was subtracted from the Flexit reading to give a true azimuth. Tests were taken at a rate of 3-15 per hole, with the first test taken at a downhole depth of between 18 and 21 m, then at 50 m intervals down the hole. Dip deviations for the holes drilled by Opawica are estimated between 0.5° and 3° of flattening per 100 m. Azimuth deviations are estimated between 0.5° and 3.5° per 100 m with 100% of the holes deviating clockwise.

Year	Company	Core Size	Hole Series	Number of Holes	Length (m)
2009	Opawica	NQ	DI-09-01 to 16	16	3,926

FIGURE 11-3: LOCATION OF DINGMAN 2009 DRILL HOLE COLLARS AND TRACES

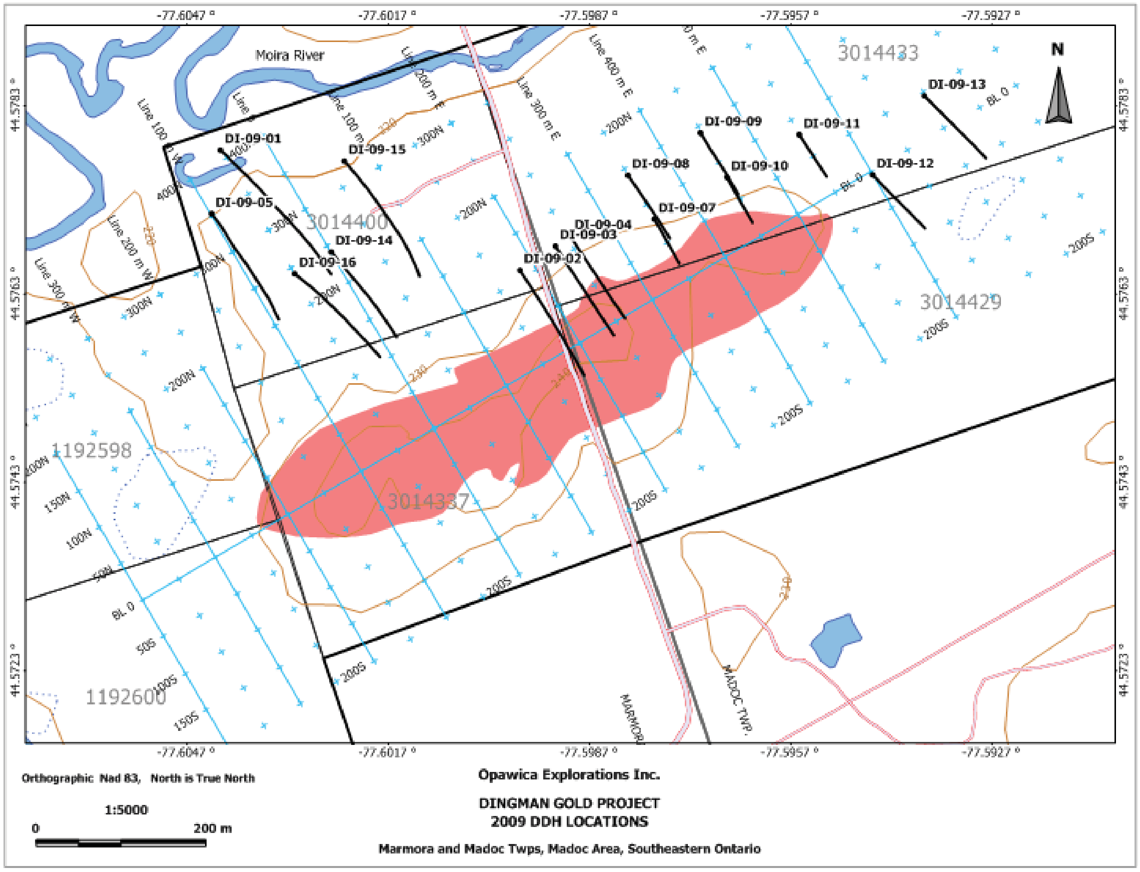
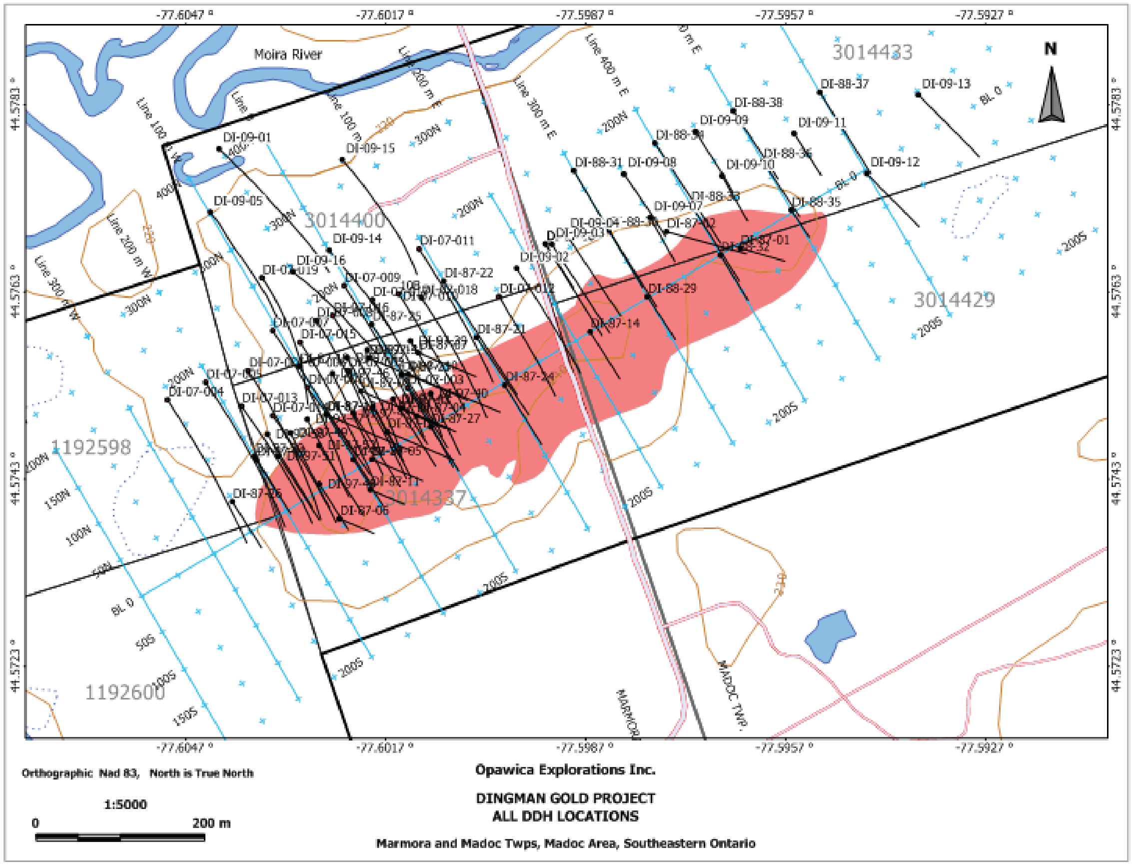


FIGURE 11-4: LOCATION OF ALL DINGMAN DRILL HOLE COLLARS AND TRACES



12.0 SAMPLING METHOD AND APPROACH

The sampling method and approaches that have been completed on the Dingman Property is based on information provided from Pope, 2008, from the previous Technical Report titled, "Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada", dated March 25, 2009 (Palmer et al., 2009), Shaft & Tunnel and Opawica, and is outlined as follows.

12.1 Noranda Exploration Company Limited, 1986 Channel Sampling Program

The channel samples were cut with a diamond saw in areas of outcrop and from bedrock exposed by shovel and backhoe trenching. The channel samples average 0.9 to 1.0 m in length and 3 to 4 cm in width (LeBaron, 1986). Backhoe trenches in overburden were filled in after the program at the request of the surface rights owner.

The channel samples were chained into the Noranda grid (LeBaron, pers. com., 2008). Channel sample locations, orientation of the samples, sample widths, sample numbers and assay results were digitized from the original Noranda channel sampling maps and verified as described in Section 14.1.

12.2 Noranda Exploration Company Limited, 1987 and 1988 Diamond Drilling

All core logging and splitting was completed in a temporary core shack on the Robinson Farm. The original paper copies of the drill logs exist for all the holes. Geological data recorded on the drill logs include lithology, texture, structure, alteration, quartz veining and sulphide mineralization. All geological and sample data was recorded in imperial units. The alteration legend used in core logging is the classification scheme developed for the Dingman granite by LeBaron (1986) from surface mapping. The core logging is considered to be of high quality and the data collected can be used in the development of geological and mineralization models.

The entire interval of granite intersected in each drill hole was sampled and assayed (King, 1988). In addition, bracket samples were generally taken from the hanging wall and footwall sedimentary lithologies and from intervals of sediments containing an appreciable amount of sulphides.

The sample lengths are generally 1 m, but individual samples were adjusted to accommodate lithologic contacts, changes in alteration, quartz veining and sulphide mineralization. Core was split in half, presumably with a mechanical or hydraulic core splitter, and the half core samples sent to the laboratory. Contiguous sample series were used for the core sampling. Sample intervals were recorded on standard Noranda diamond drill core assay record sheets. The collar, downhole survey, lithology and sample data was later entered into computerized drill logs in Borsurv format and the data (with the exception of the gold assays) converted from imperial to metric units.

The remaining half of the core was left in the core box and initially stored at the temporary core shack on the property. Some time after the program, most of the split core was transported to Timmins, Ontario, and stored at the former Aunor mine site. After Deloro worked on the property, the core stored at the Aunor mine site was transported to the core storage area at the Holloway mine site, located 60 km east of Matheson, Ontario. Core from 16 Noranda holes (DI-87-02 to 06, 09, 11-13, 17-18, 20, 25, DI-88-31, 33 and 34) is stored at the Holloway mine site (Kusins, 2008). Core from 5 Noranda holes (DI-07-07, DI-88-27, 28, 30 and 35) is stored at the MNDM core library in Tweed, Ontario. Parts of holes DI-87-22, 24, DI-88-28, 29, 34 and 35 still remain on the Robinson Farm, but consist primarily of marble and are in poor condition.

12.3 Deloro Minerals Ltd. Re-logging and Check Sampling of Noranda Drill Core in 1997

A program of re-logging and check sampling of the Noranda drill core was undertaken by Deloro in the summer of 1997. The re-logging and check sampling was carried out at the Aunor mine site in Timmins, Ontario. The work focused on 22 of the Noranda drill holes collared in the better mineralized western portion of the property. The purpose of the re-logging program was to identify the key geological controls for the gold mineralization within the Dingman granite and develop a detailed core logging and coding scheme for use in the upcoming drill program.

Geological data was recorded as codes or percentages for each sample interval as well as for sections not sampled. Geological data recorded on the drill logs include lithology, texture, grain size, replacement sericite, quartz vein volume percent, pink kspar and total sulphide percent. The replacement sericite coding scheme was derived from the alteration legend developed by Noranda.

The purpose of the check sampling was to provide a check on the gold values reported by Noranda from the two laboratories used during the 1987 and 1988 drill programs, and to determine the best sample preparation and assay procedures to use for the 1997 drill program. Deloro took a total of 135 check samples of the Noranda drill core. The samples collected were from the remaining half of the split core, covering the exact interval as the original samples.

Drill core was picked up from the drill rig twice a day by Deloro employees and taken to a nearby location for logging, photographing and splitting (Pope, 2008). The original drill logs in table format exist for all the holes.

Geological data was recorded as codes or percentages for each sample interval as well as for sections not sampled. Geological data recorded on the drill logs include lithology, colour, colour intensity, texture/deformation, grain size, foliation/bedding, quartz eyes, biotite content, replacement sericite, quartz vein numbers, quartz vein volume percent, thickest quartz vein, pervasive silicification, pink feldspar, lilac feldspar, 2^{nd} calcite, fluorite, total sulphide percent, pyrrhotite-pyrite ratio, average sulphide size, maximum sulphide size, other sulphides and oxidation of sulphides. All geological and sample data was recorded in metric units.

The coding system developed by Deloro for each sample interval, primarily sericite alteration, quartz veining and total sulphides, is considered critical in undertaking the geological interpretation, and in constructing the 3D lithology, structure and mineralization models described in Section 17.

The entire interval of granite intersected in each drill hole was sampled and assayed. In addition, bracket samples were taken from the hanging wall and footwall sedimentary lithologies and from intervals of sediments containing an appreciable amount of sulphides.

The sample length was generally 1 m, but individual samples were adjusted to accommodate lithologic contacts or abrupt changes in quartz veining and sulphide mineralization. Core was split in half, presumably with a mechanical or hydraulic core splitter, and the half core samples sent to the laboratory. Contiguous sample series were used for the core sampling. Sample intervals were recorded on sample ticket books and on the standard drill core log described above. All logging information recorded on the drill logs was later entered into computerized drill logs in a spreadsheet format.

The remaining half of the core was left in the core box and stored on the Robinson Farm. Core for 8 of the Deloro holes (DI-97-40 to DI-97-47) still remain on the Robinson Farm and are in good condition. Core from the remaining 5 Deloro holes are stored at the MNDM core library in Tweed, Ontario as confirmed by the District Geologist for Southeastern Ontario (LeBaron, 2009).

12.4 Opawica Explorations Inc. 2007 Diamond Drilling

After the core was placed in wooden trays at the drill site, it was loaded directly into the back of covered trucks, and delivered in batches of three holes by Opawica personnel to the core shack in Matachewan, Ontario. All core logging and core splitting was completed in the Opawica core shack. Before logging, the drill core was fit together, measured and then marked every 1 m, and then photographed. The drill core was logged by Fred Sharpley, P.Geo., the Opawica QP for the Dingman Property. Geological data recorded on the drill logs include lithology, texture, structure, alteration, quartz veining and sulphide mineralization. All logging information was recorded directly into a laptop computer in Microsoft Excel spreadsheet format. All geological and sample data was recorded in metric units.

Core intervals identified for sampling were marked with wax crayons, with sample tags placed at the end of a sample interval. The sample length was generally 1 m, but individual samples were adjusted to accommodate lithologic contacts. The entire interval of granite intersected was sampled and assayed. In addition, bracket samples were taken from the enclosing sedimentary lithologies and from intervals of sediments containing an appreciable amount of sulphides.

Core was sawn in half using a Vancon diamond rock saw, with a sample tag placed in the bag and the bag sealed with a plastic tie strap. The remaining half of the core was left in the core box and stored in core racks beside the Opawica core shack in Matachewan.

Contiguous sample series were used for the core sampling. Sample intervals were recorded on sample ticket books and entered directly into a laptop computer. Geological data recorded for each sample interval included sericite alteration index, quartz vein percent and total sulphide percent. The Dingman drill hole logging legend used by Opawica is modified from the work completed by Noranda and Deloro and presented in Tables 12-1 to 12-5.

12.5 Opawica Explorations Inc. 2009 Diamond Drilling

After removal from the drill hole, the core was placed in wooden trays at the drill site, it was loaded into a covered truck, and delivered in batches by Opawica personnel to the core shack in Matachewan, Ontario.

All core logging and core splitting was completed in the Opawica core shack. Before logging, the drill core was fit together, measured and then marked every 1 m. The drill core was logged by Terry Link and reviewed by Robert Laakso, P.Eng. All logging information was recorded directly into a laptop computer in Microsoft Excel spreadsheet format. All geological and sample data was recorded in metric units.

Core intervals identified for sampling were marked with wax crayons, with sample tags placed at the end of a sample interval. The sample length was generally 1 m, but individual samples were adjusted to accommodate lithologic contacts. The entire interval of granite intersected was sampled and assayed. The core recovery for Dingman is between 95% - 100% and does not significantly affect sampling of the core. In addition, bracket samples were taken from the enclosing sedimentary lithologies and from intervals of sediments containing an appreciable amount of sulphides.

Core was sawn in half using a Vancon diamond rock saw, with a sample tag placed in the bag and the bag sealed with a plastic tie strap. The remaining half of the core was left in the core box and stored in core racks beside the Opawica core shack in Matachewan.

Contiguous sample series were used for the core sampling. Sample intervals were recorded on sample ticket books and entered directly into a laptop computer. The Dingman drill hole logging legend used by Opawica is modified from the work completed by Noranda and Deloro and presented in Tables 12-1 to 12-5.

Length weighted average composite assay intervals were calculated after QA/QC and reported in Opawica's press releases. Subsequently, length weighted average composite assay intervals were calculated and used to define polygon outlines on sections for resource calculations. Two drill hole composites used in resource modelling are 49 m of 1.34 g/t Au from 609 m to 658 m in drill hole DI-09-01, and 69 m of 1.44 g/t from 453 m to 522 m in drill hole DI-09-05. A table of 2009 drill holes with all composites used in the resource estimate is included in Appendix B.

TABLE 12-1: DINGMAN DRILL HOLE LEGEND - LITHOLOGY CODES AND COLOUR SCHEME

Rock Code	Rock Type	Label	Colour
0	Overburden / Casing	OVB	Brown
1	Pellitic Carbonate	PCARB	Blue
2	Carbonate(Undifferentiated)	CARB	Blue
3	Marble	MARB	Blue
4	Carbonate / Marble (Undifferentiated)	CM	Blue
5	Granite (Granite, Granodiorite and Diorite Undifferentiated)	GRAN	Orange
6	Felsite Or Syenite	FEL OR SYN	Orange
7	Aplite	APL	Orange
8	Lamprophyre / Fault / Altered / Contact Zone (Skarn)	LAMFZ	Purple
9	Diorite	DIO	Orange
10	Skarn Or Contact Zone	CZ	Green
11	Unidentified Igneous	IGN	Orange
12	Quartz Diorite	QDIO	Orange
13	Chert	CH	Blue
90	Massive Sulphide	MS	Red
98	Lost Core	LC	Grey
99	Unknown	UNK	Grey

TABLE 12-2: DINGMAN DRILL HOLE AND CHANNEL SAMPLE LEGEND AU ASSAY COLOUR SCHEME

Assay Values From (Au g/t)	Assay Values To (Au g/t)	Colour
0	0.099	Black
0.1	0.499	Blue
0.5	0.999	Green
Greater Than Or Equal To 1		Red

TABLE 12-3: DINGMAN DRILL HOLE LEGEND - SERICITE ALTERATION INDEX

Deloro Replacement Sericite Code	Noranda Equivalent Alteration Type	Sericite Alteration Intensity	Colour
0	G1	Weak	Black
1	G1-G2	Weak To Moderate	Blue
2	G2 OR G1-G2 WITH G3	Moderate	Green
3	G2-G3 OR G3 WITH G1-G2	Moderate To Strong	Red
4	G3	Strong	Red
5	RARELY USED (G3)	Intense	Red

**TABLE 12-4: DINGMAN DRILL HOLE LEGEND
QUARTZ VEINS VOLUME % COLOUR SCHEME**

Quartz Veins From Volume %	Quartz Veins To Volume %	Colour
0	0.9	Black
1	9.9	Blue
10	19.9	Green
Greater than or equal to 20		Red

**TABLE 12-5: DINGMAN DRILL HOLE LEGEND
TOTAL SULPHIDE % COLOUR SCHEME**

Sulphide % (From)	Sulphide % (To)	Colour
0	0.49	Black
0.5	1.9	Blue
2	2.9	Green
Greater than or equal to 3		Red

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The sampling preparation, analyses and security that have been completed on the Dingman Property is based on information provided from Pope, 2008, from the previous Technical Report titled, "Technical Report on the Dingman Gold Property, Madoc, Ontario, Canada", dated March 25, 2009 (Palmer et al., 2009), Shaft & Tunnel and Opawica, and is outlined as follows.

13.1 Historical Sample Security, Storage and Shipment

No information exists with respect to sample security, storage and shipping arrangements for the Noranda channel sampling and diamond drill programs.

In the 1997 Deloro drill program, core splitting and bagging of samples were carried out by personnel under contract to Deloro and under the supervision of a member of Deloro's management (Pope, 2008). The first and last sample batches were delivered by Deloro personnel to the Chemex laboratory in Mississauga, Ontario. The remainder of the sample batches were picked up from the project site and transported to the laboratory by Chemex employees. The samples were kept in a secure building until they were delivered to Chemex.

13.2 Opawica Sample Security, Storage and Shipment

In the 2009 Opawica drill program, sawn core samples were collected and processed by personnel under contract to Opawica and under the supervision of Robert Laakso, P.Eng., at the Matachewan core logging facility. After sawing and bagging, the sealed individual samples were placed in shipping bags, which in turn were sealed with plastic tie straps. The bags remained sealed until they were opened by Swastika personnel in Swastika, Ontario. The samples were kept in the locked Opawica core shack in Matachewan and delivered on a regular basis by Opawica personnel to the laboratory in Swastika.

13.3 Historical Sample Preparation and Assay Procedures

13.3.1 Noranda Exploration Company Limited Protocol, 1986 Channel Samples

There are no records of the sample preparation and assaying procedures used by Noranda for the surface channel samples taken in their 1986 program, or the laboratory at which the assaying was performed. No assay laboratory certificates are available for the channel samples.

13.3.2 Noranda Exploration Company Limited Protocols, 1987 and 1988 Drill Core Samples

Split core samples from the first phase program, drill holes DI-87-01 to DI-87-13, were submitted to Technical Service Laboratories (TSL) in Mississauga, Ontario. There are no records of the sample preparation and assay procedures used by TSL. Assay certificates exist for the samples from holes DI-87-01 to DI-87-06 and DI-87-09 to DI-87-13. All core samples submitted to TSL were assayed for gold; the majority of the samples returning gold values greater than 0.05 oz/t were also assayed for silver. Gold assays are reported on the certificates in ppb; values greater than 1,000 ppb are reported in oz/t. A few samples returning values slightly less than 1,000 ppb are also reported in oz/t. The detection limit for gold was 5 ppb for values reported in ppb and 0.005 oz/t for values reported in oz/t.

A total of 20 samples from the first phase drill program were analyzed for major oxides, S%, CO_2%, and base metals (King, 1988). There are no records of the sample preparation and analytical procedures used by Noranda for these samples, or the laboratory at which the work was performed.

Split core samples from the second phase program, drill holes DI-87-14 to DI-87-28 and DI-88-29 to DI-88-38, were submitted to Lakefield Research (Lakefield) in Lakefield, Ontario. There are no records of the sample preparation and assay procedures used by Lakefield. Assay certificates exist for all the core samples submitted to Lakefield. All core samples submitted to Lakefield were assayed for gold, with samples returning gold values greater than 0.05 oz/t also assayed for silver. Gold assays are reported on the certificates in oz/t with a detection limit of 0.001 oz/t.

A total of 39 samples from the second phase drill program were analyzed for major oxides, S%, CO_2%, arsenic, chromium, and the base metals copper, zinc and lead at Lakefield. There are no records of the sample preparation and analytical procedures used by Lakefield. Assay certificates exist for all but nine of the S%, CO_2%, arsenic, chromium, copper, zinc and lead results.

13.3.3 Deloro Minerals Ltd. Protocols, Check Sampling of Noranda Drill Core in 1997

Samples of split core from the Noranda drilling were submitted by Deloro to Chemex in two separate shipments in the summer of 1997. The first shipment, submitted in July 1997, consisted of 30 split core samples. Sample preparation was performed at the Chemex laboratory in Timmins, Ontario. Assay certificates exist for all the core samples in the first shipment submitted to Chemex.

The split core samples from the first shipment were crushed in their entirety to better than 80% passing a 0.85 mm or 20 mesh screen (Chemex Procedure 226a). All the crushed material was pulverized using a jumbo chrome steel ring mill to better than 90% passing a 0.10 mm or 150 mesh screen (Chemex Procedure 3288). The pulverized material was split using a riffle splitter into two sub samples: 1) sub sample A weighing 500 grams; and 2) sub sample B weighing 1,000 grams.

Samples drawn from sub sample A were analyzed for gold using a variety of procedures at Chemex, Intertek Testing Services (Chimitec or Bondar Clegg) (Intertek) and Cone Geochemical Inc. (Cone Geochemical). Fire assaying was performed on two 30-gram samples (30 g sample fusion #1 and 2) drawn from the pulp A at the Chemex laboratory in Mississauga, Ontario. The gold bead was assayed using atomic absorption spectrometry (Chemex Procedure 494). Gold assays are reported on the certificates in g/t. The detection limit for this method is 0.005 g/t.

Fire assaying was performed on a 50 g sample (50 g sample fusion) drawn from pulp A at the Chemex laboratory in Vancouver, British Columbia. The gold bead was assayed using atomic absorption spectrometry (Chemex Procedure 3599). Gold assays are reported on the certificates in g/t. The detection limit for this method is 0.07 g/t.

Fire assaying was performed on a 30 g sample drawn from pulp A at the Intertek laboratory in Val d'Or, Quebec. There is no record of the assay finish procedure. Gold assays are reported on the certificates in g/t. The detection limit for the procedure used by Intertek is 0.03 g/t.

Fire assaying was performed on a 30 g sample drawn from pulp A at the Cone Geochemical laboratory in Lakewood, Colorado. The gold bead was assayed using atomic absorption spectrometry. Gold assays are reported on the certificates in g/t. The detection limit for the procedure used by Cone Geochemical is 0.001 g/t.

Sub sample B was analyzed for gold at the Chemex laboratory in Mississauga, Ontario using a metallic or screen assay technique (Chemex Procedure G803a). The plus 150 mesh portion was screened out and assayed entirely. The minus fraction was homogenized and two 30 g fusions used to determine its grade. Each fraction was weighed and the calculated grade of the sample is the weighted average of both fractions.

The second shipment, submitted in August 1997, consisted of 105 split core samples. Sample preparation was performed at the Chemex laboratory in Timmins, Ontario. Assay certificates exist for all the core samples in the second shipment submitted to Chemex.

The split core samples from the second shipment were crushed in their entirety to better than 80% passing a 0.85 mm or 20 mesh screen (Chemex Procedure 226a). The crushed material was split using a riffle splitter to arrive at a sub sample of 1,000 g. The sub sample was pulverized using a jumbo chrome steel ring mill to better than 90% passing a 0.10 mm or 150 mesh screen (Chemex Procedure 3288).

Fire assaying was performed on two 50 g samples (samples A and B) drawn from the pulp at the Chemex laboratory in Mississauga, Ontario. The gold bead was assayed using atomic absorption spectrometry (Chemex Procedure 3583). Gold assays are reported on the certificates in ppb. Values below the detection limit of 5 ppb are reported as less than the detection limit. Samples returning assays above 10,000 ppb were automatically re-assayed on a 1 assay ton sample (29.2 g) by gravimetric technique (Chemex Procedure 997). The detection limit for the gravimetric assays is 0.07 g/t.

13.3.4 Deloro Minerals Ltd. Protocol, 1997 Drill Core Samples

Split core samples from the Deloro drilling in 1997 were submitted to the Chemex laboratory in Mississauga, Ontario. All core samples submitted to Chemex were assayed for gold. Assay certificates exist for all the core samples submitted to Chemex.

The split core samples were crushed in their entirety to better than 80% passing a 0.85 mm or 20 mesh screen (Chemex Procedure 226a). The crushed material was split using a riffle splitter to arrive at a sub sample of 1,000 g. The sub sample was pulverized using a jumbo chrome steel ring mill to better than 90% passing a 0.10 mm or 150 mesh screen (Chemex Procedure 3288).

Fire assaying was performed on a 30 g sample drawn from the pulp at either the Chemex laboratory in Vancouver, British Columbia or Mississauga, Ontario. The gold bead was assayed using atomic absorption spectrometry (Chemex Procedure 494). Gold assays are reported on the certificates in g/t. Values below the detection limit of 0.005 g/t are reported as less than the detection limit. Samples returning assays above 12 g/t were automatically re-assayed on a 1 assay ton sample (29.2 g) by gravimetric technique (Chemex Procedure 997). The detection limit for the gravimetric assays is 0.07 g/t.

13.4 2007 Opawica Sample Preparation and Assay Procedures

Core samples from the 2007 drill program were submitted to Swastika. Swastika was not an ISO certified laboratory in 2007, but has been conducting assaying for the gold mining industry for many years and has a very good reputation for quality of work.

All core samples submitted to Swastika were assayed for gold. The sawn core samples were crushed in their entirety to arrive at a prepared sample of between 6 and 10 mesh (3.35 mm and 2 mm) (Procedure SP-1). The mesh size depends on the hardness and texture of the rock material. The crushed material was split successively in a riffle divider to arrive at a sub sample of between 300 and 400 g. The sub sample was pulverized in a ring and puck pulverizer to ensure a minimum of 90% of the material passed through a 0.15 mm or 100 mesh screen (Procedure SP-1).

Fire assaying was performed on a 30 g sample drawn from the pulp (Procedure FA-1). The gold bead was assayed using either atomic absorption spectrometry or gravimetric technique; the technique chosen was based on a visual assessment of the bead by the assayer (Procedure GA-1). Gold values are reported on the certificates in g/t. Values below the detection limit of 0.01 g/t are reported as nil.

Pulp samples from a majority of the drill holes (1,944 samples out of a total of 2,283 assayed for gold) were also analyzed for silver at Swastika. Pulp samples from sections of core containing chalcopyrite, sphalerite or galena were also analyzed for the base metals copper (40 samples), zinc (191 samples) or lead (78 samples) at Swastika. Silver, copper, zinc and lead were determined using a 0.5 g sub sample drawn from the pulp and subjected to Geochemical Procedure BMA-G1. This method uses aqua regia to digest the sample and the diluted solution analyzed using atomic absorption spectrometry. Silver values are reported on the certificates in g/t with a detection limit of 0.1 g/t. Copper, zinc and lead values are reported on the certificates in ppm. The detection limit for copper, zinc and lead is 1 ppm. Silver, copper, zinc and lead values below detection limit are reported as the detection limit value on the certificates.

A total of 11 pulp samples were submitted to Assayers Canada in Vancouver, British Columbia for whole rock assay and multi-element geochemical analysis. The pulps consisted of 6 samples of granite and 5 samples of carbonate sediments selected from holes DI-07-016 to DI-07-018. The samples were analyzed for major oxides, C%, S% and 12 trace elements using lithium metaborate fusion, followed by dissolution in aqua regia and analysis by ICP-AES. The samples were also analyzed for 30 elements using aqua regia digestion and analysis by ICP-AES.

13.5 2009 Opawica Sample Preparation and Assay Procedures

Core samples from the 2009 drill program were submitted to Swastika. Swastika was not an ISO certified laboratory at the time of submitting the samples for assay, but has been conducting assaying for the gold mining industry for many years and has a very good reputation for quality of work. All core samples submitted to Swastika were assayed for gold. The sawn core samples were crushed in their entirety to arrive at a prepared sample of between 6 and 10 mesh (3.35 mm and 2 mm) (Procedure SP-1). The mesh size depends on the hardness and texture of the rock material. The crushed material was split successively in a riffle divider to arrive at a sub sample of between 300 and 400 g. The sub sample was pulverized in a ring and puck pulverizer to ensure a minimum of 90% of the material passed through a 0.15 mm or 100 mesh screen (Procedure SP-1).

Fire assaying was performed on a 30 g sample drawn from the pulp (Procedure FA-1). The gold bead was assayed using either atomic absorption spectrometry or gravimetric technique; the technique chosen was based on a visual assessment of the bead by the assayer (Procedure GA-1). Gold values are reported on the certificates in g/t. Values below the detection limit of 0.01 g/t are reported as nil.

13.6 Historical Quality Control Programs

13.6.1 Noranda Exploration Company Limited Protocols, 1987 and 1988 Drill Core Samples

Check sampling undertaken by Noranda included: 1) re-assaying of pulps and/or rejects from three holes at either Chemex or Min-En Laboratories Ltd. (Min-En); 2) check assays and assays reported in both ppb and oz/t for samples from hole DI-07-13 at TSL; and 3) metallic or screen assays performed on samples with visible gold in holes DI-87-19 and DI-87-20 at Lakefield.

A total of 116 samples from drill hole DI-87-12 were submitted to the Chemex laboratory in Mississauga, Ontario. The assay certificate exists for the samples submitted to Chemex. The only information that exists regarding sample preparation or assaying procedures is that the fire assaying was performed on a 1/2 assay ton sample (Pope, 2008). Gold assays are reported on the certificate in oz/t with a detection limit of 0.002 oz/t. In addition, 153 check assays from hole DI-07-12 performed by Min-En are recorded on the Borsurv version of the assay log. No certificate exists for these assays.

A total of 289 reject samples from holes DI-87-18 and DI-87-20 were submitted to the Min-En laboratory in Vancouver, British Columbia. The assay certificate exists for these holes, although there is no information regarding sample preparation or assaying procedures. Gold assays are reported on the certificate in both g/t (minimum value reported is 0.01 g/t) and oz/t (minimum value reported is 0.001 oz/t).

The assays from 120 samples (out of a total of l22 samples) in hole DI-87-13 analyzed at TSL are reported on the certificate in both ppb and oz/t. In addition, duplicate assays were performed on 38 samples (31% of the samples in the hole).

Metallic or screen assays were performed at Lakefield on samples with visible gold in holes DI-87-19 (1 sample) and DI-87-20 (7 samples). The plus 100 mesh portion and minus 100 mesh portion were assayed and weighed; with the calculated grade of the sample being the weighted average of both fractions.

There is no record of internal quality assurance/quality control (QA/QC) measures in place at TSL or Lakefield (now SGS) at the time, although, with prevailing industry standards in place at all commercial laboratories, QA/QC processes were most likely conducted at both laboratories.

13.6.2 Deloro Minerals Ltd. Protocols, 1997 Check Sampling of Noranda Drill Core

Deloro submitted two batches of Noranda drill core samples (other core half) to Chemex in 1997 to confirm Au mineralization from the Noranda historical drilling. A total of 135 drill core halves were submitted from 15 drill holes (DI-87-04 to 06, DI-87-08 to 13, DI-87-17 to 19, DI-87-23 and DI-87-25 to 26).

As part of the Noranda duplicate sample checks, each sample was assayed twice by Chemex and a general comparison between the two datasets indicates that Au values occur in both datasets at values that are consistent with the Au mineralization of the deposit. There is some variation between individual Au assay values when compared (Deloro vs. Noranda and Deloro vs. Deloro) against each other, but the trend is not biased toward one set of data. However, the mean Au assay values for the Noranda samples are generally higher than the duplicate samples assayed by Deloro. This variation can be attributed to different factors such as the "nugget" gold nature of the deposit and the different analytical techniques used at each assay facility (TSL, Lakefield and Chemex).

13.6.3 Deloro Minerals Ltd. Protocol, 1997 Drill Core Samples

There are no records of external QA/QC measures or check assaying undertaken by Deloro on the core samples from the 1997 drill program. Deloro requested that Chemex ship all the pulps and rejects from the 1997 drill program to the MNDM core storage facility in Tweed, Ontario.

Chemex made available the results of their internal QC data generated during the assaying of core samples from the 1997 drill program to Deloro. QC data certificates exist for all the core samples submitted to Chemex. The Chemex internal quality control measures included the addition of standards into the sample stream at a rate of one in twenty (5%) overall. In addition, blanks were added at a rate of one in forty, and duplicate assays performed on every 40th sample.

Most of the standards in use at Chemex were secondary reference materials from internationally recognized sources (Chemex quotation dated September 3, 1997). Chemex used five different standards for the work.

All the blanks returned assays below the detection limit of 0.005 g/t.

Chemex re-assayed a total of 42 pulps during the 1997 drill program.

13.7 2007 Opawica Quality Control Programs

External QA/QC procedures for the 2007 drill program were conducted. Standards and blanks were inserted into the sample stream by the core cutter during sample packing, prior to shipment to the laboratory, at a rate of one standard and one blank for every twenty samples, according to the assay log.

In addition, an empty sample bag with a tag marked duplicate was inserted into the sample stream by the core cutter at a rate of one duplicate for every twenty samples, according to the assay log. For the duplicate samples, the laboratory was instructed to perform a duplicate assay on the reject material from the preceding sample.

Opawica used five standards during the program with gold values ranging between 0.597 g/t and 4.041 g/t. In addition, one sample also had a silver grade of 33.25 g/t. Two of the standards were prepared by Ore Research and Exploration Pty. Ltd. (OREAS) of Australia and supplied by Analytical Solutions Ltd. of Toronto, Ontario. Three of the standards were prepared by Rocklabs of Auckland, New Zealand and supplied by Mine Assay Supplies of Kirkland Lake, Ontario. The blank material used by Opawica was prepared by Rocklabs and consisted of pulp-sized material.

All standard and blank assays were reviewed on a regular basis under the supervision of Fred Sharpley, P.Geo., the Opawica QP for the project. Simple rules for accepting quality control data were established at the start of the program and adhered to: 1) if a gold standard falls beyond 10% of the accepted value, the standard is classified as a failure (accuracy); and 2) if a blank returns more than three times the detection limit, the analytical batch is classified as a failure (contamination). All the standard and blank failure batches were re-analyzed and passed the QC criteria on re-analysis. Only assay data that passed the QC criteria was accepted for entry into the drill logs and database.

Opawica submitted a total of 125 reject duplicates for routine re-analysis by Swastika during the 2007 drill program.

Internal quality control procedures by Swastika consisted of standards, blanks and duplicate samples. Standards and blanks were inserted at rate of one per batch that consisted of between 15 and 79 samples. In addition, 10% of the samples were re-assayed on the original pulp. Swastika reported the results of the internal quality control data with each dataset sent to Opawica and on the final certificates.

Swastika used one standard during the program with a grade of 3.557 g/t. The standard was prepared by Rocklabs of Auckland, New Zealand and supplied by Mine Assay Supplies of Kirkland Lake, Ontario. The blank material used by Swastika was also supplied by Mine Assay Supplies and consisted of silica sand.

Accepted values of the standard and blank used by Swastika are summarized in Table 13-1 as well as the standards and blanks inserted by Opawica.

TABLE 13-1: ACCEPTED VALUES OF STANDARDS AND BLANKS

Standard Reference Material	Supplied By	Inserted By	Accepted Value Au g/t	Accepted Value Cu %	Accepted Value Ag g/t
OREAS 7Pb	Analytical Solutions	Opawica	2.77		
OREAS 50Pb	Analytical Solutions	Opawica	0.841	0.744	
OREAS 52Pb	Analytical Solutions	Opawica	0.307	0.3338	
OREAS 53Pb	Analytical Solutions	Opawica	0.623	0.546	
OREAS 54Pa	Analytical Solutions	Opawica	2.9	1.55	
SE29	Rocklabs	Opawica	0.597		
SI25	Rocklabs	Opawica	1.801		33.25
SK33	Rocklabs	Opawica	4.041		
AUBLANK9	Rocklabs	Opawica	0.003		
Marble Blank		Opawica			
OxK48	Rocklabs	Swastika	3.557		

Swastika re-assayed a total of 233 pulps during the 2007 drill program.

A review of the duplicate samples submitted by Opawica and Swastika are provided as scatter plots in Appendix B. The individual Au sample variance indicated on these plots is similar to that of “nugget” type Au deposit with the mean Au assay value for each data set difference less than 5%.

13.8 2009 Opawica Quality Control Programs

External QA/QC procedures for the 2009 drill program were conducted. Standards and blanks were inserted into the sample stream by the core cutter during sample packing, prior to shipment to the laboratory, at a rate of one standard and one blank for every twenty samples, according to the assay log.

In addition, an empty sample bag with a tag marked duplicate was inserted into the sample stream by the core cutter at a rate of one duplicate for every twenty samples, according to the assay log. For the duplicate samples, the laboratory was instructed to perform a duplicate assay on the reject material from the preceding sample.

Opawica used four standards during the program with gold values ranging between 0.307 g/t and 2.9 g/t. The four standards were prepared by Ore Research and Exploration Pty. Ltd. (OREAS) of Australia and supplied by Analytical Solutions Ltd. of Toronto, Ontario. The blank material used by Opawica was decorative marble stone supplied by Home Hardware.

All standard and blank assays were reviewed on a regular basis under the supervision of Robert Laakso, P.Eng., the Opawica QP for the project. Simple rules for accepting quality control data were established at the start of the program and adhered to: 1) if a gold standard falls beyond 15% of the accepted value, the standard is classified as a failure (accuracy); and 2) if a blank returns more than five times the detection limit, the analytical batch is classified as a failure (contamination). All the standard and blanks passed the QC criteria. Only assay data that passed the QC criteria was accepted for entry into the drill logs and database.

Opawica submitted a total of 84 reject duplicates for routine re-analysis by Swastika during the 2009 drill program.

Internal quality control procedures by Swastika consisted of standards, blanks and duplicate samples. Standards and blanks were inserted at rate of one per batch that consisted of between 3 and 74 samples. In addition, 15% of the samples were re-assayed on the original pulp. Swastika reported the results of the internal quality control data with each dataset sent to Opawica and on the final certificates.

Swastika used three standards during the program with gold values ranging from 1.258 to 3.583 g/t. The standards were prepared by Rocklabs of Auckland, New Zealand and supplied by Mine Assay Supplies of Kirkland Lake, Ontario. The blank material used by Swastika was also supplied by Mine Assay Supplies and consisted of silica sand.

Accepted values of the standard and blank used by Swastika are summarized in Table 13-2 as well as the standards and blanks inserted by Opawica.

TABLE 13-2: ACCEPTED VALUES OF STANDARDS AND BLANKS FOR THE 2009 DRILL PROGRAM

Standard Reference Material	Supplied By	Inserted By	Accepted Value Au g/t	Accepted Value Cu %	Accepted Value Ag g/t
OREAS 50Pb	Analytical Solutions	Opawica	0.841	0.744	
OREAS 52Pb	Analytical Solutions	Opawica	0.307	0.3338	
OREAS 53Pb	Analytical Solutions	Opawica	0.623	0.546	
OREAS 54Pa	Analytical Solutions	Opawica	2.9	1.55	
Marble Blank	Home Hardware	Opawica			
OxJ64	Rocklabs	Swastika	2.366		
OxK69	Rocklabs	Swastika	3.583		
OxH66	Rocklabs	Swastika	1.258		
Blank	Mines Assay	Swastika			

Swastika re-assayed a total of 240 pulps during the 2009 drill program.

A review of the duplicate samples submitted by Opawica and Swastika are provided as scatter plots in Appendix A. The individual Au sample variance indicated on these plots is similar to that of "nugget" type Au deposit with the mean Au assay value for each data set difference less than 10%.

13.9 Bulk Density (SG)

Collection of bulk density data from the Opawica drill core was carried out in two phases. The first phase, carried out in January 2008, consisted of 11 samples of whole core about 20 cm in length that were selected after the geological logging was completed and before the core was sampled. The second phase, carried out in July 2008, consisted of 115 samples of sawn core about 20 cm in length that were selected after the assays were returned and the initial geological interpretation completed.

Approximately 90% of the samples were selected from the various mineralized zones within the Dingman granite, typically consisting of one sample per zone per drill hole. Samples were also collected from mineralized zones containing a range of sericite alteration, quartz veining and sulphide mineralization. Approximately 10% of the samples were selected from: 1) sections of weakly mineralized or barren granite; and 2) carbonate sediments in the immediate hanging wall or footwall of the granite.

The samples were submitted to Swastika where bulk density determinations were performed using the water immersion procedure.

No additional samples for bulk density testing were completed for the 2009 samples.

Mineralized zones, unmineralized granite and carbonated sediments drilled during the 2007 and 2009 drilling programs are similar. The bulk density values for granite and carbonated sediments used to estimate the January 29, 2009 resource tonnage have been used to estimate the July 16, 2009 resource tonnage.

14.0 DATA VERIFICATION

A selection of data verification checks were completed for the previous mineral resource estimate by Golder (Palmer et al., 2009), dated January 29, 2009, on the historical data and the Opawica drill hole sample data from 2005 to 2008 exploration programs. Therefore, only verification work was completed on the 2009 exploration data since sufficient verification checks were previously completed. A summary of the verification checks previously completed by Golder have been included in the following sections and the new verification checks on the 2009 data are also included.

14.1 Historical Data Verification (Palmer et al, 2009)

Data verification undertaken in 2007 focused on the digital Dingman diamond drill hole and channel sample database (Barnes database) received from Barnes. The Barnes database contains 536.1 of the 563.2 m of channel sampling completed by Noranda and 36 of the 52 diamond drill holes completed by Noranda and Deloro. The channel samples and drill holes contained in the Barnes database are from the western portion of the Dingman Property. Barnes received the drill hole data from Deloro as an electronic file and did not verify the data (Barnes, 1998). The channel sample data was digitized by Barnes from plan maps provided by Deloro.

Verification of the drill hole data in the Barnes database involved checking the collars, downhole surveys, lithology, alteration, quartz veining, sulphides and assays with the incomplete set of drill logs, reports, maps and cross sections available in 2007. Verification of the channel sample data was not possible due to the unavailability of the historical data.

The Noranda and Deloro data were made available to Opawica in February 2008. The historical data includes the Noranda drill logs and diamond drill core assay record sheets with assays, most of the Noranda drill core assay certificates, electronic data files of the Noranda drill holes in Borsurv format (a commercial drill hole collection and display software, now obsolete), Deloro re-logging and check sampling notes of the Noranda core, drill logs and assay certificates, Deloro downhole surveys notes, Deloro drill logs and assay certificates, Deloro core photographs, photographs of Noranda core taken by Deloro, Noranda geology, geochemical survey, channel sample and geophysical survey maps, Noranda geological and exploration reports, Noranda and Deloro resource and economic studies, Noranda mineralogical studies, Noranda metallurgical studies and assessments, and Noranda environmental studies.

14.1.1 Historical Drill Hole Collar Data (Palmer et al., 2009)

Verification of the Noranda drill hole collar data initially involved checking the grid coordinates and elevations of the holes contained in the Barnes database with the historical Noranda data including: 1) drill logs; 2) Borsurv files; 3) Appendix I in King (1988); 4) LeBaron’s 1:1000 scale geology map showing the drill hole collars and traces; and 5) plans and cross-sections used in the resource estimate by Huska (1989). The historical Noranda drill hole collar elevation data were also checked with the available elevation data from a survey by Miller in 2007.

With a few exceptions, the collar coordinates of the Noranda holes in the Barnes database could be verified with the various historical Noranda data sources. A small (typically less than 0.1 m) discrepancy in the collar coordinates between the drill logs and the Borsurv files for holes DI-87-01 to DI-87-13 is not considered significant and the collar coordinates in the drill logs were used in the database. The Noranda drill hole collar data for the holes missing from the Barnes database were entered into the database from the drill logs and Borsurv files and verified using all the available historical Noranda data. Numerous discrepancies were found between the various Noranda data sources and the Barnes database for the Noranda drill hole collar elevations.

Verification of the Deloro drill hole collar data initially involved comparison of the grid coordinates and elevations of the holes in the Barnes database with the historical Deloro data including: drill logs, field notes and tables and a copy of LeBaron's geology map. The collar coordinates of the Deloro holes in the Barnes database could be verified with the various historical Deloro data sources with the exception of holes DI-97-43 and DI-97-44. The Deloro drill hole collar elevations were also checked with the elevation data from the surveying completed by Miller in 2007. A number of discrepancies ranging up to 5 m were found between the historical Deloro and 2007 Miller elevation data.

Two of the 1987 Noranda drill hole collars were located in the field and surveyed in the UTM Zone 18 NAD 83 projection system by Miller during the May 2008 survey. Drill hole DI-87-06, located on line 150W in the western part of the property, consists of a B-size collar in outcrop with the correct azimuth and dip. Drill hole DI-87-02, located at approximately 350E in the eastern part of the property, consists of a BW casing in place with the correct azimuth and dip. Three wood stakes from the original Noranda baseline located on the eastern part of the property were identified and surveyed by Miller.

From the May 2008 survey, it was determined that the Rajong baseline surveyed in 2007 was not the original Noranda baseline, but was at a small angle, such that the Rajong baseline diverges from the original Noranda baseline by approximately 20 m south at line 200W.

The original Noranda grid was reconstructed from the Miller survey data of the two Noranda drill hole collars and the three wood stakes on the Noranda baseline. Collar coordinates in the UTM Zone 18 NAD 83 projection system of the remaining 36 Noranda drill holes were calculated using the original historical Noranda grid coordinates and the elevations were determined by projecting the collar location to a topography surface compiled from the 2007 and 2008 Miller survey data. A geodetic elevation of 221 m was set as equivalent to the Noranda grid elevation of 1,000 m in order to establish the elevation of the Noranda, Deloro and Opawica drill hole and channel sample data. The azimuth of the Noranda baseline is now set at 061.42°, slightly different from the azimuth of 060° shown on the Noranda maps. This discrepancy may be due to a magnetic declination of 10º West shown on the Noranda maps, which is different from the calculated magnetic declination for the property in 1987 of 11º 49' West (government of Canada website).

All 14 Deloro drill hole collars were located in the field in May 2008, and the collar coordinates surveyed in the UTM Zone 18 NAD 83 projection system by Miller. The Noranda grid coordinates of the Deloro drill hole collars were then re-calculated based on the results of the 2008 Miller survey. The surveyed collar coordinates for holes DI-97-43 and DI-97-44 agree with the locations on the historical maps and cross-sections, but not the drill logs.

Verification of the Opawica drill hole collar data involved a comparison of the UTM Zone 18 NAD 83 coordinates and elevations in the Opawica drill logs and Opawica Surpac database with the 2007 and 2008 Miller survey data. The Noranda grid coordinates of the Opawica collars were re-calculated based upon the results of the 2008 Miller survey. The reconciliation required an adjustment of the Opawica drill collars by up to 11 m grid west and 29 m grid south from their original planned locations. Therefore, the three twin holes originally drilled by Opawica are not twinned holes as planned.

14.1.2 Historical Downhole Survey Data (Palmer et al., 2009)

Verification of the Noranda downhole survey data involved checking the data contained in the Barnes database with the drill logs and Borsurv files. Numerous minor imperial to metric conversion errors in the Barnes database were found and corrected by Opawica. The Noranda downhole data for the holes missing from the Barnes database were entered into the database from the drill logs and verified with the Borsurv files. No spurious dip measurements exist in the Noranda downhole survey data.

Verification of the Deloro downhole survey data initially involved checking the data contained in the Barnes database with the drill logs and downhole survey field notes. Numerous minor imperial to metric conversion errors, discrepancies in correction for magnetic declination and spurious downhole survey results were detected and corrected by Opawica.

The Deloro downhole survey data was re-entered into the database from the original field notes. The imperial to metric conversion errors were corrected. The azimuth readings had been corrected by Deloro using a magnetic declination of 9º 23' West from grid or UTM North, apparently taken from the adjoining NTS map sheet 31 C/11, and did not account for the angle between grid north and astronomic north. The calculated magnetic declination for the property in 1997, from the government of Canada website, is 12º 14' West. The azimuth readings were corrected using a magnetic declination of 12º West by Opawica. Tests with spurious downhole azimuth readings were not used in the database.

There is no record of the azimuth direction or the inclination at the collar for the Deloro drill holes. The collar azimuth of the Deloro holes is assumed to be the same as the first reliable downhole azimuth reading. With few exceptions, the dip at the collar for most of the holes in the Barnes database is -45°, and the first downhole dip test typically varies between 1-3° from a 45° collar dip. The potential error in assuming the first azimuth reading or the 45° dip at the collar of the drill hole is considered negligible since the first downhole test was taken at 3 runs or 9 m past the casing.

14.2 2007 Opawica Downhole Survey Data (Palmer et al., 2009)

Verification of the Opawica downhole survey data involved checking the data in the drill logs with the original Flexit test records. The calculated magnetic declination for the property in 2008 from the government of Canada website is 12º 6' West. The azimuth readings were corrected using a magnetic declination of 12º West. Tests with spurious downhole azimuth readings were not used in the database. The collar azimuth of the Opawica 2007 holes is assumed to be the same as the first reliable downhole azimuth reading. The potential error in assuming this collar azimuth on the trace of the drill holes is considered negligible since the first test was taken at between 18 and 21 m down the hole. With few exceptions, the first downhole test, taken at between 18 and 21 m down the hole, is within 1° of the planned collar dip or inclination.

During the June 2008 site visit for the previous mineral resource estimate by Golder (Palmer et al., 2009), a selection of Opawica drill hole collars from the 2007 drilling program were surveyed with a hand-held GPS and matched with the drill hole collar co-ordinates from the Miller survey.

14.3 2009 Opawica Downhole Survey Data

The verification of the Opawica 2009 downhole survey data involved checking the data in the drill logs with the original Flexit test records. The calculated magnetic declination for the property in 2009 from the government of Canada website is 12º 6' West. The azimuth readings were corrected using a magnetic declination of 12.1º West. Tests with spurious downhole azimuth readings were not used in the database. The collar azimuth of the Opawica holes is assumed to be the same as the first reliable downhole azimuth reading. The potential error in assuming this collar azimuth on the trace of the drill holes is considered negligible since the first test was taken at between 18 and 21 m down the hole.

14.4 Historical Drill Hole Lithology or Rock Type Data (Palmer et al., 2009)

The Noranda lithology or rock type data was extracted from the Barnes database and checked with the original drill logs and Borsurv files. The lithology for the Noranda drill holes had been coded by Deloro during the program of re-logging, discussed in Section 12.3**,** and according to the scheme shown in Table 12-1. Although Deloro developed a more detailed lithology legend or scheme than the one used by Noranda, no significant changes to the original Noranda lithology units were found in the data, and the Deloro scheme was maintained with only minor modifications.

Numerous minor imperial to metric conversion errors in the Noranda lithology in the Barnes database were found and corrected by Opawica. Lithology data for the Noranda holes missing from the Barnes database were entered into the database from the drill logs and verified with the Borsurv files. As a final check, the lithology distances were compared with the sample intervals, and only a few instances of sampling across lithology contacts were found.

The Deloro lithology data were extracted from the rock type codes for each sample contained in the Barnes database. The extracted lithology data were then verified with the rock type codes in the original Deloro drill logs.

14.5 Opawica Drill Hole Lithology or Rock Type Data

Verification of the Opawica drill core logging and sampling procedures was carried out over five days in the period between January and March 2008. The work was carried out at the Opawica core shack in Matachewan, Ontario by Pat Pope, P.Geo., on behalf of Opawica. Key sections of 19 of the 20 holes completed by Opawica were reviewed to determine if the geological interpretation proposed in 2007 was valid and the data collection sufficient for undertaking the re-interpretation and constructing the 3D geologic model required for the resource estimate.

Verification of the Opawica 2009 core logging and sampling procedures was carried out by Terry Link and Bob Laakso, P. Eng., between February and May 2009. The work was carried out at the Opawica core shack in Matachewan, Ontario.

14.6 Historical Drill Hole Sample and Assay Data (Palmer et al., 2009)

A detailed review of the historical drill hole sample and assay data against Opawica's digital database was completed by Pat Pope on Opawica's behalf prior to providing the database for the previous mineral resource estimate (Palmer et al., 2009), dated January 29, 2009. A selection of historical logs from Noranda and Deloro against Opawica's database was completed and no transcription errors were indicated. The following section is a detailed description of the verification checks completed by Pat Pope.

Verification of the Noranda drill hole sample and assay data involved checking the data contained in the Barnes database with the Noranda diamond drill core assay record sheets, Borsurv files and assay certificates. Noranda sample and assay data for the holes missing from the Barnes database was entered into the database using the Noranda drill logs, drill core assay record sheets, Borsurv files and assay certificates. Check assay data collected by Noranda, discussed in Sections 13.2 and 13.3 above, was entered into the database from the assay certificates and the oz/t values converted to g/t using the conversion in Section 2. Assay data from the check sampling of Noranda core by Deloro, discussed in Sections 12.3, 13.2 and 13.3 above, was entered into the database using the Deloro field notes and assay certificates.

Verification of the sample data in the Barnes database for drill holes DI-87-03 to DI-87-13, submitted to TSL for assay, initially involved comparing the sample distances with the Noranda drill core assay record sheets and Borsurv files. Numerous small imperial to metric conversion errors were found in both the Barnes database and the Borsurv files. The un-sampled intervals in the Barnes database were deleted. The sample numbers were entered and the sample distances re-entered for all these holes and the distances converted from imperial to metric using the conversion of 1 foot = 0.3048 m. The sericite, quartz vein volume percent and total sulphide percent coding completed during the re-logging by Deloro for each sample interval was not verified with the descriptions in the Noranda drill logs. A final check of the sample intervals involved comparing the rock type codes of the sample intervals with the lithology distances in the drill logs and Borsurv files.

The Noranda sample data for holes DI-87-01 and DI-87-02, missing from the Barnes database, were entered into the 2008 database using the Borsurv data following the same procedures used for holes DI-87-03 to DI-87-13. Rock type, sericite, quartz vein volume percent and total sulphide percent was coded for each sample interval using the descriptions in the Noranda drill logs. A final check of the sample intervals involved comparing the rock type codes of the sample intervals with the lithology distances in the drill logs and Borsurv files.

Verification of the sample data in the Barnes database for drill holes DI-87-17 to DI-87-25, DI-87-27 and DI-87-28, submitted to Lakefield for assay, initially involved comparing the sample distances with the drill core assay record sheets and Borsurv files. Numerous small imperial to metric conversion errors were found in the Barnes database; the Borsurv data contained only a few distance errors and was subsequently used to create the final 2008 database. The un-sampled intervals in the Barnes database were deleted. The sample numbers were entered for all these holes, and the necessary imperial to metric conversion corrections made to the sample intervals in the Borsurv files. The sericite, quartz vein volume percent and total sulphide percent coding completed during the re-logging by Deloro for each sample interval was not verified with the descriptions in the Noranda drill logs. A final check of the sample intervals involved comparing the rock type codes of the sample intervals with the lithology distances in the drill logs and Borsurv files.

The Noranda sample data for holes DI-87-14 to DI-87-16, DI-87-26 and DI-88-29 to DI-88-38, missing from the Barnes database, were entered into the 2008 database using the Borsurv data and following the same procedures used for holes DI-87-17 to DI-87-25, DI-87-27 and DI-87-28. Rock type, sericite, quartz vein volume percent and total sulphide percent were coded for each sample interval using the descriptions in the Noranda drill logs. A final check of the sample intervals involved comparing the rock type codes of the sample intervals with the lithology distances in the drill logs and Borsurv files.

Verification of the assay data in the Barnes database for drill holes DI-87-03 to DI-87-13, submitted to TSL for assay, involved comparing the assays in g/t with the values in the Noranda drill core assay record sheets, TSL assay certificates and Borsurv files converted from oz/t to g/t using a conversion factor of 1 troy ounce = 31.1035 g. Although there were few discrepancies between the assays in the Barnes database and Borsurv files, small errors had been introduced by the original values in ppb being converted to oz/t and then to g/t. The assays in ppb or oz/t were re-entered from the assay certificates for these holes, with the values in oz/t taking precedence. Assays in ppb were converted directly to g/t with values below the detection limit of 5 ppb entered as 1/2 the detection limit or 0.003 g/t. The oz/t values were converted to g/t using the conversion factor of 1 troy ounce = 31.1035 g.

The Noranda assay data for holes DI-87-01 and DI-87-02, missing from the Barnes database, were entered into the 2008 database using the same procedure as for holes DI-87-03 to DI-87-13.

Verification of the assay data in the Barnes database for drill holes DI-87-17 to DI-87-25, DI-87-27 and DI-87-28, submitted to Lakefield for assay, involved comparing the assays in g/t with the values in the Noranda drill core assay record sheets, Lakefield assay certificates and Borsurv files converted from oz/t to g/t using the conversion factor of 1 troy ounce = 31.1035 g. With the exception of hole DI-87-20, there were few discrepancies between the assays in the Barnes database, Borsurv files and assay certificates. All the assay values in these holes were checked with the assay certificates and the necessary corrections made. Values below the detection limit of 0.001 oz/t were entered as 1/2 the detection limit or 0.0005 oz/t (0.017 g/t). The oz/t values were converted to g/tonne using the conversion factor of 1 troy ounce = 31.1035 g.

The Noranda assay data for holes DI-87-14 to DI-87-16, DI-87-26 and DI-88-29 to DI-88-38, missing from the Barnes database, were entered into the database using the Borsurv data and following the same procedures used for holes DI-87-17 to DI-87-25, DI-87-27 and DI-87-28. Verification of the Deloro drill hole sample data in the Barnes database involved checking 5% the data with the original drill logs. Sample numbers were entered for all the Deloro holes and the un-sampled intervals were deleted from the database. No errors were found in the sample data in the Barnes database.

Verification of the Deloro drill hole assay data in the Barnes database involved checking 5% the assays and all the values greater than 10 g/t with the assay certificates. One error detected in the values below 10 g/t was corrected by Opawica. A number of small errors found in the values greater the 10 g/t were corrected. Values below the detection limit of 0.005 g/t were entered as 1/2 the detection limit or 0.003 g/t. The internal Chemex QC data, discussed in Section 13.3 of this report, was entered into the database from the assay certificates.

14.7 2007 Opawica Drill Hole Sample and Assay Data (Palmer et al., 2009)

A detailed review of Opawica's drill hole sample and assay data against Opawica's digital database was completed by Pat Pope on Opawica's behalf prior to providing the database for the previous mineral resource estimate by (Palmer et al., 2009). A selection of Opawica's Microsoft Excel core logs was reviewed against Opawica's database and no transcription errors were indicated. The following section is a detailed description of the verification checks completed by Pat Pope and Golder.

Verification of the Opawica sample data in core involved selective comparison of the sample tags and downhole distances. In one instance, a sample interval was corrected in the drill log by Opawica. Bracket sampling of the hanging wall and footwall sedimentary lithologies was generally confined to one or two samples; however, sediments containing any appreciable amount of sulphides were typically sampled.

Verification of the Opawica sample data in core also included checking the coding of sericite alteration, quartz vein percent and total sulphide percent. This work focussed primarily on the sericite alteration; a significant amount of re-coding was completed to ensure consistency with the logging legend and the Noranda and Deloro data. The quartz vein and total sulphide data were not checked to the same level of detail as the sericite. The quartz vein and sulphide data appeared reasonable and only a few revisions were made. The rock type codes for the sample intervals were checked to ensure consistency with the lithology units.

Verification of the Opawica assay data in the drill logs involved checking 5% the assays and all the values greater than 10 g/t with the assay certificates. Values below the detection limit of 0.01 g/t are entered as 1/2 the detection limit or 0.005 g/t. No errors were found in the checks of the assay data against the original laboratory certificates or Opawica's Microsoft Excel core logs when compared to Opawica's database. The verification checks were completed by Pat Pope and Golder.

Golder received the corrected drill hole database, containing all historical drill holes and the Opawica 2007 drill holes, from Opawica in four digital files (collars, downhole surveys, lithology and assays) using a Comma Separated Value (CSV) text format.

As part of the Golder verification, approximately 10% (243) of the 2007 assays supplied in the CSV files to the assay certificates from Swastika were checked by randomly selecting values from the assay CSV file and comparing to the assay certificate from Swastika. The assay certificates were supplied to Golder in PDF format. All checked values in the logs matched the values in the individual assay certificates. An additional check of 123 assays above 5 g/t Au was conducted by comparing the assays in the supplied CSV files to the assay certificates from the 1987-1988, and the 1997 drilling campaigns. No transcription errors were found in the database assay entries checked.

14.8 2009 Opawica Drill Hole Sample and Assay Data

Robert Laakso, P.Eng., reviewed a selection of Opawica's Microsoft Excel core logs against Opawica's database and did not find any transcription errors.

Verification of the Opawica sample data in core involved selective comparison of the sample tags and downhole distances. Bracket sampling of the hanging wall and footwall sedimentary lithologies was generally confined to a few samples; however, sediments containing any appreciable amount of sulphides were typically sampled. The rock type codes for the sample intervals were checked to ensure consistency with the lithology units.

Verification of the Opawica assay data in the drill logs involved checking 10% of the assays and all the values greater than 2 g/t with the assay certificates. Values below the detection limit of 0.01 g/t are entered as 1/2 the detection limit or 0.005 g/t. No errors were found in the checks of the assay data against the original laboratory certificates or Opawica's Microsoft Excel core logs when compared to Opawica's database. The verification checks were completed by Robert Laakso, P.Eng., and Terry Link.

Shaft & Tunnel received the drill hole database, containing Opawica 2009 drill holes, from Opawica in four digital files (collars, surveys, lithology and assays) using a Comma Separated Value (CSV) text format.

As part of the Shaft & Tunnel verification, approximately 10% (150) of the 2009 assays supplied in the CSV files were checked by randomly selecting values from the assay CSV file and comparing to the assay certificate from Swastika. The assay certificates were supplied to Shaft & Tunnel in PDF format. All checked values in the assay file matched the values in the individual assay certificates.

14.8.1 Channel Sample Data (Palmer et al., 2009)

Verification of the Noranda channel sample data contained in the Barnes database initially involved plotting the channel sample traces with sample numbers and assays and comparing them against the original Noranda channel sample maps (West and Middle Sheets). The channel sample traces were also compared with the locations shown on the 1:1000 scale geology map of LeBaron (1986). The channel sample traces and assays showed a very close correspondence with the original maps and no errors were found.

The Noranda channel sample elevation data contained in the Barnes database was also checked with the limited elevation data from the 2007 Miller survey. Numerous discrepancies, ranging up to 7 m, were found between the Barnes and Miller data for the Noranda channel sample elevations.

Channel sample data for the eastern portion of the Dingman Property was digitized from the original Noranda channel sample map (East Sheet). Initial verification of the data involved comparing the digitized data with the original channel map and the channel locations shown on the 1:1000 scale geology map of LeBaron (1986). Elevations of the channel samples were initially estimated from the nearest available drill collar, and the inclination of all the channels assumed to be horizontal.

Approximately 35% (34 out of the 96 total) of the Noranda channels were located in the field in May 2008, and the coordinates surveyed in the UTM Zone 18 NAD 83 projection system by Miller. In addition, the bends and end points of the channels were also surveyed where possible, and the data used to calculate the azimuth and inclinations of the channel traces.

Collar coordinates for the remaining 62 channels were converted to the UTM Zone 18 NAD 83 projection system using the original historical Noranda grid coordinates and the reconstructed Noranda grid. The collar elevations of the 62 Noranda channel samples were calculated by projecting them to the topography surface created in Surpac 6.0 from 2007 and 2008 Miller surveys. The traces of the 62 Noranda channels were defined by using the azimuths digitized from the original maps with the inclinations re-calculated so the channels lie on the topographic surface.

As a final check, the channel sample traces were plotted and compared to the original Noranda channel sample maps by Opawica. In the western part of the grid, the surveyed channels generally plot within 1-2 m of the original locations and the channel traces compare reasonably well with the historic data, suggesting that the reconstructed Noranda grid is well located in the UTM Zone 18 NAD 83 projection system. In the eastern part of the grid, the surveyed channels vary between 2 and 10 m from the original locations, with much of the discrepancy in the easting coordinates. The reason for this discrepancy is not known, but suggests that the reconstructed Noranda grid is not as accurate in the eastern part of the property.

Gold assays are plotted on the original Noranda channel sample maps in oz/t with a minimum value of 0.001 oz/t. In the Barnes database, the gold assays are in oz/t, sections of the channels with no samples or no assays are minus one, and nil values are zero. No assay certificates exist for the channel samples; therefore, a detection limit of 0.001 oz/t was assumed and the nil values assigned a value of one half the detection limit. Channel sections with no sample or assay value were assigned a zero value. The oz/t values were converted to g/t using the conversion factor of 1 troy ounce = 31.1035 g. 1 troy ounce / ton (short) = 34.28 g / tonne. Since no assay certificates were available for the Noranda Channel samples this data was not included in the 2009 mineral resource estimate.

14.9 2007 Check Assaying of Opawica Pulp and Reject Samples at Swastika (Palmer et al., 2009)

Following the completion of the 2007 Opawica drill program, 223 pulp and 114 reject samples were submitted for re-analysis at Swastika. The pulp and reject samples were chosen at random from the total of 2,283 samples analyzed during the 2007 drill program, representing approximately 10% of pulps and 5% of rejects. All samples were assigned a new sample number, and were re-submitted to Swastika for fire assay using the same sample preparation and assay procedures as the drill core samples.

External QA/QC procedures for the check assay program at Swastika were provided by standards inserted by Opawica into the sample stream prior to shipment to the laboratory, at a rate of one for every twenty samples. Opawica used three standards during the check assay program with gold values ranging between 0.597 g/t and 4.041 g/t. The standards were prepared by Rocklabs of Auckland, New Zealand and supplied by Mine Assay Supplies of Kirkland Lake, Ontario.

Internal quality control procedures by Swastika consisted of standards, blanks and duplicate samples. Standards and blanks were inserted at rate of one per batch that consisted of approximately 60 samples. In addition, 20% of the samples were re-assayed on the original pulp. Swastika reported the results of the internal quality control data with each dataset sent to Opawica and on the final certificates.

Swastika used one standard during the check assay program with a grade of 2.366 g/t Au. The standard was prepared by Rocklabs of Auckland, New Zealand and supplied by Mine Assay Supplies of Kirkland Lake, Ontario. The blank material used by Swastika was also supplied by Mine Assay Supplies and consisted of silica sand.

Swastika re-assayed a total of 43 pulps during the check assay program.

Swastika re-assayed a total of 20 pulps prepared from the reject samples during the check assay program.

14.10 2007 Check Assaying of Opawica Pulp and Reject Samples at Activation Laboratories Ltd. (Palmer et al., 2009)

Following the completion of the 2007 Opawica drill program, 226 pulp and 110 reject samples were submitted for re-analysis at Activation Laboratories Ltd. (Actlabs). Actlabs is an ISO 17025 certified laboratory. The pulp and reject samples were chosen at random from the total of 2,283 samples analyzed during the 2007 drill program, representing approximately 10% of pulps and 5% of rejects.

The pulp and reject samples were submitted to the Actlabs sample preparation laboratory in Timmins, Ontario. Sieve tests were performed on 10 pulp samples, or approximately every 20th sample. Four of the ten pulp samples failed to meet the normal Actlabs standard of 95% passing a 0.10 mm or 150 mesh screen. The pulp samples were re-homogenized and shipped for analysis at the main Actlabs laboratory in Ancaster, Ontario.

Sieve tests were performed on ten reject samples, or approximately every 10th sample. All the reject samples failed to meet the normal Actlabs standard of 75% passing a 2 mm or 10 mesh screen. The reject material was mechanically or riffle split to arrive at a sub sample of 250 g. The sub sample was pulverized to ensure a minimum of 95% of the material passed through a 0.10 mm or 150 mesh screen (Procedure RX1). The pulp samples prepared from the rejects were shipped for analysis at the main Actlabs laboratory in Ancaster, Ontario.

Fire assaying was performed on a 30 g sample drawn from the pulp. The gold bead was assayed using either atomic absorption spectrometry (Procedure 1A2) or a gravimetric technique for values greater than 3,000 ppb (Procedure 1A3). Gold values are reported on the certificates in ppb for assays performed by atomic absorption spectrometry and in g/t for assays performed by gravimetric technique. Values below the detection limit of 5 ppb are reported as below detection limit.

External QA/QC procedures for the check assay program at Actlabs were provided by standards inserted by Opawica into the sample stream prior to shipment to the laboratory. Standards were inserted into the sample stream at the rate of one for every twenty samples. Opawica used three standards during the check assay program with gold values ranging between 0.597 g/t and 4.041 g/t. The standards were prepared by Rocklabs of Auckland, New Zealand and supplied by Mine Assay Supplies of Kirkland Lake, Ontario.

Internal quality control procedures by Actlabs consisted of standards, blanks and duplicate samples. Each batch of 42 samples contained 3 standards, 2 blanks and 3 pulp duplicate samples. Actlabs reported the results of the internal QC data with each batch sent to Opawica and on the final certificates.

Actlabs used seven standards during the check assay program with gold values ranging between 0.77 g/t and 18.14 g/t. Five of the standards were prepared by CDN Resource Laboratories Ltd. of Delta, British Columbia and two of the standards were prepared by Rocklabs of Auckland, New Zealand.

Actlabs re-assayed a total of 17 pulps during the check assay program.

Actlabs re-assayed a total of 9 pulps prepared from the reject samples during the check assay program.

14.11 2008 and 2009 Shaft & Tunnel Site Visits

Shaft & Tunnel collected and delivered all 2009 core during the 2009 drill program. Robert Laakso, as QP, visited the Matachewan core logging and storage facility. Logging practices, core splitting, review of quality assurance procedures and storage facilities were observed. A review of the logging procedures showed that the core boxes are clearly labelled with the drill hole number and downhole distances, and are covered in a secure storage area. Mr. Laakso also made a number of visits to the Dingman Property during the 2009 drilling program to confirm that the work was carried out and to review drill holes. Illustrated on Figures 14-1 through to 14-10 are a selection of figures of the Matachewan core logging facility and the Dingman Property.

FIGURE 14-1: MATACHEWAN CORE STORAGE FACILITIES

FIGURE 14-2: DINGMAN CORE TRAYS WITH METAL TAGS

FIGURE 14-3: DINGMAN DRILL HOLE TAGS ON CORE TRAYS IDENTIFYING HOLE NUMBER AND DOWNHOLE DISTANCE

FIGURE 14-4: DINGMAN SPLIT CORE – HOLE # DI-07-07 - MARKED IN PREPARATION FOR LOGGING

FIGURE 14-5: DINGMAN SAMPLES PREPARED FOR DELIVERY TO ASSAY LAB

FIGURE 14-6: CORE HALVES WITH SAMPLE TAGS AT THE MATACHEWAN FACILITY

FIGURE 14-7: DRILLING DI-09-01

FIGURE 14-8: DRILLING DI-09-02

FIGURE 14-9: EXAMPLE OF BOLT/ALUMINUM TAG ASSEMBLY FOR BHID

FIGURE 14-10: DRILLING DI-09-09

15.0 ADJACENT PROPERTIES

There are no immediately adjacent properties to the Dingman Property containing significant gold mineralization. There are records of historical properties in the general area outlined by regional geology map OGS Map P.3402, and the Mono-Bannockburn gold property outlined in an exploration proposal by J. Baldry, P.Eng., dated December 31, 1986.:

- Gawley Shaft – gold property occurrence located west of Dingman Property.
- Sovereign Mine – gold mine located south of Dingman Property.
- Deloro Mine – gold mine located south of Dingman Property.
- Richardson Mine – gold mine located east of Dingman Property.
- Mono-Bannockburn – gold property located north east of Dingman.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Historical Mineral Process and Metallurgical Testing

16.1.1 1987-1989 Metallurgical Testwork (RPA, 1997)

Outlined in the RPA report from January 10, 1997 (RPA, 1997) for Rajong is a summary of metallurgical testwork that has been completed on the Dingman Property in 1987, 1988 and 1989 by two commercial laboratories (Witteck and Lakefield).

Summarized in the RPA report are descriptions of conventional bottle roll cyanidation testwork on 9 samples, and leach column tests (1 at Witteck and 4 at Lakefield) on 5 composited samples. Sample test results form the bottle roll testing (material crushed to ½ inch) indicated Au extractions ranged from 39.3 %to 64.9% with increased extracting occurring for finer crushed material.

The results from the 5 column leach tests (for heap leaching) indicated gold extractions ranged from 41.4% to 86.5% (material crushed to minus ½ inch) with increased extraction occurring for finer crushed material.

Estimates from the testwork by Rajong indicated Au recoveries on the order of 70% for material crushed to less than 3/8 inch.

In addition, RPA identified that the calculated head grades (from bottle roll and column leach testing) were more representative of the true gold grades compared to initial head assay because of the larger weight of samples tested (1 kg for bottle test and 8 kg or more for column leach). Also, samples that were a created from a composite of drill core samples had an Au average value closer to the calculated Au head grades.

16.1.2 2005 Metallurgical Testwork (Dymov, 2005)

In 2005, Edward Neczkar submitted 17 reject samples of Deloro drill core to Lakefield for metallurgical testwork (Dymov, 2005). The samples were inventoried and weighed by Lakefield. No information was available to determine which drill holes from the property were selected for the testwork.

The 17 samples selected for the testwork were from 4 drill holes (DI-97-41, DI-97-42, DI-97-43 and DI-97-46) located in the central area of the deposit (between -20E and -70E Noranda grid). The range of gold values for these samples was between 0.985 g/t and 7.76 g/t with a combined average grade of 2.53 g/t.

Lakefield created a single bulk sample by splitting the 17 samples in half. One half of each sample was retained and the other half was used to create the bulk sample. Two head samples assays for gold were collected for the bulk sample and were 1.96 g/t and 1.99 g/t, respectively.

The bulk sample was submitted for a standard "bottle roll" cyanidation test. The procedure provided by Lakefield describing the "bottle roll" cyanidation test includes adding lime and NaCN to the pulped bulk sample (grind size was 98% passing -200 mesh) in water allowing a leach of 48 hours. During the test, the pH and NaCN concentration is maintained. The pregnant solution and residue bulk sample from the test are assayed for Au.

The results of the test work indicated that the recovery of gold was 97.5% leaving a bulk sample residue of 0.05-0.06 g/t. The calculated head grade from the cyanidation test metallurgical balance was 2.19 g/t.

No additional historical test work has been completed on the Dingman Property.

16.2 Opawica Mineral Process and Metallurgical Testing (Palmer et al., 2009)

In March 2008, Opawica completed initial and limited whole rock analysis on portions of drill core from the 2007 drilling at Dingman. This initial testing was completed by Swastika Laboratories and the results identified that arsenic, mercury and antimony content were virtually at undetectable levels. The conclusions of this initial testing are consistent with an independent valuation report by RPA (1997), in which the Dingman rocks are determined to be substantially free of such deleterious elements from limited and initial historic tests.

In July, 2008, Opawica contracted Gekko in Ballarat, Australia to conduct testwork on Dingman samples to establish the amenability of Dingman material to crushing, gravity flotation, intense cyanidation and cyanide leaching. A summary of the testwork is provided by a document authored by Michael Braaksma of Gekko (Braaksam, 2008) and excerpts are taken from that document and from email correspondence from Gekko to Opawica in order to provide a summary of testwork completed at the effective date of the Technical Report (Gekko, 2009).

It is Gekko's recommendation that a combination of gravity and flotation recovery stages of Dingman material can produce a gold concentrate that contains 97.7% of the head material tested and 93.1% of the sulphur. Flotation tail grades on testing completed to date are 0.09 g/t and 0.06% sulphur. It is recommended by Gekko that flotation optimization testwork should be completed to improve concentration grades (Gekko, 2009). The flotation optimization studies, recommended by Gekko, are still ongoing and results are pending.

16.2.1 Summary of Gekko Mineral Process and Metallurgical Testwork (Palmer et al., 2009)

The following summary is based on a review test work provided by a document authored by Michael Braaksma of Gekko (Braaksam, 2008) and includes a Microsoft Excel spreadsheet showing the Gravity flotation results. Only factual test information is provided and any conclusions ore recommendations provided by Gekko.

A single bulk sample (LOPA-B) was provided to Gekko by Opawica and it was comprised of 173 samples from 10 Opawica drill holes for a total of 171 m of core sample length. The purpose of the testwork was to establish the amenability of the Dingman material to crushing, gravity concentrate, flotation, intense cyanidation and cyanide leaching.

A review of the head assay grade, size analysis grades and single pass gravity tabling testing grades is summarized in Table 16-1.

TABLE 16-1: HEAD GRADE SUMMARY FOR SAMPLE LOPA-B (BRAAKSAM, 2008)

Test	Calculated Head Grades			
	Au (g/t)	S (%)	Cu (ppm)	Ag (g/t)
Average Head Grade	2.15	0.70	43	<1.0
Size Analysis	2.04	0.68	45	1.0
Single Pass Tabling Test	1.53	0.54	34	1.35
Sighter Rougher Flotation Test	1.44	0.33	100	1.32

The average head grade assay was based on 4 samples tested from sample LOPA-B. The Au assay ranged between 1.65 g/t to 3.0 g/t with an average of 2.15 g/t.

As discussed by Gekko, the samples' response to Vertical Shaft Impactor (VSI), which is a method of size reduction, was poor; this was due to the extremely small breakage rate per pass measured through the test rig. The Single Pass Tabling (Wilfrey Shaking Table) test completed indicated that a gravity jigging circuit operating at a coarse crush will recover 72.3% of the gold with a mass recovery of 10.1%. A grinding mill is the recommended size reduction method.

The results from the Single Pass Tabling Test were based on a 31 kg sample with a two stage rougher/cleaner tabling test and indicated results that 52.5% of the gold and S could be captured by pulling 2.2% of the mass. Recoveries increased to 73% if 10.1% of the mass was pulled. The calculated head grades from the Single Pass Tabling Test were 1.53 g/t Au and 0.54 % S.

A Microsoft Excel spreadsheet was provided to Opawica discussing the preliminary results of the flotation test defined as a Sighter test (no. 26698F-1) – 3 stage rougher (P80=88µm). The calculated head grades from the rougher flotation testing were 1.44 g/t Au and 0.33 % S with 91.3% of the gold and S captured after the 2 flotation stages and 93.3 % of the gold and S captured after the 3 flotation stages.

A final report of all the test work completed on the LOPA-B sample from the Dingman Property is pending.

16.3 2009 Summary of Gekko Opawica Leach Test Results

In 2009 Gekko completed additional processing testing including Bond Work Index (BWi), progressive grinding tabling test and Gravity-Flotation-Intensive Leach (GFIL) on different grinds to test for gold recoveries.

The BWi testwork shows that the tested material has an average hardness at 15.4 kwh/tonne. The VSI amenability test indicates that the material tested was not amenable to size reduction using a VSI Crusher.

A progressive grind tabling test, which simulates the effect of a jig circuit in the grinding circuit circulating load, was performed at P_80 300, 150 and 106 µm. The Progressive Grind Tabling Test was followed by conventional flotation with 1, 3 and 6 minute concentrates pulled.

The laboratory was able to concentrate 96.1% of the gold in the feed to a mass of 5.3%.

The original concentrate sample used was leached, without additional grinding, in 2% NaCN with 2 kg/tonne PbNO3 and oxygen for 24 hours. The results from this leach test showed 78% gold dissolution in this time, which was contrary to the geological assessment that the gold was predominantly free. A second leach test on table concentrate was performed at the same conditions as the initial leach test with an additional grinding stage bringing the particle size to P_100 53 µm. This leach test had much improved dissolution rate of 97.7% after 2 hours of leaching time.

The overall GFIL recovery of the ore is expected to be 93.9% (Crowie, 2009).

Gekko's final report on the above testing can be found in Appendix C.

17.0 MINERAL RESOURCE ESTIMATES

17.1 Introduction

The July 16, 2009 Dingman Property Mineral Resource Estimate was calculated under the direction of Robert Laakso, P.Eng., QP. This is the first update (August 2009) to the original independent NI 43-101 mineral resource estimate completed for the Dingman Property (March 2009). The updated additional mineral resource estimate was for gold and was based on mineralized zones created using Surpac software.

The additional mineralized zone limits are 112.5 m W to 112.5 m E, 30 m N to 300 m N and 300 m to 890 m elevation (local grid coordinates).

The overall mineralized zone limits are 280 m W to 490 m E, 120 m S to 300 m N, and 300 m to 1,030 m (local grid coordinates).

The Mineral Resources defined from the 2009 drilling was added to the initial Mineral Resource Estimate, in the Inferred category only, as outlined in Table 17-3 and the July 16, 2009 Mineral Resource Estimate is summarized in Table 17-4 based on a 0.40 g/t Au Eq cut-off grade and Au capped grades of 30.0 g/t.

17.2 Drill-Hole Data

The Dingman 2009 drill hole database used in the additional resource estimate consisted of a total of 16 holes drilled by Opawica in 2009.

The drill hole data was provided to Shaft & Tunnel by Opawica in four CSV files, as provided in Table 17-1.

TABLE 17-1: 2009 DRILL HOLE DATA FILES

File name	Content	Date
DINGMAN_COLLAR_2009_DDH_Only.CSV	Drill hole collar data	June 30, 2009
DINGMAN_SURVEYS_2009_DDH_Only.CSV	Drill hole survey data	June 5, 2009
DINGMAN_LITHOLOGY_2009_DDH_Only.CSV	Drill hole lithology data	June 4, 2009
DINGMAN_ASSAY_2009_DDH_Only.CSV	Drill hole assay data	June 30, 2009

The data was imported into Surpac. Local grid coordinates were used for drill hole collars. A summary of the data is presented in Table 17-2.

TABLE 17-2: 2009 DRILL HOLE INFORMATION

Campaign	Year	Type	Number	Metres
Opawica	2009	Drill holes	16	3,926

The 2009 diamond drill program was designed to explore the Dingman granite at depth below all previous drilling between section 100W and 100E to a vertical depth of 700 m, to complete infill drilling from sections 150E to 250E, and to delineate additional aggregate resources from 350E to 700E. A detailed description of the drilling campaigns is given in Section 11. A list of drill hole names and collar locations is provided in Appendix B.

17.3 January 29, 2009 Estimation Methodology

The January 29, 2009 Dingman Property Mineral Resource Estimate was calculated by Greg Warren of Golder under the direction of Paul Palmer, P.Geo., P.Eng., and Greg Greenough, P.Geo., of Golder as the QPs. This was the first time an independent NI 43-101 mineral resource estimate had been completed for the Dingman Property.

The mineral resource estimate was based on a total of 72 drill holes, including 52 from historical drilling and 20 from the 2007 Opawica drilling program, with a total length of 11,812 m.

A single continuous mineralized envelope was constructed, largely constrained by the Dingman granite, using an approximate cut-off grade of 0.40 g/t Au. 6,278 samples, with a total length of 6,141 m, lay inside the mineralized envelope. Samples were composited to 2 m lengths.

Block model limits are 280 m W to 490 m E, 120 m S to 190 m N and 660 m to 1,030 m elevation (local grid coordinates). The block model was orthogonal to the local grid system and the blocks measure 10 m East by 5 m North by 10 m elevation.

The Ordinary Kriging interpolation method was used for resource estimation of Au g/t using variogram parameters defined from a geostatistical analysis.

The Indicated Resource for the Dingman Property includes a total of 8,801,000 tonnes at 0.97 g/t Au and the Inferred Resource includes 5,673,000 tonnes at 0.76 g/t Au. These resources are based on a 0.40 g/t Au cut-off grade and a capping Au strategy of 30 g/t. Provided in Appendix A are tables with a range of Au cut-off grades from 0.1 g/t to 1.0 g/t for 0.05 g/t increments using two Au capping strategies: 30 g/t and 15 g/t.

The Au cut-off grade is based on cost information provided by Opawica (Robert Laakso, P.Eng., QP) and assumes the deposit will be mined by open pit methods. No recoveries (mining or metal) or dilution factors have been considered in these estimates, and the results should be considered strictly in situ.

TABLE 17-3: JANUARY 29, 2009 DINGMAN PROPERTY MINERAL RESOURCE ESTIMATE SAMPLES CAPPED TO 30 AU G/T

Classification	Cut-off Grade Au (g/t)	Tonnes	Au (g/t)	Au (oz)
Indicated	0.40	8,801,000	0.97	275,000
Inferred	0.40	5,673,000	0.76	138,000

Note: Tonnes and ounces are rounded to the nearest 1000.

17.4 Estimation Methodology 2009 Program (July 16, 2009)

Additional mineralized zone grades for Au were estimated using inverse distance squared in Surpac Version 6.1.2 software. The additional mineralized zones were based on 16 drill holes from the 2009 Opawica drilling program, with a total length of 3,926 m. The additional mineralized zones were modelled, within the Dingman granite, using an approximate cut-off grade of 0.40 g/t Au. 291 samples, with a total length of 289.6 m, lay inside the additional mineralized zones. Outlines of mineralized zones were created using drill hole assay composites and guided by geology and upper mineralized zone interpretations. The outlines were created every 25 m starting at 100 W to 100 E along the local grid coordinates. The software calculated a weighted average Au grade for each outlined zone and then, using section width and bulk density, calculated a weighted average grade and tonnage for each section. Final calculations using the section tonnages and grades gave a total weighted average grade and tonnage of an additional Inferred resource of 5,628,000 tonnes at 1.2 g/t Au.

17.5 July 16, 2009 Updated Resource Statement

The Indicated Resource for the Dingman Property remains at 8,801,000 tonnes at 0.97 g/t Au, from the previous resource estimate as outlined in Table 17-3. The July 16, 2009 addition to the Inferred Resource has been estimated to be 5,628,000 tonnes at 1.2 g/t Au (Table 17-4) and, when combined with the previous resource of 5,673,000 tonnes at 0.76 g/t Au (Table 17-3), the total Inferred Resource is increased to 11,301,000 tonnes at 0.98 g/t Au as outlined in Table 17-5. These resources are based on a 0.40 g/t Au cut-off grade and a capping Au strategy of 30 g/t.

No recoveries (mining or metal) or dilution factors have been considered in these estimates, and the results should be considered strictly in situ.

TABLE 17-4 JULY 16, 2009 DINGMAN PROPERTY ADDITIONAL INFERRED MINERAL RESOURCES SAMPLES CAPPED TO 30 AU G/T

Classification	Cut-off Grade Au (g/t)	Tonnes	Au (g/t)	Au (oz)
Inferred	0.40	5,628,000	1.2	217,000

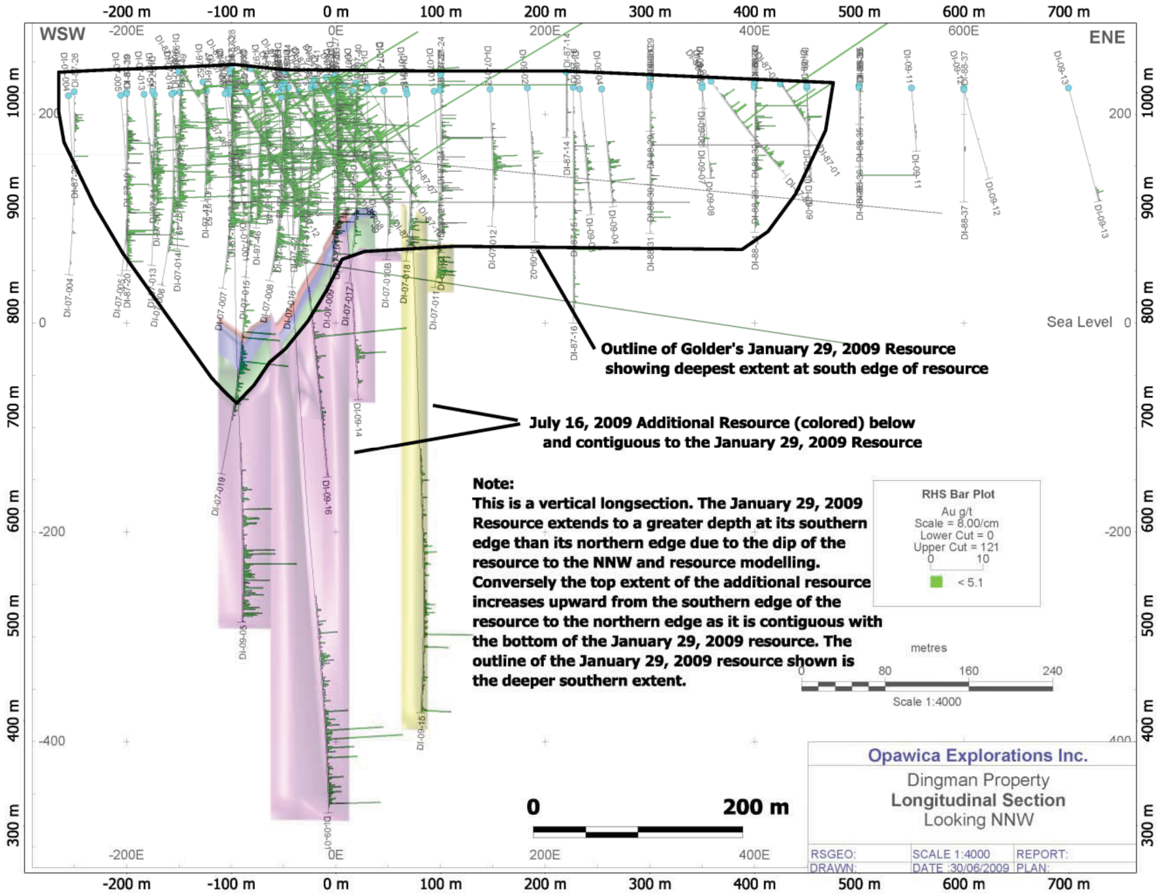
Note: Tonnes and ounces are rounded to the nearest 1000.

TABLE 17-5 JULY 16, 2009, DINGMAN PROPERTY UPDATED MINERAL RESOURCE ESTIMATE SAMPLES CAPPED TO 30 AU G/T

Classification	Cut-off Grade Au (g/t)	Tonnes	Au (g/t)	Au (oz)
Indicated	0.40	8,801,000	0.97	275,000
Inferred	0.40	11,301,000	0.98	355,000

Note: Tonnes and ounces are rounded to the nearest 1000.

FIGURE 17-1: LONGITUDINAL SECTION SHOWING JULY 16, 2009 ADDITIONAL RESOURCE BELOW THE JANUARY 29, 2009 RESOURCE



17.6 July 16, 2009 Revised Aggregate Resource Statement

As of July 16, 2009, Mr. Robert Laakso, P.Eng., reports that, as a result of the 2009 exploration program on the gold bearing granite, additional information regarding the host rocks have also been determined.

The granite intrusive stock has forced its way through a limestone (marble) formation that is a known source for limestone in the area. There is very little overburden covering the limestone while the granite is totally exposed as a prominent hill in the area. Studies carried out by Robert Laakso have defined a 75,000,000 tonnes Indicated Resource of granite and limestone aggregate exist on the property.

This aggregate resource has been calculated using 56 drill holes, of which 16 were drilled in 2009, 18 were drilled by Opawica in 2007 and 22 are historic holes drilled by previous operators Noranda and Deloro. The aggregate resource has been calculated using the drill holes and outcrop between grid lines 350 W to 800 E (1150 m total), between 400 N to 100 S (500 m total) and from vertical depths to 200 m.

During the 2009 exploration program, it became apparent that the revenue from the waste rock has a positive impact on the economics of this property. Field mapping to identify the bedrock, as well as a number of additional holes were drilled specifically to confirm the extent of the limestone (marble) at the East end of the claims. The mining of limestone and granite out to the boundaries of the claims has been considered and, as such, the aggregate resource increased substantially from the initial 14 million tonnes (originally considered as open pit waste on a ratio of 1:1), to 75 million tonnes of valuable and readily available aggregate.

Studies carried out by Robert Laakso have found that the limestone is a marketable aggregate product throughout the Toronto corridor from Burlington to Kingston, Ontario, as aggregate for road base, concrete as well as rip-rap (fill) along Lake Ontario shore areas and to a lesser extent as ornamental stone. The Dingman property is within economic trucking range of the Greater Toronto Area ("GTA").

In addition to the limestone, a quantity of the waste rock from stripping of the potential open pit operations will be portions of the granite itself. Based on mineralogy tests to date, the granite is sufficiently free of deleterious minerals, which enables it to become a premium aggregate for the manufacture of asphalt.

Some principal quarries in the inner GTA are becoming depleted or are closing due to dust and noise issues near Toronto suburbs. Quarries in the outer GTA are less obtrusive to residents living in the inner GTA.

The Dingman property is much better located for premium aggregate compared to the granite quarries found further from the GTA in Huntsville, Ontario, and areas north of there.

Recent valuations for the above products range from $CDN 8 to $CDN 10 per tonne F.O.B. (freight on board) at the Dingman Property site. The $CDN 10 figure is for aggregate destined to asphalt plants.

Marble and limestone products used for building materials such as chips for stucco, terrazzo, and stone sell for $100 to $130 a tonne. In 2006, six of the top ten quarries in Ontario mining limestone for cement and aggregate produced over 15 million tonnes. The Dingman marble which is quite pure appears to be suitable as a source for most of the product mentioned above.

The value and volume of selected industrial mineral production in the Southern Ontario Region are highlighted below to indicate the magnitude of the industry (Source: Sangster, P.J., Laidlaw, D.A., Lebaron, P.S., Steele, K.G., Lee, C.R., Carter, T.R., and Lazorek, M.R., 2008. Ontario Geological Survey, Open File Report 6222, 60p.).

Structural Materials	Quantity ($x10^6$ tonnes)	Value ($ million)
Cement	5.9	650
Lime	0.9	118
Sand & Gravel	99.6	490
Stone	61.8	629
Totals	**168.2**	**1,887**

Quotations from various local mine contractors have outlined their services to drill, blast and crush the rock to required sizes, stockpile same and load trucks on demand. The cost of producing aggregate in this area ranges in price from a low of $CDN 4.30 to a high of $CDN 5.45 per tonne.

The above costs and quotations for aggregate products are current estimates only and do not constitute executed contracts with mine contractors or for the purchase of aggregate products at this time. Such contracts cannot be completed until mine or quarry permitting has been obtained and mining operations have been commissioned. Mine or quarry permits have not been obtained by Opawica at this time.

18.0 OTHER RELEVANT DATA AND INFORMATION

To the knowledge of the authors of this report, there is no other relevant data and information concerning the current property.

19.0 CONCLUSIONS

Opawica has completed 16 additional drill holes (in 2009) on the Dingman Property which have increased the mineral resource. The Inverse Distance interpolation method was used for additional resource estimation of zones outlined by the 2009 drill holes.

The core logging and QA/QC sampling program implemented by Opawica for the 2009 drilling campaign was reviewed and included property and core logging facility site visits by the Shaft & Tunnel QP. Opawica's drill hole database was verified against logs, surveys, and assay certificates. The results of these reviews indicated that Opawica's database and QA/QC sampling program are to industry standards and acceptable for mineral resource evaluation.

Gekko has completed additional processing testing including: Bond Work Index that shows that the material tested had an average hardness at 15.3 kwh/tonne, progressive grind tabling tests followed by conventional flotation that resulted in concentration of 96.1% of the gold to a mass of 5.3%, and Gravity-Flotation-Intensive Leach on different grinds with an overall gold recovery of 93.9% expected.

As a result of the 2009 diamond drilling program in the gold zones, the Inferred Resource for Au has increased by 5,628,000 Tonnes (1.2g/t Au) for a total in this category of 11,301,000 Tonnes (0.98 g/t Au). The Indicated resource estimate has remained unchanged. Drilling to the east to confirm continuity of the limestone (marble) has raised the aggregate resource to 75,000,000 Tonnes of granite and limestone. This may have a positive effect on the overall property development.

20.0 RECOMMENDATIONS

Opawica has completed an Updated Technical Report of the Mineral Resource Estimate for the Dingman Property. As the project advances, the following steps are recommended:

- Exploration drilling is required to:
  - Infill drilling of the Indicated Resource “higher” grade core between 200 West and 100 East with the purpose of potentially increasing the confidence in these areas to a higher resource classification.
  - Infill drilling of the Inferred Resource east of the Indicated Resource with the purpose of potentially increasing the confidence in these areas to a higher resource classification.
  - Deeper drilling to confirm the down-dip potential of the resource.
- Development of a robust drilling database system. Currently, all data is stored in a Microsoft Excel file which does not include built-in data verification.
- Bulk sample of approximately 40,000 tonnes.
- Duplicate samples should be collected in all samples above 5 g/t to determine variability in these samples based on review of QA/QC data.
- Future mineral resource estimation should review Au capping strategy currently used.
- A more detailed assessment for all categories of aggregate resources located on the property should be conducted once the current drilling program has been completed east of 150 m East.
- Costs associated with maintaining and/or acquiring surface rights.
- Baseline studies and monitoring for permitting and environmental work.
- Collection of geotechnical data to support a preliminary open pit mining assessment.
- Preliminary open pit mining assessment of the Au resource and the aggregate resource.
- Survey the 2009 drill program collars by an Ontario Land Surveyor.

20.1 Estimate Cost of 2010 Exploration Programs

PHASE I	
Drilling	
At least 4 deep holes at up to 700 m per hole and 10 shallow holes at 150 m per hole, say 4,500 m – at $100/m all in cost	$ 450,000
Contingency in-fill drilling on strike and to depth, 25 holes averaging 300 m per hole, say 7,500 m – at $100/m all in cost	$ 750,000
Personnel	
Project Manager – 9 months at $10,000 / month	$ 90,000
Geologist – 6 months at $7,000 / month	$ 42,000
Field Technician-assistants – 2 at 6 months at $5,000 / month	$ 60,000
Transportation	
Vehicle rentals and core transport – 6 months	$ 15,000
Room and Board	
Lodging and Meals – 6 months	$ 30,000
Drill Core and Assay Costs	
Shipping – 12 months	$ 3,000
Assays – up to 4,000; 1 m interval samples	$ 80,000
Core Splitting/Storage – 12 months	$ 40,000
Reports/Supplies and Other	
Reports	$ 70,000
Supplies	$ 25,000
Communications	$ 15,000
SUBTOTAL PHASE-I WITH CONTINGENCY DRILLING	**$ 1,670,000**
Contingency (10%)	$ 167,000
PROPOSED TOTAL	**$ 1,837,000**
PHASE II	
Preliminary Scoping and Baseline Studies (2009-2010)	
Contingent upon positive results from above programs: Preliminary Scoping Studies with final in-fill drilling/Baseline/Environmental Study Costs (2009-2010)	**$ 1,250,000**

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22.0 CERTIFICATE OF QUALIFIED PERSON

Certificate of Qualified Person can be found in Appendix D.

23.0 DATE AND SIGNATURE PAGE

The report was prepared and signed by Robert W. Laakso, P.Eng., of Shaft & Tunnel. The effective date of this technical report is August 31, 2009.

SHAFT & TUNNEL ENGINEERING SERVICES LTD.

Original signed and stamped by:

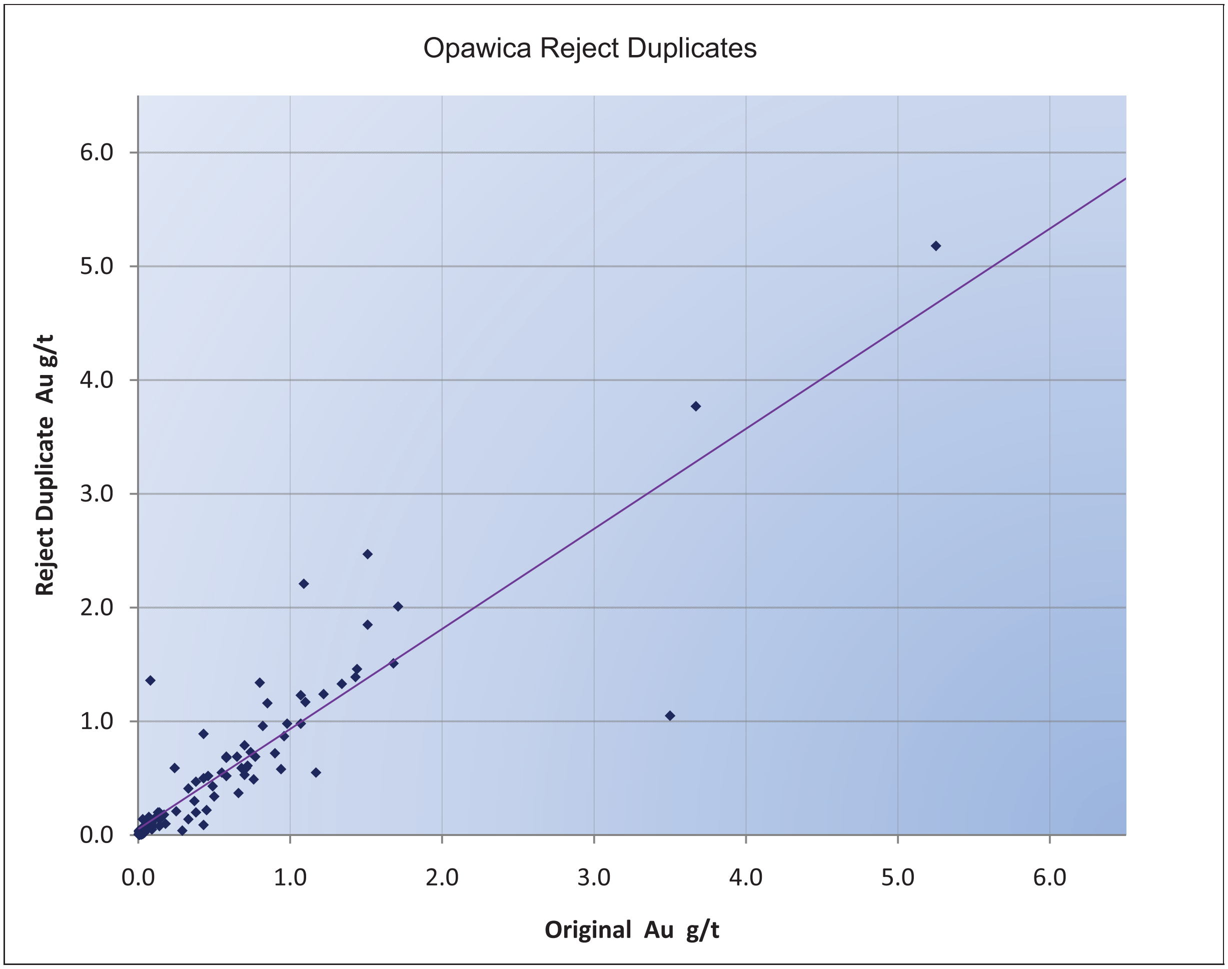
Robert W. Laakso, P.Eng.
Mining Consultant

APPENDIX A

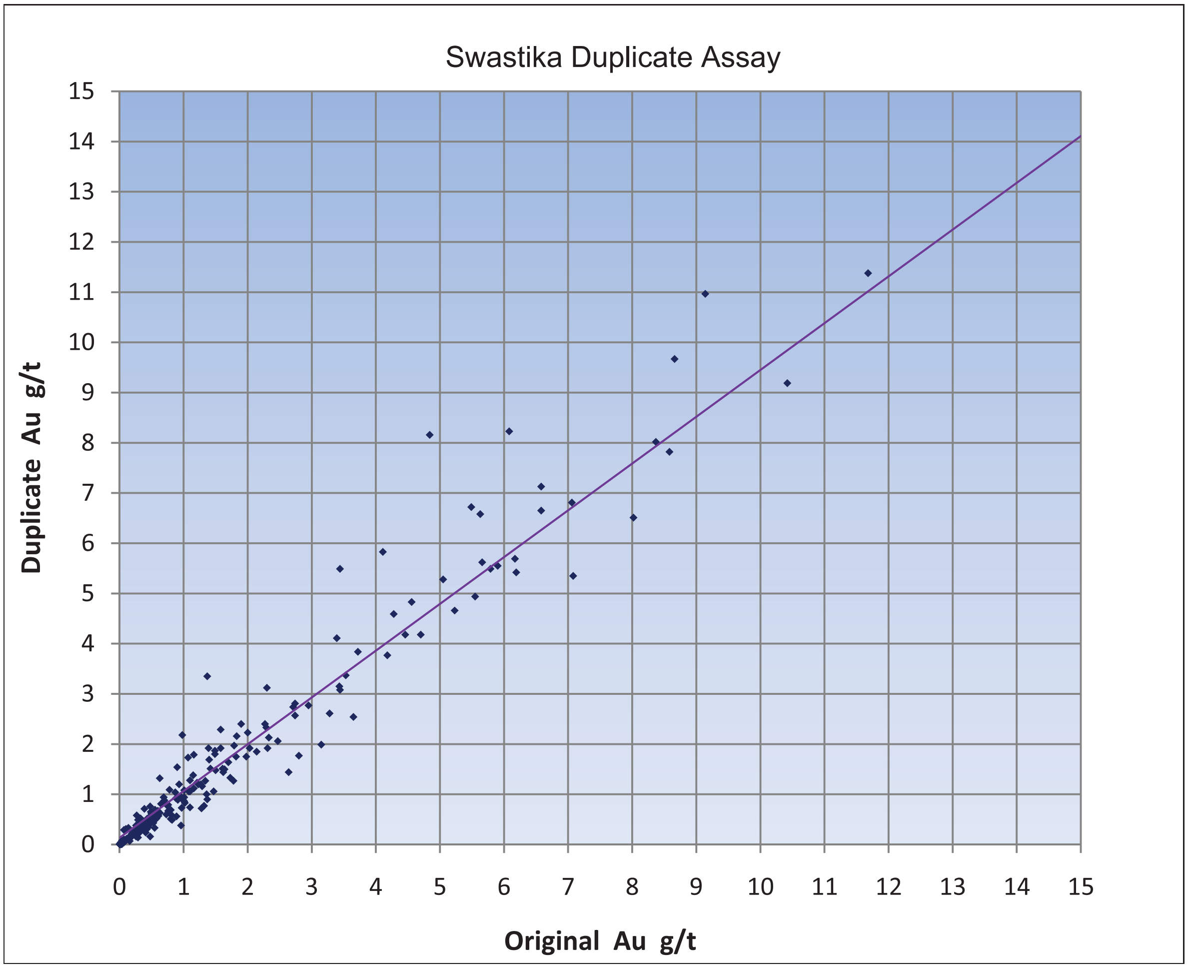
DUPLICATE SAMPLE ASSAY PLOTS

APPENDIX A

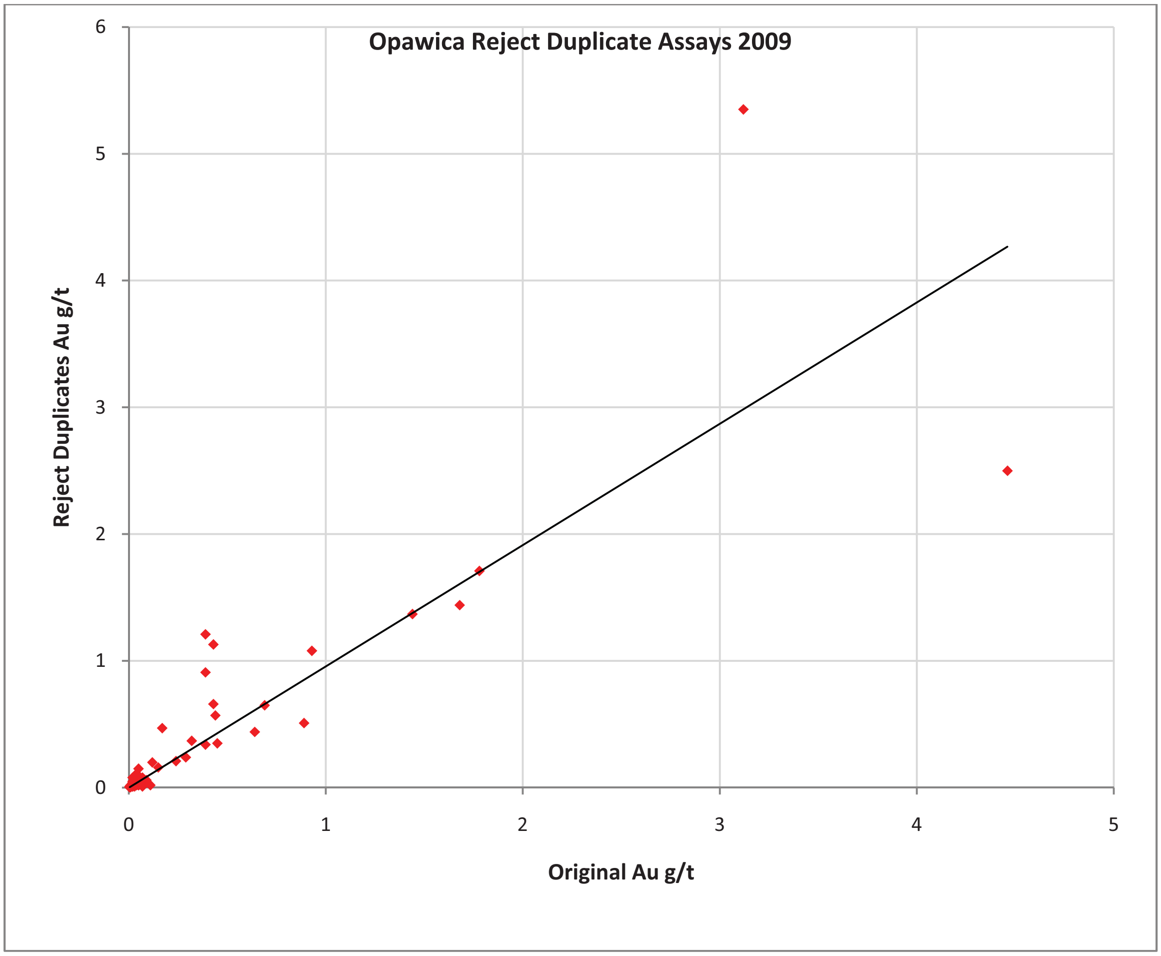
FIGURE A-1: SWASTIKA LABORATORY DUPLICATE ASSAY CHARTS



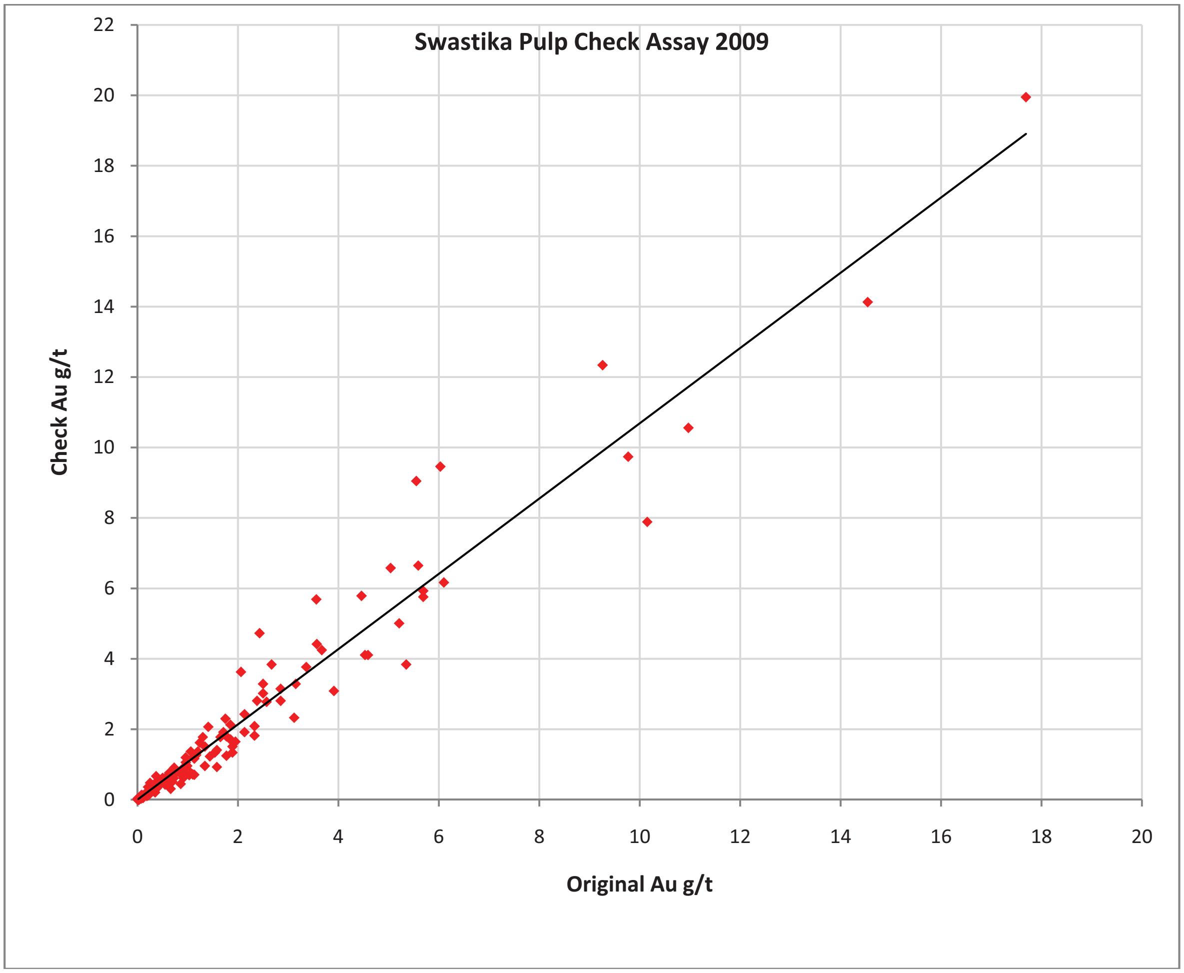
Original		Reject Duplicate	
Count	125	Count	125
Minimum	0.005	Minimum	0.005
Maximum	6.510	Maximum	5.490
Mean	0.483	Mean	0.477
Median	0.090	Median	0.100
StdDev	0.9195	StdDev	0.8650



Original		Duplicate	
Count	233	Count	233
Minimum	0.005	Minimum	0.000
Maximum	115.540	Maximum	106.970
Mean	2.248	Mean	2.227
Median	0.760	Median	0.720
StdDev	7.9180	StdDev	7.4021



Original		Reject Duplicate	
Count	84	Count	84
Minimum	0.005	Minimum	0.005
Maximum	4.46	Maximum	5.35
Mean	0.260	Mean	0.284
Median	0.03	Median	0.03
StdDev	0.6648	StdDev	0.7181



Original		Duplicate	
Count	240	Count	240
Minimum	0.005	Minimum	0.005
Maximum	17.69	Maximum	19.95
Mean	1.138	Mean	1.209
Median	0.25	Median	0.245
StdDev	2.2526	StdDev	2.4682

APPENDIX B

DRILL HOLES WITH COLLAR LOCATIONS AND COMPOSITE ASSAYS

APPENDIX B

TABLE B-1: DINGMAN 2009 DRILL HOLES AND COMPOSITE ASSAY INTERVALS USED IN RESOURCE ESTIMATION

BHID	XCOLLAR	YCOLLAR	ZCOLLAR	Depth (m)	Composite From (m)	Composite To (m)	Length (m)	Composite Grade Au g/t	Company
DI-09-01	-51.7	411.3	995	714					Opawica
					518	531	13	0.587	
					551	562	11	1.173	
					579	584	5	0.794	
					609	658	49	1.34	
					687	705	18	1.109	
DI-09-02	182.5	109.5	1004	207					Opawica
					85	89	4	0.88	
DI-09-03	233.2	112.6	1003	174					Opawica
					79	84	5	0.59	
					108	114	6	0.7	
DI-09-04	254.2	113.7	1003	162					Opawica
					72.4	75	2.6	0.701	
					96	109	13	0.965	
					126	130	4	1.518	
DI-09-05	-98.7	351.8	998	528					Opawica
					389	393	4	0.945	
					411	427	16	1.246	
					453	522	69	1.444	
DI-09-06	350.3	80.9	1007	31.5					Opawica
DI-09-07	350.4	80.9	1007	84					Opawica
DI-09-08	349.5	141.2	1004	123					Opawica
DI-09-09	448.9	141.1	1004	123					Opawica
DI-09-10	449.4	80.2	1006	84					Opawica
DI-09-11	548.7	80	1004	84					Opawica
DI-09-12	599.6	-4.6	1004	123					Opawica
DI-09-13	699.7	44.9	1004	150					Opawica
DI-09-14	0.6	241	1004	324					Opawica
					267	281	14	1.196	
					304	307	3	0.66	

BHID	XCOLLAR	YCOLLAR	ZCOLLAR	Depth (m)	Composite From (m)	Composite To (m)	Length (m)	Composite Grade Au g/t	Company
DI-09-15	68.4	325.8	998	615					Opawica
					444	453	9	0.625	
					503	515	12	0.757	
					534	548	14	1.103	
					611	613	2	3.205	
DI-09-16	-49.5	241	1004	399					Opawica
					227	229	2	2.51	
					249	256	7	3.12	
					274	277	3	0.923	
					295	297	2	0.775	
					331	333	2	0.72	

APPENDIX C

GEKKO REPORT VERSION: 2.0 DATED AUGUST 28, 2009

Quote: TO455 – Testwork Quote

Dingman

Laboratory Testwork Report

Opawica Exploration

Approved by:
Michael Braaksma
Technical Coordinator
Accepted by:
Tad Crowie

Version: 2.0
Date: 2009/08/28
Status: Final

For more information

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About this document

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Revision History

Version	Date	Author	Description
1.0	14/11/2008	Michael Braaksma	Preliminary Testwork Report
1.1	15/01/2009	Michael Braaksma	Preliminary Testwork Report
2	28/08/2009	Tad Crowie	Final Testwork Report

Executive Summary

This report describes the testwork results currently available on the **Dingman ore** sample. The head grades of the sample are summarised in Table A;

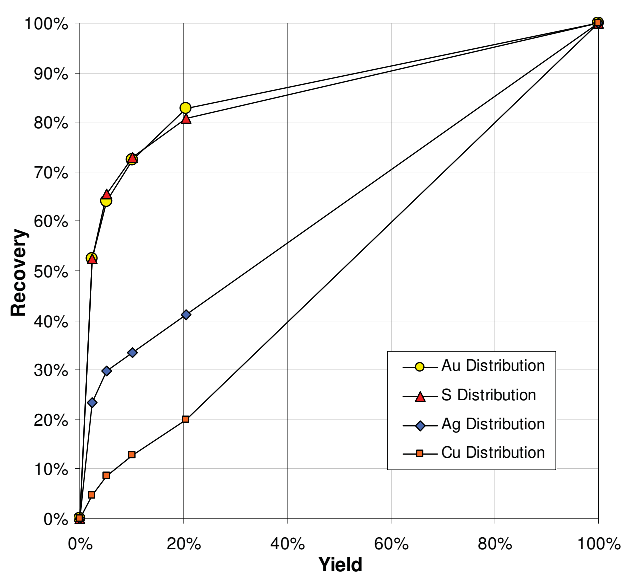
Table A: Head Grade Summary

Test	**Calculated Head Grades**			
	Au (ppm)	S (%)	Cu (ppm)	Ag (ppm)
Average Head Assay	2.15	0.70	43	<1.0
Size Analysis	2.04	0.68	45	1.0
Single Pass Tabling Test	1.53	0.54	34	1.4

The response to the VSI amenability test was poor. The ore tested cannot be crushed using a Vertical Shaft Impactor. The Bond Ball Mill Work Index is 15.4 kwh/tonne which is in the medium range for typical ores.

The response of the material to the gravity separation in the circulating load of a grinding circuit is shown by the results in Graph A. Over 50% of the gold can be recovered at mass yields less then 5%.

Graph A: Progressive Grind Tabling Results



A single flotation test conducted at a P80 of 88 µm was very successful in boosting Gold and Sulphur recoveries to high levels. Flotation tail grades of 0.09ppm Au were achieved. Table B summarises the results.

Table B: Combined Gravity-Flotation Test Summary

		Copper		Gold		Sulphur	
	Wt%	ppm	Dist%	ppm	Dist%	%	Dist%
T.C. 1	2.3	70	1.4	35.3	36.3	12.5	46.6
T.C.1+F.C.1	3.0	312	8.1	66	88.7	16.1	78.6
T.C.1+F.C.1&2	3.9	334	11.4	53	94.5	13.4	86.5
T.C.1+F.C.1,2&3	5.3	377	17.5	40	96.1	10.4	90.7

Further flotation test optimisation work should be carried out on the gravity tails samples or fresh ore samples. This should be directed at maximising grades and recoveries.

The leach testing showed that the gold contained in the gravity and flotation concentrates is very fine. The dissolution on a sample without additional grinding was 78%. A sample of the gravity concentrate was reground to P_{100} 53µm and then leached in standard bottle roll conditions. This leach test achieved 97.7% dissolution.

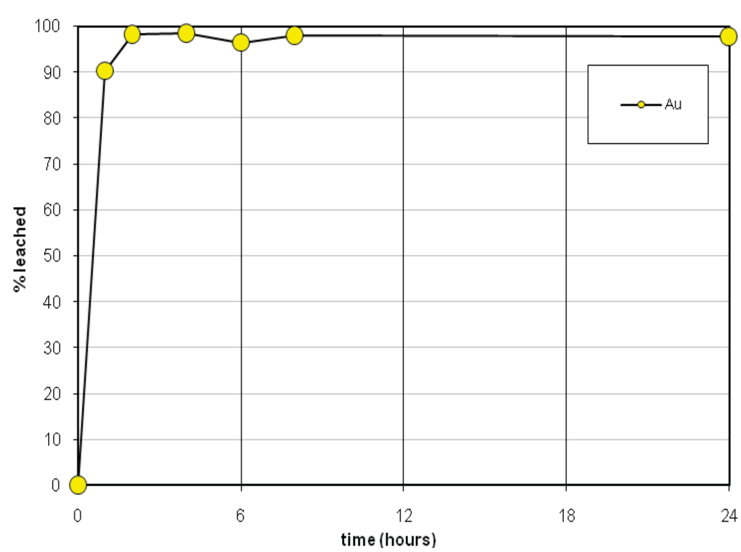


Table of Contents

1. Introduction

1.1. Scope

The purpose of the testwork was to establish the amenability of the ore to crushing, gravity concentration, flotation, intense cyanidation and cyanide leaching.

2. Sample Receipt and Preparation

2.1. Sample Receipt

The assigned name for the sample was:

- **LOPA - B**

2.2. Sample Preparation

The sample was prepared to P100 - 850µm by using jaw crusher, Rolls Crusher and a VSI. All the material was combined, homogenised and working samples split out using a laboratory scale rotary splitter.

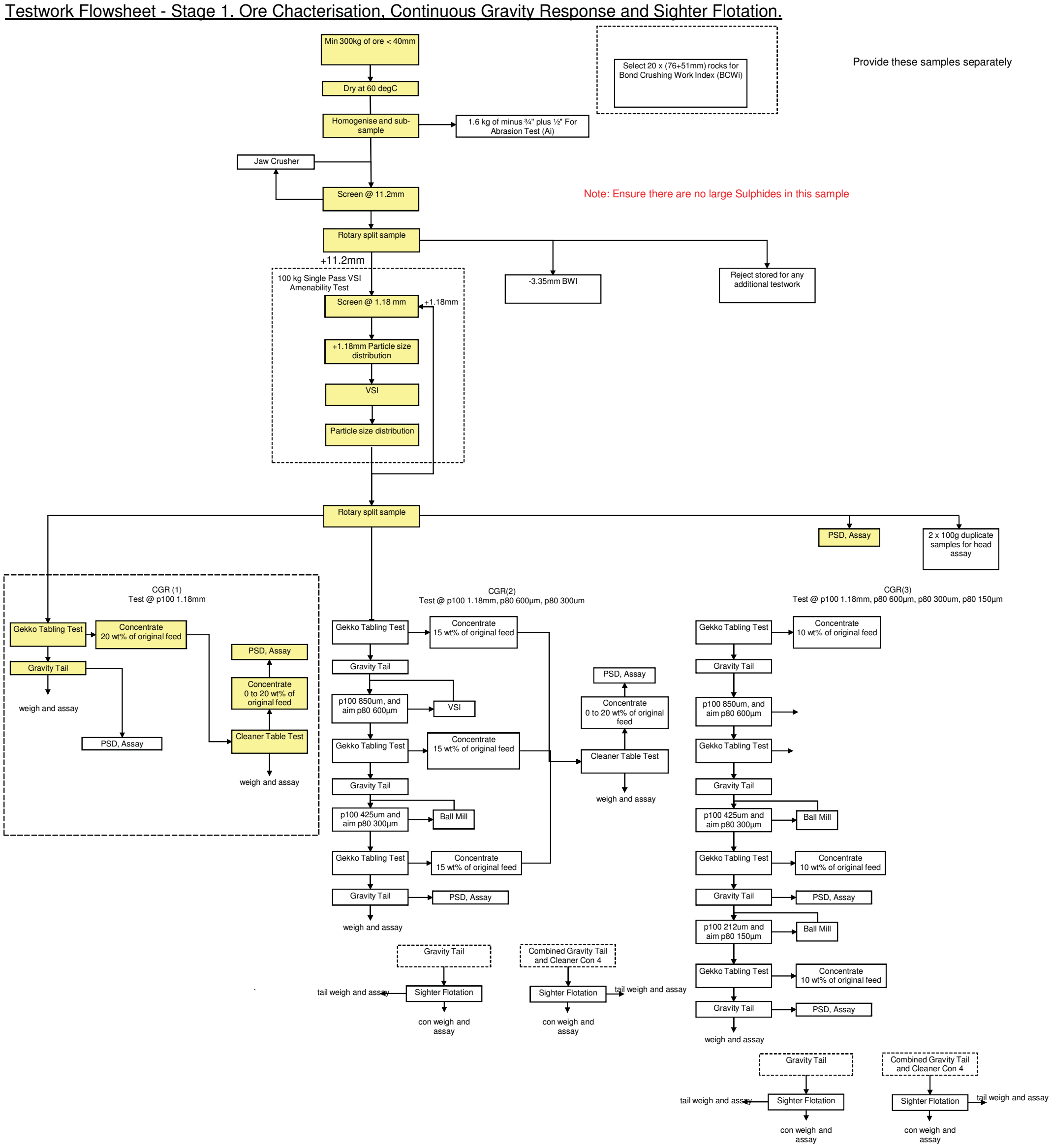
3. Metallurgical Tests

3.1. Testwork Flowsheet

The testwork flowsheet outlining all steps of the program completed to date is shown in Figure 1 below.

Table 1: Testwork Flowsheet

Testwork Flowsheet - Stage 1. Ore Chacterisation, Continuous Gravity Response and Sighter Flotation.



3.2. Sizing Analysis Method

The "as received/prepared" sample was subjected to a size by grade distribution analysis. The sieve sizes used (where applicable) were 13200µm, 9500µm, 8000µm, 6700µm, 4750µm, 2360µm, 1700µm, 1180µm 850µm 850µm, 600µm, 425µm 300µm, 212µm 150µm, 106µm 75µm, 53µm and 38µm. The -75µm, 53µm and the -38µm fractions were obtained via washing 100% of the sample over a 75µm and a 38µm screen using a pneumatic sieve.

The +75µm fraction was dried in a low (50 ºC) temperature oven and subsequently dry sieved through the nest of sieves (as listed above) using an electric shaker for 20 minutes.

The weight of each sample fraction was recorded and each size fraction was sent for assay.

3.1. VSI Amenability Test

This test is conducted to determine the amenability of processing the sample through a vertical shaft impactor crusher. The test involves screening the sample at sieve size of 850 µm and then passing the oversize through a lab sized VSI. The feed and product size ranges are then measured and compared. The amount of fines generated (<0.85 mm) is one measure of the viability of using this method of size reduction.

3.2. Progressive Grind Tabling Method

A laboratory size Wilfrey shaking table is used for the tabling test. The sample is initially slurried before being carefully added to the top of the table. The angle of the table is then adjusted to produce the required mass recovery to the concentrate streams. A thin film of town water is applied to the shaking surface and several concentrate ports are available to separate the products. The tails are then taken and reduced in size using either a VSI or rod mill, and then re-passed over the table. The tails of the second stage can then be reduced in size again for third pass over the table. The concentrates from each pass are then re-tabled to produce a number of concentrate products.

The products are collected in large tubs that are not allowed to overflow during the test.

3.3. Flotation Testing

The gravity tails were combined with Table Conc. 2, 3 and 4, and this composite was used for the flotation test. A laboratory Denver flotation cell (2.5L) was used for a rougher flotation test using the following conditions:

- No regrind. As received.
- 10 min condition time with PAX and Copper Sulphate
- Three rougher float concentrates taken. Additional conditioning time before each float commenced (2, 1 and 1 minutes).
- 50 g/t Pax and 200 g/t $CuSO_4$

The tests were undertaken with a view to recover all sulphides floatable, using a strong unselective collector.

3.4. Intensive Cyanidation Test Method

The intensive cyanidation tests are conducted in rolling bottles with samples taken at regular intervals to determine leach kinetics. Under standard conditions bottled oxygen is added to the leach to provide a high oxygen environment for leaching and lead nitrate is added to minimise the effects of sulphides on the leach. The leaching conditions are shown:

Intensive Cyanidation Conditions

Conditions	Test 1	Test 3
Weight	300g	325g
% NaCN	2	2
% Solids	30	30
Oxidant	Oxygen	Oxygen
Lead Nitrate	Yes	Yes
Duration	24 hours	24 hours
Regrind	No	Yes

4. Results

4.1. Head Grade Determination

Four samples weighing approx 100 grams each were split out from the prepared sample (P_{80} 850 µm) and sent for head grade determination. Samples were sent to AMMTEC Burnie Laboratory in Tasmania, Australia for analysis. Table 1 displays the results.

Table 1: Head Grades

	Repeat (1)	Repeat (2)	Repeat (3)	Repeat (4)	**Average**
Ag	<2	<2	<2	<2	<2
Au	1.65	3	2.17	1.79	**2.15**
Al	5.02%	5.02%	5.33%	5.11%	5.12%
Ba	520	480	539	504	511
Bi	<10	<10	<10	<10	
Ca	1.34%	1.35%	1.36%	1.28%	1.33%
Cd	<5	<5	<5	<5	
Co	6	5	<5	<5	6
Cr	11	<10	11	<10	11
Cu	47	43	42	41	43
Fe	1.70%	1.62%	1.65%	1.59%	1.64%
K	2.67%	2.87%	3.03%	2.98%	2.89%
Li	12	11	12	11	12
Mg	0.44%	0.41%	0.44%	0.42%	0.43%
Mn	203	198	211	204	204
Mo	<5	<5	<5	5	5.00
Na	0.59%	0.53%	0.57%	0.55%	0.56%
Ni	11	7	8	6	8
P	166	166	165	183	170
Pb	123	108	129	113	118
S	0.69	0.67	0.7	0.72	**0.70**
Sr	170	163	178	169	170
Ti	939	879	924	872	904
V	14	13	14	13	14
Y	11	10	12	11	11
Zn	79	87	84	79	82
Zr	212	194	214	190	203

Table 2 displays a summary of the calculated head grades from all tests conducted.

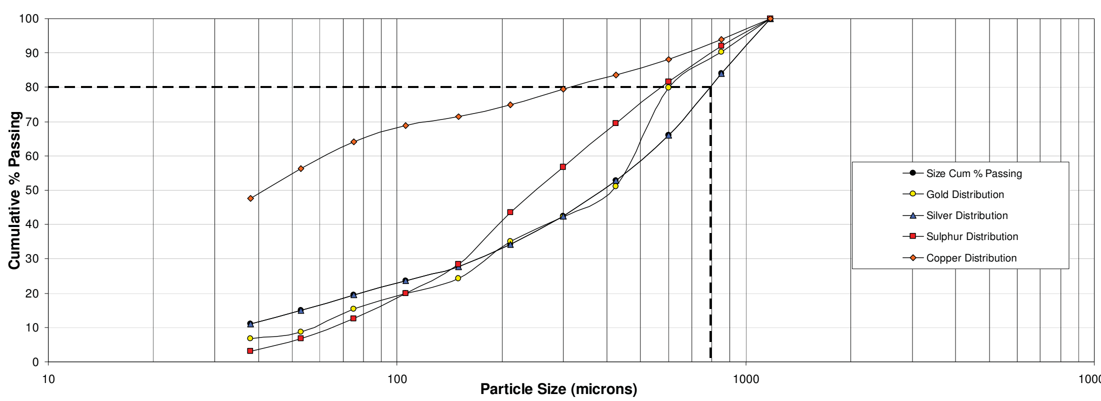
Table 2: Calculated Head Grades

Test	**Calculated Head Grades**			
	Au (ppm)	S (%)	Cu (ppm)	Ag (ppm)
Average Head Assay	2.15	0.70	43	<1.0
Size Analysis	2.04	0.68	45	1.0
Single Pass Tabling Test	1.53	0.54	34	1.4

4.2. Size by Grade Analysis

The particle size range was P_{80} 793µm. The Copper values occurred predominately in the finer fractions. The particle and elemental distributions are shown in Graph 1 below. Refer to Appendix A for the detailed results.

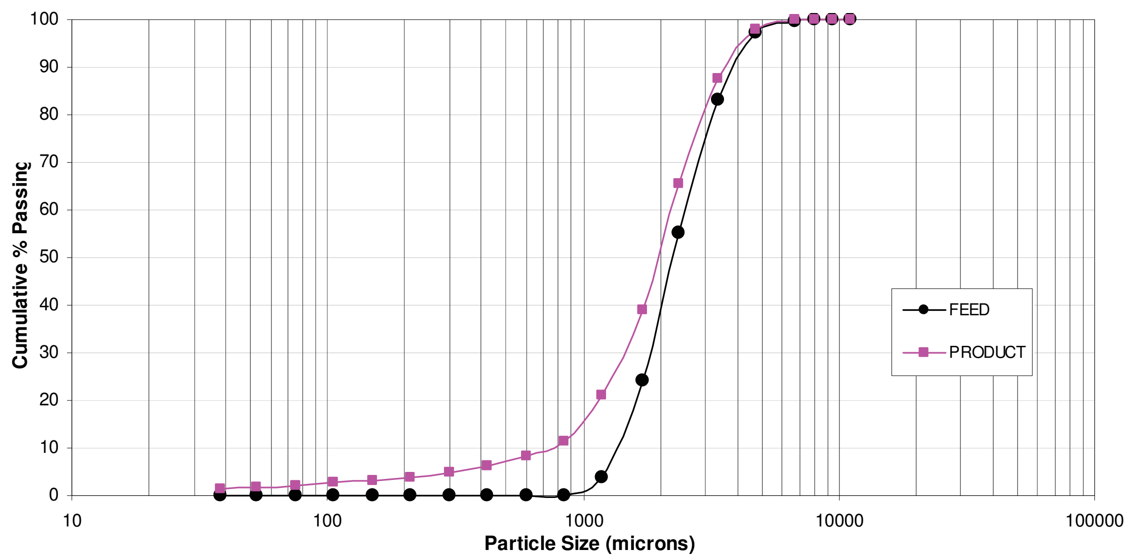
Graph 1: Table Feed Size Analysis



4.3. VSI Amenability Test

The results of the test are displayed in Graph 2. The amount of fines (<850 µm) generated in one pass was just 11. 5 %. This represents a circulating load in excess of 900%. The VSI isn't suitable for use in crushing this material. Refer to Appendix B for the full results.

Graph 2: VSI Test Sizing's



4.4. Bond Ball Mill Work Index

The BWi is 15.4 kwh/tonne. This is in the medium range for typical ores.

4.5. Progressive Grind Tabling Test

Approximately **31** kg of sample was used for a single pass, two stage rougher/cleaner, tabling test, to produce a number of concentrates at various mass yields.

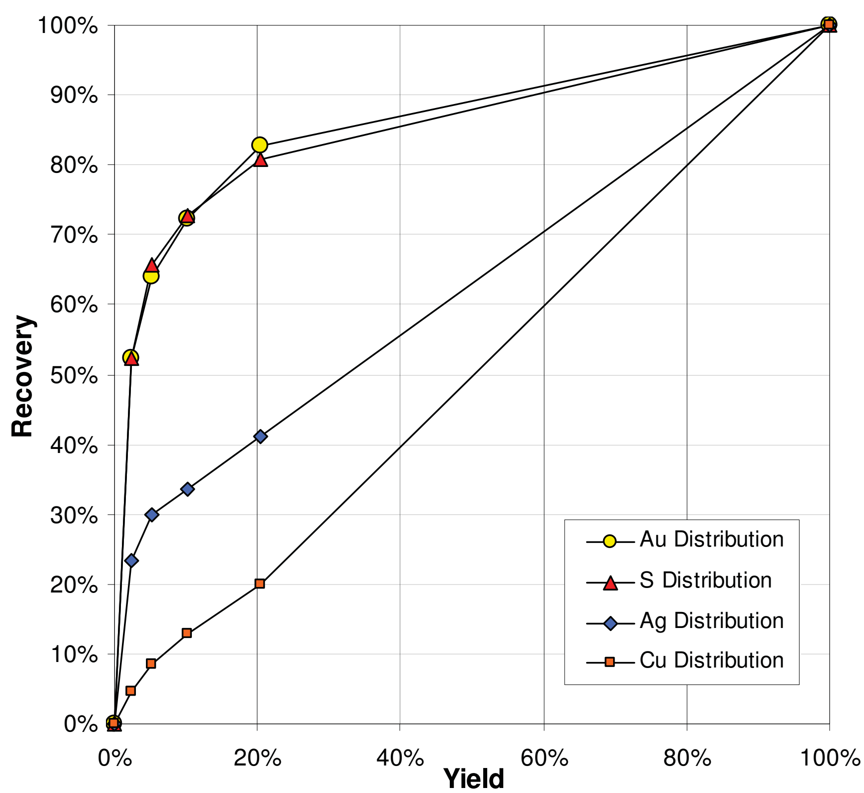
The results indicated that **52.4%** of the Gold and **52.4%** of the Sulphur could be captured by pulling **2.3%** of the mass. The recoveries increase to **72.3%** and **72.8%** respectively when the mass pull is increased to **10.1%.** The results are summarised in Table 3 below.

Table 3: Tabling Test Result Summary

		Copper		Gold		Sulphur	
	Wt%	ppm	Dist%	Ppm	Dist%	%	Dist%
T.C. 1	2.3	70	4.7	35.3	52.4	12.5	52.4
T.C. 1-2	5.1	57	8.6	19.0	63.9	6.9	65.6
T.C. 1-3	10.1	43	12.9	10.9	72.3	3.9	72.8
T.C. 1-4	20.5	33	20.0	6.2	82.8	2.1	80.9

Graph 3 displays the concentrate grade for a particular mass yield. Refer to Appendix B for full details.

Graph 3: Single Pass Tabling Test (P_{100} 822 µm)



4.6. Flotation Test

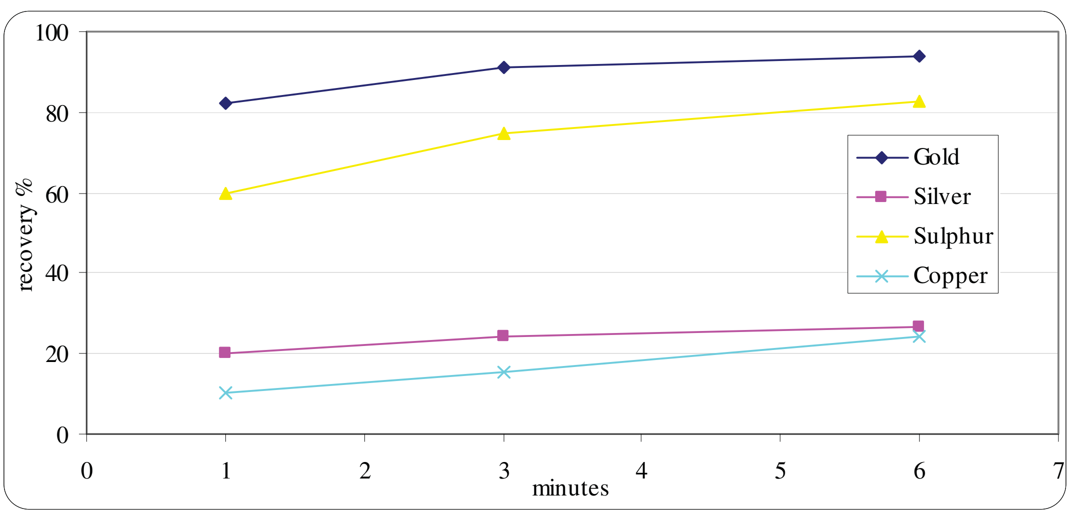
The gravity tabling tails were combined with table concentrates 2, 3 and 4. The sample was floated (2mins of polishing only) at the as received sizing of P80 88µm. The sample was then sent to Metcon Laboratories in Sydney for a basic rougher flotation test.

Table 4 below summarises the flotation result. A flotation tail assayed **0.09 ppm Au** and 0.06% S. Graph 4 depicts the fast kinetics of the float test with the majority of the Au and Cu recovered after 3 mins. Refer to Appendix D for the full results.

Table 4: Flotation Test Summary

		Copper		Gold		Sulphur	
	Wt%	ppm	Dist%	ppm	Dist%	%	Dist%
Rghr Conc. 1	0.7	1100	10.2	166	82.3	28.0	60.0
Rghr Conc. 1 & 2	1.7	700	15.3	77	91.3	14.6	74.7
Rghr Conc. 1,2 & 3	3.1	600	24.5	43	93.9	8.8	82.5

Graph 4: Flotation Kinetics



4.7. Combined Gravity - Flotation Test

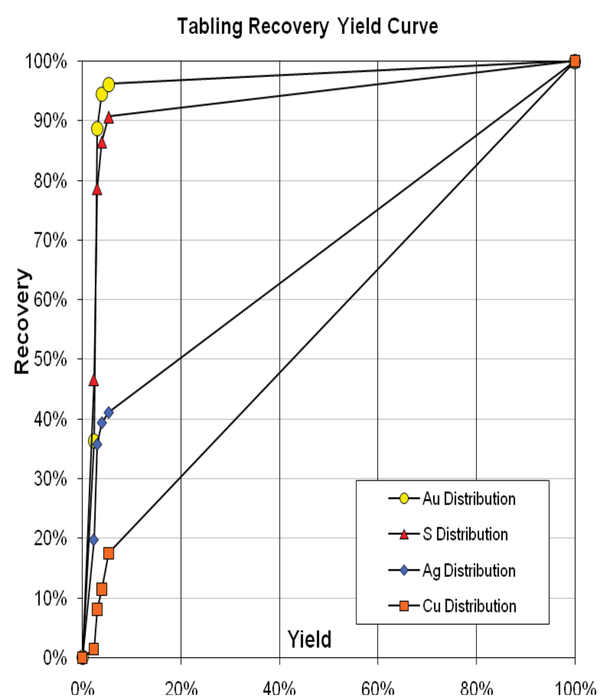
The results of the combined Gravity-Flotation test are displayed in table 5 and Graph 5 below. Detailed results are shown in Appendix E.

Table 5: Combined Gravity-Flotation Test Summary

		Copper		Gold		Sulphur	
	Wt%	ppm	Dist%	ppm	Dist%	%	Dist%
T.C. 1	2.3	70	1.4	35.3	36.3	12.5	46.6
T.C.1+F.C.1	3.0	312	8.1	66	88.7	16.1	78.6
T.C.1+F.C.1&2	3.9	334	11.4	53	94.5	13.4	86.5
T.C.1+F.C.1,2&3	5.3	377	17.5	40	96.1	10.4	90.7

The results show that **94.5%** of the Gold and **86.5%** of the Sulphur can be recovered into a concentrate containing **3.9%** of the mass, at a grade of 53ppm Au. The recoveries increase to **96.1%** and **90.7%** respectively when the mass pull is increased to **5.3%**, at a grade of 40 ppm Au.

Graph 5: Gravity-Flotation Recovery vs. Yield Curve



4.8. Leach Testing

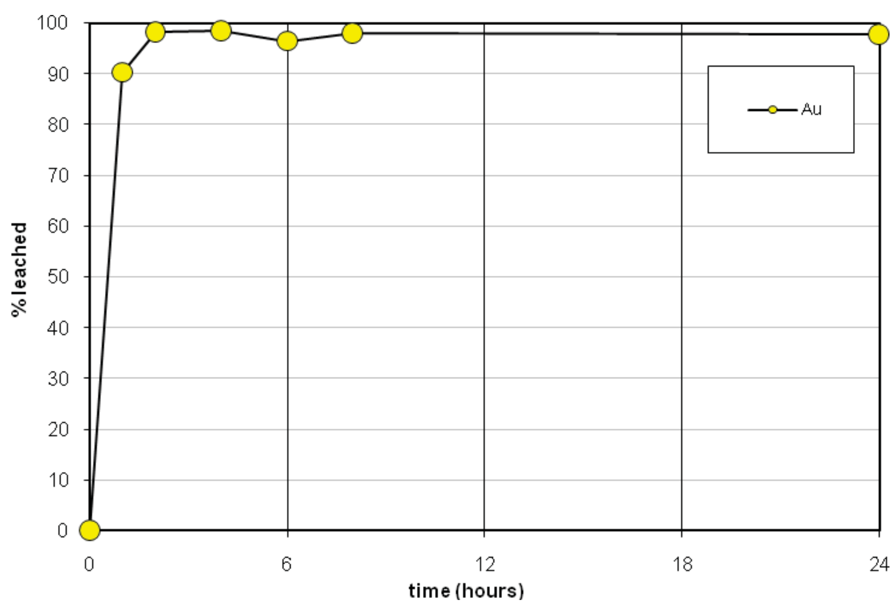
The results of the leach test03 are displayed in graph 6 below. Detailed results are shown in Appendix F.

The gold present in the gravity and flotation concentrates is very fine, but easily leachable when liberated. The original leach test (named leach test 01) was performed without regrinding the concentrate sample. The dissolution was maxed out at 78%.

Upon discussions with Dan Clark, it was decided to re-leach the gravity concentrate at a much finer grind size. After grinding, the dissolution of gold in the sample was approximately 98%.

The leach kinetics are very fast with the leach effectively complete within 2 hours of the start of the test.

Graph 6: Leach Test Results (sample re-ground)



5. Conclusions

The following conclusions can be made:

- The sample is not amenable to crushing with a VSI due to the extremely small breakage rate per pass (11%) measured through the test rig.
- The progressive grind tabling test indicated that a gravity jigging circuit operating in the circulating load of mill, will recover approximately **36.3%** of the Gold into a 35.3ppm concentrate grade, at a mass pull of 2.3%.
- Flotation on the gravity tail is effective in recovering the bulk of the gold and sulphur. Flotation tails of 0.09ppm were achieved.
- A combination of gravity and flotation recovery stages can produce a concentrate that contains **96.1%** of the Au and **90.7%** of the S. The flotation tails grades were and 0.09 ppm Au. 0.06% S
- A very fine grind size is necessary for recovery by leaching.

6. Recommendations

The following recommendations can be made:

- As the ore isn't suitable for processing in a VSI, a conventional crushing circuit is required to prepare the material for the grinding circuit.
- Flotation optimisation testwork should be completed to improve concentrate grades.
- The gravity concentrates recovered had a concentrate grade of 35 g/t which is at the minimum limit for continuous intensive cyanidation processing. It would be recommended to determine the effect of flotation only or possibly centrifugal concentrator followed by flotation (although these methods do target the same gold size ranges).

7. Appendix

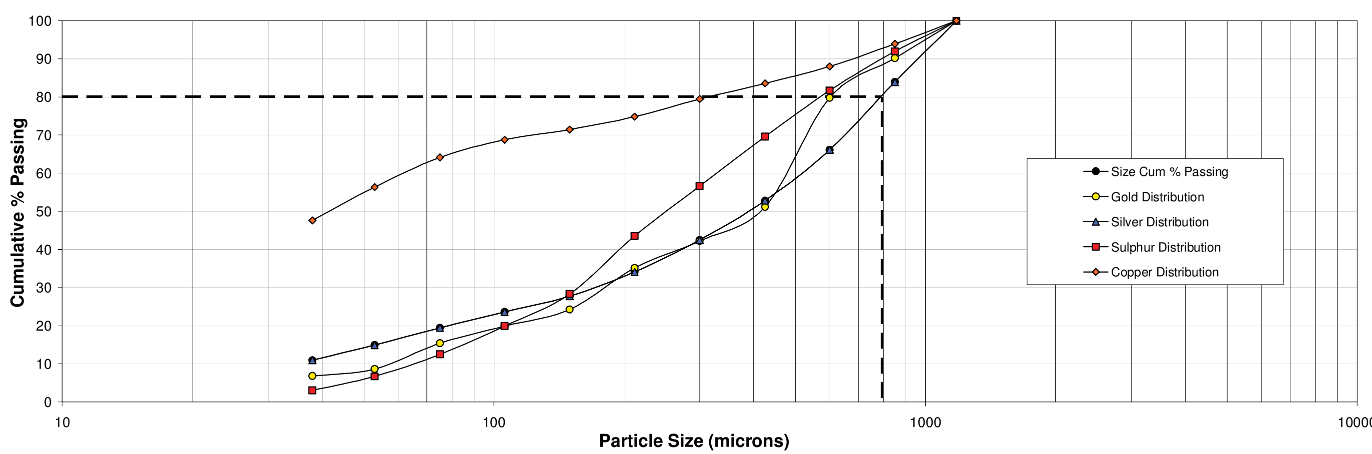
Appendix A – Prepared Sample Size Distribution

GEKKO SYSTEMS

LABORATORY TEST WORK RESULTS

Particle Size Analysis
Project: **Opawica - B "Dingman"**
Stream: **Sample Prepared to P100 -1.18mm**
Date: 22/10/2008

	Particle Size Distribution			Gold Distribution				Silver Distribution				Sulphur Distribution				Copper Distribution			
Size	Wt Retained (g)	Wt %	Size Cum %	Assay	Mass	Wt%	Cum %	Assay	Mass	Wt%	Cum %	Assay	Mass	Wt%	Cum %	Assay	Mass	Wt%	Cum %
(microns)	Sizing 1	Retained	Passing	(g/t)	(Units)	Retained	Passing	(g/t)	(Units)	Retained	Passing	(%)	(Units)	Retained	Passing	(g/t)	(Units)	Retained	Passing
1180	0.00	0.00	100.00	0.00	0.00	0.00	100.00	0.00	0.00	0.00	100.00	0.00	0.00	0.00	100.00	0.00	0.00	0.00	100.00
850	174.20	16.07	83.93	1.25	0.20	9.83	90.17	1.00	0.16	16.07	83.93	0.34	0.05	8.05	91.95	17	2.73	6.06	93.94
600	193.20	17.82	66.11	1.19	0.21	10.38	79.79	1.00	0.18	17.82	66.11	0.39	0.07	10.24	81.71	15	2.67	5.93	88.01
425	144.10	13.29	52.81	4.40	0.58	28.62	51.17	1.00	0.13	13.29	52.81	0.62	0.08	12.15	69.56	15	1.99	4.42	83.59
300	112.00	10.33	42.48	1.77	0.18	8.95	42.22	1.00	0.10	10.33	42.48	0.85	0.09	12.94	56.62	18	1.86	4.13	79.46
212	90.60	8.36	34.12	1.73	0.14	7.08	35.15	1.00	0.08	8.36	34.12	1.06	0.09	13.05	43.56	25	2.09	4.64	74.82
150	69.10	6.37	27.75	3.49	0.22	10.89	24.26	1.00	0.06	6.37	27.75	1.62	0.10	15.22	28.35	24	1.53	3.39	71.43
106	44.90	4.14	23.61	2.16	0.09	4.38	19.88	1.00	0.04	4.14	23.61	1.38	0.06	8.42	19.92	29	1.20	2.66	68.77
75	45.30	4.18	19.43	2.20	0.09	4.50	15.38	1.00	0.04	4.18	19.43	1.21	0.05	7.45	12.47	50	2.09	4.64	64.13
53	48.9	4.51	14.92	3.06	0.14	6.75	8.63	1.00	0.05	4.51	14.92	0.87	0.04	5.78	6.69	78	3.52	7.81	56.32
38	43.0	3.97	10.95	0.94	0.04	1.82	6.80	1.00	0.04	3.97	10.95	0.62	0.02	3.62	3.07	99	3.93	8.71	47.61
0	118.7	10.95	0.00	1.27	0.14	6.80	0.00	1.00	0.11	10.95	0.00	0.19	0.02	3.07	0.00	196	21.46	47.61	0.00
Total	1084	100.00		Head Grade	2.04	100.00		Head Grade	1.00	100.00		Head Grade	0.68	100.00		Head Grade	45	100.00	
Initial Wt	1090	Reconcile	101%	Assay Head	2.15			Assay Head	1.00			Assay Head	0.70			Assay Head	43		
D80	**793**	**D50**	**389**	**D80**	**604**	**D50**	**408**	**D80**	**793**	**D50**	**389**	**D80**	**573**	**D50**	**254**	**D80**	**314**	**D50**	**42**



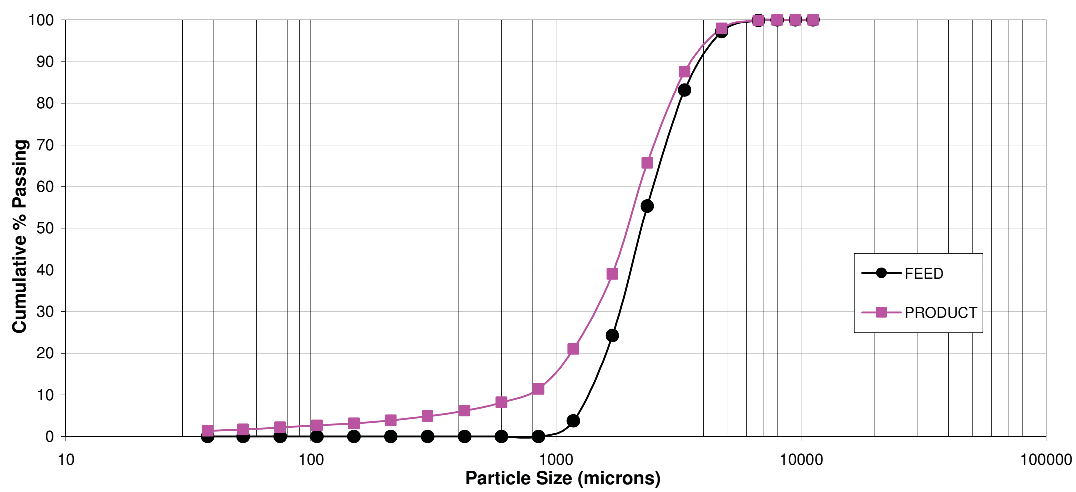
Appendix B – VSI Amenability Testwork

GEKKO SYSTEMS LABORATORY TEST WORK RESULTS

Particle Size Analysis
Project: Opawica - Dingman
Test: Single Pass VSI Amenability Test
Stream: Feed screened at 1.18 mm
Date: 17/10/2008

	FEED			PRODUCT		
	Particle Size Distribution			Particle Size Distribution		
Size (microns)	Wt Retained (g) Sizing 1	Wt % Retained	Size Cum % Passing	Wt Retained (g) Sizing 2	Wt % Retained	Size Cum % Passing
11200	0.00	0.00	100.00	0.00	0.00	100.00
9500	0.0	0.00	100.00	0.0	0.00	100.00
8000	3.4	0.08	99.92	3.3	0.07	99.93
6700	5.2	0.12	99.80	4.2	0.09	99.85
4750	115.7	2.66	97.15	91.7	1.88	97.96
3350	611.4	14.04	83.11	507.9	10.42	87.55
2360	1211.0	27.81	55.30	1067.5	21.90	65.65
1700	1354.0	31.09	24.20	1296.8	26.60	39.04
1180	889.7	20.43	3.77	879.2	18.04	21.01
850	164.3	3.77	0.00	465.9	9.56	11.45
600	0.00	0.00	0.00	156.8	3.22	8.23
425	0.00	0.00	0.00	98.0	2.01	6.22
300	0.00	0.00	0.00	65.2	1.34	4.88
212	0.00	0.00	0.00	50.5	1.04	3.85
150	0.00	0.00	0.00	34.8	0.71	3.13
106	0.00	0.00	0.00	23.3	0.48	2.66
75	0.00	0.00	0.00	21.3	0.44	2.22
53	0.00	0.00	0.00	23.5	0.48	1.74
38	0.00	0.00	0.00	19.1	0.39	1.35
0	0.00	0.00	0.00	65.6	1.35	0.00
Total	4355	100.00		4875	100.00	
Initial Wt	4360			4880		

p80 3239 um **p80 3009 um**



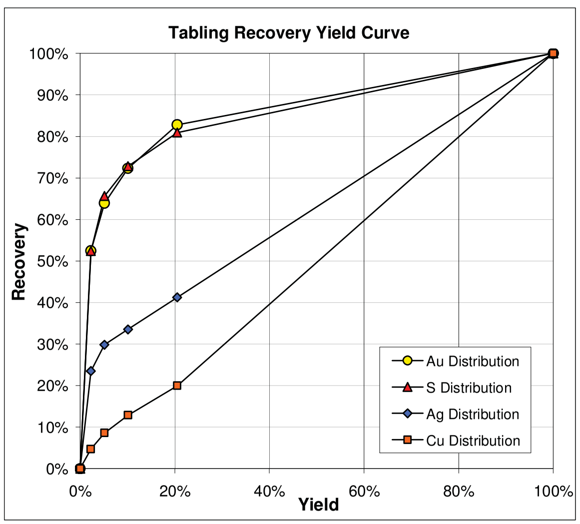
Appendix C – Gravity Testwork

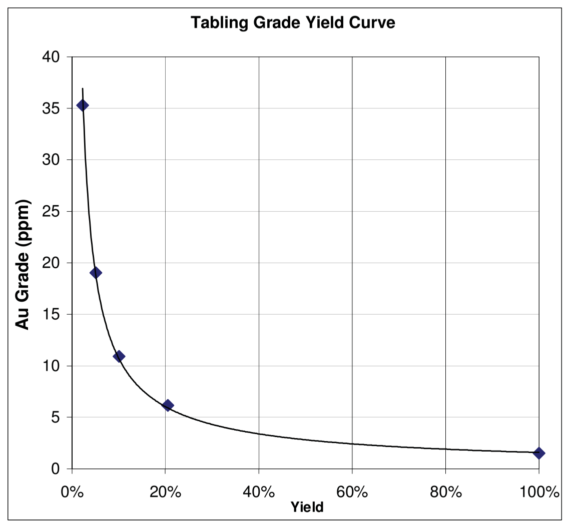
LABORATORY TEST WORK RESULTS

Gravity Separation Test

Project: **Opawica - Dingman**
Stream: Three Stage Progressive Grind Table Test (P80 - 300µm, 150µm & 106µm)
Date: 7/11/2008
Results Date: 14/11/2008

Sample	Mass Yield			Gold Distribution				Sulphur Distribution				Silver Distribution			
	g	%	cumulative %	Assay ppm	Distribution %	Cumulative Distribution	Cumulative grade ppm	Assay %	Distribution %	Cumulative Distribution	Cumulative grade %	Assay ppm	Distribution %	Cumulative Distribution	Cumulative grade ppm
Concentrate 1	683.5	2.27%	2.27%	35.30	52.42%	52.42%	35.30	12.50	52.40%	52.40%	12.50	14.00	23.48%	23.48%	14.00
Concentrate 2	863.2	2.86%	5.13%	6.13	11.50%	63.92%	19.02	2.50	13.24%	65.64%	6.92	3.00	6.35%	29.83%	7.86
Concentrate 3	1497.9	4.97%	10.10%	2.58	8.40%	72.31%	10.93	0.78	7.17%	72.80%	3.90	1.00	3.68%	33.51%	4.49
Concentrate 4	3140.7	10.42%	20.52%	1.54	10.51%	82.82%	6.16	0.42	8.09%	80.90%	2.13	1.00	7.71%	41.21%	2.72
Table Tails	23960	79.48%	100.00%	0.33	17.18%	100.00%	1.53	0.13	19.10%	100.00%	0.54	1.00	58.79%	100.00%	1.35
Calc'd Feed	30145	100%		1.53	100%		1.53	0.54	100%		0.54	1.35	100%		1.35
Assay Feed	30280	100.00		2.15	100.00	100.00		0.70	100.00	100.00		1.00	100.00	100.00	





Copper Distribution			
Assay ppm	Distribution %	Cumulative Distribution	Cumulative grade ppm
70	4.70%	4.70%	70.00
46	3.90%	8.60%	56.61
29	4.27%	12.87%	43.02
23	7.10%	19.97%	32.86
34	80.03%	100.00%	33.77
34	100%		33.77
43.00	100.00	100.00	

Appendix D – Flotation Testwork

GEKKO SYSTEMS **OPAWICA FLOTATION TEST RESULT SHEET** **METCON LABORATORIES**

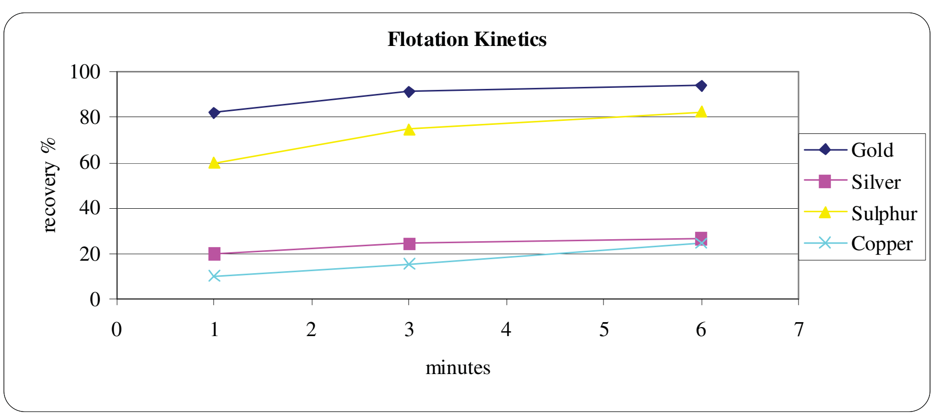
SAMPLE	26698
OBJECTIVE	**Sighter test - 3 stage rougher (P80 = 88µm, as received).**
JOB NUMBER	M1823
TEST NUMBER	26698F-1
DATE	1/12/08
OPERATOR	Andre
P80 (µm)	88
SS ROD MILL GRIND	no grind

Stage	Denver cell # 1 litres	rpm	air l/min	impellor
rghr	2.5	1100	6	small
clnr				
reclnr				

TEST PRODUCT	grams	wt %	Gold ppm	Gold dist %	Silver ppm	Silver dist %	Sulphur %	Sulphur dist %	Copper %	Copper dist %
rghr conc 1	7.1	0.7	166	82.3	37	20.0	28.0	60.0	0.11	10.2
rghr conc 2	9.9	1.0	13.1	9.1	6	4.5	4.91	14.7	0.04	5.2
rghr conc 3	14.1	1.4	2.64	2.6	2	2.1	1.84	7.8	0.05	9.2
rghr tail	966.2	96.9	0.09	6.1	1	73.4	0.06	17.5	0.01	75.5
calc. head	997.3	100.0	1.44	100.0	1	100.0	0.33	100.0	0.01	100.0
					actual tail Ag assay <1ppm					
			rghr tail dup. Au assays 0.10, 0.08							

COMBINED PRODUCTS	grams	wt %	Gold ppm	Gold dist %	Silver ppm	Silver dist %	Sulphur %	Sulphur dist %	Copper %	Copper dist %
rghr conc 1	7.1	0.7	166	82.3	37.0	20.0	28.0	60.0	0.11	10.2
rghr cons 1 & 2	17.0	1.7	77.0	91.3	18.9	24.5	14.6	74.7	0.07	15.3
rghr cons 1 to 3	31.1	3.1	43.3	93.9	11.3	26.6	8.79	82.5	0.06	24.5

FLOTATION CONDITIONS		CuSO4 g/t	PAX g/t	MIBC g/t	pH	Eh	mins condit	mins float	cum mins float
repulp					8.0	40	10		
rougher 1				20	8.1	5	2	1	1
	strong sulphides								
rougher 2				5			1	2	3
rougher 3					8.3	-5	1	3	6
Total		0	0	25				6	



Appendix E – Combined Gravity-Flotation Testwork

GEKKO SYSTEMS

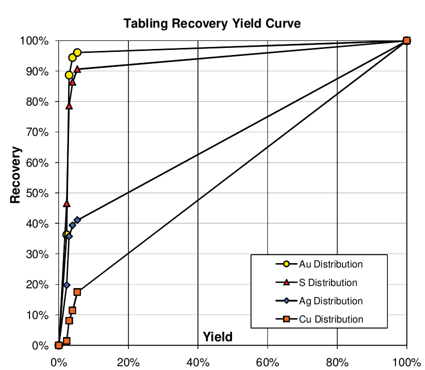
LABORATORY TEST WORK RESULTS

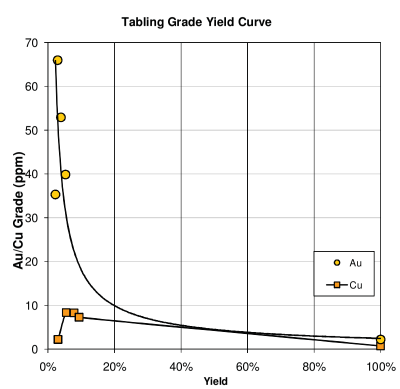
Gravity Separation Test

Project: **Opawica - Dingman**
Stream: PGT Gravity test + Rougher Flotation
Date: 1/12/2008
Results Date:

Sample	Mass Yield			Gold Distribution				Sulphur Distribution				Silver Distribution			
	g	%	cumulative %	Assay ppm	Distribution %	Cumulative Distribution	Cumulative grade ppm	Assay %	Distribution %	Cumulative Distribution	Cumulative grade %	Assay ppm	Distribution %	Cumulative Distribution	Cumulative grade ppm
Grav Conc 1	683.5	2.27%	2.27%	35.30	36.31%	36.3%	35.3	12.50	46.61%	46.61%	12.50	14.0	19.75%	19.7%	14.00
Float Conc 1	209.7	0.70%	2.96%	166.0	52.40%	88.7%	66.0	28.00	32.04%	78.64%	16.14	37.0	16.01%	35.8%	19.40
Float Conc 2	292.5	0.97%	3.93%	13.1	5.77%	94.5%	52.9	4.91	7.83%	86.48%	13.37	6.0	3.62%	39.4%	16.10
Float Conc 3	416.5	1.38%	5.32%	2.6	1.65%	96.1%	39.9	1.84	4.18%	90.66%	10.37	2.0	1.72%	41.1%	12.43
Float Tails	28543	94.68%	100.00%	0.1	3.87%	100.0%	2.2	0.06	9.34%	100.00%	0.61	1.0	58.90%	100.0%	1.61
Calc'd Feed	30145	100%		2.20	100%		2.2	0.61	100%		0.61	1.61	100%		1.61
Assay Feed	31480	100.00		1.56	100.00	100.00		2.15	100.00	100.00		5.00	100.00	100.00	
	1335														

Copper Distribution			
Assay ppm	Distribution %	Cumulative Distribution	Cumulative grade ppm
70.0	1.38%	1.4%	70.0
1100	6.67%	8.1%	311.9
400	3.38%	11.4%	333.6
500	6.02%	17.5%	376.9
100	82.54%	100.0%	114.7
114.72	100%		114.7
0.95	100.00	100.00	





Appendix F – Leach Test Results

LABORATORY TEST WORK RESULTS

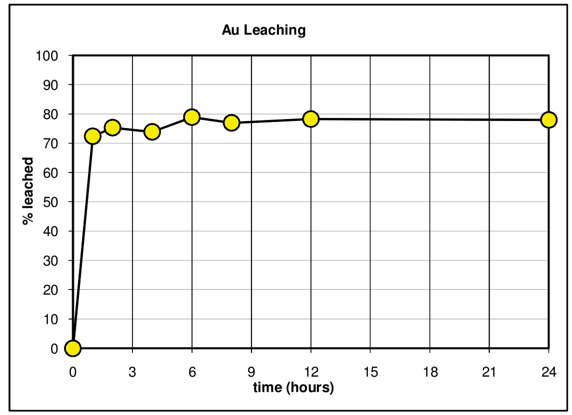
Intensive Cyanidation Test

Project: Opawica - Dingman
Stream: LOPA - B (1) Table Concentrate 1
Test: LOPA - B (01)
Date: 14/05/2009

Grind p80 = NA

Conditions: **2% NaCN, 2g/kg Pb(NO3)2 and Bottled O2 addition**

Wt % Solids	**30.0%**		**NaCN** consumed	**4.09 g** **13.6 kg/t**
H2O2 addition	0.00 **g** 0.00 **kg/t**		Net NaCN added	17.90 g 59.6 kg/t
Pb(NO3)2 addition	0.60 **g** 2.00 **kg/t**		NaCN residual	1.97 % 13.81 g
pH	natural 8.06 initial 11.32 final 11.26		**NaOH** addition	0.00 **g** 0.00 **kg/t**



SAMPLE NAME	Wt. OR VOLUME	SOLUTION SUB/ADD		ASSAYS Au	**Recovery Au**	**DO2**	**pH**	**Sodium Cyanide level**	**added**	**removed**	sample Au	leached Au
	g	g	g	ppm	**%**	**ppm**		**%w/v**	**g**	**g**	ug	ug
Sampled Head	300.5			35.3								
Solutions hours						initial	8.06					
0	701				**0.00**		11.32	2.00	14.00	0.00		
1	701	50	50	10.0	**72.39**	41.3	11.14	1.45	4.58	0.72	499	7011
2	701	50	59	9.7	**75.35**	33.9	11.26	1.82	2.17	0.91	485	7298
4	710	50	40	8.7	**73.89**	31.7	11.28	1.78	2.43	0.89	433	7157
6	700	50	50	8.9	**78.94**	36.5	11.20	2.01	0.00	1.00	444	7646
8	700	51	51	8.0	**77.03**	33.2	11.24	2.01	0.00	1.02	405	7461
12	700	50	51	7.6	**78.32**	30.0	11.21	1.97	1.19	0.99	382	7586
24	701	48	0	7.0	**78.00**	30.2	11.26	1.97	0.00	0.94	334	7554
Leach residue	297.60			7.16			Total		24.37	6.47		2131
Calculated Head	300.5			**32.2**								9685

GEKKO

LABORATORY TEST WORK RESULTS

Intensive Cyanidation Test

Project: Opawica Explorations - Dingman
Stream: LOPA - B Table Concentrate 1
Test: LOPA - B (03)
Date: 11/08/2009

Grind: p100 -53um

Conditions: **2% NaCN, 2g/kg Lead Nitrate and bottled oxygen addition.**

Wt % Solids: **30.0%**

NaCN consumed **3.45 g** **10.6 kg/t**

Lime addition 0.00 **g** 0.00 **kg/t**

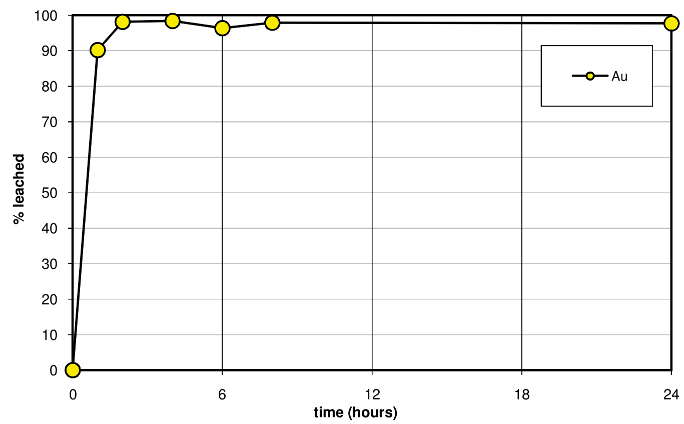
Net NaCN added 15.36 g 47.3 kg/t

Pb(NO3)2: addition 0.65 **g** 2.00 **kg/t**

NaCN residual 1.57 % 11.92 g

pH natural 7.48 initial 11.89 final 11.81

NaOH addition 0.73 **g** 2.25 **kg/t**



SAMPLE NAME	Wt. OR VOLUME	SOLUTION SUB/ADD		ASSAYS	Recovery	DO2	pH	Sodium Cyanide			REMOVED IN SAMPLE	TOTAL LEACHED UNITS
				Au	Au			level	added	removed	Au	Au
	g	g	g	ppm	%	ppm		%w/v	g	g	ug	ug
Sampled Head	325.1			35.3								
Solutions hours							7.48					
0	759				**0.0**		11.89	2.00	15.20	0.00		
1	759	50	50	13.8	**90.2**	46.6	12.06	1.94	1.43	0.97	695	10494
2	759	51	52	14.1	**98.2**	45.6	12.02	1.82	2.28	0.94	728	11422
4	759	50	56	13.2	**98.4**	40.2	11.88	2.03	0.00	1.02	667	11450
6	764	53	47	11.9	**96.4**	40.5	11.80	1.60	1.28	0.84	627	11214
8	759	52	53	11.4	**97.9**	17.9	11.75	2.01	0.00	1.05	599	11390
24	759	50		10.6	**97.7**	40.2	11.81	1.57	0.00		530	11370
Leach residue	325.0			0.81			Total		20.19	4.83		263
Calculated Head	325.1			**35.8**								11633

Appendix F – Disclaimer

Gekko has undertaken test work to characterize the response of your ore to certain separation techniques and/or to help your own experts make a decision as to whether you wish to purchase our product and, if so, the number and type.

It is important that you understand that:

- *Our testing is preliminary only.*
- *You should obtain, independent advice from all relevant specialists, including a metallurgist, before acquiring any equipment and before committing to and proceeding with your project.*
- *You must have your own experts examine the detailed analysis in our report to decide its applicability to your project.*
- *We analyse only the sample you provide. Any one of a number of factors may cause that sample inaccurately to reflect the ore body. You must determine the extent to which the sample represents the ore body. That includes the detection limits and confidence intervals relevant to our results.*

At all times we endeavour to provide accurate test work outcomes but you should not use our results as a basis for your broader business decisions about your project.

If we have not exercised due care with our tests, the limit of our liability, both at common law and under any statute, will be to provide a further set of test results to you free of charge. You indemnify us with respect to all other loss and damage of every kind, including, without limitation:

- *damage to or loss of property;*
- *injury to or death of any person; and*
- *economic and consequential loss arising from the negligent act or omission of us or any one else in connection with our test*

APPENDIX D

CERTIFICATE OF QUALIFICATIONS

ROBERT LAAKSO
CERTIFICATE OF QUALIFICATION

I, Robert Laakso, P.Eng. do hereby certify that:

1. I am the principal owner of Shaft & Tunnel Engineering Services Ltd. at:

   P.O. Box 1659
   Holland Landing, Ontario, Canada
   L9N 1P2
   Telephone: (905) 836-4836
   Fax: (905) 836-4110

2. I graduated with a degree in Geological Engineering.
3. I am a member in good standing of the ASSOCIATION of PROFESSIONAL ENGINEERS OF THE PROVINCE OF ONTARIO since 1965.
4. I have practised my profession for 45 years since graduation.
5. I am responsible for all sections of the technical report titled, Technical Report On The Dingman Gold Property, Madoc, Ontario, Canada with revised date of August 31, 2009 (the "Technical Report"). I visited the Dingman Property site on a weekly basis during March, April and May of 2009.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.
7. I have had prior involvement with the property since November of 2005, where acting as the Qualified Person, I managed the field exploration and diamond drilling programs as well as logging and handling of core from site to the core splitting facility in Matachewan and final shipping to the assay laboratories.
8. I am not aware of any material fact or change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the issuer applying all the tests in section 1.4 of the National Instrument (NI) 43-101.
10. I have read NI 43-101 and form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 31st Day of August, 2009.

SHAFT & TUNNEL ENGINEERING SERVICES LTD.

Original Signed and Stamped by:

Robert Laakso, P.Eng.