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OMEGA PROPERTY
MCVITTIE TOWNSHIP
ONTARIO, CANADA
TECHNICAL REPORT
for
MISTANGO RIVER RESOURCES INC.

Prepared by AMC Mining Consultants (Canada) Ltd. In accordance with the requirements of National Instrument 43-101, "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators

Qualified Persons:
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AMC 912003
Effective date 31 August 2012
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1 SUMMARY

This Technical Report on the Omega Property in McVittie Township of the Larder Lake Mining Division, Ontario, has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) Toronto, Ontario office on behalf of Mistango River Resources Inc. (Mistango) of Kirkland Lake, Ontario. It has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators (CSA) for lodgment on CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR).

This report is a statement of Mineral Resources, as defined in NI 43-101, as at 31 August 2012 and updates an earlier Technical Report titled "Technical Review of the Omega Gold Mine Property, Ontario, Canada for Mistango River Resources Inc." dated 30 November 2011.

History, Location and Ownership

The Omega Property (the Property) is located in McVittie Township of the Larder Lake Mining Division, Ontario, and covers a total of 104 hectares. The Mistango holdings in Ontario encompass The Omega, Southwest Zone and Lake Zone group of claims which are centred on UTM Zone 17, 596,700 mE and 5,329,350 mN. The Property is 100% owned by Mistango and consists of seven leased claims and is adjacent to an additional eight leased claims and two patents, held within two adjoining blocks. There are also six optioned leased claims abutting the two of the blocks. The Property hosts the Omega Mine that historically produced gold from 1921 to 1929 and from 1936 to 1947.

Within the boundaries of the old Omega Mine there are two tailings disposal facilities containing waste from the flotation concentrate and the mill tailings. Both are located to the west of the number 2 shaft of the former Omega Mine. Under the terms of the lease agreement with the government, Mistango is required to prevent any environmental contamination originating from either of these facilities.

The flotation tailings facility contains an estimated 118,000 tonnes of material with an elevated level of cyanide. Mistango has therefore signed a memorandum of understanding with United Commodity AG (UC) of Thun of Switzerland regarding the reprocessing of the tailings from the former Omega Mine site. These will be removed and processed at a specialist facility. This agreement is dated 20 September 2012.

The second tailing facility contains mill tailings which are considered inert and Mistango will need to develop a closure plan for full rehabilitation, including revegetation.

Historical infrastructure on closure of the Omega Mine consisted of two shafts 305 m (1,000 ft) and 457 m (1,550 ft) deep and a winze down to 610 m (2,000 ft). The two shafts have been capped. All other surface infrastructure has been removed.

Mining and mineral exploration, equipment fabrication, construction trades, transportation, tourism, and forestry are the main sources of employment in the area. The Property is located in an active mining belt. The area businesses offer most supplies and services. The area has a substantial professional workforce that is experienced in mining and related activities, however the current high-level of mining activity could affect immediate workforce availability.

The area has well developed infrastructure, therefore availability of power, transportation and water are not likely to impact on the project.

Geology and Mineralization

The Property is located within the Abitibi Greenstone Belt of the Superior Province of the Precambrian Shield. Part of the Cadillac-Larder Lake Deformation Zone (C-LLDZ), a regional scale shear, passes through the northern part of the Property.

The oldest sequence in the Kirkland Lake - Larder Lake area is Precambrian Abitibi volcanics interbedded with slate and chert, dated between 2747 Ma and 2705 Ma. This sequence contains long, narrow bodies of diorite and gabbro, as well as coarser-grained flows and was subsequently deformed into a series of regionally ESE-WNW trending folds. Timiskaming Group interbedded sediments and alkali volcanics dated circa 2680 Ma, unconformably overlie the older volcanics and their deposition is spatially associated with the Larder Lake-Cadillac Break (LLCB). The Timiskaming Group sediments are comprised of two sequences, one non-marine fluvial in origin and the other of sub-marine fans, intercalated with several volcanic sequences, varying in composition from intermediate to basic, and suggestive of an island arc origin. These units form a long, relatively narrow, east-west trending belt which was intruded by a number of syenite and porphyry stocks and dykes dated 2673 Ma. Contemporaneous lamprophyre and diabase dykes are widespread throughout the region. Most of the diabase is of the "Matachewan" swarm of north-striking dykes dated at2485 Ma.

Undeformed Proterozoic age Huronian Supergroup sedimentary rocks, primarily of the Cobalt Group, unconformably overlie the Achaean basement, which are in turn intruded by Nipissing diabase dykes dated at 2200 Ma.

The Property lies on the southern limb of an overturned anticline which has its axis lying sub-parallel to the northern boundary of the property. The anticline is sharply folded and overturned to the north and is broken by a thrust fault following the strike of the fold, suggesting a roll-over anticline at the leading edge of the thrust. The rocks along this limb face north and are overturned, dipping at ~60° south. At least a part of the movement is post-ore indicated by vein fragments in the fault gouge and by the drag of the ore along the fault plane.

The two most prominent gold-bearing structures in the region are the C-LLDZ and the Kirkland Lake "Main Break" (KLMB). The C-LLDZ is a regionally extensive shear zone, characterized by the development of mica schists and locally marked by hydrothermal alteration (silicification, sulphidation and carbonatization), and the development of quartz stockworks and breccia. Green mica (fuchsite) is commonly developed where alteration overprints ultramafic rocks. This structure is considered to be the western extension of the Malartic-Cadillac Deformation Zone, making this structure more than 160 kilometres long. The zone has the appearance of being a south-dipping reverse fault, in which the south-side seems to have moved upwards and eastward relative to the north-side. However, the zone has also been described as a slightly overturned normal fault structure. The KLMB is a fault zone branching north-westerly from the C-LLDZ near Kenogami Lake. This structure has been identified in all the gold mines in Kirkland Lake, down to depths of more than 2 kilometres. The structure varies from a single plane to multiple bifurcating planes. The widest ore bodies occur where the cross-over faults and the tension fractures between the planes are most numerous.

Exploration

Since early 2011 Mistango has undertaken four phases of exploration on the Omega Group and Southwest Group of claims. This included geophysical surveys, soil sampling and drilling. During 2011 Mistango carried out phase one exploration on the Omega Group of claims consisting of line cutting, magnetometer and deep induced polarization (IP) surveys, with limited soil sampling to profile the IP anomalies. The other three subsequent phases consisted of drilling. Mistango commenced a drill program to outline the potential of the Omega Group of claims in early 2011 and have subsequently completed three phases of drilling. The holes are drilled either using BTW or NQ sized core.

Data Management

The orientation of the drilling is in two primary directions, approximately perpendicular to the strike of the mineralization zone. Twenty holes have been drilled from the footwall side and have an azimuth of about 145^0, with the remaining 132 holes having an approximate azimuth of 325^0. To date, nearly 13,000 drill-core samples from the Omega project have been analysed. AMC reviewed the core handing, quality assurance and quality control (QA/QC) procedures and data collection during a site visit between 7 and 9 August 2012. AMC is satisfied that data is collected to industry standards.

A quality control (QC) program of standards, blanks and duplicate samples has been used since 2011 for all of the drill samples analysed. The blank material is a marble sourced locally. The standard samples are provided by Oreas-Ore Research & Exploration Pty Ltd. The standards contain low, high and moderate gold grades.

During 2012 Mistango commissioned SGS Mineral Services (SGS) to complete a preliminary economic assessment (PEA) (scoping–level, metallurgical test program to establish the basic processing parameters for the treatment of a composite sample shipped by Mistango to the SGS Lakefield facility. The metallurgical investigation undertaken on the composite sample provided some understanding of the sample nature and metallurgical behaviour.

The sample composite contained 3.58 g/t gold based on direct head assaying by pulp/metallic protocol. The silver grade was determined to be < 2 g/t. The composite sample also yielded a sulphide grade of 3.54%.

Initial whole ore cyanidation testing of the composite, after 48 hours of leaching, showed recoveries ranging from 76% to 86%. Cyanide consumption was 0.53 kg/t to 1.38 kg/t of NaCN. Lime consumption was low at 0.40 kg/t to 0.45 kg/t. Gravity separation testing on the Omega composite at a P_{80} size of 125 microns showed a very low result for gold recovery of 3%.

Gravity tailing cyanidation testing of samples leached showed similar recoveries after 48 hours of leaching, as observed in the whole ore leaches. Gold recoveries after 48 hours of leaching ranged from 74% to 84% while cyanide consumption was 0.54 kg/t to 1.39 kg/t of NaCN. Lime consumption was low at 0.41 kg/t to 0.46 kg/t. The combined gravity plus gravity tailing cyanidation gold recoveries for the composite ranged from 75% to 84% showing no real increase due to the very low gravity recovery of gold.

Gravity tailing flotation testing of samples showed excellent gold recoveries for all tests conducted. Gold recoveries for all three tests performed were reported at 99%. While the Omega composite head silver grade was reported as < 2 g/t, there was a significant improvement in recovery observed in the finer grind tests. For the tests, silver recoveries were shown to be 48% for the test at a P_{80} size of 125 microns, 66% for the test at a P_{80} size of 85 microns and 70% for the test at a P_{80} size of 52 microns.

The diagnostic leach program showed an initial 84.2% gold recovery of readily leachable gold. A further 3.2% of the gold was extracted from possible gold association with iron-arsenic compounds or bismuth minerals and 2.6% of the gold was further extracted from possible gold associations with weak acid soluble compounds. A total of 7.4% of the gold was observed to be from possible gold association with or occluded by sulphide minerals, pyrite and arsenopyrite. The remaining 2.5% of the gold remaining in the final leach residue was deemed to be the gold mainly associated with silicates or fine sulphides locked in silicates. The results from the diagnostic leach program should be viewed as an indication of general trends only.

Mineral Resource

A summary of the results of the estimated Inferred Mineral Resource, at cut-offs of 0.5 g/t Au for mineralization above an elevation of 130 m above sea level (masl), representing open-pit potential and for a cut-off of 3 g/t Au below 130 masl, representing underground potential are shown in Table 1.1.

Table 1.1 Summary of Inferred Mineral Resources as at 31 August 2011

Cut-off grade and location	Tonnes (Mt)	Au (g/t)	Contained Au ounces
0.5 g/t Au above 130 masl	3.8	2.50	306,100
3 g/t Au below 130 masl	1.2	4.33	166,000
Total	5.0	2.93	472,100

Note: A constant bulk density of 2.89 t/m^3 has been used.

AMC completed an independent Mineral Resource, as defined in NI 43-101, estimate based on original wireframes provided by Mistango and subsequently modified by AMC. These wireframes were modelled using the drill data to identify the zones over 1 g/t Au that was used to delineate the individual zones. In addition, the stope outlines of the previously mined areas were used to identify the trends in the mineralization.

Modelling of the 13 zones was carried out as follows:

- All wireframes in DXF format and drillhole files were imported into CAE Datamine software.
- Wireframes were sub-divided into the individual zones.

- Samples within each zone were composited to 1 m.
- Statistical and variogram analysis of the composited sample grades was carried out.
- A block model with blocks 25 m wide in the east and vertically, and 2 m thick in the north direction was prepared.
- Each individual zone was filled with blocks using sub-cells down to 2.5 m in the east and vertical directions, and 0.25 m in the north direction.
- Gold grades were estimated into each block within the zones, and the whole model, using ordinary kriging.
- The blocks located within the areas of previous mining were identified.
- The individual zones and background area models were combined into one model.

A total of 14,427 composites were available for the whole model with 975 composite samples selected from within the zone wireframes and used for the variogram analysis and estimation of the blocks gold grades within the zones.

Bulk density measurements have not been systematically collected with values only available for the mineralized zones. Based on the 115 samples collected an average density of 2.89 has been used for the resource estimate.

A statistical analysis and variography was carried out for the 13 zones combined.

Exploration Potential

There is significant exploration potential at the Omega project. Parts of the zones have not been sufficiently drilled to enable their continuity to be assessed. A number of drillholes have failed to penetrate to the other side of the areas previously mined. These areas will need infill drilling.

There also remains down-dip potential for many of the zones, along with their potential extension along strike.

Conclusions and Recommendations

The exploration programs run by Mistango have been conducted at generally good industry standards and the resulting data is appropriate for the estimation of Mineral Resources. Some historic 1980s drilling data was used for resource estimation purposes after validation by twinning of some of these drillholes. Additional information relating to old underground workings has also been incorporated into the model in order to ensure the estimate has taken into account the material already mined.

The geological interpretation of the deposit agrees with the style of mineralization found in this area. The mineralization occurs within a series of narrow zones adjacent to the hanging wall (south contact) of the ultramafic rocks with altered basaltic volcanics along the LLCB. The majority of the mineralization is deposited along the main fault, which defines the hanging and foot walls at the former Omega Mine. There is probably repetition of the zones due to stacking of the zones during movement along the LLCB.

All known errors have been removed from the drillhole database.

There are a number of recommendations:

- The location of the stopes needs to be improved with addition of new drillhole data.
- There are issues with the topography control currently being used and it is recommended Mistango purchase a more accurate satellite digital topographical map. The estimated cost is C$1,000.
- Better sample control must be implemented for any further drilling in order to reduce the mixing up of the standards as noted in the QA/QC review. Mistango has already put in place a QC system that is tracked by one person and this will help control this source of error.
- Continuous sampling of the drillholes should be employed. Better structural logging of the core would also aid in identifying if the faults impact on the continuity of the individual zones.
- Additional bulk density measurements should be collected from all rock- types within the drill core in order to generate values that have a good spatial coverage of the deposit. This could be undertaken as part of the sampling process on-site.
- Additional infill drilling should be undertaken within the apparent gaps, based on the current 50 m x 50 m grid. A total of 1,800 m is proposed, which at a cost of C$150 per metre, has an estimated cost o C$270,000.

TABLE OF CONTENTS

TABLES

FIGURES

APPENDICES

Distribution list:

2 copies to Mr Bob Kasner, Mistango River Resources Inc.
1 copy to AMC Toronto Office

2 INTRODUCTION

This Technical Report on the Omega Property (the Property) in McVittie township of the Larder Lake Mining Division, Ontario, has been prepared by AMC Mining Consultants (Canada) Ltd (AMC) Toronto, Ontario office on behalf of Mistango River Resources Inc (Mistango) of Kirkland Lake, Canada. It has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators (CSA) for lodgment on CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR).

This report is a statement of Mineral Resources as at 31 August 2012 and updates the earlier Technical Report entitled "Technical Review of the Omega Gold Mine Property, Ontario, Canada for Mistango River Resources Inc." prepared by David Power-Fardy, P.Geo., of Watts Griffis and McOuat (WGM), dated 30 November 2011.

The names and details of persons who prepared, or contributed to, the Technical Report are listed in Table 2.1. The Qualified Persons meet the requirements of independence as defined in NI 43-101.

Table 2.1 Persons who Prepared or Contributed to this Technical Report

Qualified Person	Position	Employer	Independent of Mistango	Date of Site Visit	Professional Designation	Sections of Report
Ms C Pitman, P.Geo.(Ontario)	Senior Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	7-9 August 2012	M.Sc., B.Sc, P.Geo.	All Sections
Mr R Webster, M.AIG	Principal Geologist	AMC Mining Consultants (Canada) Ltd	Yes	No visit	B appSc, M.AIMM, M.AIG	Part of Section 14

An inspection of the property was undertaken by Qualified Person C. Pitman between 7 and 9 August 2012. During the visit a review of the data collection and all geology aspects of the project was carried out. It also included the inspection of drill core, data handling and sampling procedures.

Mistango was provided with a draft of this report to review for factual content and conformity with the brief.

This report is effective as of 31 August 2012.

2.1 Units of Measurement and Conversion Factors

The Metric System or System International (SI) is the primary system of measure and length used in this report. Conversions from the Metric System to the Imperial System are provided below for general guidance.

Metals and minerals acronyms in this report conform to mineral industry accepted usage. Further information is available online from a number of sources, including website: http://www.maden.hacettepe.edu.tr/dmmrt/index.html.

The following conversion factors are used in this report:

- 1 hectare = 2.471 acres
- 1 hectare = 0.00386 square miles
- 1 square kilometre = 0.3861 square miles
- 1 metre = 3.28084 feet
- 1 kilometre = 0.62137 miles
- 1 gram = 0.03215 troy ounces
- 1 troy ounce = 31.1035 grams
- 1 kilogram = 2.205 pounds
- 1 tonne = 1.1023 short tonnes
- 1 gram/tonne = 0.0292 troy ounces/short tonne

A more complete list of conversion factors can be found on the following website: www.empr.gov.bc.ca/Mining/Geolsurv/MINFILE/manuals/coding/Hardcopy/appdvii.htm.

The term gram/tonne or g/t is equivalent to 1 ppm (part per million) = 1000 ppb (part per billion). Other abbreviations include:

- oz/t = ounce per long ton
- Moz = million ounces
- Mt = million tonnes
- t = tonne (1000 kilograms)
- wt% = percent by weight
- % = ppm/10,000
- m = metre
- km^2 = square kilometres
- ha = hectare
- BD = bulk density
- SG = specific gravity
- lb/t = pounds/tone

Dollars are expressed in Canadian currency (C$) unless otherwise stated.

3 RELIANCE ON OTHER EXPERTS

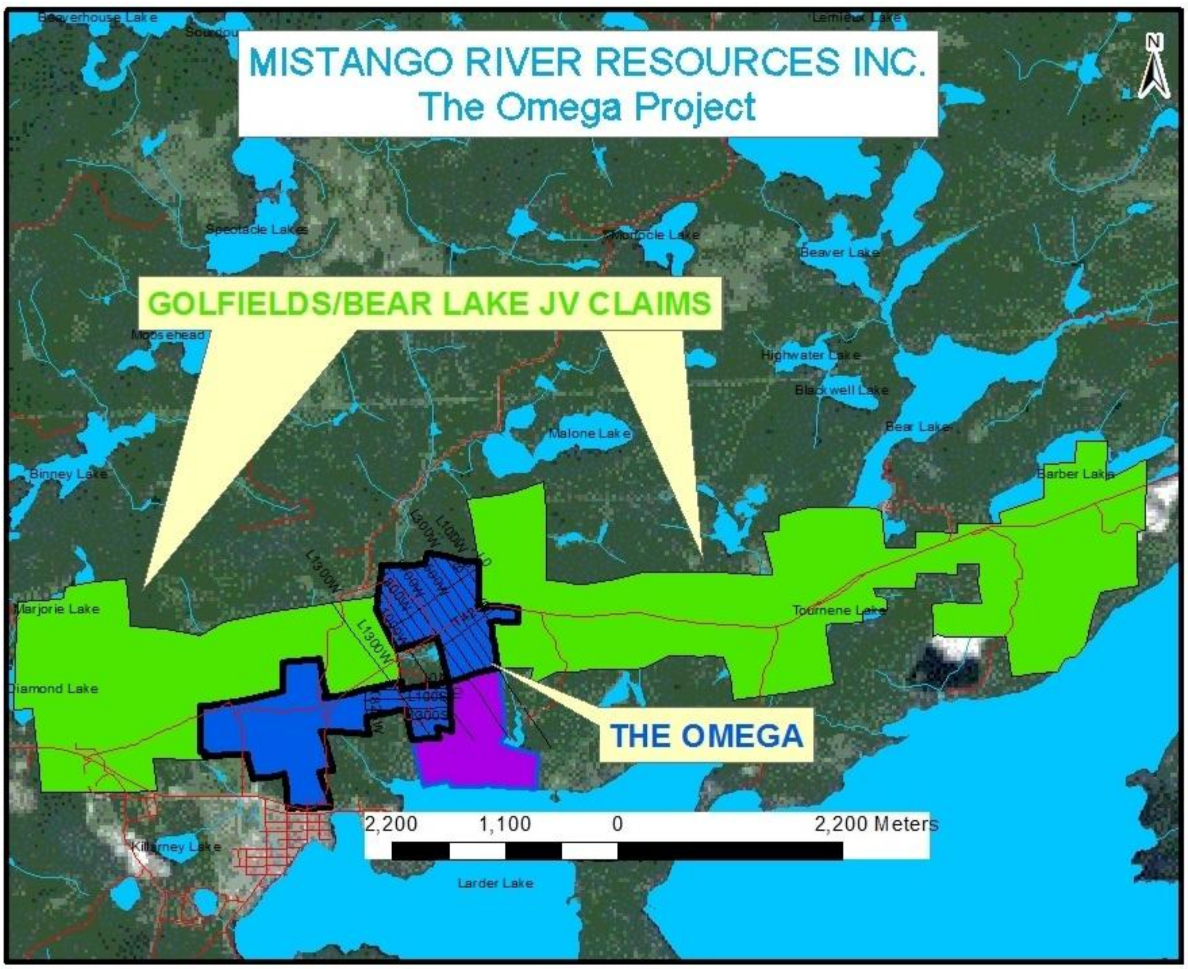
AMC has relied on the information for title and ownership provided by Mistango and confirmed by the Ministry of Northern Development and Mines (MNDM) website (http://www.mndm.gov.on.ca).

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Mistango properties in Ontario are located in McVittie Township of the Larder Lake Mining Division, Ontario. They cover a total of approximately 275 hectares, and encompass The Omega, Southwest Zone and Lake Zone group of claims, which are centred on UTM Zone 17, 596,700mE and 5,329,350mN (Figure 4.1).

Figure 4.1 Location Map



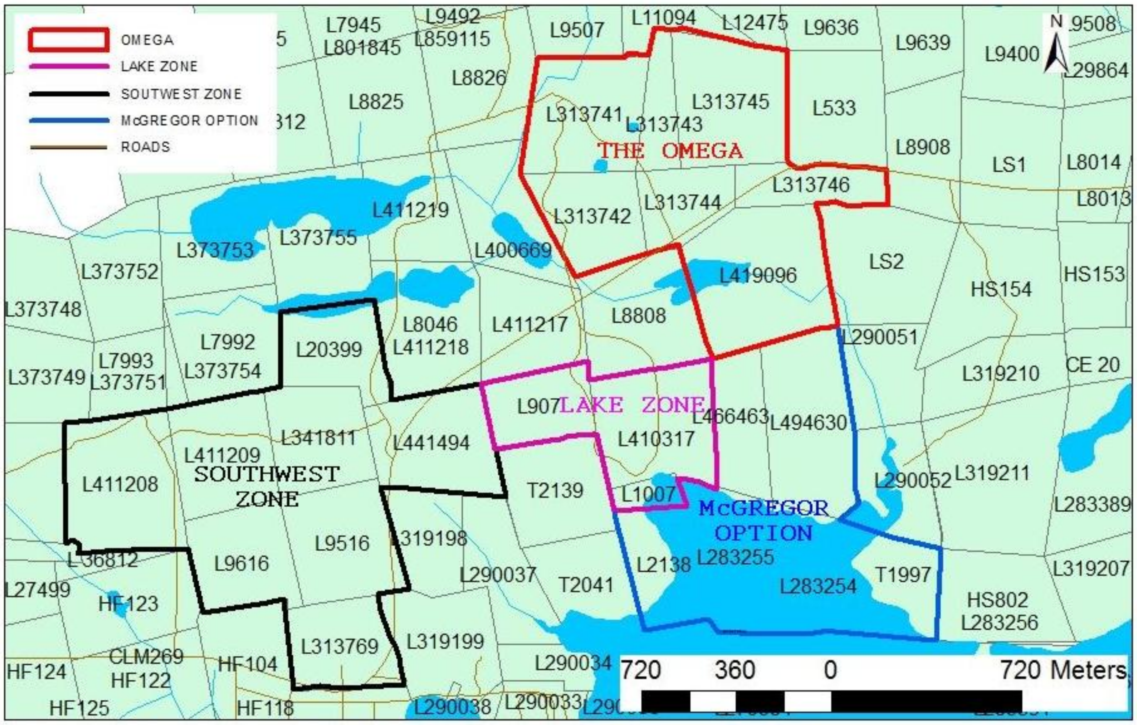
Source Mistango River Resources Inc., 2012
Note: only the blue claims are 100% owned and only The Omega is the subject of this report.

4.2 Property Description and Ownership

The Property is 100% owned by Mistango and consists of two mining leases containing 7 mining claims, L313741 to L313746 and L419096, (Figure 4.2) covering a total of 104 hectares. It is adjacent to an additional eight leased claims and two patented claims held by Mistango that form the adjoining Lake Zone and Southwest Zone blocks. There are also six optioned leased mining claims, McGregor Option, abutting The Omega and Lake Zone blocks.

The Property hosts the Omega Mine that historically produced gold from 1921 to 1929 and from 1936 to 1947. This is discussed in more detail in Section 6 of this report.

Figure 4.2 Mistango Ownership Map



Source: Mistango River Resources Inc., 2012

In July 2011, Mistango entered into an option agreement to acquire a 100% interest in six adjacent claims totalling 98.5 ha from Skead Holdings Ltd (Robert McGregor).

To acquire the 100% interest, the Company must pay a total of C$150,000 in cash and issue 300,000 shares as well as a work commitment of C$500,000 to be completed by June 30, 2014, with the vendor retaining a 3% Net Smelter Return Royalty (NSRR). A payment of C$25,000 and 50,000 shares was required on execution of the agreement. An additional cash payment of C$25,000 and 50,000 shares was issued on 30 June 2012 and cash payments of C$50,000 and 100,000 shares are to be issued on 30 June 2013 and again on 30 June 2014.

An initial work commitment of C$100,000 was met by 30 June 2012 and future work commitments are C$100,000 by 30 June 2013, and C$300,000 by 30 June 2014. Mistango has the first right of refusal to purchase the NSRR from Robert McGregor.

The complete list of Mistango claims, including the options, is detailed in Table 4.1.

Table 4.1 Mistango Claims and Options

Claim Number	Block	Owner	Issued	Expires	Area (Ha)
L313741	The Omega	MRR	01-Jan-05	31-Dec-25	21.70
L313742	The Omega	MRR	01-Jan-05	31-Dec-25	12.95
L313743	The Omega	MRR	01-Jan-05	31-Dec-25	5.60
L313744	The Omega	MRR	01-Nov-03	30-Oct-24	10.02
L313745	The Omega	MRR	01-Jan-05	31-Dec-25	20.72
L313746	The Omega	MRR	01-Nov-03	30-Oct-24	9.91
L419096	The Omega	MRR	01-Nov-03	30-Oct-24	23.14
Sub-total	**The Property**				**104.04**
L410317	Lake Zone	MRR	02-Jan-05	31-Jan-26	20.70
L907	Lake Zone	MRR		Patent	10.24
L1007	Lake Zone	MRR		Patent	10.24
Sub-total	**Lake Zone**				**41.18**
L20399	Southwest Zone	MRR		Patent	12.83
L313769	Southwest Zone	MRR	01-Apr-05	31-Mar-26	15.53
L341811	Southwest Zone	MRR	01-Apr-05	31-Mar-26	15.62
L411208	Southwest Zone	MRR	01-Apr-05	31-Mar-26	19.95
L411209	Southwest Zone	MRR	01-Apr-05	31-Mar-26	15.76
L9616	Southwest Zone	MRR	01-Apr-05	31-Mar-26	17.12
L9516	Southwest Zone	MRR	01-Apr-05	31-Mar-26	15.37
L441494	Southwest Zone	MRR	01-Apr-05	31-Mar-26	16.31
Sub-total	**Southwest Zone**				**128.48**
L466463	McGregor Option	SHL	01-Mar-04	28-Feb-25	10.89
L494630	McGregor Option	SHL	01-Mar-04	28-Feb-25	20.61
L2138	McGregor Option	SHL		Patent	11.05
T1997	McGregor Option	SHL		Patent	12.71
L283254	McGregor Option	SHL	01-Apr-05	31-Oct-25	27.20
L283255	McGregor Option	SHL	01-Apr-05	31-Mar-26	15.99
Sub-total	**McGregor Option**				**98.46**

AMC has checked the title ownership and status against information obtained from the MNDM claims website. All of the claims are listed as in good standing as at the date of this report.

For the purposes of this Technical Report only the Omega Group of claims were considered.

4.3 Existing Environmental Liabilities

Within the boundaries of the old Omega Mine there are two tailings disposal facilities containing waste from the flotation concentrate and the mill tailings. Both are located to the west of the number 2 shaft of the old Omega Mine. Under the terms of the lease agreement with the government, Mistango is required to prevent any environmental contamination originating from either of these facilities.

The flotation tailings facility contains an estimated 118,000 tonnes of material with an elevated level of cyanide. Mistango has therefore signed a memorandum of understanding with United Commodity AG (UC) of Thun of Switzerland regarding the reprocessing of the tailings from the former Omega Mine site. These will be removed and processed at a specialist facility. This agreement is dated 20 September 2012.

The second tailing facility contains mill tailings which are considered inert and Mistango will need to develop a closure plan for full rehabilitation, including revegetation. Due to their proximity, this cost will be shared with the neighbouring Bear Lake Property, which covers approximately one third of the area.

Historical infrastructure on closure of the mine consisted of two shafts 305 m (1000 ft) and 457 m (1,550 ft) deep and a winze down to 610 m (2,000 ft). The two shafts have been capped. All other surface infrastructure has been removed.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Property is located 30 km east of Kirkland Lake and 3 km east of Larder Lake, along Highway 66. The area is serviced by Ontario Northland bus and railway services, with a train station situated at Swastika. The Property is crossed by Trans-Canada Highway 66 which connects Kirkland Lake to Rouyn-Noranda. Highway 11 from North Bay to Cochrane crosses Highway 66, approximately 15 kms to the west of Kirkland Lake. The Property is easily accessible through several local service roads.

Kirkland Lake, Rouyn Noranda and Timmins Airports are serviced by both national and local carriers, with frequent daily services to major Canadian cities.

5.2 Climate

Summer temperatures average 20ºC which falls to -15ºC in the winter. Temperature extremes reported over the period of 1913 through to 2000 are -53 °C (1914) and +37°C (1921). The average precipitation is 90 mm with snowfall averaging 200 cm, mostly falling in December and January.

Drilling, most exploration activities, and potential mining operations can be conducted year round on the Property. Surface mapping is the main exploration activity that is limited by snow-cover.

5.3 Local Resources and Infrastructure

There is a long history of mining and exploration within the region. Kirkland Lake Gold (KLG) has recently expanded its mining operations. Other mining and exploration companies include Queenston Mining Inc., Armistice Resources Corp. and St Andrew Goldfields. The latter is now operating the reopened Holt-McDermott gold mine north of Kirkland Lake.

Mining and mineral exploration, equipment fabrication, construction trades, transportation, tourism and forestry are the main sources of employment in the area. The Property is located in an active mining belt. The area offers a substantial professional work force experienced in mining and related activities. It also offers most supplies and services. The current high level of mining activity could affect immediate availability of skilled labour.

The area has well developed infrastructure therefore the availability of power, transportation and water are not likely to impact on the project.

5.4 Physiography and Vegetation

In the area of the Property there are small hills, typical of the Canadian Shield, along with the eskers and moraines associated with the last ice age. The elevation varies between 290 m and 330 m above sea level in the vicinity of the old mine. All timber has been cleared and vegetation is limited to spruce with jack pine and alders in regrowth.

The local terrain varies from flat to hilly, mostly wooded, with coniferous forests and numerous lakes and streams. The dominant tree varieties include black spruce, jack pine and trembling aspen, as well as white birch, alder and white spruce. The dominant forest form is black spruce–feathermoss climax forest which characteristically exhibits moderately dense canopy and a forest floor of feathermoss. There are numerous kettle lakes that were developed during the last ice age.

A local landform known as the "the height of land" is within the area of Larder Lake, at an elevation of 318 masl. This elevation marks the "divide" between the Arctic watershed where the drainages flow northwards into Hudson Bay and James Bay, and Atlantic watershed where the drainages flow southerly into the Great Lakes - St. Lawrence River drainage system.

6 HISTORY

6.1 Prior Ownership

The following table gives a summary of the ownership and activities of the Omega and Southwest Group of claims. A more complete description is contained in the previously published Technical Report (Fardy, 2011).

Table 6.1 Summary of Ownership

Date	Company	Status
1914	Jack Costello	Discovered #1 Ore Zone on claim L1794
1921	Crown Reserve Mining Company	Adjacent claims to east of 31 Ore Zone staked
1921	Canadian Associated Goldfields	Costello claim sold to company
1926	Canadian Associated Goldfields	Built mill and started production
1928	Crown Reserve & CA Goldfields	Production ceased
1936	Omega Gold Mines	Production began and increased to 450 tons per day (tpd)
1947	Omega Gold Mines	Production and milling ceased
1950	Lomega Gold Mines	Restructuring and single deep hole drilled intersecting deep mineralization zone
1974	Davy Lowe	Discovery on claim L341811
1975	Grasset Lake Mines Ltd	Completed 6 hole drill program on claim L341811
1979	Lenora Explorations Ltd	Omega claims and additional Southwest claims combined into one company
1987	AXR Resources	Argentex Resources Exploration Corporation amalgamated with Sholia Resources Limited to form AXR Resources
1988	Greater Lenora Resources Corporation	Lenora, Mary Ellen Resources Limited ("Mary Ellen") and AXR Resources Limited ("AXR") amalgamated to form the Greater Lenora Resources Corporation
2001	MinCo Inc.	Greater Lenora Resources Corporation and 3796299 Canada Inc. Amalgamated to form MinCo (3851419 Canada) Inc.
2003	GLR Resources Inc.	Minco transferred 15 claim leases to GLR Resources Inc., comprising the Omega Group and the Southwest Group
2011	Mistango River Resources Inc.	GLR Resources Inc. changed its name to Mistango River Resources Inc.

6.2 Exploration and Development

In 1980, Greater Lenora Resources Corporation (Lenora) carried out a drilling program on the "West" group of claims (L20399, L411208, L411209, L341811, L441494, L419377, L313769 and L313770). The drilling consisted of 11 holes totalling 1,135 m. The program was designed to test the gold-bearing carbonate rock on Claims L341811 and L441494. Gold values were intersected in two distinct carbonate horizons within ultramafic volcanics. The results of the drilling indicated that that the gold mineralization had a steep plunge to the west and was controlled by block faulting (Hinse, 1981).

In 1982, Lenora carried out an exploration program on the "Lake" claim (L410317). This program consisted of trenching, channel sampling and drilling. A total of 111 channel samples were taken along a length of 84.5 m and approximately 376 m of drilling was completed. Several drillholes returned anomalous gold values, therefore the drilling was continued to further outline the mineralized zone (Hinse, 1983).

An extensive surface exploration program was carried out on the Property between January and December 1983. Work consisted of bulk sampling in the Lake and Southwest Zones, on claims L410317 and L341811, in the Southwest Group of claims. Also detailed geophysical (magnetic) surveys on the Omega Group claims, along with test pitting, surface trenching, channel sampling and diamond drilling on the Omega Group and Southwest Group of claims were done.

6.3 Historic Resources

The results from the 1983 drilling program on the Omega Group of claims were considered highly encouraging. A resource of 164,154 tonnes at a grade of 5.48 g/t Au was outlined for the combined No. 4 and No. 17 ore zones of the project. The old Omega Mine crown pillar was calculated to contain approximately 91,391 tonnes at a grade of 5.28 g/t Au (Hinse, 1984).

It should be noted that these resources are historical, are not being treated as current mineral resources or reserves, and are not compliant with the NI 43-101 guidelines and therefore should not be relied upon.

Since the Hinse, 1984 report the western crown pillar has been mined by Balmoral Mines (pers. comm., R. Kasner).

6.4 Prior Production

The Omega Mine was in production for two periods during the last 100 years. Estimates of the production during the 1920’s are treatment of approximately 20,480 tonnes of ore yielding over 2,500 oz Au ($52,295 value at the fixed price of $20.67/oz of Au).

During the second period of production, between 1935 and 1947, the mine milled an estimated 1.436 Mt averaging 5.41 g/t Au yielding 215,000 oz Au, (Hinse G. , 1986).

7 GEOLOGICAL SETTING AND MINERALIZATION

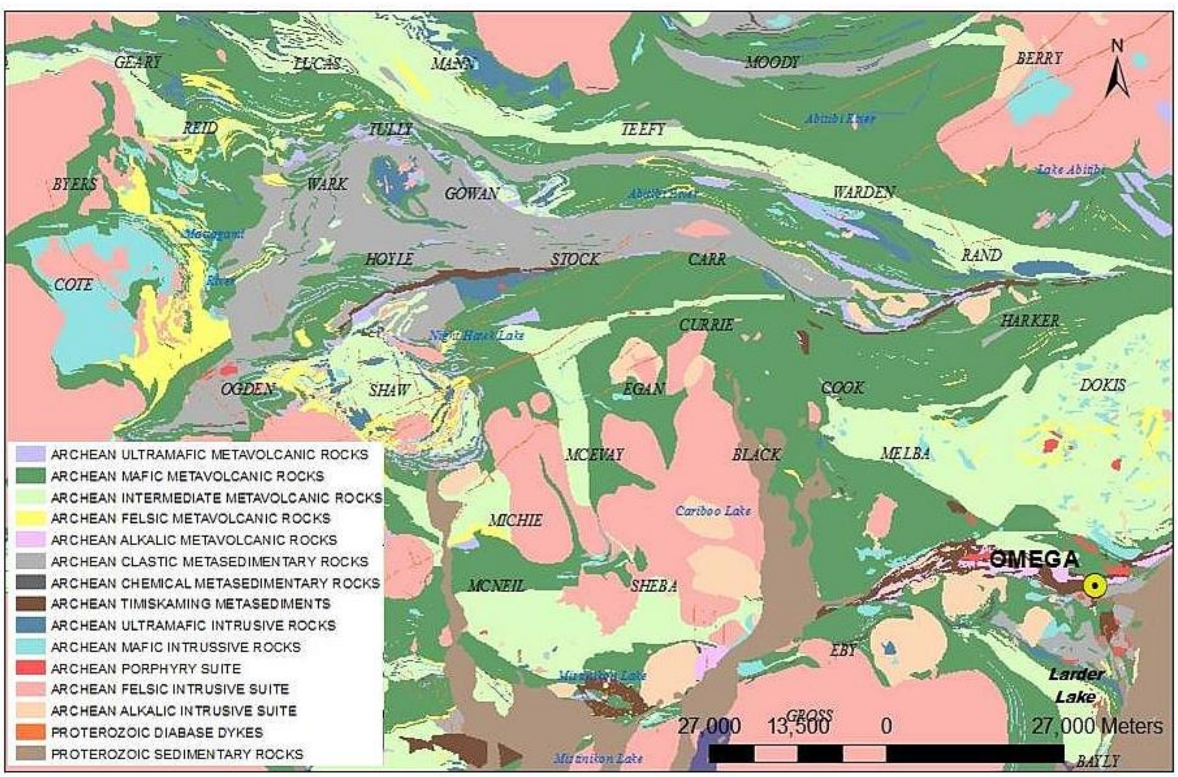
The Property is located within the Abitibi Greenstone Belt of the Superior Province of the Precambrian Shield. It is a part of the Cadillac-Larder Lake Deformation Zone (C-LLDZ), a regional scale shear that passes through the northern part of the Property.

7.1 Regional Geology

The Abitibi Greenstone belt is a sub-province of the Archean Superior Province of the Canadian Shield. The Abitibi Belt is truncated by the Grenville Structural Zone on its southeast side and the Kapuskasing Structural Zone to the west. The Opatica gneiss belt marks the northern boundary and the Huronian Supergroup sediments overlie the rocks of the Abitibi to the south side of the belt (Powell, 1991) (Figure 7.1).

The southern volcanic zone (SVZ) of the Late Archean Abitibi belt of the Superior province of Canada is dominated by komatiitic to tholeiitic volcanic plateaus and large, bimodal, mafic-felsic volcanic centres. These volcanic rocks were erupted between about 2717 Ma and 2700 Ma in a series of rift basins that formed as a result of wrench-fault tectonics. They overlie and juxtapose a volcano-plutonic assemblage characterized in the northern Abitibi belt. The age of the assemblage is about 2720 Ma or older, and it comprises basaltic to andesitic and dacitic subaqueous massive volcanics, cored by comagmatic sills and layered anorthositic complexes. They are overlain by felsic pyroclastic rocks that were comagmatic with the emplacement of tonalitic plutons at 2717 ± 2 Ma. (Lunden. J. and Hubert, 2007)

Figure 7.1 Geology of the Abitibi Belt Region



Source: Mistango River Resources Inc., 2012

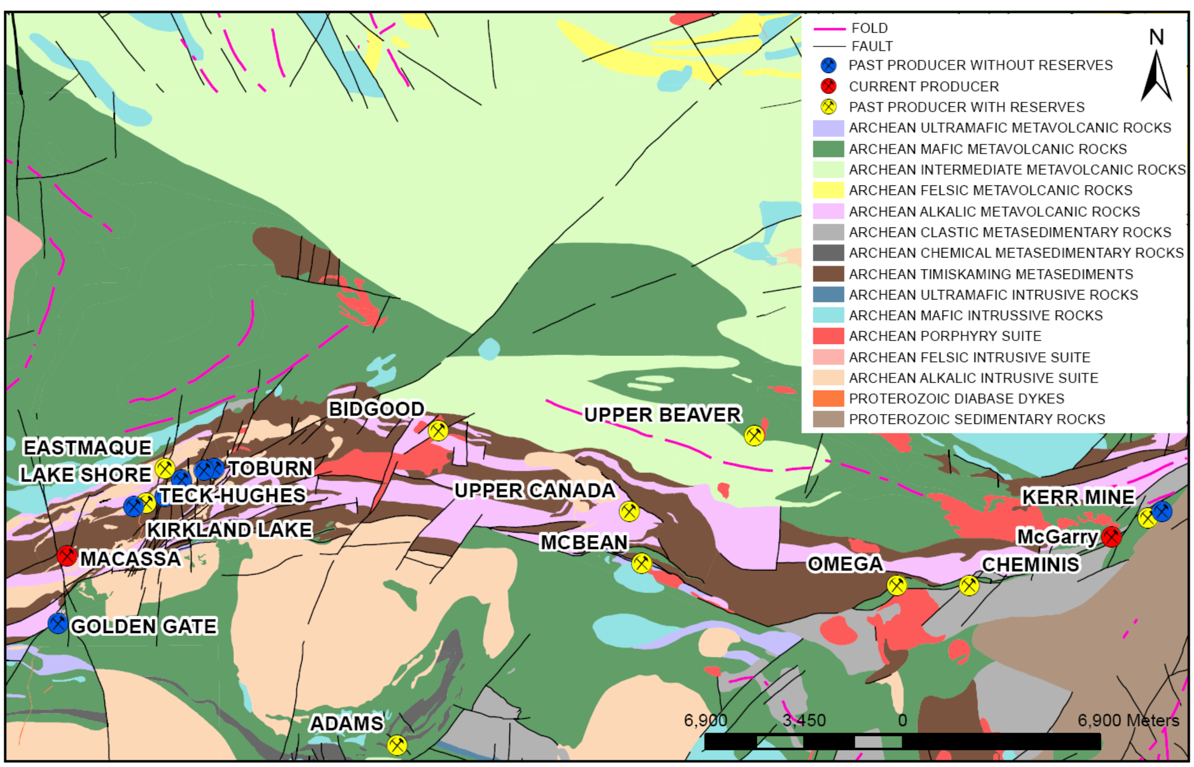
7.2 Local Geology

The oldest sequence in the Kirkland Lake - Larder Lake area is Precambrian Abitibi volcanics interbedded with slate and chert, dated between 2747 Ma and 2705 Ma. They range in composition from komatiites and tholeiites at the stratigraphic base to calc-alkaline volcanics at the top. This sequence contains long narrow bodies of diorite and gabbro, as well as coarser-grained flows and was subsequently deformed into a series of regionally ESE-WNW trending folds (Powell, 1991). Refer to Figure 7.2.

Timiskaming Group interbedded sediments and alkali volcanics dated circa 2680 Ma, unconformably overlie the older volcanics and their deposition is spatially associated with the Larder Lake-Cadillac Break (LLCB). The Timiskaming Group sediments are comprised of two sequences, one non-marine fluvial in origin and the other of sub-marine fans, intercalated with several volcanic sequences varying in composition from intermediate to basic and suggestive of an island arc origin (Powell, 1991). These units form a long, relatively narrow, east-west trending belt which was intruded by a number of syenite and porphyry stocks and dykes dated 2673 Ma. Contemporaneous lamprophyre and diabase dykes are widespread throughout the region. Most of the diabase is of the "Matachewan" swarm of north-striking dykes dated at 2485 Ma (Heaman, 1988).

Undeformed Proterozoic age Huronian Supergroup sedimentary rocks, primarily of the Cobalt Group, unconformably overlie the Achaean basement, which are in turn are intruded by Nipissing diabase dykes dated at 2200 Ma (MNDM, Kirkland Lake Resident Geologist, 2002).

Figure 7.2 Geology of Kirkland Lake Area

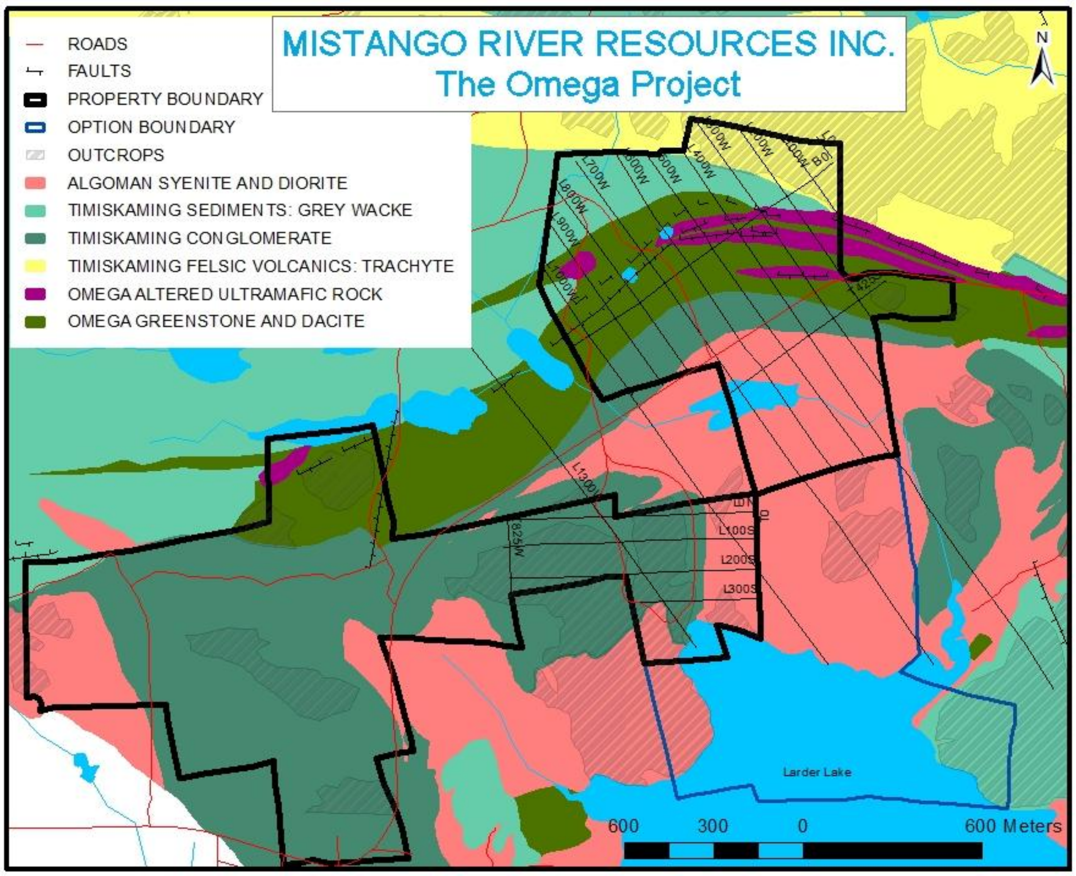


Source: Mistango River Resources Inc., 2012

7.3 Property Geology

The Property lies on the southern limb of an overturned anticline which has its axis lying sub-parallel to the northern boundary of the property (Hinse, 1981). The anticline is sharply folded and overturned to the north and is broken by a thrust fault following the strike of the fold, suggesting a roll-over anticline at the leading edge of the thrust. The rocks along this limb face north and are overturned, dipping ~60° south (Jenney, 1941). Jenney, 1941 also states that the south side is displaced upwards, but the amount of movement is not known. At least a part of the movement is post-ore as indicated by vein fragments in the fault gouge and by the drag of the ore along the fault plane (Figure 7.3).

Figure 7.3 Geology of the Omega Project



Source: Mistango River Resources Inc., 2012

7.3.1 Structural Geology

The C-LLDZ is expressed as the main fault within the Omega Group, the Larder Lake Break (LLB), which roughly parallels the contact between the older volcanics and younger Timiskaming rocks. To a depth of 165 m it has an average dip of ~ 60^0 to the south-east, but this shallows with depth and below 320 m the dip has flattened to about 40^0. Within the Property the LLC has been traced down to 600 m to the southwest of No. I shaft, but further to the east it divides into a number of weaker branches. The fault plane is irregular and sinuous in certain parts of the mine.

The rollover anticline is broken-up into three main blocks by three or four faults that are roughly parallel to each other and trending north-south. The Omega Group mineralization horizons are contained within these three fault blocks.

The alteration patterns within the Property are divided according to the primary lithology type.

The ultramafic volcanics show early stage alteration to talc-chlorite with minor amounts of carbonates and varying quartz content through to chlorite-carbonate with weak silicification. With increasing alteration this progresses through alteration to a brown carbonate with ankerite-dolomite-silica and varying amounts of sericite and finally a green carbonate containing fuschite mica-silica-grey carbonate with varying amounts of sericite. The latter lithology type is associated with gold mineralization occurring as visible free gold where there is a relationship between the degree of alteration and the amount of gold. Within the Property however, there is very limited gold found within this lithology type.

The mafic volcanics exhibit an initial stage chlorite-albite-muscovite assemblage which increases to a carbonate-albite-muscovite-quartz-leucoxene assemblage as the alteration advances. Two domains are observable within the altered mafic tholeiites: hyaloclastite breccia and sulphide breccia. The hyaloclastite breccia consists of quartz-feldspar clasts sealed by secondary feldspar, paragonitic phengitic mica, pyrite and carbonate (dolomite-ankerite). The sulphide breccia hosts the mineralization as electrum within sulphides (mainly pyrite) sealed by carbonate, mica feldspar, graphite and strained quartz (Renaud, 2011).

Sedimentary rocks within the Property display carbonatization and albitization as well as late stage silicification. Mylonite and cataclastite develops in the zones that have undergone high levels of strain such as along the C-LLDZ. All alteration of the sedimentary units is associated with mineralization.

7.4 Mineralization

The two most prominent gold-bearing structures in the region are the C-LLDZ and the Kirkland Lake -*Main Break* (KLMB). The C-LLDZ is a regionally extensive shear zone, characterized by the development of mica schists and locally marked by hydrothermal alteration (silicification, sulphidation and carbonatization), and the development of quartz stockwork and breccia. Green mica (fuchsite) is commonly developed where alteration overprints ultramafic rocks. This structure is considered to be the western extension of the Malartic-Cadillac Deformation Zone, making this structure more than 160 km long. The zone has the appearance of being a south-dipping reverse fault, in which the south-side seems to have moved upwards and eastward relative to the north-side. However, the zone has also been described as a slightly overturned normal fault structure. The KLMB is a fault zone branching north-westerly from the C-LLDZ near Kenogami Lake. This structure has been identified in all the gold mines in Kirkland Lake down to depths of more than 2 kms. The structure varies from a single plane to multiple bifurcating planes. The widest ore bodies occur where the cross-over faults and the tension fractures between the planes are most numerous (MNDM Resident Geologist, 2002).

7.4.1 Mineralization Styles

There are three main mineralization styles recognizable in the Omega deposit. These are:

1. Flow Ore: Related to metasomatism of mafic, Fe-rich, variolitic, hyaloclastite tholeiites. Gold is present in the form of electrum within pyrite and minor arsenopyrite, hosted in altered dacite (silica, albite, sericite and carbonate) as a disseminated stockwork. There is increased leucoxene concentration in the mineralized zone. This alteration and mineralization is not limited to tholeiitic volcanics, as close to surface (down to about 150 m) it is also found within altered argillites interlayered with tuffs and siltstones.
2. Green Carbonate: Free gold is associated with altered, deformed ultramafic komatiite flows. Within the Project area this type of mineralization is not extensive and represents a minor amount of the mineralization.
3. Later quartz filled veins with visible gold, cross-cutting the volcanics, representing a later remobilization of the gold during periods of deformation.

7.4.2 Distribution of Mineralization

The main gold mineralization at Omega is in the form of electrum (80% Au, 19% Ag and 1% Hg) found within the pyrite, in zones containing 20% to 40% sulphides (Renaud, 2011). The mineralization occurs adjacent to the hanging wall (south contact) of the ultramafic rocks with altered basaltic volcanics along the LLB. The majority of the mineralization is deposited along the main fault, which defines the hanging and foot walls at the old Omega Mine. The No.1 Zone is situated in the hanging wall and the No.2 Zone in the foot wall relative to the fault. Due to similar alteration and rock type assemblages hosting these two zones, they most likely represent the same ore body being faulted and up-thrusted upon itself, with the No.1 Zone being originally stratigraphically lower. This is supported by vein fragments in the fault gouge and by the drag of the ore along the fault plane (Thompson, 1941). Close to surface the pyrite is usually oxidized to hematite. Historically this has been noted by calling the oxidized areas *Red Ore* and the non-oxidized, albitized areas *Grey Ore*.

A number of other previously recognised ore zones are also located within the hanging wall, all of which conform to the same style of mineralization. To date, none of the zones identified fall within the green mica-carbonate altered (green carbonate) units.

Secondary mineralization in the form of well-defined visible gold bearing quartz veins lie within, or closely associated with, the No. I orebody and appear to have been introduced post faulting.

8 DEPOSIT TYPES

The tectonic setting of the local geology has been interpreted as a back-arc environment where mineralization is emplaced in response to successive arc rifting, back-arc basin development, and exhumation of the adjoining accretionary complexes along major arc-parallel thrust faults. The deposit type has been classified as Orogenic gold, a classification based on epigenetic compressional regimes and common structural control proximal to convergent plate boundaries (Groves, 1998). This type of orogenic gold deposit corresponds to structurally controlled complex epigenetic deposits characterized by simple to complex networks of gold-bearing, laminated quartz-carbonate fault-fill zones. These zones are hosted by moderately to steeply dipping, compressional brittle-ductile shear zones and faults with locally associated shallow-dipping extensional veins and hydrothermal breccias. The deposits are hosted by greenschist to locally amphibolite-facies metamorphic rocks of dominantly mafic composition and formed at intermediate depth (5 kms to 10 kms). The mineralization is syn- to late-deformation and typically post-peak greenschist-facies or syn-peak amphibolite-facies metamorphism. They are typically associated with iron-carbonate alteration. Gold is largely confined to the quartz-carbonate zone network. However gold may also be present in significant amounts within iron-rich sulphidized wall-rock selvages or within silicified and arsenopyrite-rich replacement zones (Dube, 2007).

Greenstone-hosted quartz-carbonate vein deposits, within the orogenic gold classification, typically occur in deformed greenstone belts of all ages, especially those with variolitic tholeiitic basalts and ultramafic komatiitic flows intruded by intermediate to felsic porphyry intrusions, and sometimes with swarms of albitite or lamprophyre dyke. They are distributed along major compressional to transtensional crustal-scale fault zones in deformed greenstone terranes commonly marking the convergent margins between major lithological boundaries, such as volcano-plutonic and sedimentary domains. The large greenstone hosted quartz-carbonate vein deposits are commonly spatially associated with fluvio-alluvial conglomerate (e.g.Timiskaming conglomerate) distributed along major crustal fault zones (e.g. Destor Porcupine Fault). This association suggests an empirical time and space relationship between large-scale deposits and regional unconformities. These types of deposits are most abundant and significant, in terms of total gold content, in Achaean terranes (Dube, 2007).

Although the gold deposits along the C-LLDZ are broadly classified as vein- or lode-type (orogenic), they are highly variable in character. They range from discrete quartz-carbonate veins carrying native gold and associated minerals within various host rocks, though auriferous pyritic and cherty zones containing erratic veining, to mineralized veins and fracture systems in sialic to mafic porphyritic rocks. Varying ore types often exist within a single deposit (Hinse G. , 1986).

9 EXPLORATION

Since early 2011, Mistango has undertaken four phases of exploration on the Omega Group and Southwest Group of claims. This included geophysical surveys, soil sampling and drilling. The following is a discussion on Phase 1, as the other three phases consisted of drilling alone, which will be covered in Chapter 10 of this report.

9.1 Phase 1 – April to October 2011

During 2011, Mistango carried out Phase I exploration on the Omega Property consisting of line cutting, magnetometer and deep induced polarization (IP) surveys, with limited soil sampling to profile the IP anomalies.

The IP survey was conducted over the Omega grid by Larder Geophysics Ltd (LGL) in April 2011, using a 10 channel Elrec Pro receiver with a VIP 3000 (3kw) transmitter. The deep IP survey configuration was used for the survey. A 19.34 km grid was established prior to the survey, with lines spaced at 100 m intervals and stations at 25 m. The baseline was oriented at 55^0 (along the strike of the mineralization zones) for 1.3 km. Four lines of deep IP were performed along lines 400 W, 700 W, 1000 W and 1300 W. These results were plotted as both raw data sets and with a 3D inversion performed. Penetration at the Omega Mine site was 100 m but at grid line 1200 S it is approximately 450 m. Plans of High Definition Induced Polarization (HDIP) resistivity and chargeability were provided every 50 m down to 450 m. Sections of HDIP resistivity and chargeability were provided on lines at 400 W, 700 W, 1000 W and 1300 W (See Appendix A for the results). The Lake Zone grid was surveyed by LGL in August 2011, with an induced polarization survey, using the dipole-dipole array. The lines were spaced at 100 m intervals with stations at 25 m. They were oriented at 90^0 and the baseline at 360^0. Results are shown in Appendix A.

LGL also undertook a total field magnetic survey over the Omega grid in July 2011. This survey was conducted with a GSM-19 v7 Overhauser magnetometer with a second GMS-19 magnetometer for a base station mode for diurnal correction. A total of 6.225 line kilometres of the property was surveyed, with readings at 12.5 m intervals and lines spaced at 100 m intervals. Results are shown in Appendix A.

A total of 85 soil samples were collected on the Omega grid line 400W and the Lake Zone grid, in order to profile the IP anomalies for gold. The samples were taken in September 2011 with an auger at a depth of 30 cm. They were analysed for gold and 37 other elements using the enzyme leach method by Activation Labs in Ancaster, Ontario. The sample program returned anomalously high Au in the Lake Zone at grid coordinates 0S and 450W (327 ppb Au) and at 300S and 325W (1,280 ppb Au). Those two sample locations also returned anomalous zinc content of 109 ppm Zn and 74 ppm Zn respectively. Copper content for location 0S and 450W was 992 ppm and the same location also had elevated silver content of 0.9 ppm.

10 DRILLING

Mistango commenced a drill program to outline the potential of the Omega Group of claims in early 2011 and have subsequently completed three phases of drilling. The holes are drilled either using BTW or NQ core sizes and put into 1.5 m wooden core boxes at the rig. The lids are taped down and the boxes are brought back to the core shack on a daily basis using the company vehicle.

Core recovery for the shallow holes (BTW) averages 95%, with the average recovered length being 2.8 m. RQD measurements average 74% on these core intervals. AMC do not consider that core recovery materially impact on the accuracy of the results. It must be noted however, that in the areas where the drilling intersects stopes there have been issues with the recording of the missing intervals A closer analysis of recording of missing intervals with Mistango has resolved the issue and AMC considers that overall there has been minimal effect on the reliability of the results.

Drill core is logged and sampled under the supervision of F. Sharpley, P.Geo. and I. Iliev, G.I.T.

10.1 Phase 1 – April to October 2011

Diamond drilling consisted of 11,866 m in 48 holes to investigate the potential for open-pit and the potential down-plunge extension at depth below the mine workings on the Omega Group of claims. A total of 6,273 samples were taken and assayed.

During Phase 1, Huard Drilling of New Liskeard, Ontario, drilled 40 diamond drillholes totalling 6,848 m producing BTW core from the Omega Property. This program was aimed at assessing the open-pit potential around the surface expression of the No. 1 and No. 2 zones and the upper workings. These zones had been previously mined over a strike length of 225 m from section 650W to 875W and to within a vertical depth of approximately 10 m below surface in one place. These holes were drilled to a vertical depth of 150 m at 50 m intervals along a strike length of 750 m from section 200W to 950W. Significant intercepts from these holes are shown in Table 10.1. Figure 10.1 shows the grid locations of all the collars from Phase 1 drilling. For a summary of all the drill data refer to Appendix B.

Table 10.1 Significant Intercepts from Phase 1 Shallow Holes

BHID	Section line	Zone	Au (g/t)	Composite Length (m)
OM-11-05	350W	1/2	5.37	14.0
OM-11-04	550W	1/2	1.24	13.2
OM-11-14	650W	1/2	1.41	32.0
OM-11-21	650W	1/2	2.84	19.3
OM-11-23	600W	1/2	2.69	22.0
OM-11-34	800W	1/2	2.45	24.0

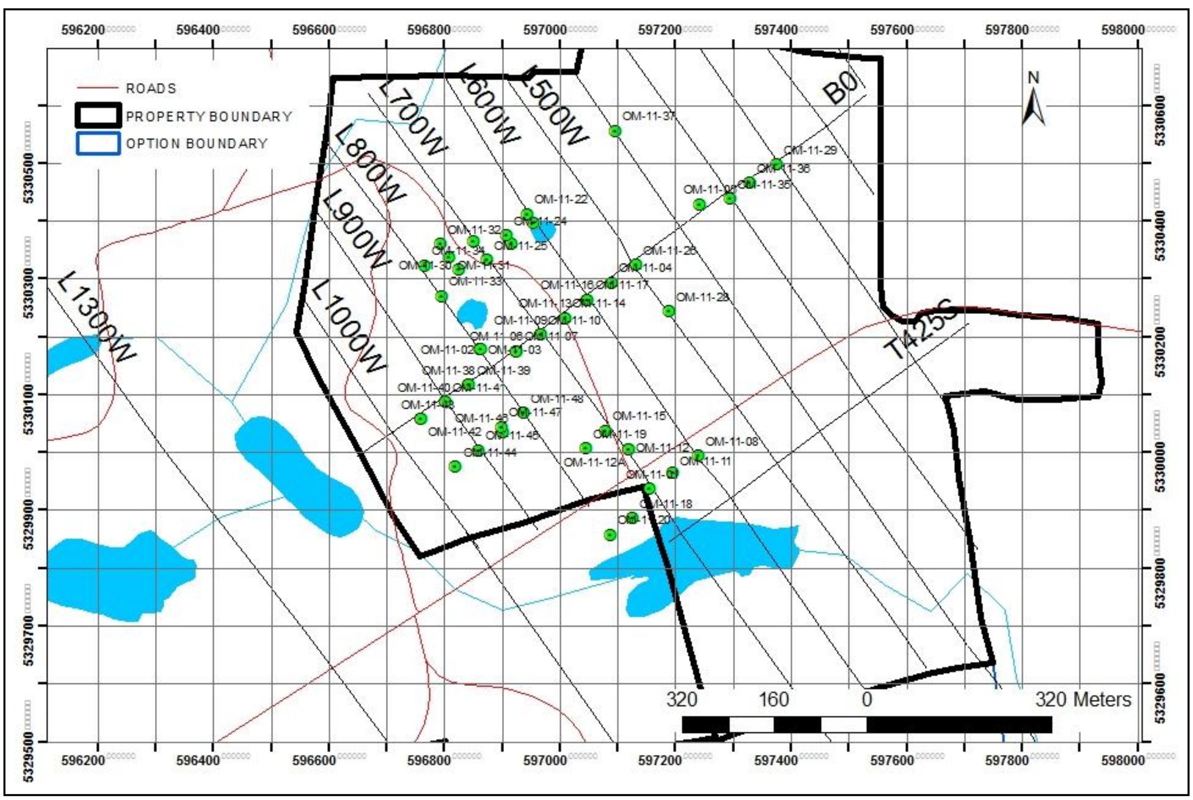
Eight deep diamond drillholes, totalling 5,018 m, were drilled by Laframboise Drilling of Earlton, Ontario, using NQ core. These were aimed at investigating the down plunge extension of the Omega deposit below the old workings and they were spaced at 50 m intervals. Two holes did not reach the target due to either being lost in a stope or flattening. Significant intercepts from these holes are shown in Table 10.2.

Table 10.2 Significant Intercepts from Phase 1 Deep Holes

BHID	Section line	Zone	Au (g/t)	Composite Length (m)
OM-11-08	600W	1/2	3.44	10.0
OM-11-11	650W	1/2	4.51	13.0
OM-11-15	700W	1/2	3.13	7.8
OM-11-19	750W	1/2	8.06	9.2
OM-11-01	700W	-	4.06	3.0
OM-11-19	750W	-	2.94	13.0
OM-11-20	800W	-	3.71	4.8
OM-11-20	800W	-	3.07	8.5

Both OM-11-18 and OM-11-20 were collared outside of the Property boundaries, as the exact boundary had not been surveyed at that time. The significant intercepts within these two holes however do fall within the claim limits.

Figure 10.1 Collar Locations for Phase 1 Drilling



Source: Mistango River Resources Inc.

10.2 Phase 2 – November 2011 – February 2012

During Phase 2, a total of 33 diamond drillholes were completed. This included 18 shallow and 15 deeper, for a total length of 11,480 m, with 4,454 samples taken from the core and assayed.

A total of 3,630.2 m were drilled by Huard Drilling of New Liskeard, Ontario, producing BTW core for the purposes of assessing the surface expression of the No. 1 and No. 2 Zones, the No.17 Zone near surface, and the upper workings. One hole (OM-12-80) was drilled on section 550W to determine the close to surface mineralization potential of No. 17 zone. These holes ranged along strike between 450W and 950W (500 m), at 50 m intervals and were drilled down to about 200 m vertical depth. Significant intercepts from these holes are shown in Table 10.3. Figure 10.2 shows the grid locations of all the collars from the Phase 2 drilling. For a summary of all the drill data refer to Appendix B.

Table 10.3 Significant Intercepts from Phase 2 Shallow Holes

BHID	Section Line	Zone	Au (g/t)	CompositeLength (m)
OM-11-50	900W	1/2	2.77	5.0
OM-11-54	850W	1/2	3.11	8.0
OM-11-58	550W	1/2	3.59*	4.0
OM-11-62	550W	1/2	2.49*	5.0
OM-11-65	550W	1/2	1.04	16.0
OM-11-69	500W	1/2	2.69	12.0
OM-11-70	650W	1/2	2.421	10.0
OM-11-70	650W	1/2	3.251	6.0
OM-11-70	650W	1/2	4.491	2.0
OM-11-70	650W	1/2	2.871	4.0
OM-11-76	600W	1/2	2.892	4.0
OM-11-76	600W	1/2	4.672	2.0

Note: *Including visible gold

A total of 15 deep diamond drillholes, totalling 7,850 m were drilled by Laframboise Drilling of Earlton, Ontario, using NQ core. These were aimed at investigating the down plunge extension of the Omega deposit below the old workings and they were spaced at 50 m intervals. One hole did not reach the target due to intersecting a stope. Significant intercepts from these holes are shown in Table 10.4.

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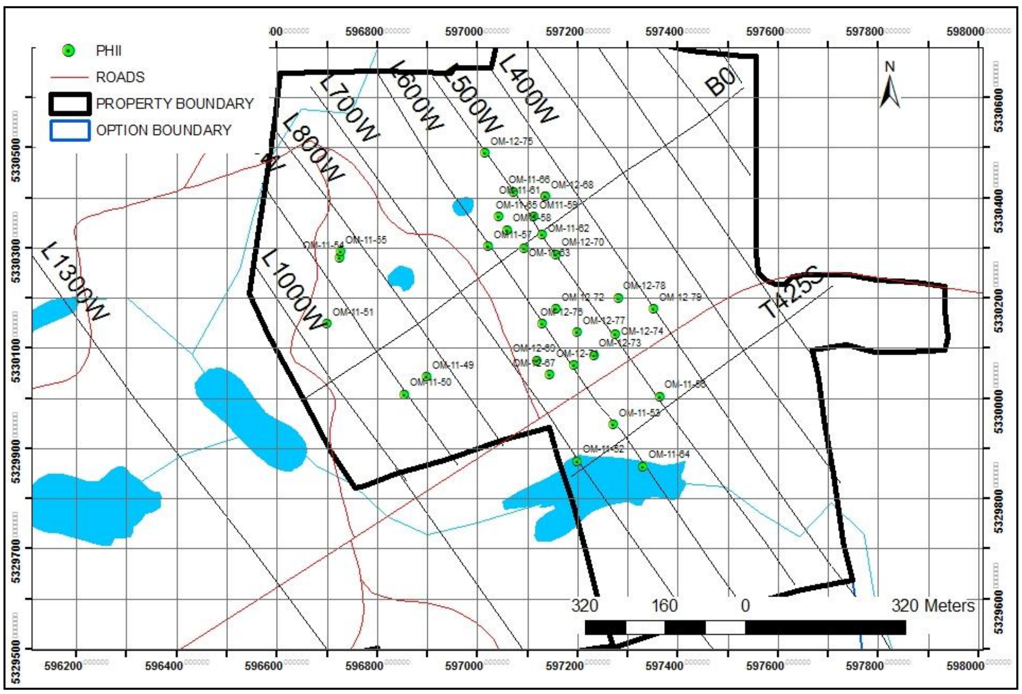
Table 10.4 Significant Intercepts from Phase 2 Deep Holes

BHID	Section Line	Zone	Au (g/t)	Length (m)
OM-12-76	600W	1/2	0.80	11.0
OM-12-77	600W	1/2	1.01	7.0
OM-12-78	600W	1/2	1.59	5.0
OM-11-53	600W	1/2	4.90	7.0
OM-11-64	600W	1/2	0.92	6.0
OM-11-64	600W	1/2	1.22	19.0
OM-11-64	600W	1/2	1.85	12.0
OM-12-67	650W	1/2	1.65	13.0
OM-12-67	650W	1/2	1.78	16.0
OM-12-68	450W	1/2	2.68	12.0
OM-12-69	650W	1/2	1.35	8.0

Figure 10.2 Collar Locations for Phase 2 Drilling



Source: Mistango River Resources Inc.

10.3 Phase 3 – March 2012 – July 2012

During Phase 3 of the program a total of 10 diamond drillholes were drilled. All holes were shallow with a total length of 2,324 m, and 1,375 samples being assayed. This part of the exploration program focused on infill drilling for the 50 m grid, with shallow holes going down to a maximum depth of 200 m.

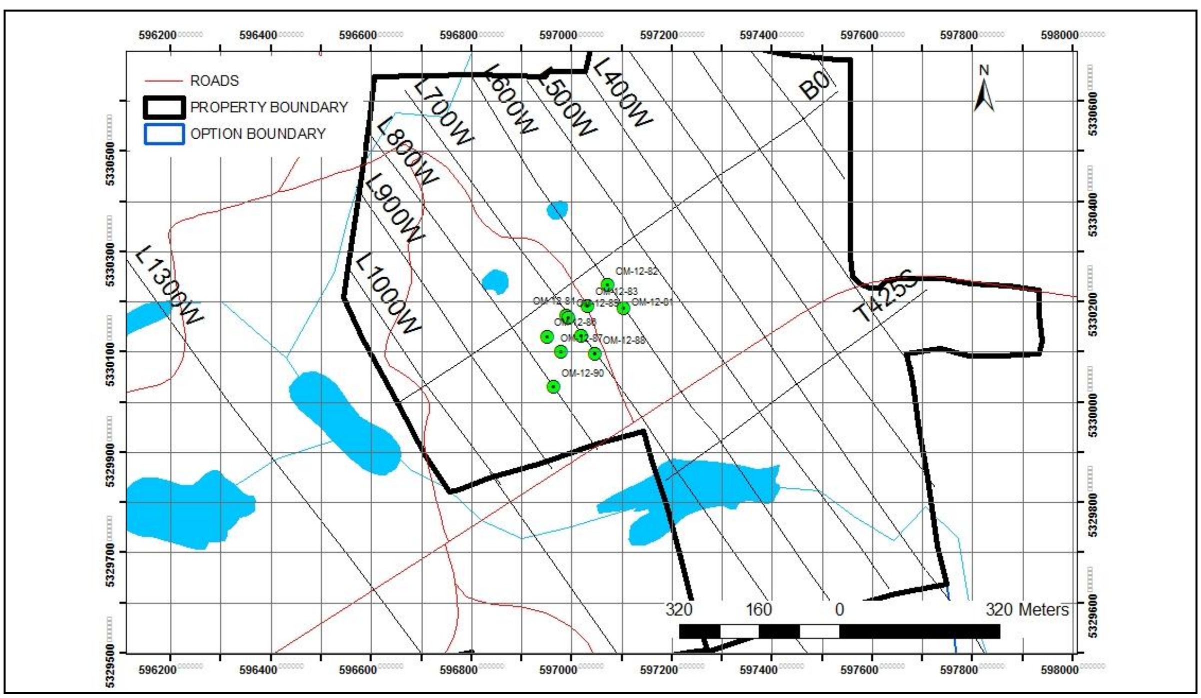
All drilling was done by Huard Drilling of New Liskeard, Ontario, using BTW core size. One hole was terminated short as it intersected a stope. Significant intercepts from these holes are shown in Table 10.5. Figure 10.3 shows the grid locations of all the collars from Phase 3 drilling.

Table 10.5 Significant Intercepts from Phase 3 Shallow Holes

BHID	Section line	Zone	Au (g/t)	Composite Length (m)
OM-12-81	600W	1/2	3.81	12
OM-12-82	600W	1/2	1.07	9.2
OM-12-82	600W	1/2	1.88	4.5
OM-12-83	650W	1/2	3.4	3.0
OM-12-84	700W	1/2	1.06	28.0
OM-12-85	700W	1/2	2.66	4.0
OM-12-86	750W	1/2	4.13	3.0
OM-12-86	750W	1/2	3.01	3.0
OM-12-87	750W	1/2	1.10	11.0
OM-12-88	750W	1/2	2.25[1]	4.8
OM-12-88	750W	1/2	1.04[1]	4.0
OM-12-88	750W	1/2	1.14[1]	10.0
OM-12-90	800W	1/2	2.01	4.0

Note [1] With an overall interval for hole OM-12-88 of 44 m at 0.81 g/t Au

Figure 10.3 Collar Locations for Phase 3 Drilling



Source: Mistango River Resources Inc.

10.4 Phase 4 – August 2012 – September 2012

During Phase 4 of the program, 8 diamond drillholes were completed, all of which were shallow. The program had a total length of 2,182 m, with 847 samples assayed. This part of the exploration program focused on infill drilling for the 50 m grid, with shallow holes going down to a maximum depth of 200 m.

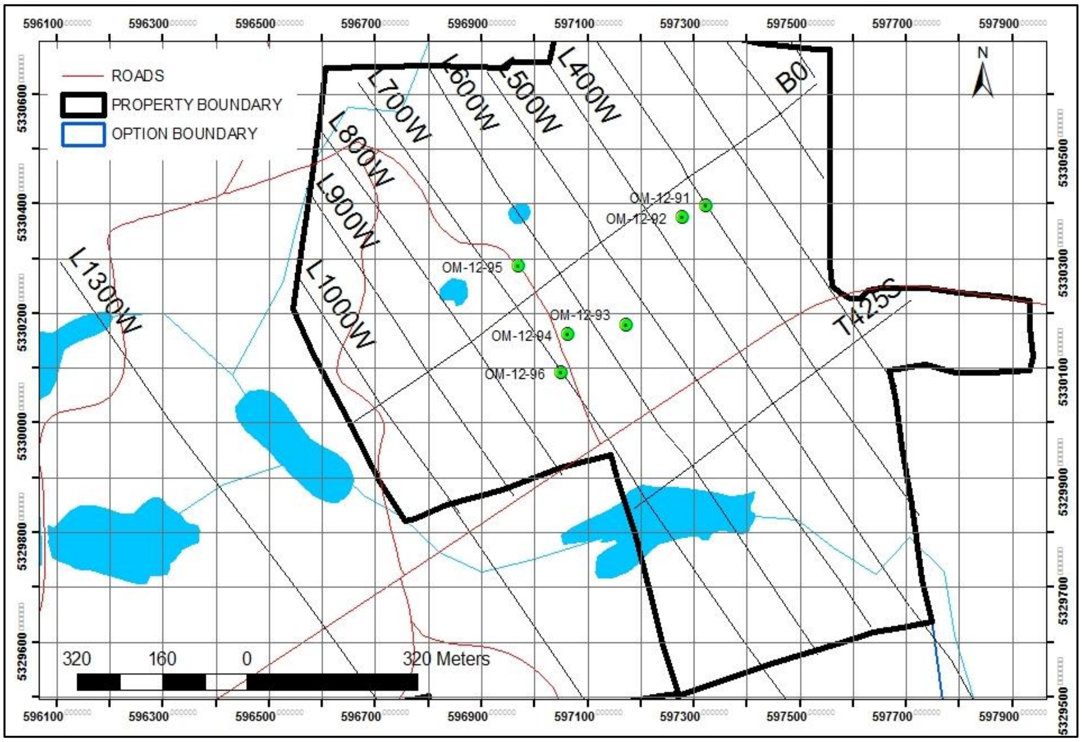
All drilling was done by Huard Drilling of New Liskeard, Ontario, using BTW core size. Significant intercepts from these holes are shown in Table 10.6. Figure 10.4 shows the grid location of all the collars from Phase 4 drilling.

For a summary of the drill data from all four Phases see Appendix B.

Table 10.6 Significant Intercepts from Phase 4 Shallow Holes

BHID	Section line	Zone	Au (g/t)	Composite Length (m)
OM-12-91	300W	1/2	1.53	1
OM-12-92	350W	1/2	2.42	19.5
OM-12-93	550W	1/2	1.49	55
OM-12-93	650W	1/2	1.15	16.5
OM-12-94	650W	1/2	4.95	8.0
OM-12-95	650W	1/2	2.63	9.0
OM-12-96	700W	-	1.13	13.5
OM-12-98	950W	1/2	1.48	3.0

Figure 10.4 Collar Locations for Phase 4 Drilling



Source: Mistango River Resources Inc.

10.5 Lithological Codes

AMC noted there are a large number of different lithological codes that have been used over the years. Mistango has developed a simplified rock code that has been put in place for current and future logging. It is recommended that all the past drill data be reconciled with the new codes.

10.6 Collar Elevations

Plotting of the drill hole collar elevations in three dimensions (3D) has shown there is a minor discrepancy between the topographic surface being used for the model and the elevations recorded by the survey company. This is attributable to the standard error encountered with topographic maps. The collars from the 1983 drilling have had their collar elevations taken from the topographic map and therefore they do not exactly coincide with the more recent drilling. AMC recommends that a better topographic survey of the area is obtained before the next level of study is undertaken and that the drill collars are all referenced correctly to this. This will also necessitate some minor remodelling once this has been undertaken.

10.7 Twinned Holes

Five holes from the 2011 drilling campaign were twinned with the 1983 drilling in order to validate the historic holes. This would enable the inclusion of the 1983 drilling results, within the dataset, to be used for the resource estimation. Based on the comparisons of the grades, the grade distribution and the average between them and AMC has made the following conclusions:

- Minimum length for the 1983 samples is 10 cm and for the 2011 samples is 20 cm
- Maximum length for the 1983 samples is 7.2 m and for 2011 is 3 m
- Average length of samples is 1.16 m for 1983 and for 2011 is 1.0 m
- Average grade of samples is 1.11 g/t m for 1983 and for 2011 is 0.91 g/t

Figure 10.5 shows a grade comparison of holes OM-11-25 and OM-83-37 and Figure 10.6 is a probability plot for all available data for the twinned holes.

Figure 10.5 Grade Comparison Between OM-11-25 and OM-83-37

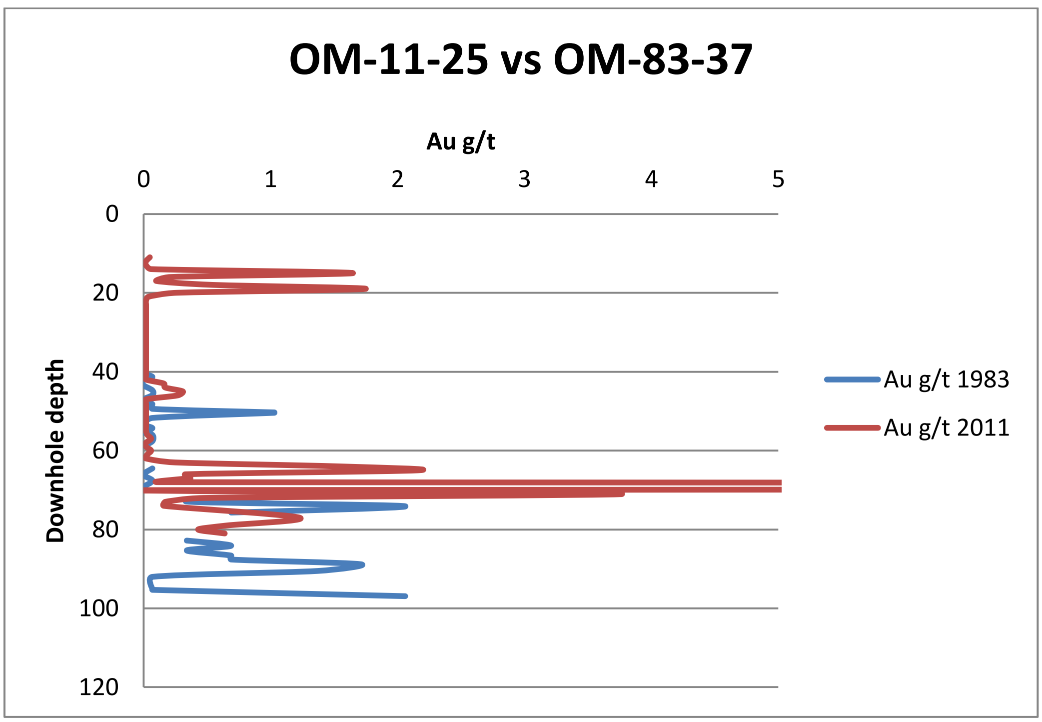
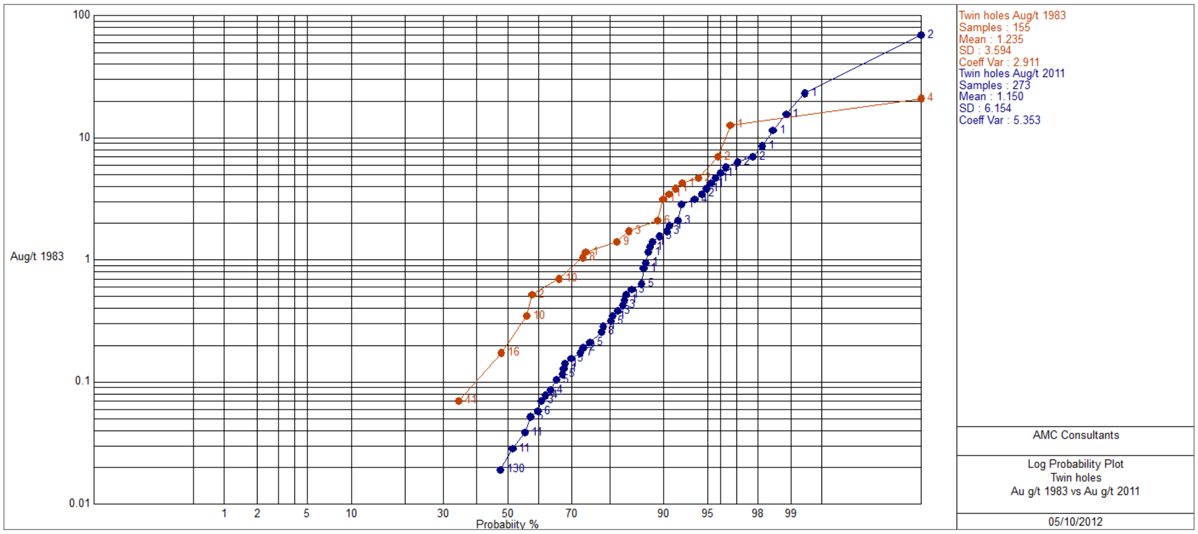
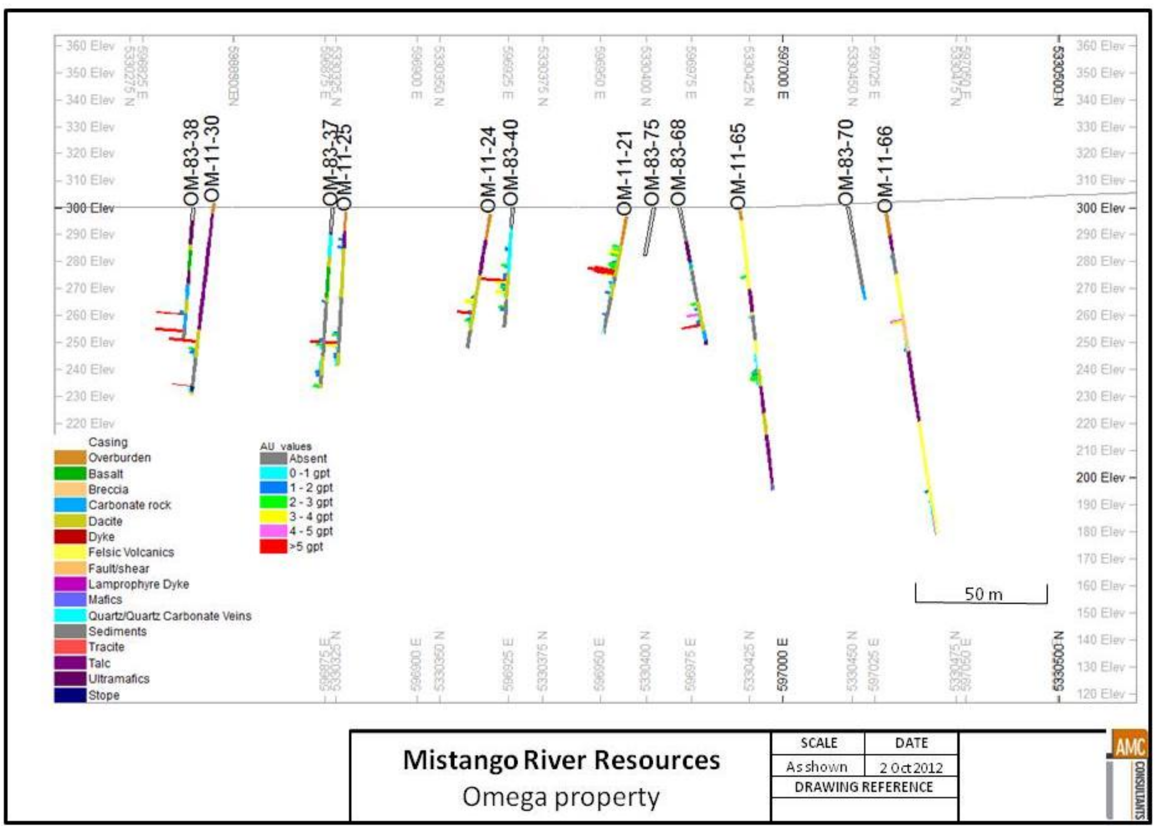


Figure 10.6 Log Probability Plot for All Twinned Hole Samples



Further comparison has shown that the mineralized horizons lie on comparable elevations, with minor discrepancies being attributable to the difference in elevation between the topography used to locate the 1983 collars and the elevations currently being recorded by the survey company. Figure 10.7 shows all the twinned holes.

Figure 10.7 Vertical Section Through Twinned Holes

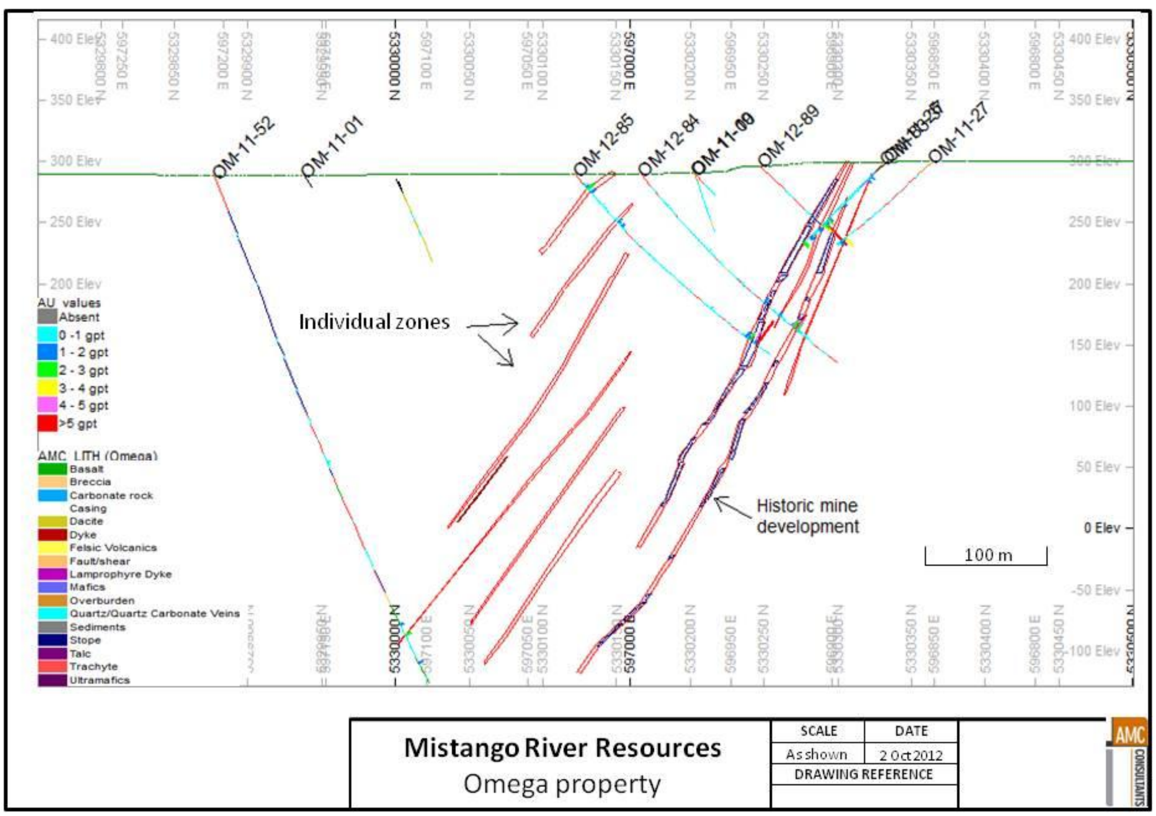


Taking into consideration the natural variability of gold grades and the slight discrepancy between the elevations, AMC considers that the 1983 assays can be included within the dataset used for the resource calculations as any differences are unlikely to have a significant effect on the estimate.

10.8 True Width of Mineralization

The orientation of the drilling is in two primary directions, approximately perpendicular to the strike of the mineralized zone. Twenty holes have been drilled from the footwall side and have an azimuth of around 145^0, with the remainder having an approximate azimuth of 325^0. Using the TrueDip process in CAE Datamine® and averaging the results, it was found that there was a 68% reduction from the apparent width of the mineralization zones (length of sample) to the true width, for the holes drilled from the footwall side. For the holes drilled from the hanging wall the difference between the apparent dip and the true dip is reduced to approximately 5%. Figure 10.8 shows a cross-section through the deposit along the 700W section, illustrating the relationship between the direction of the drilling and the orientation of the mineralization.

Figure 10.8 Section Through the Deposit at 700W Looking South-west



10.9 Specific Gravity Measurements

Specific gravity measurements have been completed on 115 samples. These, have been classified according to whether they are footwall (FW), hanging wall (HW) or mineralized zone (MZ). Table 10.7 shows the average densities for these samples.

Table 10.7 Specific Gravity Averages

Zone	Number of Measurements	Average (t/m^3)
FW	26	2.84
HW	27	2.79
MZ	61	2.95

AMC notes that although there were 115 samples tested, there were 37 different rock type codes. These need to be re-coded using the simplified lithology scheme and then a more representative number for each main lithology type and mineralization zone needs to be obtained.

Although these measurements are for specific gravity and not bulk density, in view of the early stage of this project these numbers have been used to generate the average rock density. AMC suggests that, as there are a number of volcanic and sedimentary units within the project area, the actual bulk density value for each of the units is collected and used as the Project moves forward.

For the purpose of this Mineral Resource estimate an average for all of the results of 2.89 t/m^3 was used.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

AMC reviewed the core handing, QA/QC procedures and data collection during a visit to site between 7 and 9 August 2012. AMC is satisfied that data has been collected in an appropriate industry standard manner. The core handling facility is a large shed with lockable doors. The facility was clean and well organized and Mistango have tried to make sure that the chain of custody from collection of the core trays from the drill rig to the processing and subsequent delivery of the samples to the laboratories remains with one person.

11.1 Sample Preparation and Security

Core is delivered to the core facility, where it is then orientated correctly in the trays, prior to being marked up for sample interval selection and total core recovery (TCR) and rock quality designation (RQD) measurement. Photographs are then taken of the wet core. The selected samples are then removed and cut with a diamond saw, with the half core from the same side of the hole being used for assaying each time in order to reduce bias. One half of the sample is bagged for delivery to the laboratory and analysis and the remaining half is retained in the core tray.

The nominal length of the sample is 1 m. If a run of mineralized core is marginally less than 3 m, the core is divided into three and each length is designated a 1 m sample. Only mineralized intervals are generally sampled, although Mistango has sampled all of the different lithologies within OM-11-01.

AMC recommends that all samples are assigned their exact length with no rounding, to minimize biasing the results.

After cutting, the core sample is sealed with a plastic cable tie in labelled plastic bags with its corresponding sample tag. The plastic sample bags are placed in large rice sacks and secured with tape and a plastic cable tie for shipping to the laboratory. The drillhole and sample numbers are also labelled on the outside of each rice sack and checked against the contents, prior to sealing the sacks. Standards and blanks are inserted into the sample sequence, prior to shipping, at a rate of one standard and one blank per 20 samples.

The remaining core is left in the boxes, the lids are sealed back on with tape and the boxes are labelled with the hole ID and drilling interval. Archive boxes of core, along with the pulps and rejects, from the 2011 and 2012 drilling programs are permanently stored at a facility within the Omega property boundaries. The core is stored on racks for ease of access and the pulps and rejects are filed and stored in a sea container and a trailer, both of which are kept locked.

A total of 6,273 samples were taken and assayed for Phase 1, 4,454 samples were taken for Phase 2, 1,375 samples were taken for Phase 3 and 847 samples for Phase 4.

Once there are a full batch of samples minimum of 5 bags, they are delivered to the laboratory by the person in charge of sampling, using a company vehicle. Samples are either delivered to Expert Laboratory (Expert) in Rouyn-Noranda, Quebec or Swastika Laboratory (Swastika) in Swastika, Ontario.

11.2 Assay Methods

To date, nearly 13,000 drill-core samples from the Omega project have been analysed. All of the initial assays were 30 g fire assays. Gold determinations of the fire assay buttons were by Atomic Absorption (FAAA), with a request to re-assay samples grading over 1,000 ppm, using fire assay (FA) with gold determined by Gravimetric Analyses, (gravimetric). For the resource estimate any gravimetric finishes were used over the original FAAA assays being deemed more accurate.

11.3 Laboratories

Mistango currently uses either Expert or Swastika depending on the work load of each. Neither laboratory is certified. In order to reconcile the returns from each of the laboratories Mistango has required a number of umpire samples be assayed by both laboratories. Refer to Figures 11.1 to 11.4.

Expert states their FA Au detection limit as 0.03 g/t and Swastika states theirs as 0.01 g/t.

Figure 11.1 to Figure 11.4 are four charts showing a comparison of the assay results from each laboratory. First review of the complete data set indicated that there was a problem with some of the umpire sample assays. Twelve assays greater than 0.5 g/t, assayed by Expert, were assayed to be below detection limit by Swastika. Mistango is currently assessing this.

The plots in Figure 11.1 to Figure 11.4 have removed all pairs of grades that have a mean grade less than 0.15 g/t. This is to allow for the increase in variability as the sample grade is closer to the detection limit.

Figure 11.1 Scatter Plot – Expert Assays with Swastika as Umpire Laboratory

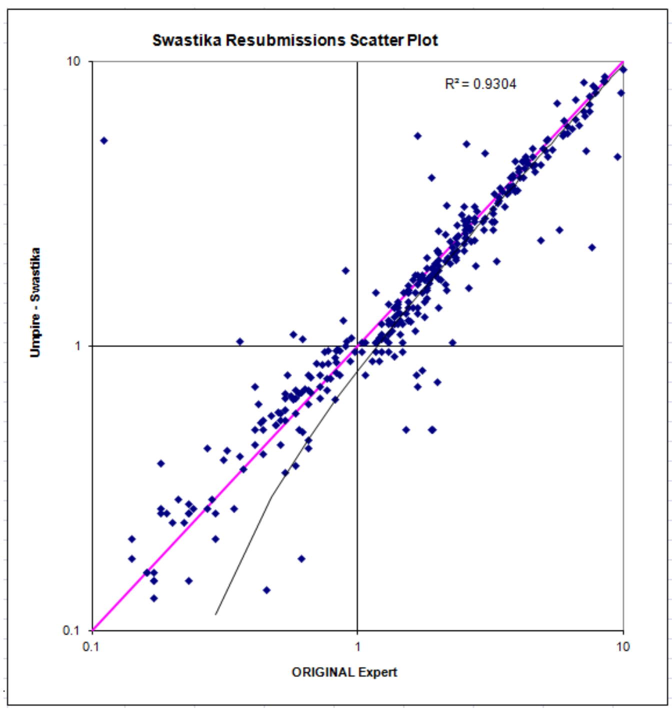


Figure 11.2 RPD Plot - Expert Assays with Swastika as Umpire

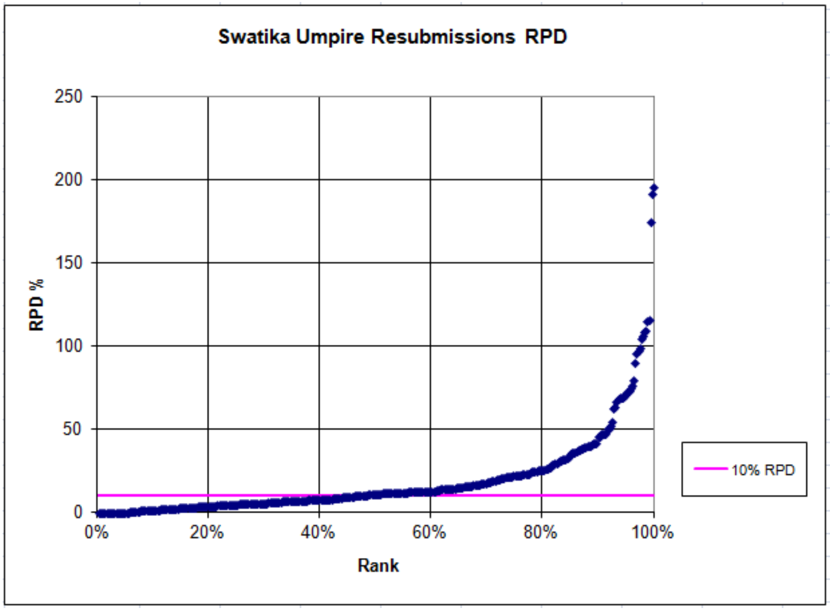


Figure 11.1 and Figure 11.2 indicate that there is a low correlation between the two sets of assays when Swastika was used as the Umpire laboratory. Only 41% of the samples have a Relative Percent Difference (RPD) of less than 10%. There is a slight negative bias for all the data. It is expected that around 80% of the data would have an RPD less than 10% for gold analyses.

Figure 11.3 Scatter Plot - Swastika Assays with Expert as Umpire Laboratory

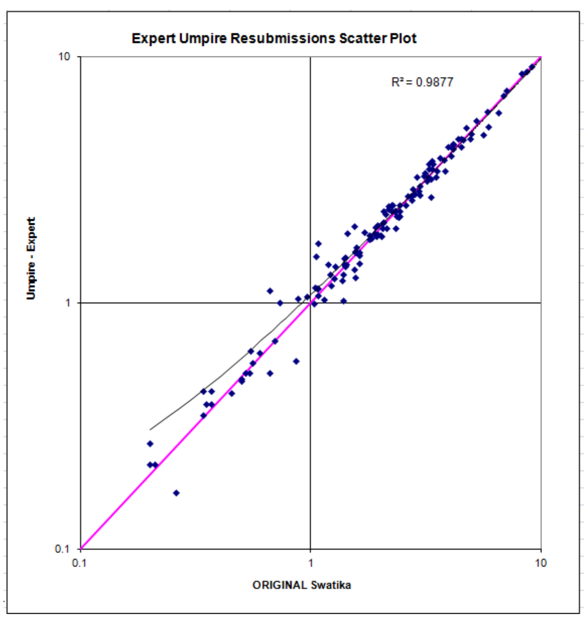


Figure 11.4 RPD Plot - Swastika Assays with Expert as Umpire Laboratory

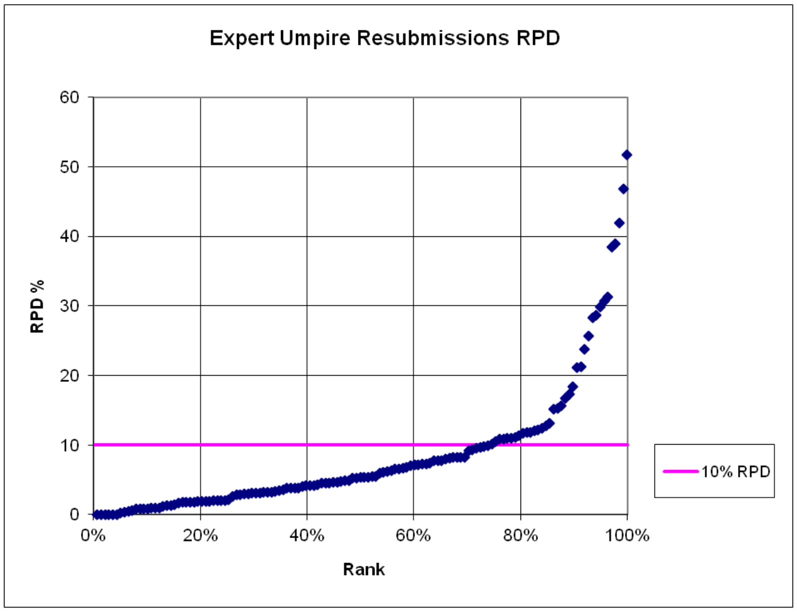


Figure 11.3 and Figure 11.4 indicate that there is a much better correlation between the two sets of assays when Expert was used as the Umpire laboratory. A total of 72% of the samples had a Relative Percent Difference RPD of less than 10%. There is no bias for the grades higher than the mean and only a minimal positive bias for assays below the mean.

11.4 Recommendations

AMC considers that Mistango have developed and used a reliable sampling procedure and that samples are kept secure throughout the process.

Although neither of the laboratories used by Mistango are ISO certified, both are covered by the *Proficiency Testing Program for Mineral Analysis Laboratories* (PTP-MAL) and have been assessed satisfactory for gold. The following section analyzes the QA/QC data from the blanks, standards and duplicates submitted by Mistango.

12 DATA VERIFICATION

A quality control (QC) program of standards, blanks and duplicate samples has been used since 2011 for all of the drill samples analysed. The blank material is a barren marble sourced locally.

12.1 Standards

The standard samples are provided by Oreas-Ore Research & Exploration Pty Ltd. (OREAS). The standards contain low, high and moderate grades of gold mineralization. Over the three phases of drilling, nine different certified standards (SRM) were used. The standards, which are pulps, are inserted on-site, prior to shipping of the samples to the laboratories.

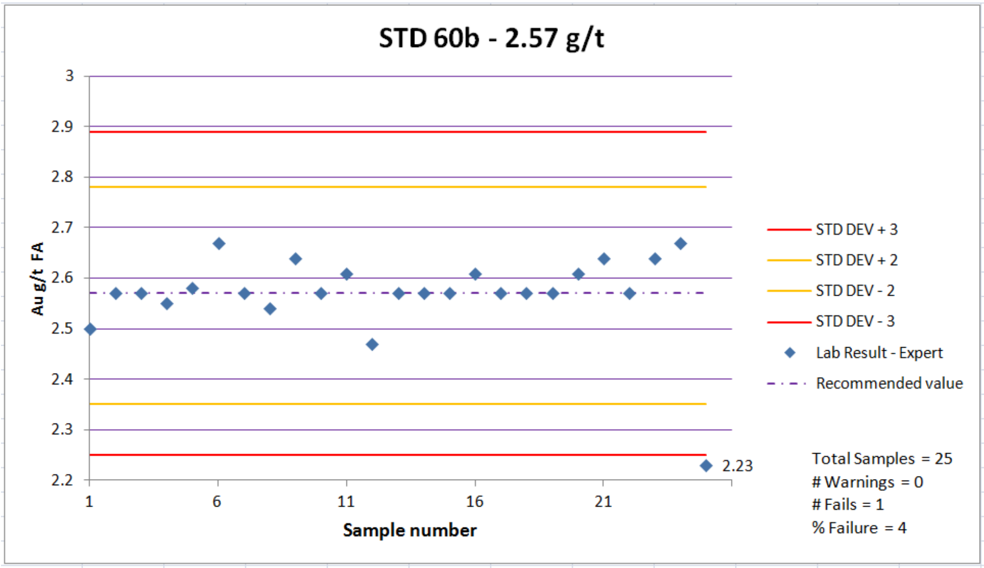
Table 12.1 lists each standard used during 2011 and 2012. AMC considers that a sample fails if it is greater than three standard deviations from the recommended value and there is a warning if it is greater than two standard deviations from the recommended value.

Table 12.1 Standard Reference Materials Used

SRM	Au (g/t)	STDev (g/t)	Sample Type	Year
OREAS_60b	2.57	0.11	Au/Ag	2011/2012
OREAS_52Pb	0.307	0.008	Au/Cu	2011/2012
OREAS_61d	4.76	0.14	Au/Ag	2011/2012
OREAS_16a	1.81	0.06	Au	2011/2012
OREAS_16b	2.21	0.07	Au	2011/2012
OREAS_15f	0.334	0.016	Au	2011/2012
OREAS_19a	5.49	0.1	Au	2011/2012
OREAS_15h	1.019	0.025	Au	2011/2012
OREAS_10c	6.6	0.16	Au	2012

The QA/QC program has identified occasional instances where there has been switching of the samples. Figure 12.1 to Figure 12.12 show the assay results for the SRMs used during the 2011 and 2012 drilling.

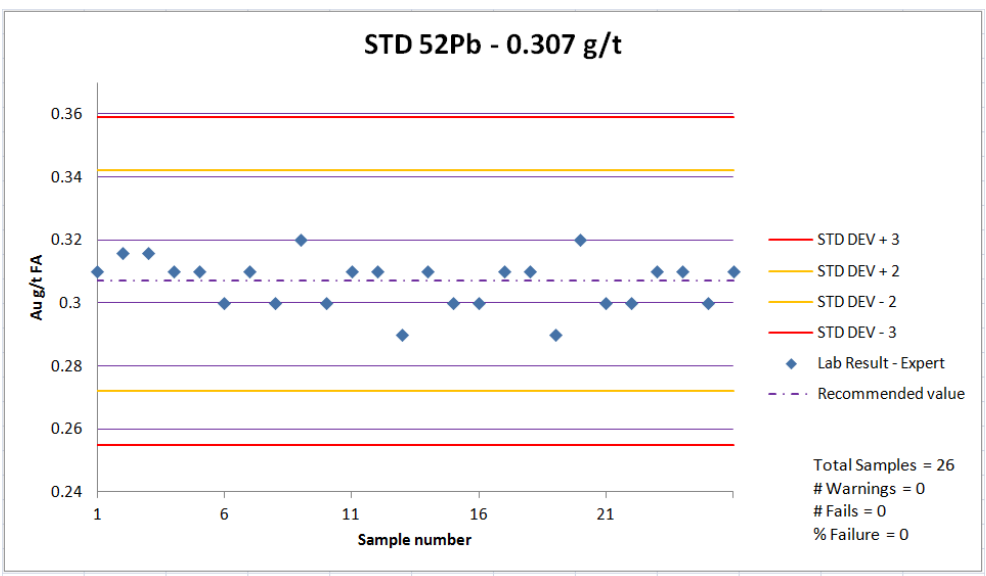
Figure 12.1 Standard OREAS 60b



For the OREAS 60b there was only one sample out of the 25 which failed and there were no warnings. The failed sample is within the grade range of Standard 16b and is considered to have been mislabelled.

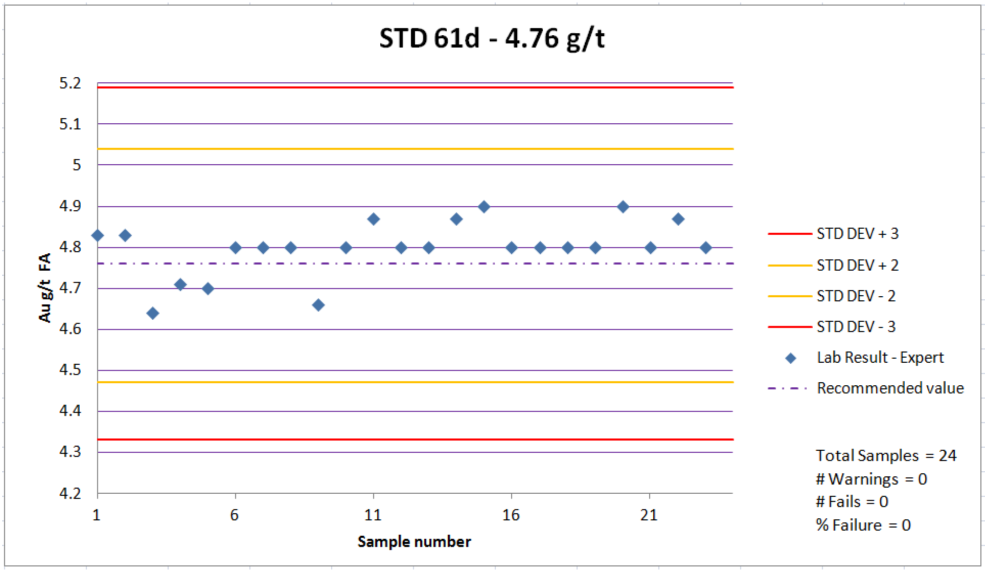
As the one mislabelled sample represents less than 5% of the population, AMC considers it does not have a material impact on the Mineral Resource estimate.

Figure 12.2 Standard OREAS 52Pb



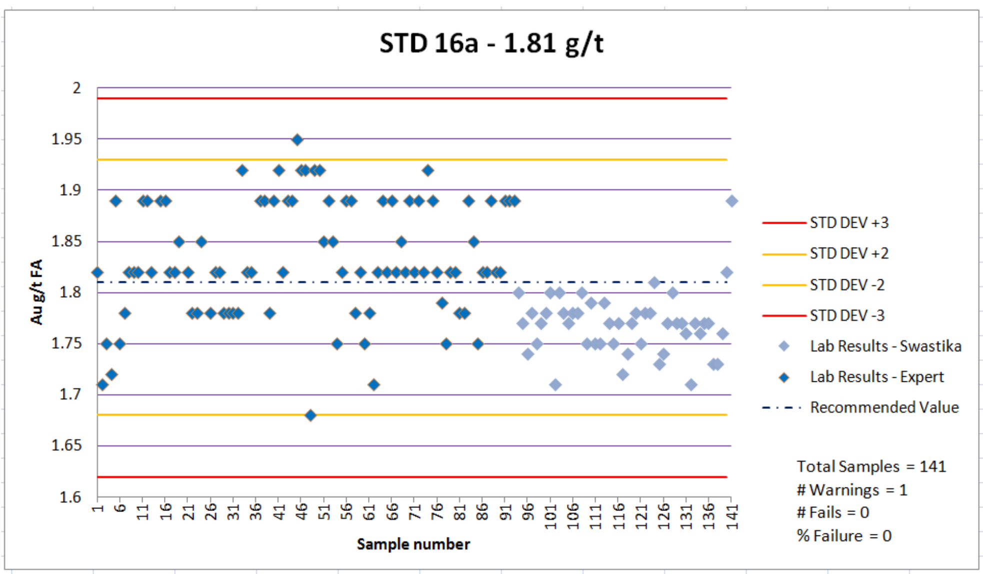
For the OREAS 52Pb there were no failures or warnings out of the 26 samples submitted.

Figure 12.3 Standard OREAS 61d



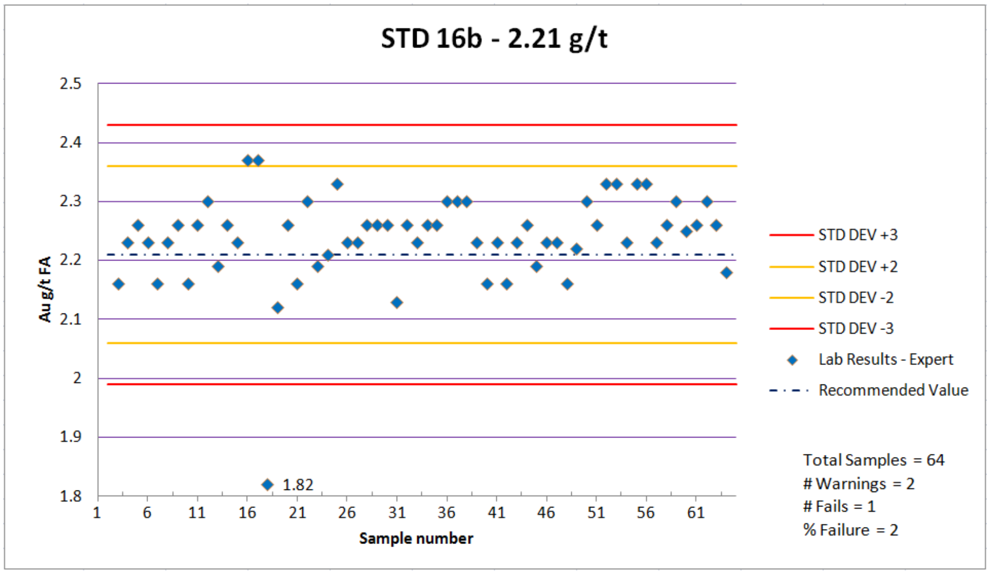
For the OREAS 61d there were no failures or warnings out of the 24 samples submitted.

Figure 12.4 Standard OREAS 16a



For the OREAS 16a there were no failures and just one warning out of the 141 samples submitted. However, it must be noted that the results from Swastika show a strong negative bias, while Expert show a slight positive bias.

Figure 12.5 Standard OREAS 16b



For the OREAS 16b there was one failure and two warnings out of the 64 samples submitted. The failed sample appears to be a mislabelled OREAS 16a standard. No re-sampling of the batch was undertaken. As the one sample represents less than 5% of the population this is not considered to have a material impact on the Mineral Resource estimate.

Figure 12.6 Standard OREAS 15f – All Data

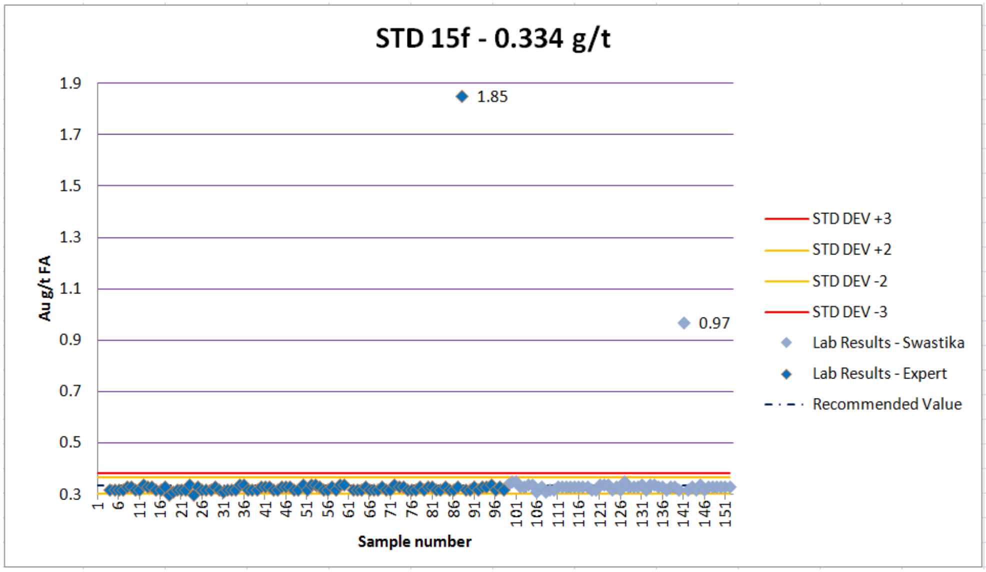
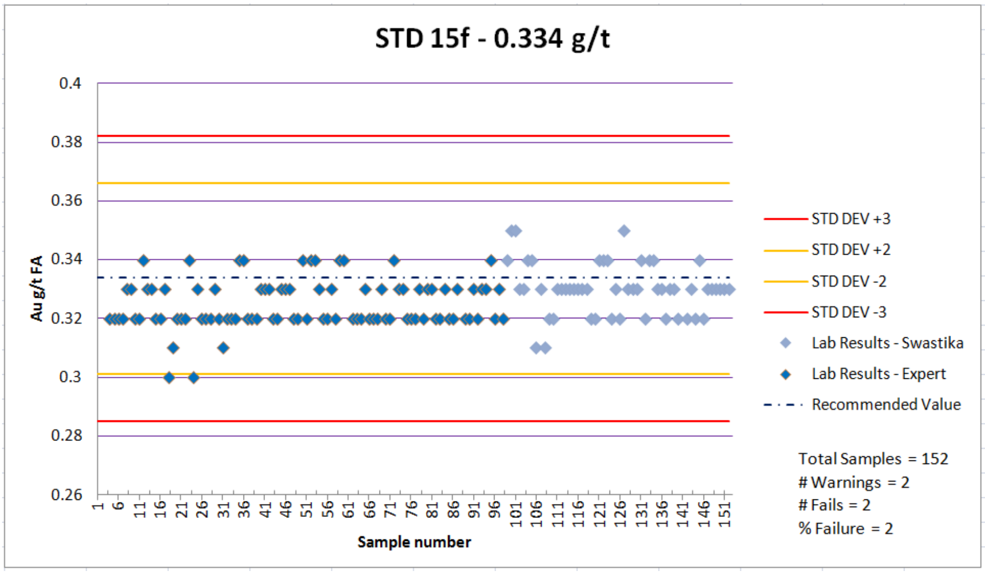


Figure 12.7 Standard OREAS 15f - Detail



For the OREAS 15f there were two failures and two warnings out of the 152 samples submitted. One of the failed samples is within the grade range of Standard 16b. It must also be noted that the values plot in lines which indicates that the assaying method used is not sensitive enough to get the results into the third decimal place range.

Figure 12.8 Standard OREAS 19a – All Data

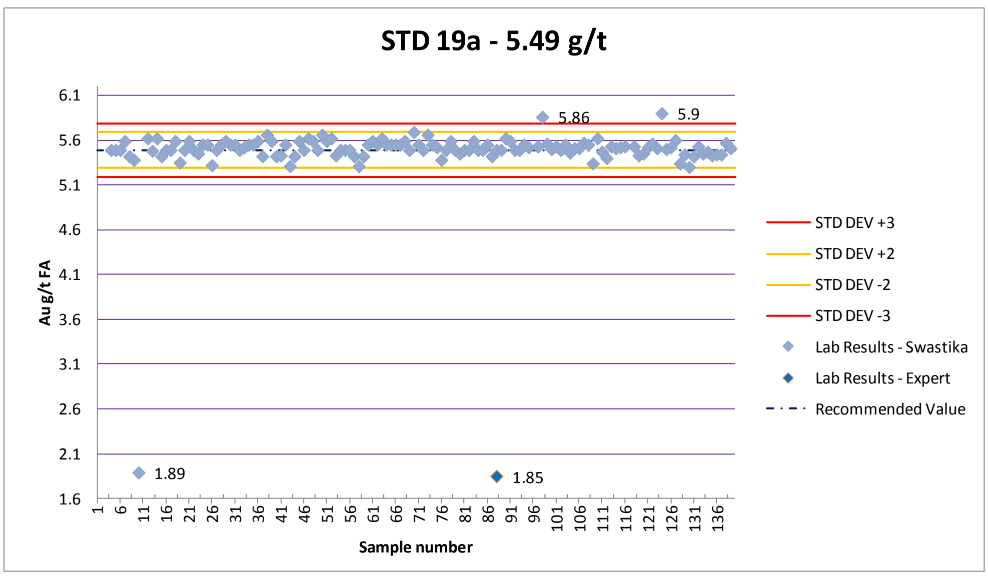
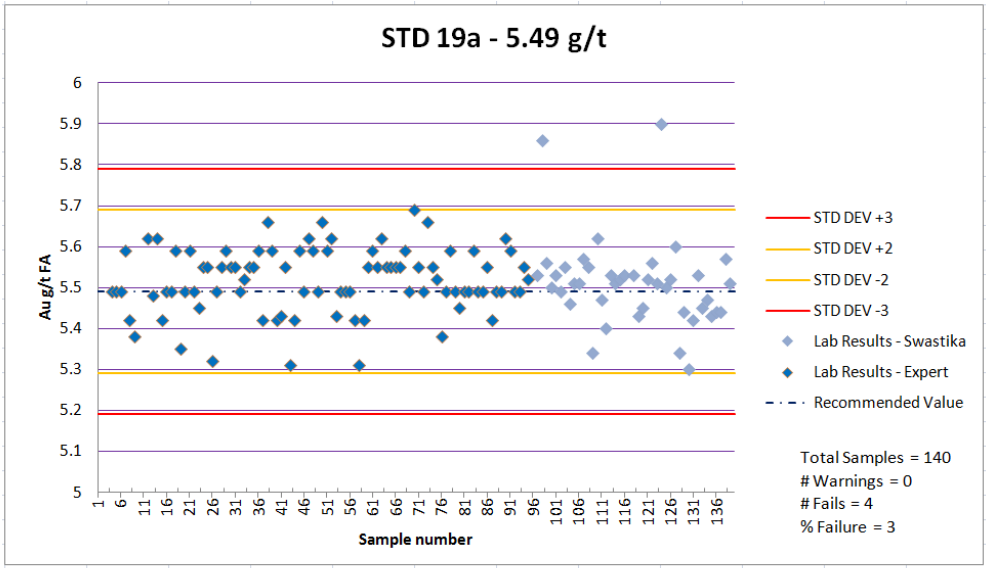


Figure 12.9 Standard OREAS 19a – Detail



For the OREAS 19a there were four failures and no warnings out of the 140 samples submitted. Two of the failed samples are within the grade range of Standard 16b indicating mislabelling of standards.

Figure 12.10 Standard OREAS 15h – All Data

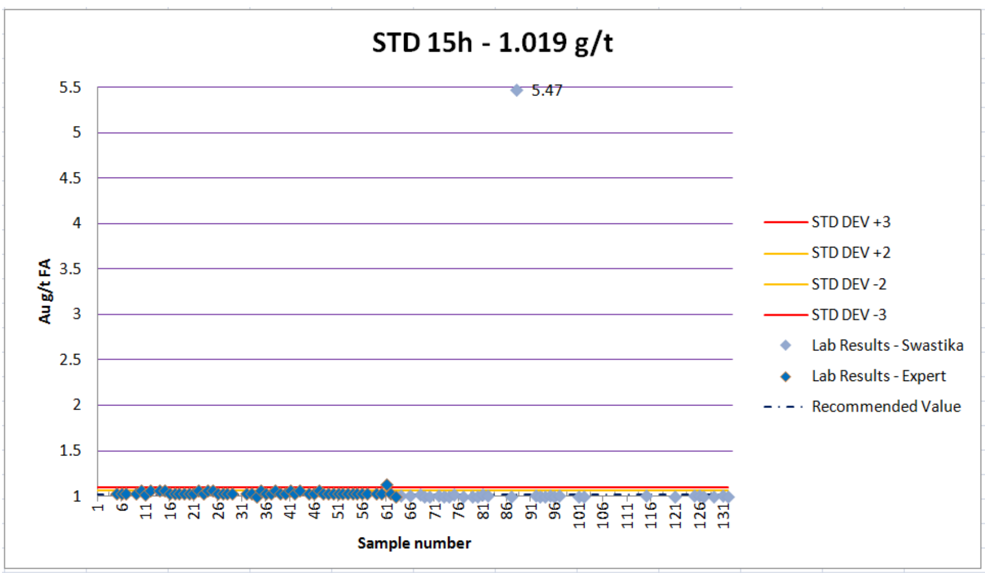
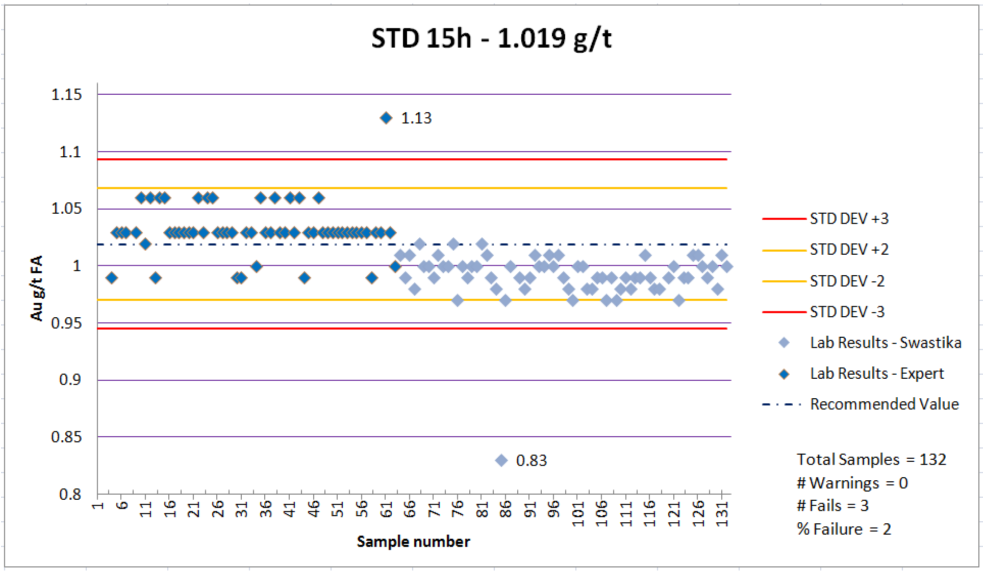
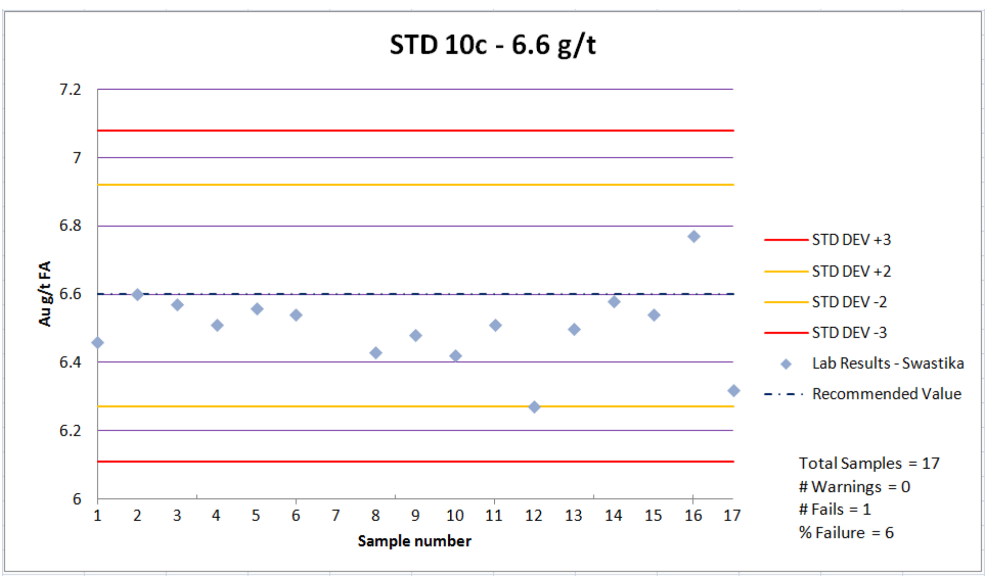


Figure 12.11 Standard OREAS 15h – Detail



For the OREAS 15h there were three failures and no warnings out of the 132 samples submitted. One of the failed samples is within the grade range of Standard 19a. It must also be noted that the results from Swastika are showing a strong negative bias and Expert a positive bias.

Figure 12.12 Standard OREAS 10c



For the OREAS 10c there was one failure and no warnings out of the 17 samples submitted. The failed sample came in as below detection limit which may be due to the mislabelling of a barren sample.

12.2 Blanks

Blank material used by Mistango is locally sourced barren marble. Blanks are submitted at a rate of 1 sample per batch of 20. As there are no obvious high grade zones within the mineralization they are submitted in a regular sequence. In total, Mistango has received returns for 676 samples with just two failures and one warning. It should however be noted that the detection limit for Expert is three times greater than for Swastika. Figures 12.13 and 12.14 show the blank assays for the 2011 and 2012 drilling.

Figure 12.13 Blanks – Expert

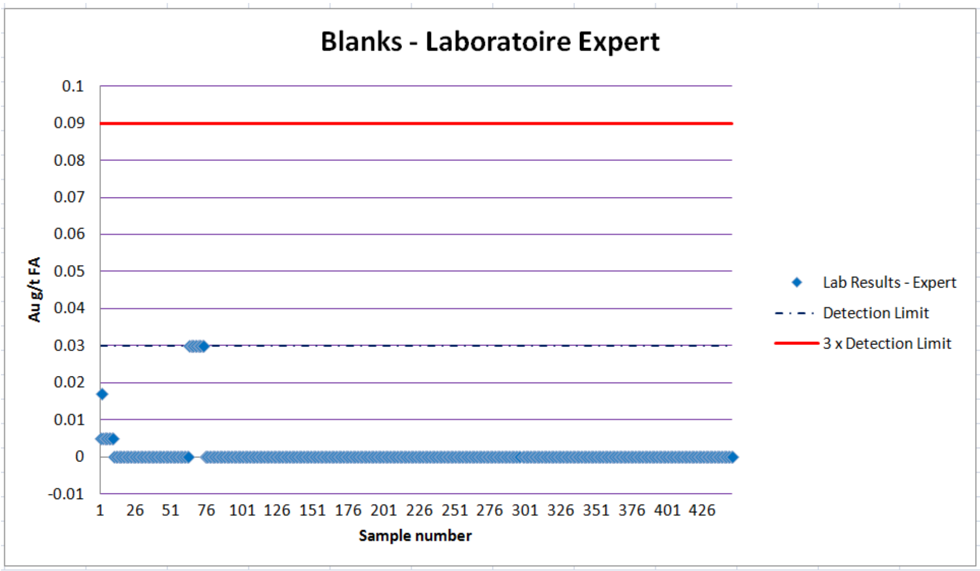
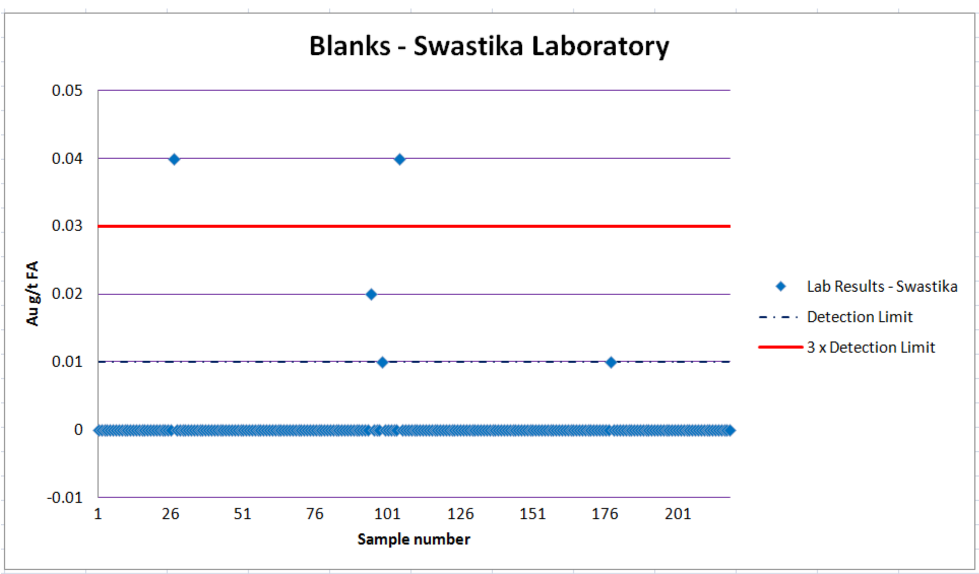


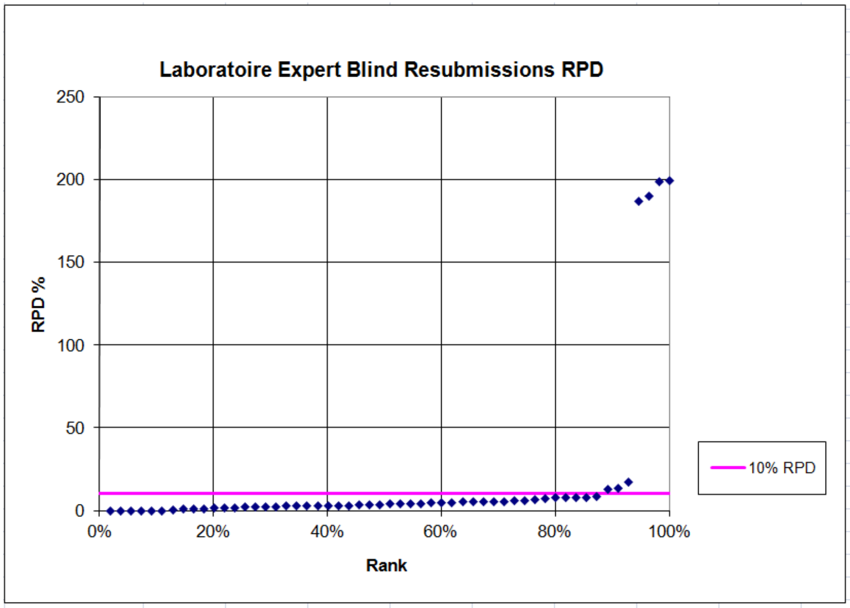
Figure 12.14 Blanks– Swastika



12.3 Duplicates

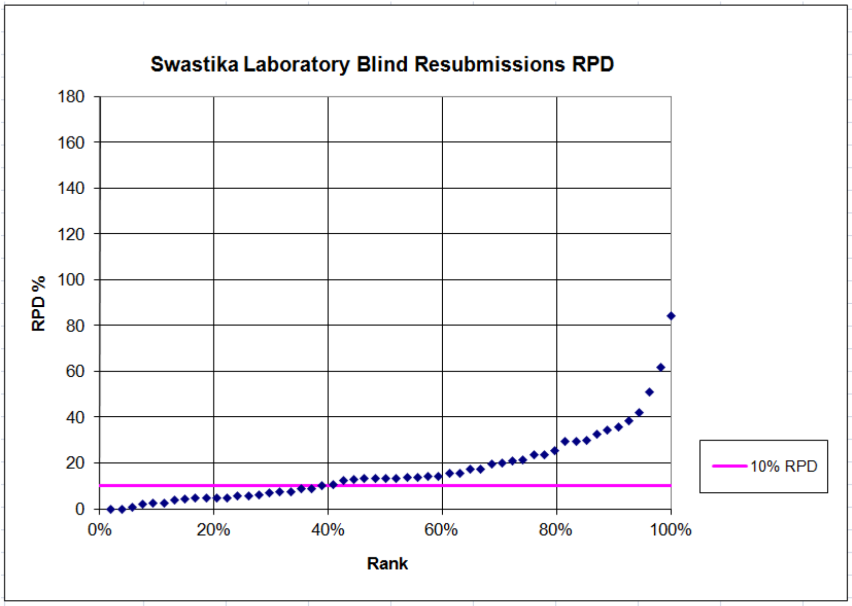
Mistango systematically resubmits some pulps as duplicates. To date they have paired assays for 444 samples from Expert and 186 from Swastika. Figures 12.15 and 12.16 show the RPD plots for each of the labs.

Figure 12.15 Duplicate Assays – Expert



There were 55 sample pairs with a mean greater than 0.15 g/t selected from an original population of 444 processed by Expert. The duplicates have 87% of the samples with less than 10% RPD, which is well within the industry standard of at least 80% of samples lying below the 10% difference value. However, Figure 12.15 shows that four samples have a difference of greater than 185%. There is no bias for the sample population.

Figure 12.16 Duplicate Assays – Swastika

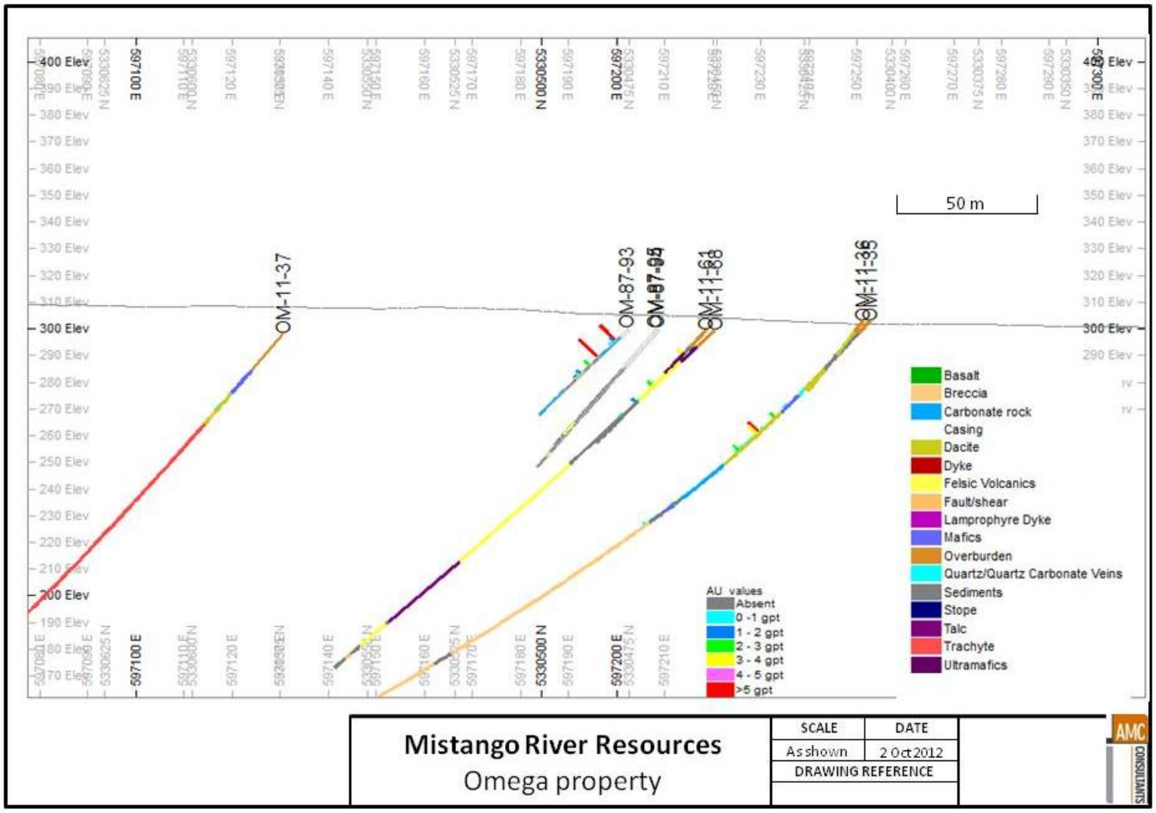


There were 54 sample pairs with a mean greater than 0.15 g/t selected from an original population of 186 processed by Swastika. The duplicates have only 37% of the samples with less than 10% RPD, which is well below acceptable industry standards. There is a negative bias of -9% for samples averaging lower than the mean gold grade of 0.9 g/t. (Figure 12.16).

12.4 1987 Drilling

Although no correlation of the five holes that were drilled in 1987 has taken place, a visual check of the position of mineralized zones by AMC indicates that three lie in the correct horizons and the grades returned are similar to expected values (Figure 12.17). Two holes, OM-87-96 and OM-87-97 were removed as their collar coordinates are incorrect.

Figure 12.17 Comparison of the 1987 Drilling to Nearest 2011 Drillholes



12.5 Conclusions

AMC makes the following conclusions:

- Although Mistango have put in place a QA/QC protocol which meets industry standards, the company still needs to better monitor the insertion of the blanks and standards.
- There appears to have been some mix up in the sampling labeling or packing process.
- Because of this mix up Mistango have recently instigated a one person chain of command for the sampling, so that any errors can quickly be identified and rectified.
- There is a strong negative bias exhibited by Swastika and positive bias by Expert for the standards between 1 and 2 g/t.
- Swastika have a low amount of repeatability for their blind duplicates, with a negative bias.
- The two laboratories used for the assaying appear to have performed poorly in the role of umpiring each other, but have a very low failure and warning rate for the standards. It is therefore considered that the assay results for the 1983, 2011 and 2012 drilling can be included in the resource estimate.
- For this level of study the procedures and the results are adequate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

During 2012 Mistango commissioned SGS Mineral Services (SGS) to complete a preliminary economic assessment (PEA) (scoping) level metallurgical test program. This program was to establish the basic processing parameters for the treatment of a composite sample shipped by Mistango to the Lakefield facility. The following is the Executive Summary from the report (DiLauro, 2012). For a more detailed account refer to the full report which is appended as Appendix C.at the end of this report. (Appendix C).*Executive Summary from the SGS Metallurgical Report*

"The metallurgical investigation undertaken on the Omega deposit ore composite sample for Mistango River Resources Inc. has provided some understanding of the sample nature and metallurgical behaviour.

A gold-bearing composite sample was examined at the SGS Mineral Services Lakefield site. The Omega sample composite contained 3.58 g/t gold based on direct head assaying by pulp and metallic protocol. The silver grade was determined to be < 2 g/t. The composite sample also yielded a sulphide sulphur grade of 3.54%.

Initial whole ore cyanidation testing of the composite sample leached showed recoveries after 48 hours of leaching ranging from 76% to 86% while cyanide consumptions were 0.53 kg/t to 1.38 kg/t of NaCN. Lime consumptions were low at 0.40 kg/t to 0.45 kg/t.

Gravity separation testing on the Omega composite at a P_{80} size of 125 microns showed a very low result of gold recovery of 3%.

Gravity tailing cyanidation testing of samples leached showed similar recoveries after 48 hours of leaching as observed in the whole ore leaches. Gold recoveries after 48 hours of leaching ranged from 74% to 84% while cyanide consumptions were 0.54 kg/t to 1.39 kg/t of NaCN. Lime consumptions were low at 0.41 kg/t to 0.46 kg/t. The combined gravity plus gravity tailing cyanidation gold recoveries for the composite ranged from 75% to 84% showing no real increase due to the very low gravity recovery of gold.

Gravity tailing flotation testing of samples showed excellent gold recoveries for all tests conducted. Gold recoveries for all three test performed were reported at 99%. While the Omega Composite head silver grade was reported at < 2 g/t there was a significant improvement in recovery observed in the finer grind tests. For the tests, silver recoveries were shown to be 48% for the test at a P_{80} size of 125 microns, 66% for the test at a P_{80} size of 85 microns and 70% for the test at a P_{80} size of 52 microns.

The diagnostic leach program showed an initial 84.2% gold recovery of readily leachable gold. 3.2% of the gold was further extracted from possible gold associations with iron-arsenic compounds or bismuth minerals and 2.6% of the gold was further extracted from possible gold associations with weak acid soluble compounds. 7.4% of the gold was observed to be from possible gold associations with or occluded by sulphide minerals, pyrite and arsenopyrite. The remaining 2.5% of the gold remaining in the final leach residue was deemed to be the gold mainly associated with silicates or fine sulphides locked in silicates. The results from the diagnostic leach program should be viewed as an indication of general trends and possibilities only."

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource has been estimated by Ms C Pitman, P.Geo.(Ontario) of AMC, who takes responsibility for the estimate, under the supervision of Mr R Webster, MAIG of AMC. All the modelling and the estimation were carried out in CAE Datamine software. Within the project area 13 zones were modelled and combined as a single block model suitable for the resource estimation. The Mineral Resources are stated as at 31 August 2012.

A summary of the results of the estimated Mineral Resource, at cut-offs of 0.5 g/t Au for mineralization above an elevation of 130 masl, representing open pit potential and for a cut-off of 3 g/t Au below 130 masl, representing underground potential are shown in Table 14.1. Both are classified as Inferred Mineral Resources.

Table 14.1 Summary of Mineral Resources as at 31 August 2011

Cut-off grade	Tonnes (Mt)	Au (g/t)	Contained Au ounces
0.5 g/t Au above 130 masl	3.8	2.50	306,100
3 g/t Au below 130 masl	1.2	4.33	166,000
Total Inferred	**5.0**	**2.93**	**472,100**

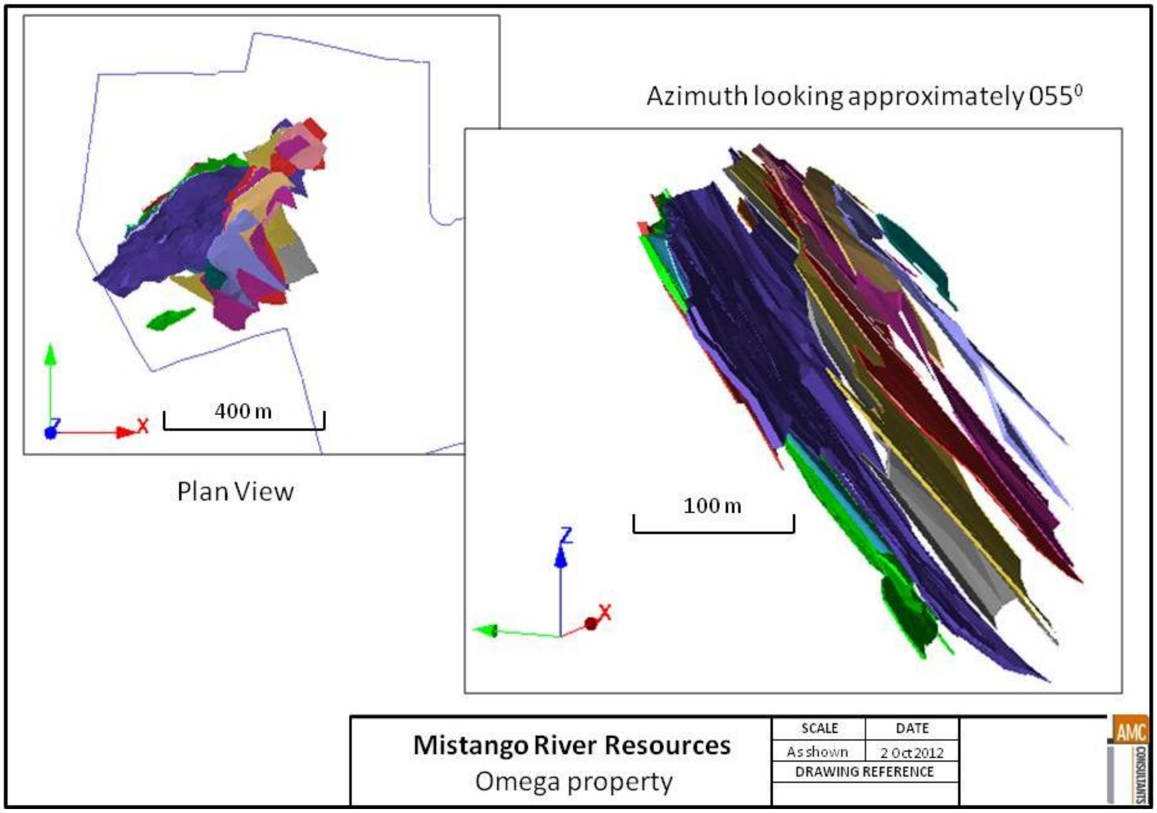
Note: A constant bulk density of 2.89 t/m^3 has been used.

14.2 Block Model Estimates

14.2.1 Introduction

AMC completed an independent Mineral Resource estimate based on zone wireframes prepared by Mistango and subsequently modified by AMC. The location of the 13 zones is shown in Figure 14.1 as plan and sectional 3D views.

Figure 14.1 Omega Zones



Note: None of the zones as modeled have been assigned numbers

14.2.2 Data Used

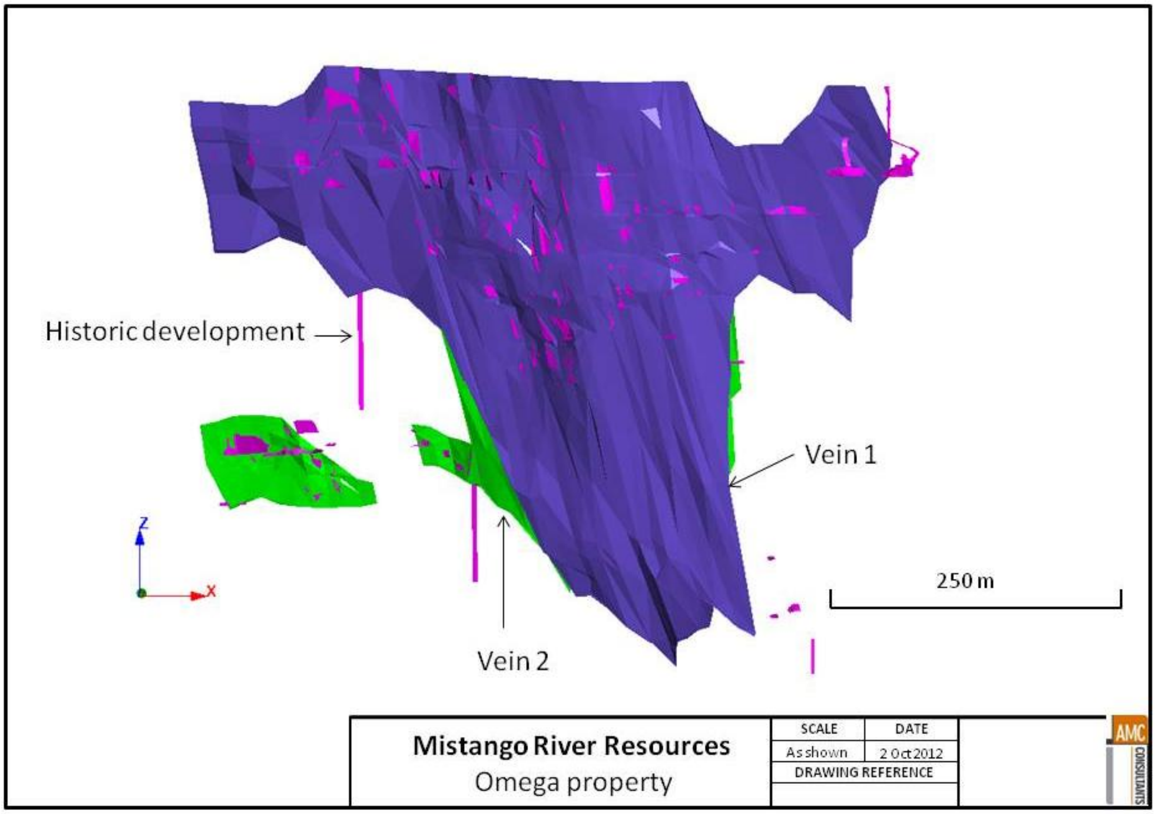
Mistango provided AMC with following data:

- Spreadsheet database containing the sample collar down hole survey, lithology and assay data.
- Initial wireframe outlines of the zones for all lenses.
- Wireframe outline of the mined areas reconstructed from limited historic mine plans.

The zone modelling was carried out initially by Mistango and then modified by AMC. These wireframes were modelled using the drill data to identify continuous zones over 1 g/t Au. These zones were used to delineate the individual zones. Also the wireframes of the mined out areas were used to identify the trends in the mineralization.

Figure 14.2 illustrates the main two zones, along with the mined out areas of the Property.

Figure 14.2 Relationship of Mined Out Areas to Modelled Zones



Note: Pink surfaces are the mined out areas

14.2.3 Grade Estimation Method

Grade estimation for the 13 zones was carried out using one block model, with the individual zones separated out for estimation and then recombined within the whole model. This involved:

- All wireframes in DXF format and drillhole files were imported into CAE Datamine.
- Individual zones were identified.
- Samples within each zone were composited to 1 m intervals.
- Statistical and variogram analysis of the grades was carried out.
- A block model with blocks 25 m wide in the east and north directions and 5 m thick in the vertical direction was prepared.
- Each individual zone was filled with blocks using sub-cells down to 5 m in the east and 1 m in the north and vertical directions.
- Block grades were estimated into each parent block within the zones, and area outside of the zones using ordinary kriging.
- The blocks located within the areas of previous mining were removed from the resource estimate.
- The individual models were combined into one final model.

14.2.4 Samples

A total of 14,427 composites were available with 975 composite samples selected from within the zone wireframes and used for the variogram analysis and estimation of the blocks within the zones.

14.2.5 Bulk Density

Bulk density measurements have not been systematically collected with samples only available for the mineralized zones. Based on the 115 samples collected and average density of 2.89 t/m^3 has been used for this estimate.

14.2.6 Statistics and Compositing

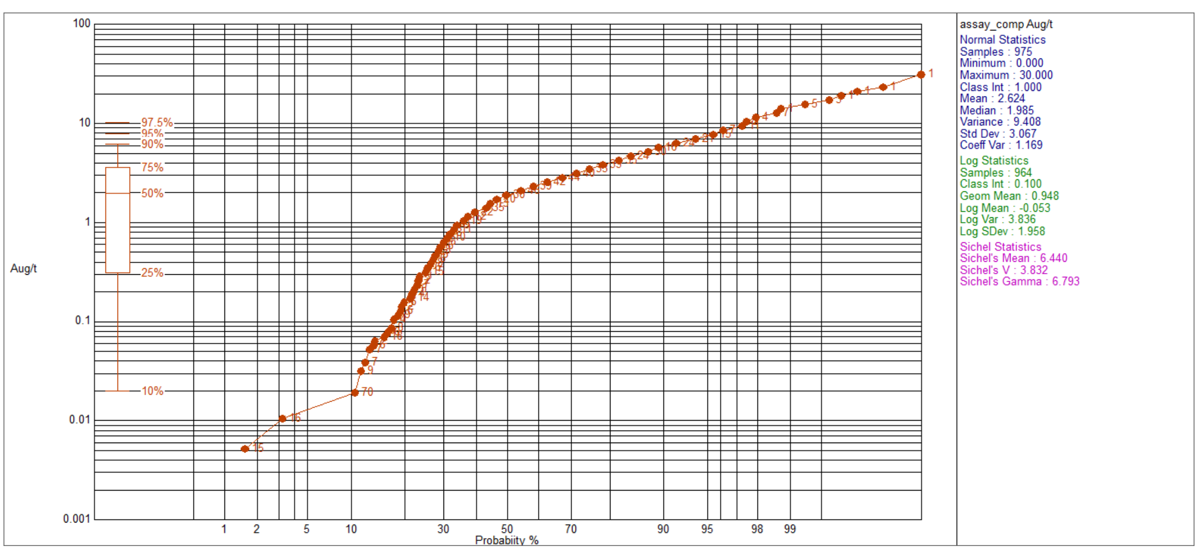
A statistical analysis and variography was carried out for the 13 zones combined due to there being insufficient data to evaluate each zone separately. Table 14.2 shows the statistics for both the raw sample data and the selected composites within the zones used for the variogram analysis.

Table 14.2 Sample Statistics

	No. Samples	Mean Au (g/t)	Median Au (g/t)	Min. Au (g/t)	Max Au (g/t)	Standard Deviation	Coefficient of Variation
Raw	14,124	0.37	0.02	0.0	30.0	1.39	1.93
Selected 1 m Comps	975	2.62	1.99	0.0	30.0	3.07	1.17

Figure 14.3 shows log probability plots of the composited gold grades within the zones. These plots indicate that the gold grade is of a single population and based on these plots no top capping was applied.

Figure 14.3 Log Probability Plot of Composite Gold Values



Variogram analysis was undertaken on the composited gold grades for all 13 zones as one data set. The variogram parameters resulting from the analysis based on a two structured variogram are shown in Table 14.3.

Table 14.3 Variogram Parameters

Variogram Parameters		Range 1 (m)				Range 2 (m)			
	Nugget	Sill 1	E-W	Vertical	N-S	Sill 2	E-W	Vertical	N-S
	0.3	0.09	43	10	48	0.61	52	10	60

14.2.7 Block Modelling

All gold grade estimation was performed into parent cells only. Blocks were discretised using 4 x 1 x 4 points (east, north and vertical).

The estimation search parameters used to estimate block gold grades for all zones and between zones were consistent as only one variogram analysis was used due to the small amount of samples within each of the individual zones. A three pass octant search was used with the three pass search parameters shown in Table 14.4. To provide block grades in areas not estimated, using these search parameters, a further estimate was carried out using a wider search ellipse and no octants.

Table 14.4 Search Parameters

Run	Search Ellipse			Rotation			Number of Samples		
	X Axis	Y Axis	Z Axis	Z Axis	Y Axis	X Axis	Min	Max	Max per hole
First	60	10	75	-54	20	40	3	6	2
Second	80	10	100	-54	20	40	3	6	2

14.2.8 Resource Classification of Block Models

The resource has been classified as Inferred for the following reasons:

- The gold is located in narrow zones.
- The location of areas of previous mining not completely known.

Continuity of the zones is not completely understood. Figure 14.4 and Figure 14.5 show the relationship between the modelled zones and the drilling to date. The zones have been infilled with blocks showing the estimated gold values.

Figure 14.4 Section Through 650W

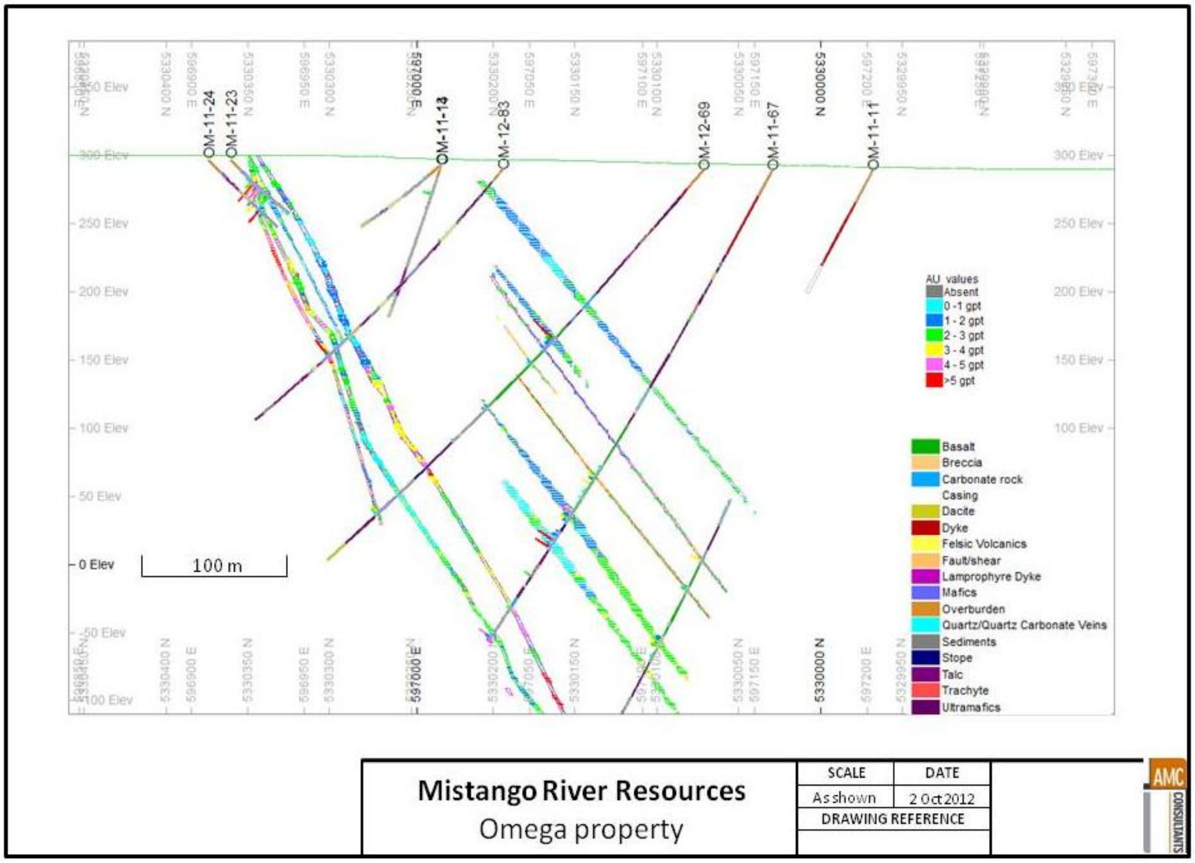
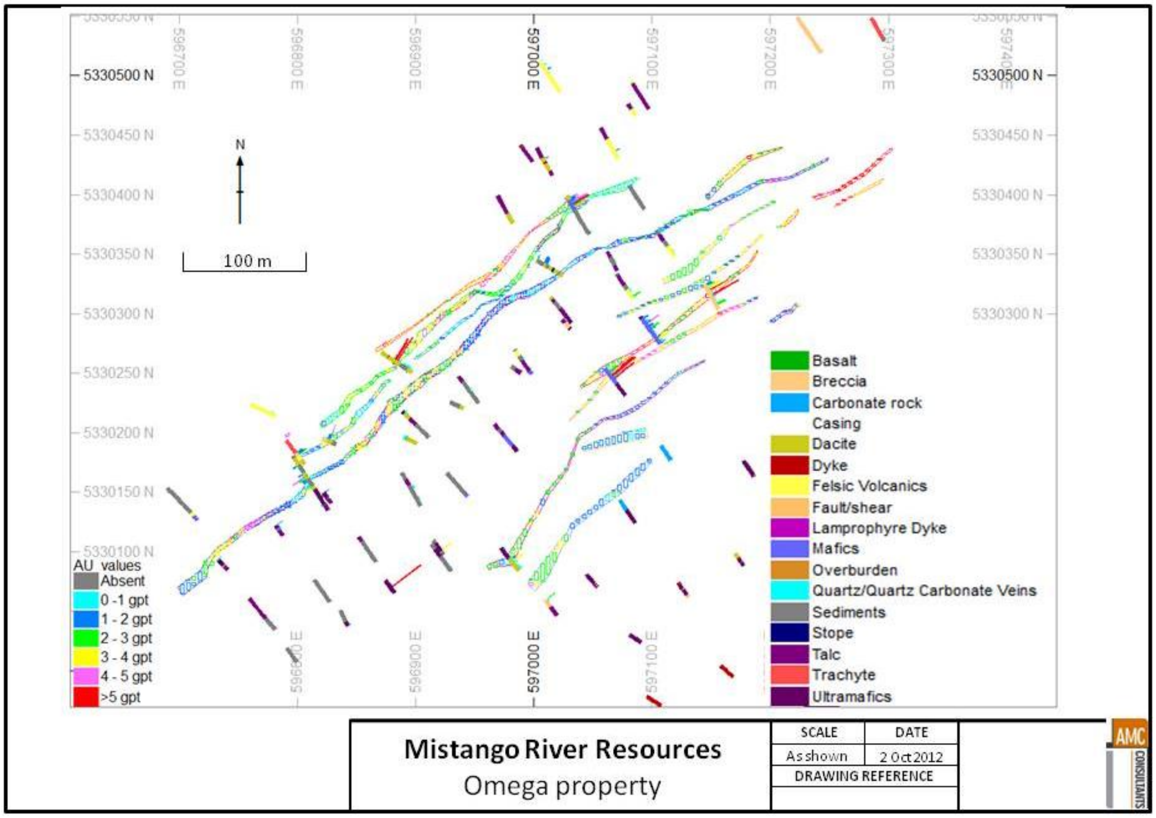


Figure 14.5 Plan View of Zones at Elevation 210 m



14.3 Mineral Resource Estimate

The Inferred Mineral Resource for zones has been reported above a 0.5 g/t Au cut-off for blocks above 130 masl and at 3 g/t Au cut-off for blocks below 130 masl. The division was created to ensure that the resource meets the reasonable prospects of economic extraction criteria. The estimated Inferred Mineral Resource as at 31 August 2012 is shown in Table 14.5.

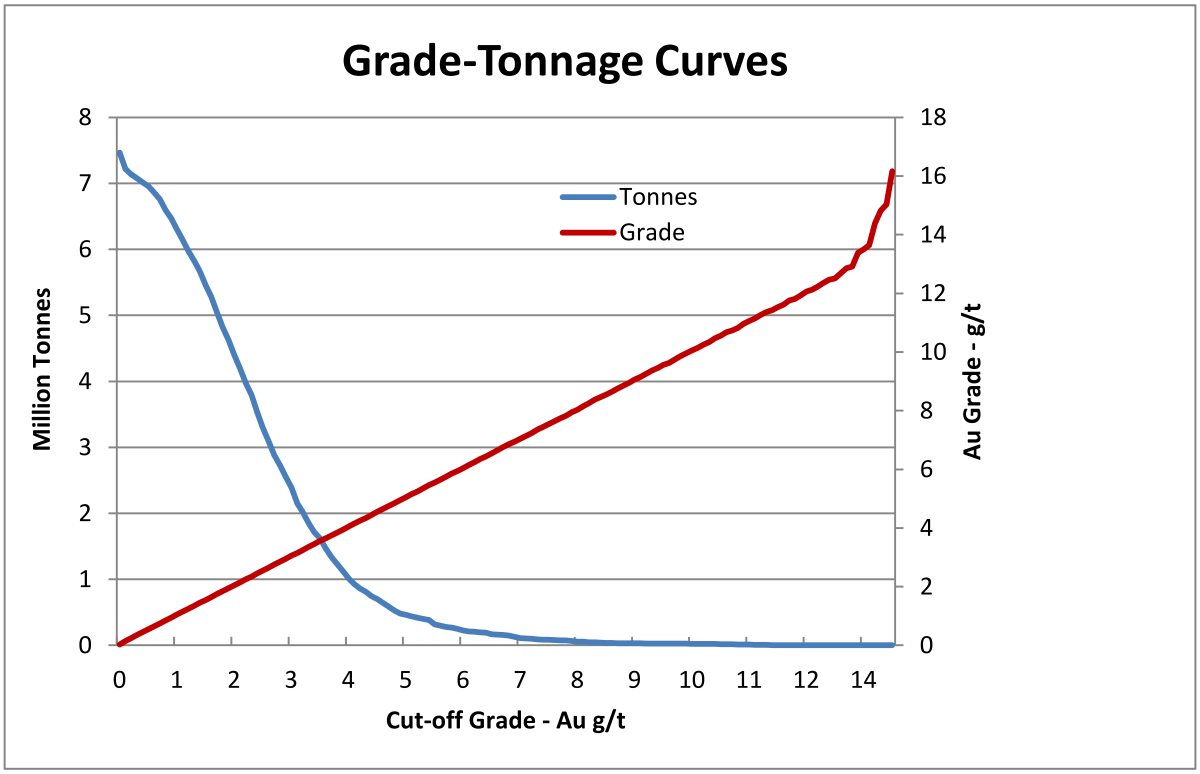
Table 14.5 Summary of Estimated Inferred Mineral Resource as at 31 August 2012

Cut-off grade	Tonnes	Au	Contained
0.5 g/t Au above 130 masl	3.8	2.5	306,100
3 g/t Au below 130 masl	1.2	4.3	166,000

AMC is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors which may materially affect the Mineral Resource.

A Grade-Tonnage curve is shown in Figure 14.6.

Figure 14.6 Grade-Tonnage Curves



14.4 Exploration Potential

There is remaining exploration potential at the Project as there are parts of the zones that have not been sufficiently drilled to gauge their continuity. Due to the location of the old Omega Mine workings there are also a number of drillholes which failed to penetrate to the other side of the workings and these areas will also need to be infill drilled.

There also remains down dip potential to many of the zones, along with their extension along strike.

15 MINERAL RESERVE ESTIMATES

Not applicable

16 MINING METHODS

Not applicable

17 RECOVERY METHODS

Not applicable

18 PROJECT INFRASTRUCTURE

Not applicable

19 MARKET STUDIES AND CONTRACTS

Not applicable

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Not applicable

21 CAPITAL AND OPERATING COSTS

Not applicable

22 ECONOMIC ANALYSIS

Not applicable

23 ADJACENT PROPERTIES

The Kirkland Lake - Larder Lake area is historically one of the most prolific gold districts in North America. Gold was first discovered in 1906, with the first mine coming into production by 1910. Gold production has been almost continuous since then and between 1910 and 1999, the gold camp produced some 1.16 million kilograms (37.3 million ounces) of gold from 25 mines. The majority of the gold mines are located on or near the C-LLDZ or on subsidiary splays and shears. Currently in the area, there are 2 mines in production and 3 properties are in the advanced stage of exploration (Fardy, 2012).

23.1 Kerr-Addison Mine

The old Kerr-Addison Mine lies 6 km east of the Omega project site. The Kerr-Addison Mine produced approximately 11 million ounces of gold during a 58 year operating life from 1938 to 1996. Gold-bearing zones within its extensive mineralized system were mined from surface to a depth of 1,312 feet, and over a strike length of about 975 m. The Kerr-Addison Mine has the same alteration assemblages and type of mineralization as Omega.

23.2 Bear Lake Gold

Bear Lake Gold Ltd is situated 2.5 km east of and along strike with the Omega project site. Their properties cover the Fern

land, Cheminis and the Bear Lake Zones. In April and June, 2011, P&E Mining Consultants issued NI 43-101 reports on the Cheminis and Bear Lake Zones. The Cheminis Zone has an Indicated Mineral Resource of 335,000 tonnes at 4.07 g/t Au and an Inferred Resource of approximately 1.39 Mt at 5.22 g/t Au. The Bear Lake Zone has an Inferred Resource of 3.75 Mt at 5.67 g/t Au. (Bear Lake Gold, 2012).

23.3 Armistice Resources Group

Armistice Resources Group have properties immediately east of the Bear Lake Gold properties and 5 km east of and along strike with the Omega project site. The company began hoisting ore from the old McGarry shaft in January 2012. Armistice Resources issued an NI 43-10I report on the McGarry Deposit with an Indicated Resource totalling some 440,000 tonnes at 7.9 g/t Au and an Inferred Resource of 156,000 tonnes at 5.83 g/t Au. In estimating the resources, high grade gold values were cut to 51.43 g/t. The resource estimate was prepared by E. Anderson, P.Eng. a non-independent Qualified Person and dated September 2011 (Drenan, 2011).

24 OTHER RELEVANT DATA AND INFORMATION

Not applicable

25 INTERPRETATION AND CONCLUSIONS

The Property was a producer of gold over two periods in the 1920's and 1950's and has been re-explored at various periods since 1975, by a number of companies. The Property is a brown-field site with the inherited problems of both contaminated land from the tailings and problems associated with the old Omega Mine workings. Both these issues will need to be dealt with as the project progresses. To remove the arsenic enriched waste, Mistango has contracted the removal and re-processing of this material to a third party.

The exploration programs run by Mistango have been conducted at generally good industry standards and the resulting data is appropriate for the estimation of Mineral Resources. Some historic 1980s drilling data was used for resource estimation purposes after validation by twinning of some of these drillholes. Additional information relating to old underground workings has also been incorporated into the model in order to ensure the estimate has taken into account the material already mined. All known errors have been removed from the drillhole database for this estimate.

The geological interpretation of the deposit agrees with the style of mineralization found in this area. The mineralization occurs within a series of narrow zones adjacent to the hanging wall (south contact) of the ultramafic rocks with altered basaltic volcanics along the LLB. The majority of the mineralization is deposited along the main fault, which defines the hanging and foot walls at the old Omega Mine. There is probably repetition of the zones due to stacking of the zones during movement along the LLB.

AMC's 31 August 2012 resource estimate is based on a total of 152 drillholes, set out on an approximate 50 m x 50 m grid. AMC have set a maximum limit of 200 m on the open pit resource as being the probable maximum depth achievable in this environment.

26 RECOMMENDATIONS

26.1 Sample and Other Data

The location of the stopes needs to be improved with addition of new drillhole data.

A reconciliation between the actual topography and the current contour map which has been used to locate the collars elevations is required. It is suggested that Mistango purchase a more accurate satellite digital topographical map. The estimated cost of this is C$1,000.

AMC recommend better sample control be implemented for any further drilling in order to reduce the mixing up of the standards as noted in the QA/QC review. Mistango has already put in place a QC system that is tracked by one person and this will help control this source of error.

AMC recommend that Mistango use continuous sampling of the holes. The gold mineralization is nuggetty and the zones which carry the gold appear to pinch and swell, therefore it may be easy to miss a particular zone during core sampling. Better structural logging of the core would also aid in identifying if the faults impact on the continuity of the individual zones.

A significant number of bulk density measurements should be collected from all rocks types within the drill core in order to generate values that have a good spatial coverage of the deposit. This could be undertaken as part of the sampling process on site.

26.2 Additional Drilling

Additional infill drilling should be undertaken within the apparent gaps based on the current 50 m x 50 m grid. These gaps have been caused by the loss of holes; primarily due to the presence of old Omega Mine workings. Also additional drilling is required to test the down-dip extension of the mineralized zones.

AMC estimates that a total of 1,800 m of additional drilling would be required to infil the 50 m x 50 m grid. Mistango estimates the total cost per metre of drilling and assaying in this region is C$150 per metre drilled. The total cost of this additional drilling is therefore estimated at C$270,000.

27 REFERENCES

Bear Lake Gold. (2012). Retrieved August 2012, from http://www.bearlakegold.com

DiLauro, P. (2012). *An Investigation into the Processing of Omega Deposit Ore.* Unpublished.

Drenan, M. (2011). *Preliminary Economic Assessmant and Mineral Resource Estimate.* Unpublished.

Dube, B. a. (2007). *Greenstone hosted quartz-carbonate vein deposits.* Geological Association of Canada.

Fardy, D. (2011). *A Technical Review of the Phase 1 Drilling Program, Omega Gold Mine Property, ON, Canada, for Mistango River Resources Inc.* Unpublished.

Groves, D. e. (1998). Orogenic gold deposits: A proposed classification in the context of their crustal distribution and relationship to other gold deposit types. *Ore Geology Reviews* , v.13, p.7-27.

Heaman, L. (1988). *A precise U-Pb zircon age for a Hearst dyke.* Geological Association of Canada.

Hinse, G. (1986). *Lenora Exploration Limited Summary report on the Larder Lake Proect, located in the McVittie Township.* Unpublished report.

Hinse, G. (1984). *Progress report on the McVittie Township Gold Property of Lenora Exploration Limited for the period of Jan 1st to Dec 31st 1983.* unpublished report.

Hinse, G. (1983). *Progress Report on the McVittie Township Gold Property of lenora Exploration Limited for the period of July 1 to December 31, 1982.* unpublished report.

Hinse, G. (1981). *Report on a Diamond Drill program done on the West Group of Lenora Exploration Limited in McVittie Township, Ontario.* unpublished report.

Hinse, G. (1982). *Report on a rock geochemical survey done on a claim group located in Dasserat Township, northwestern Quebec.* unpublished company report.

Jenney, C. (1941). Geology of the Omega mine, Larder lake, Ontario. *Economic Geology* , p.424-447.

Kirkland Lake Gold Inc. (2012, April). Retrieved Aug 2012, from http://klgold.com

Lunden. J. and Hubert, C. (2007). Geologic evolution of the Late Archean Abitibi greenstone belt of Canada. *Geology* , v.14, p.707-711.

Powell, W. (1991). *The distribution, structural history and relationship to regional metamorphism of high-strain zones forming the Larder Lake-Cadillac deformation zone, Metachewan area, Abitibi Belt.* Ontario Geological Survey, Open File Report 5789.

Renaud, J. (2011). *Petrographic and Electron Microprobe Examination of Green Mica+Carbonate and Sulpide Breccia samples from mistango River Resources' Omega Gold Project.* unpublished report.

Thompson, J. (1941). *Ontario Department of Mines Annual Report.*

28 QUALIFIED PERSONS CERTIFICATES QUALIFIED PERSONS CERTIFICATES

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1. I, Catherine Pitman, P.Geo.(Ontario), do hereby certify that I am a Senior Geologist for AMC Mining Consultants (Canada) Ltd., Suite 300, 90 Adelaide Street West, Toronto, Ontario, M5H 3V9, Canada
2. I graduated with a BSc. in Geology from University of Wales in 1982 and an M.Sc. in Mining Geology from Camborne School of Mines in 1983.
3. I am a registered member of the Association of Professional Geoscientists of Ontario.
4. I have practiced my profession continuously since 1998, and have been involved in mineral exploration and mine geology for a total of 14 years since my graduation from university. This has involved working in UK and Canada. My experience is in database management, geological interpretation and resource estimation.
5. 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 purposes of NI 43-101.
6. I am responsible for the preparation of *Omega Property, McVittie Township, Ontario, Canada, Technical Report for Mistango River Resources Inc.*, dated effective 31 August 2012 (the "Technical Report"). I have visited the Omega Property on 7-9 August 2012.
7. I have not had any prior involvement with the property that is the subject of the Technical Report.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43-101F1, and the Technical Report for which I am responsible has been prepared in compliance with that instrument and form.
10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 26 October 2012

Original signed and sealed by
Catherine Pitman P.Geo.(Ontario)

Rod Webster, M.AIG
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1. I, Rod Webster,M.AIG, do hereby certify that I am a Principal Geologist for AMC Mining Consultants Pty Ltd, Level 19, 114 William Street, Melbourne Vic 3000.
2. I graduated with a BAppSc in Geology from the Royal Melbourne Institute of Technology in 1980.
3. I am a registered member of the Australian Institute of Geoscientists, and The Australasian Institute of Mining and Metallurgy.
4. I have practiced my profession continuously since 1980, and have been involved in mineral exploration and mine geology for a total of 32 years since my graduation from university. This has involved working in Australia, United Kingdom and Canada. My experience is principally in base metals, precious metals, coal, mineral sands and uranium.
5. 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 purposes of NI 43-101.
6. I am responsible for part of Section 14 of the *Omega Property, McVittie Township, Ontario, Canada, Technical Report for Mistango River Resources Inc.*, dated effective 31 August 2012 (the "Technical Report"). I have not visited the Omega Property.
7. I have not had any prior involvement with the property that is the subject of the Technical Report.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43-101F1, and the Technical Report for which I am responsible has been prepared in compliance with that instrument and form.
10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

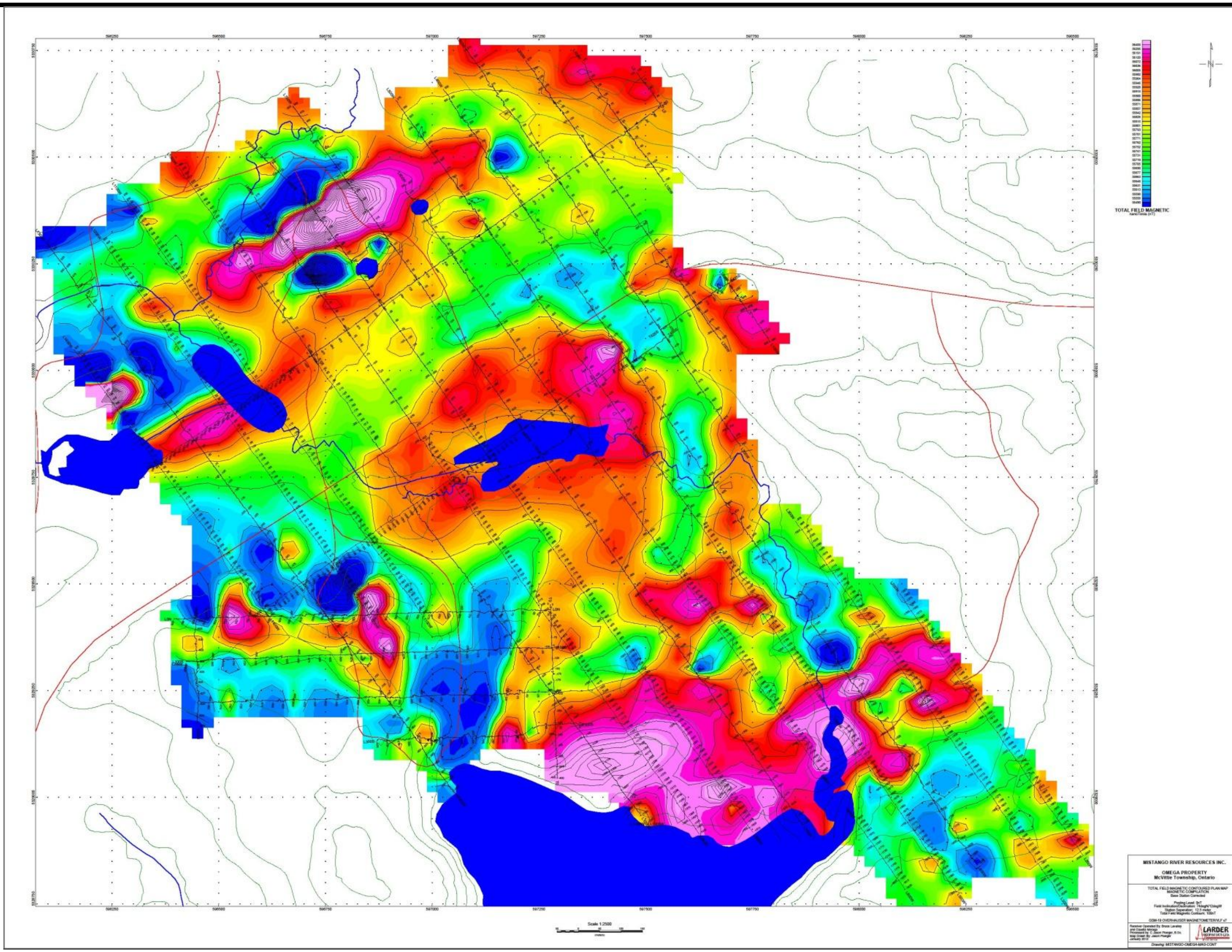
Dated this 26 October 2012

Original signed and sealed by

Rod Webster, MAIG
Principal Geologist

APPENDIX A

Geophysics Interpretation Maps



Mistango River Resources Omega property	**SCALE**	**DATE**	Ground Magnetics
	As shown	2 Oct 2012	
	DRAWING REFERENCE		
	Appendix A-1		

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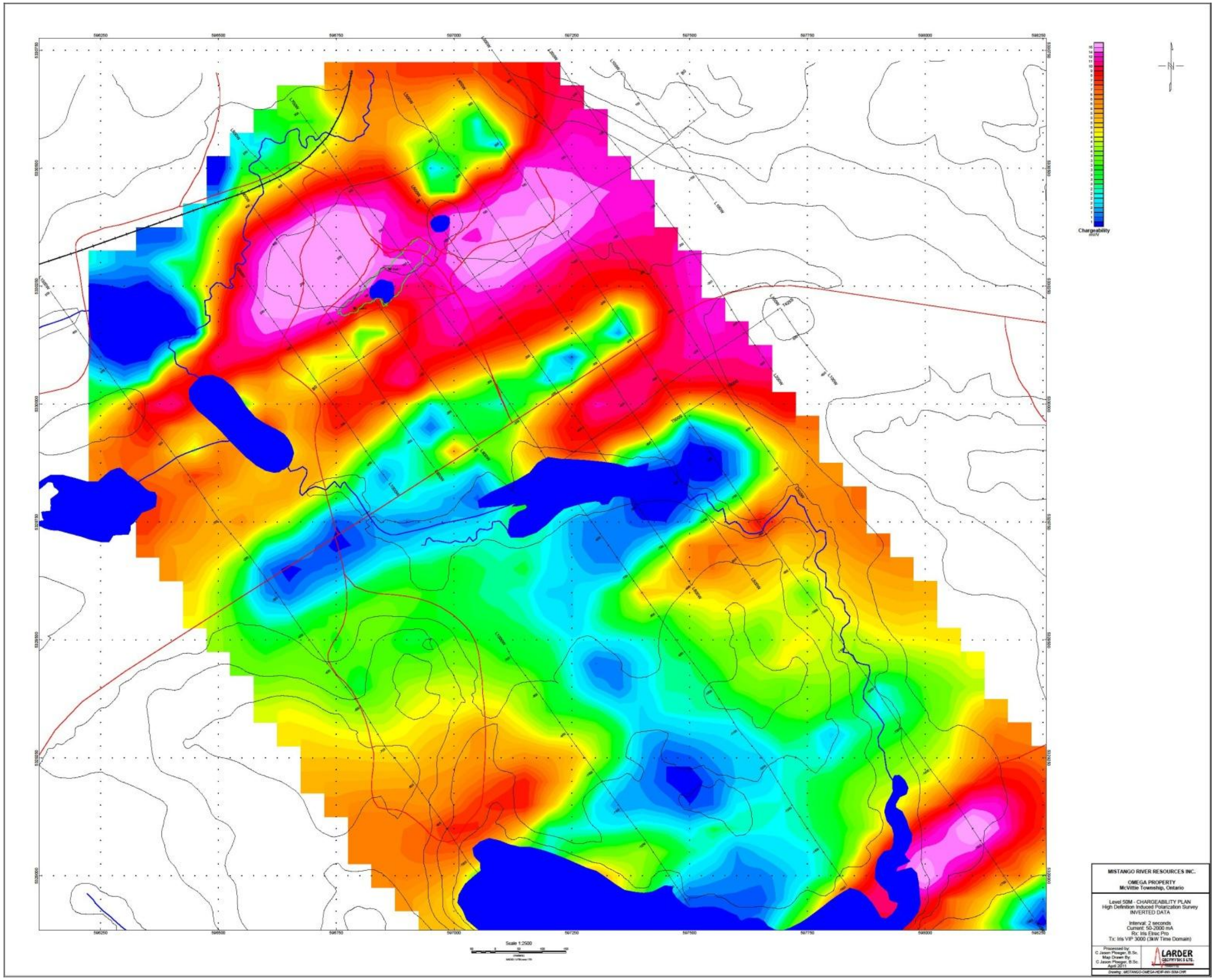

Mistango River Resources Omega property	SCALE	DATE	Induced Polarization - Chargeability
	As shown	2 Oct 2012	
	DRAWING REFERENCE		
	Appendix A-2		

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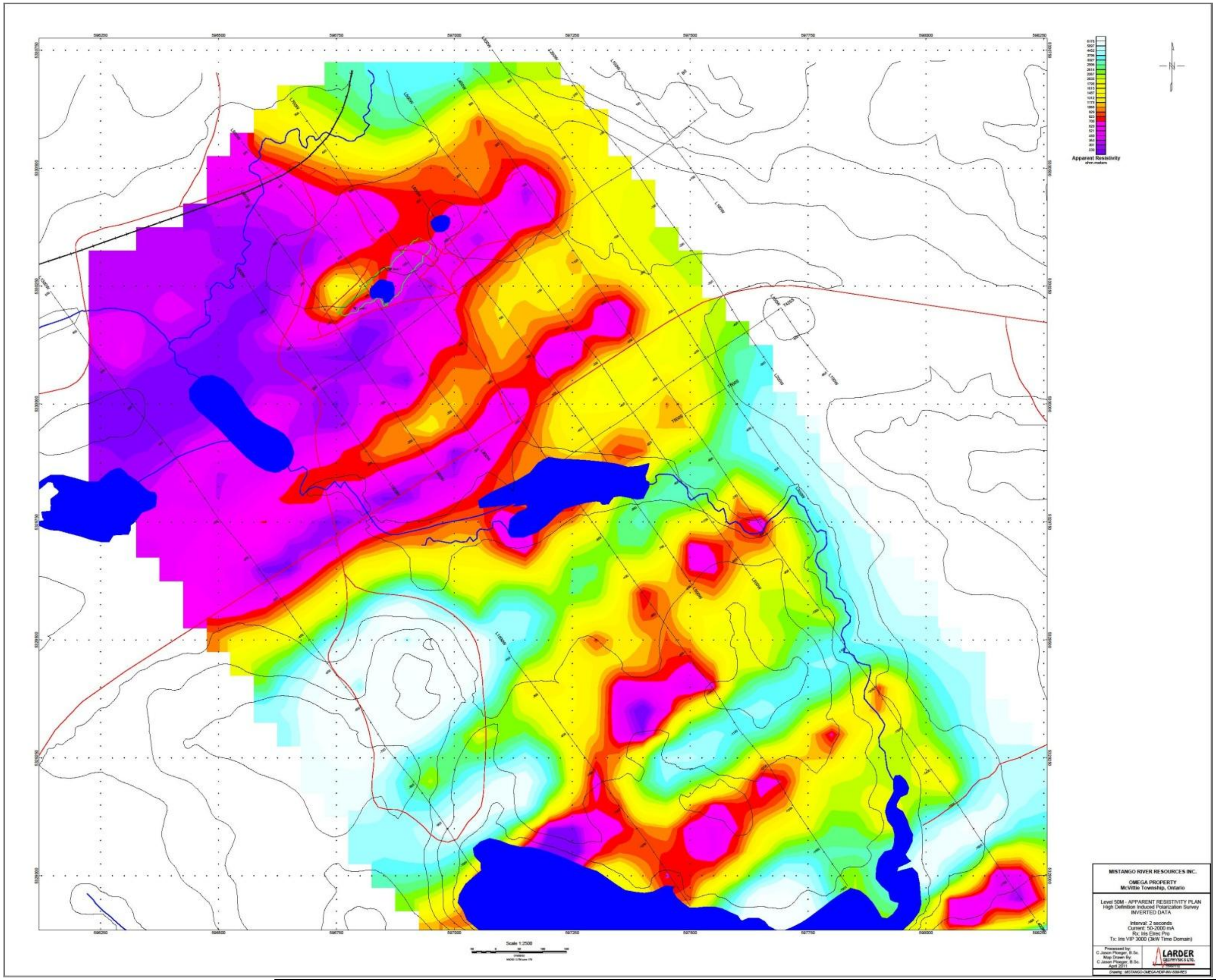

Mistango River Resources Omega property	SCALE	DATE	Induced Polarization - Resistivity
	As shown	2 Oct 2012	
	DRAWING REFERENCE		
	Appendix A-3		

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APPENDIX B

Drillhole Information

Table B.1 Phase 1 Summary

BHID	Easting	Northing	Elevation	Azimuth	Dip	Hole Length
OM-11-01	597157.216	5329935.901	287.956	325	-65	407.5
OM-11-02	596863.166	5330177.465	292.477	325	-45	150
OM-11-03	596864.033	5330177.113	292.8	325	-70	188
OM-11-04	597091.509	5330292.771	294.353	325	-50	170.5
OM-11-05	597244.544	5330428.038	301.783	325	-45	149
OM-11-06	596927.165	5330174.761	289.762	325	-45	149
OM-11-07	596927.24	5330174.642	289.725	325	-70	152
OM-11-08	597241.348	5329993.193	288.586	325	-65	606.3
OM-11-09	596968.775	5330203.077	290.329	325	-45	207
OM-11-10	596968.342	5330203.304	290.372	325	-70	137
OM-11-11	597197.713	5329963.546	288.502	325	-65	650
OM-11-12	597120.814	5330004.036	288.628	325	-65	277
OM-11-12A	597120.959	5330003.794	288.775	325	-65	47
OM-11-13	597011.328	5330231.518	292.48	325	-45	119
OM-11-14	597011.684	5330231.211	292.341	325	-70	248
OM-11-15	597080.461	5330036.373	288.803	325	-60	665
OM-11-16	597049.619	5330263.271	293.453	325	-45	117
OM-11-17	597049.881	5330262.861	293.391	325	-70	173
OM-11-18	597127.23	5329885.99	287.85	325	-65	635
OM-11-19	597046.202	5330006.277	288.937	325	-60	566
OM-11-20	597088.432	5329857.271	288.184	325	-65	662
OM-11-21	596955.113	5330397.061	296.716	145	-45	69
OM-11-22	596945.493	5330411.485	296.41	145	-45	78
OM-11-23	596918.159	5330360.376	297.282	145	-45	59
OM-11-24	596908.773	5330374.125	297.508	145	-45	72
OM-11-25	596876.018	5330331.654	298.295	145	-45	81
OM-11-26	597133.701	5330323.41	296.903	325	-50	156.5
OM-11-27	596851.921	5330364.075	299.129	145	-45	103
OM-11-28	597189.95	5330242.712	290.857	325	-50	452
OM-11-29	597376.889	5330497.535	303.514	325	-50	458
OM-11-30	596825.934	5330315.522	301.698	145	-45	100
OM-11-31	596810.873	5330337.733	301.182	145	-45	59
OM-11-32	596794.603	5330359.379	300.877	145	-45	26.1
OM-11-33	596797.475	5330269.433	304.098	145	-45	86
OM-11-34	596767	5330322	311	145	-45	146
OM-11-35	597297.102	5330437.943	302.722	325	-50	338
OM-11-36	597330.344	5330465.102	302.729	325	-55	401
OM-11-37	597098	5330555	303	325	-50	362
OM-11-38	596844.742	5330115.858	289.852	325	-50	180

BHID	Easting	Northing	Elevation	Azimuth	Dip	Hole Length
OM-11-39	596844.742	5330115.858	289.852	325	-70	200
OM-11-40	596802.635	5330086.37	290.243	325	-45	107
OM-11-41	596802.635	5330086.37	290.243	325	-70	185
OM-11-42	596761.932	5330057.065	292.023	325	-45	194
OM-11-43	596761.932	5330057.065	292.023	325	-70	137
OM-11-44	596821.458	5329974.514	288.327	325	-50	246
OM-11-45	596859.958	5330003.192	288.845	325	-50	345
OM-11-46	596903.045	5330034.047	289.105	325	-50	251
OM-11-47	596901	5330043	293	325	-65	284
OM-11-48	596938	5330068	297	325	-45	215

Table B.2 Phase 2 Summary

BHID	Easting	Northing	Elevation	Azimuth	Dip	Hole Length
OM-11-49	596901	5330043	293.00	325	-65	317
OM-11-50	596856	5330008	293.00	325	-65	356
OM-11-51	596821	5329974	288.00	325	-65	316
OM-11-52	597200.88	5329874.8	287.04	325	-70	661
OM-11-53	597272.49	5329948.16	287.53	325	-70	629
OM-11-54	596759.15	5330236.35	304.30	145	-45	83
OM-11-55	596727.33	5330280.56	302.73	145	-45	238.6
OM-11-56	597367.18	5329994.19	292.08	325	-70	644
0M-11-57	597022.08	5330303.1	296.49	325	-50	161
OM-11-58	597061.4	5330335.78	299.39	325	-50	159
OM-11-59	597112.73	5330361.88	300.42	325	-50	176
OM-11-60	597494.65	5329967.87	291.53	325	-80	698
OM-11-60A	597494.65	5329967.87	291.53	335	-80	29
OM-11-61	597140.63	5330396.18	299.06	325	-50	138.6
OM-11-62	597091.46	5330292.92	294.21	325	-50	164
OM-11-63	597091.98	5330291.98	294.39	325	-60	221
OM-11-64	597337.19	5329856.98	286.59	325	-70	800
OM-11-65	597046.89	5330356.13	298.65	325	-45	146
OM-11-66	597071.78	5330408.4	297.70	325	-45	179
OM-12-67	597153.52	5330025.17	288.67	325	-60	405.2
OM-12-68	597143.47	5330394.14	299.18	325	-45	185
OM-12-69	597123.66	5330067.55	288.62	325	-50	404
OM-12-70	597160.43	5330285.25	292.97	325	-50	259
OM-12-71	597194.8	5330057.48	288.45	325	-60	475.3
OM-12-72	597118.56	5330253.53	291.28	325	-50	227
OM-12-73	597238.44	5330081.88	288.87	325	-70	322.3
OM-12-74	597272.94	5330126.26	289.21	325	-70	491
OM-12-75	597249.95	5330158.77	289.69	325	-60	503
OM-12-76	597138.02	5330138.66	289.04	325	-60	452
OM-12-77	597209.19	5330123.51	289.13	325	-60	443
OM-12-78	597353.82	5330173.8	289.91	325	-75	422
OM-12-79	597287.66	5330185.07	290.29	325	-60	500
OM-12-80	597146.6	5330212.77	289.39	325	-50	275

Table B.3 Phase 3 Summary

BHID	Easting	Northing	Elevation	Azimuth	Dip	Hole Length
OM-12-81	597114.74	5330171.21	289.085	325	-50	248
OM-12-82	597074.21	5330228.85	289.923	325	-50	181
OM-12-83	597036.34	5330191.84	289.278	325	-50	263
OM-12-84	596993.45	5330165.94	289.228	325	-50	225
OM-12-85	597025.6	5330121.93	288.76	325	-45	252
OM-12-86	596956.2	5330133.83	288.95	325	-50	248
OM-12-87	596985.89	5330091.76	288.56	325	-50	260
OM-12-88	597026.48	5330034.54	288.62	325	-50	278
OM-12-89	596935.89	5330247.88	296.04	325	-45	58
OM-12-90	596971.96	5330023.04	288.68	325	-45	311

Table B.4 Phase 4 Summary Table

BHID	Easting	Northing	Elevation	Azimuth	Dip	Hole Length
OM-12-91	597325	5330396	301	325	-50	233
OM-12-92	597281	5330375	300	325	-50	220
OM-12-93	597175	5330179	290	325	-50	311
OM-12-94	597065	5330161	290	325	-50	248
OM-12-95	596972	5330286	297	325	-50	195
OM-12-96	597053	5330091	290	325	-50	251
OM-12-97	596885	5329967	290	325	-60	326
OM-12-98	596875	5329897	290	325	-50	398

APPENDIX C

SGS Canada Metallurgical Report

An Investigation into

THE PROCESSING OF OMEGA DEPOSIT ORE

prepared for

MISTANGO RIVER RESOURCES

Project 13634-001 – Final Report
June 25, 2012

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Table of Contents

List of Tables

List of Figures

Executive Summary

The metallurgical investigation undertaken on the Omega deposit ore composite sample for Mistango River Resources has provided some understanding of the sample nature and metallurgical behaviour.

A gold-bearing composite sample was examined at the SGS Mineral Services Lakefield site. The Omega sample composite contained 3.58 g/t gold based on direct head assaying by pulp and metallic protocol. The silver grade was determined to be < 2 g/t. The composite sample also yielded a sulphide sulphur grade of 3.54%.

Initial whole ore cyanidation testing of the composite sample leached showed recoveries after 48 hours of leaching ranging from 76% to 86% while cyanide consumptions were 0.53 kg/t to 1.38 kg/t of NaCN. Lime consumptions were low at 0.40 kg/t to 0.45 kg/t.

Gravity separation testing on the Omega composite at a P_{80} size of 125 microns showed a very low result of gold recovery of 3%.

Gravity tailing cyanidation testing of samples leached showed similar recoveries after 48 hours of leaching as observed in the whole ore leaches. Gold recoveries after 48 hours of leaching ranged from 74% to 84% while cyanide consumptions were 0.54 kg/t to 1.39 kg/t of NaCN. Lime consumptions were low at 0.41 kg/t to 0.46 kg/t. The combined gravity plus gravity tailing cyanidation gold recoveries for the composite ranged from 75% to 84% showing no real increase due to the very low gravity recovery of gold.

Gravity tailing flotation testing of samples showed excellent gold recoveries for all tests conducted. Gold recoveries for all three test performed were reported at 99%. While the Omega Composite head silver grade was reported at < 2 g/t there was a significant improvement in recovery observed in the finer grind tests. For the tests, silver recoveries were shown to be 48% for the test at a P_{80} size of 125 microns, 66% for the test at a P_{80} size of 85 microns and 70% for the test at a P_{80} size of 52 microns.

The diagnostic leach program showed an initial 84.2% gold recovery of readily leachable gold. 3.2% of the gold was further extracted from possible gold associations with iron-arsenic compounds or bismuth minerals and 2.6% of the gold was further extracted from possible gold associations with weak acid soluble compounds. 7.4% of the gold was observed to be from possible gold associations with or occluded by sulphide minerals, pyrite and arsenopyrite. The remaining 2.5% of the gold remaining in the final leach residue was deemed to be the gold mainly associated with silicates or fine sulphides locked in silicates. The results from the diagnostic leach program should be viewed as an indication of general trends and possibilities only.

Introduction

Mistango River Resources are currently investigating the processing of their Omega deposit ore. SGS Minerals Services was requested to perform a scoping level metallurgical test program to establish the basic processing parameters for the treatment of a composite sample shipped to Lakefield. The sample was received in a shipment of two skidded plastic pails on February 27th, 2012.

The test program was designed in conjunction with Mr. Bob Kasner and Mr. Fred Sharpley of Mistango River Resources and Dan Mackie of Dan Mackie & Associates. The results were forwarded as they became available via e-mail.

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Report preparation by: Pete DiLauro
Reviewed by: S. McKenzie, I. Dymov

Testwork Summary

A scoping test program was undertaken on behalf of Mistango River Resources to examine the potential for processing the Omega deposit ore to various metallurgical process options for the extraction of gold.

The composite sample was prepared for metallurgical evaluation in order to produce sufficient information to achieve a level of confidence in the process selection for the ore sample.

The samples received from the Omega deposit were previously assayed and showed a calculated gold grade of 3.82 g/t of gold. The combined samples for testing at SGS Lakefield were identified as the Omega Composite.

Testing on this Omega Composite consisted of metallurgical work. Metallurgical testing included whole ore cyanidation, gravity separation, gravity tailing cyanidation, gravity tailing flotation and diagnostic leaching.

The effect of the fineness of grind on the recovery of gold was the main parameter evaluated in this test program.

The results of the test program are summarised in the following sections, and full details of the described work are appended.

1. Sample Receipt & Characterisation

On February 27th, 2012, ore samples were received from Mistango River Resources from their Omega deposit property for metallurgical testing. The samples arrived in a single shipment in two plastic pails (SGS receipt number 0379-FEB12).

1.1. Sample Inventory

In the two pail shipment, there were twelve individual samples in plastic bags, unlabelled. The total weight for the sample was 17.0 kilograms. The full inventory list can be viewed in Appendix A.

1.2. Sample Preparation

For the composite sample it was understood that the entire contents of each pail (twelve bags) would be combined to make the composite sample for testing. The composite was crushed to minus 10 mesh and well blended. Representative portions were removed and submitted for head analyses by way of screened metallics protocol for Au, as well as for C speciation, S speciation, whole rock analysis and an

ICP scan. Test charges were also prepared and forwarded to the metallurgical testing to fulfill the program requirements.

1.3. Chemical Analysis

The gold head grade of the composite sample was determined by applying a screened metallics protocol at +/- 150 mesh. The screened undersize product was sampled by riffling and assayed in duplicate. The screened oversize product was assayed to extinction. From the assaying protocol applied it was determined that the gold head grade of the Omega Composite was 3.58 g/t Au. The detailed results from the screened metallics method can be viewed in Table 1.

Table 1 – Composite Screened Metallics Gold Determinations

Sample	Product	Amount		Assay (g/t)	Distribution (%)
		(g)	%	Au	Au
Omega Composite	Plus 150 M	24.48	4.8	5.06	6.8
	Minus 150 M	481.3	95.2	3.51*	93.2
	Head (calc)	505.8		3.58	100.0

Note: * average of duplicate assays for gold (Au) 3.55 g/t 3.47 g/t

The results showed that only about 7% of the gold present reported into the low weight coarser fraction making up approximately 5% of the total weight for the Omega Composite.

A smaller head sample was submitted for sulphur and carbon speciation and for a whole rock analysis, as well as a semi-quantitative ICP scan analysis. The results from these analyses are shown in Table 2.

The assays showed that the composite contained 4.13% sulphur, 3.54% sulphide sulphur and less than 2 g/t silver. Due to the low silver content, silver was not included in the pulp and metallics determinations.

Table 2 – Composite Head Analysis

Element		Omega Composite
Au	g/t	3.58
$C_{(t)}$	%	2.29
$C_{(g)}$	%	0.04
TOC	%	0.14
CO_3	%	10.4
S	%	4.13
$S^{=}$	%	3.54
SO_4	%	0.1
S°	%	< 0.05
ICP-Scan		
Ag	g/t	< 2
As	g/t	219
Ba	g/t	125
Be	g/t	< 0.4
Bi	g/t	< 20
Cd	g/t	< 10
Co	g/t	28
Cu	g/t	87.8
Li	g/t	< 5
Mo	g/t	< 5
Na	g/t	86
Pb	g/t	< 20
Sb	g/t	< 30
Se	g/t	< 30
Sn	g/t	< 20
Sr	g/t	130
Tl	g/t	< 30
U	g/t	< 20
Y	g/t	45.2
Zn	g/t	49
Whole Rock		
SiO_2	%	50.9
Al_2O_3	%	12.1
Fe_2O_3	%	10.1
MgO	%	3.17
CaO	%	5.41
Na_2O	%	4.99
K_2O	%	0.72
TiO_2	%	1.09
P_2O_5	%	0.16
MnO	%	0.16
Cr_2O_3	%	0.02
V_2O_5	%	0.01
LOI	%	6.07

*determined by screen metallics protocol

2. Metallurgical Testing

The metallurgical test program examined the response of the composite sample to processes including whole ore cyanidation, gravity separation, gravity tailing cyanide leaching, gravity tailing flotation and diagnostic leaching.

The metallurgical tests performed are discussed in the following sections and the details for each are given in Appendix B.

2.1. Whole Ore Cyanidation

The composite was subjected to three, 1 kilogram exploratory grinds in a laboratory rod mill in order to determine a grind curve. Bottle roll cyanidation tests were completed on 1 kilogram charges of whole ore for each composite at three P_{80} grind sizes, ~150 µm, ~75 µm and ~50 µm. The conditions which were applied were 40% solids for 48 hours with the pH maintained between 10.5 and 11.0 with lime added as calcium hydroxide. The cyanide concentration was maintained at 0.5 g/L NaCN throughout the 48 hours as well. During the tests the pregnant leach solutions were sub-sampled and submitted for assays. Upon completion of the tests, the final residues and final pregnant leach solutions were submitted for assay.

The leach results showed that there is an improvement in gold recovery the finer the grind size tested. CN-1 at a P_{80} of 112 microns reported a gold recovery of 77% after 7 hours of leaching and showed no improvement after 48 hours. CN-2 at a P_{80} of 68 microns reported a gold recovery of 85% after 7 hours of leaching and showed no improvement after 48 hours and CN-3 at a P_{80} of 57 microns reported a gold recovery of 80% after 7 hours of leaching and showed an improvement after 48 hours to a gold recovery of 86%. The final reagent consumptions ranged from 0.53 kg/t to 1.38 kg/t NaCN and 0.40 kg/t to 0.45 kg/t lime for the tests.

While the 48 hour gold recovery for tests CN-2 and CN-3 are quite similar, the kinetics appear to be slower and the cyanide consumption was 0.34 g/t more.

Table 3 shows the test results summary for the whole ore cyanidations. Figure 1 shows the recovery over time for the composite for the different grind sizes tested.

Table 3 – Whole Ore Cyanidation Results

Sample	Test No.	Size Target µm	Size Actual µm	Reagent Addition kg/t CN Feed		Regent Cons. kg/t CN Feed		% Extraction Au			Residue g/t Au	Head (calc) g/t Au	Head (dir) g/t Au
				NaCN	CaO	NaCN	CaO	7h	24h	48h			
Omega Comp	CN 1	150	112	1.24	0.47	0.53	0.45	77	77	76	0.81	3.38	3.58
	CN 2	75	68	1.67	0.44	1.04	0.43	85	84	83	0.59	3.49	
	CN 3	50	57	2.10	0.43	1.38	0.40	80	84	86	0.55	3.80	

All cyanidations were conducted at 40% solids, 0.5 g/L NaCN, pH 10.5-11.0 and for 48 hours.

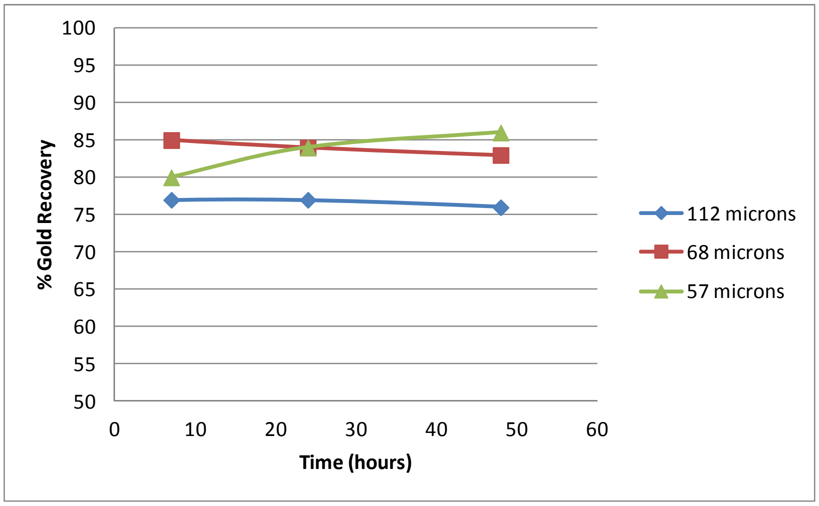


Figure 1 – Omega Composite Whole Ore CN Recovery vs. Time and Grind Size

2.2. Gravity Separation

A 10 kilogram ground charge (K_{80} ~125 µm) of the composite was processed through a Knelson MD-3 concentrator. The Knelson concentrate was recovered and upgraded further by treatment on a Mozley mineral separator. The Mozley concentrate (6 to 12-g) was assayed to extinction for Au and Ag. The Mozley and Knelson tailings were recombined, blended and divided into representative charges for downstream flotation and cyanidation testwork.

The test on the composite sample resulted in a percentage of gold reporting to the Mozley gravity concentrate of only 3%. The Mozley concentrate assayed at 109 g/t gold and 31 g/t silver, while the gravity tailing grade was reported at 3.48 g/t gold. A summary of the test results is given in Table 4.

Table 4 – Gravity Test Results Summary

Test No.	Sample	Tailing k80 (micron)	Conc. wt. (%)	Conc. Ag (g/t)	Conc. Au (g/t)	Recovery Au (%)	Tailing Au (g/t)	Head Grade Au Calc (g/t)	Head Grade Au Direct (g/t)
GV-1	Omega	125	0.10	31	109	3.1	3.48	3.58	3.58

2.3. Gravity Tailing Cyanidation

Based on the grind curves established in the whole ore cyanidation testing, 1 kilogram (dry equivalent) charges of gravity tailing were reground to ~75 µm and ~50 µm. These charges of gravity tailings along with one charge not reground (~150 µm) were subjected to cyanidation testing. The conditions which were applied were 40% solids for 48 hours with the pH maintained between 10.5 and 11.0 with lime added as calcium hydroxide. The cyanide concentration was maintained at 0.5 g/L NaCN throughout the 48 hours as well. During the tests the pregnant leach solution was sub-sampled and submitted for assay. Upon completion of the tests, the final residue and final pregnant leach solution were submitted for assay.

Results for the Omega composite samples were very similar to those determined in the whole ore leaching tests. The leach results showed that there is an improvement in gold recovery the finer the grind size tested. CN-4 at a P_{80} of 125 microns reported a gold recovery of 75% after 7 hours of leaching and showed no improvement after 48 hours. CN-5 at a P_{80} of 85 microns reported a gold recovery of 83% after 7 hours of leaching and showed no improvement after 48 hours and CN-6 at a P_{80} of 52 microns reported a gold recovery of 82% after 7 hours of leaching and showed an improvement after 48 hours to a gold recovery of 84%. The final reagent consumptions ranged from 0.54 kg/t to 1.39 kg/t NaCN and 0.41 kg/t to 0.46 kg/t lime for the tests.

When combining the gravity tailing cyanidation gold recoveries with the gravity recoveries the overall recoveries for the three sizes tested were reported as 75% for CN-4, 81% for CN-5 and 84% for CN-6.

Table 5 shows the test results summary for the gravity tailing cyanidations. Figure 4 shows the recovery over time for the composite for the different grind sizes tested.

Table 5 – Gravity Tailing Cyanidation Results

Sample	Test No.	Size Target µm	Size Actual µm	Reagent Addition kg/t CN Feed		Regent Cons. kg/t CN Feed		% Au Extraction				Residue g/t Au	Head (calc) g/t Au	Head (dir) g/t Au
				NaCN	CaO	NaCN	CaO	7h	24h	48h	Grav. & CN			
Omega	CN 4	150	125	1.27	0.47	0.54	0.46	75	75	74	75.0	0.86	3.32	3.48
	CN 5	75	85	1.71	0.45	1.06	0.43	83	81	81	81.1	0.65	3.34	
	CN 6	50	52	2.12	0.44	1.39	0.41	82	83	84	84.2	0.58	3.52	

All cyanidations were conducted at 40% solids, 0.5 g/L NaCN, pH 10.5-11.0 and for 48 hours.

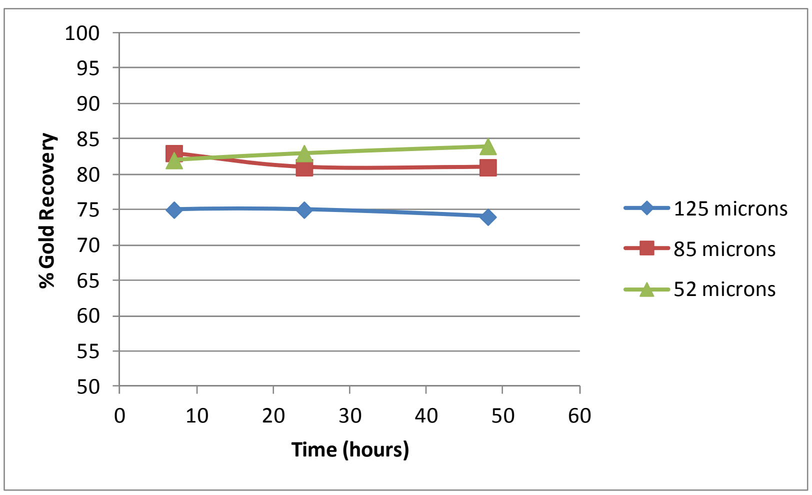


Figure 2 – Omega Composite Gravity Tailing Recovery vs. Time and Grind Size

2.4. Gravity Tailing Flotation Testing

The flotation testing was performed on the composite primarily to evaluate the impact of grind size on gold recovery. The full individual test details can be viewed in Appendix B.

As in the gravity tailing cyanidation section, based on the grind curves established in the whole ore cyanidation testing, 1 kilogram (dry equivalent) charges of gravity tailing were reground to ~75 µm and ~50 µm. This was performed twice for each size as the flotation tests were performed on 2 kilogram charges. These charges along with a two kilogram tailing charge not reground (~150 µm) were subjected to flotation testing.

For the tests a xanthate (PAX) and specific gold collector (Cytec A 208) were applied and a series of 5 timed rougher concentrates were recovered and assayed for gold, silver and sulphide sulphur. All tests were performed with total additions of 100 g/t of PAX, 25 g/t Cytec A 208 and 30-35 g/t MIBC.

For the gravity tailing flotation testing on the Omega Composite, the gold and sulphide sulphur recovery was reported as 99% for both and the silver recovery was reported as 48%, with a mass pull of 32% for the test F1 performed at a P_{80} of 125 microns. The gold and sulphide sulphur recovery was reported as 99% for both and the silver recovery was reported as 66%, with a mass pull of 33% for the test F2 performed at a P_{80} of 85 microns and 99% for both and a silver recovery of 70%, with a mass pull of 35% for the test F3 performed at a P_{80} of 52 microns. The flotation bulk concentrate grades ranged from 10 g/t to 12 g/t Au, 1 g/t to 2 g/t Ag and 11% to 12% $S^{=}$. When combining the gold recovery from the gravity concentrate with the recoveries from the gravity tailing flotation concentrates, the overall gold recoveries for the Omega Composite were 99% for all three tests. Table 6 shows the detailed summary of the flotation results from the Omega Composite gravity tailing tests.

Table 6 – Omega Composite Gravity Tailing Flotation Results

Test No.	Sample	Feed, P_{80}, µm	Flotation Conditions (g/t)	Product (cumulative)	Mass %	Assays, g/t, %			% Distribution			Au Extraction
						Au	Ag	S=	Au	Ag	S=	Gravity + Flot %
F1	Omega	Target 150	A208 (25)	Rougher Conc 1	9.2	34.8	1.80	34.6	89.9	25.3	81.4	**98.9**
			PAX (100)	Rougher Conc 1+2	16.0	21.6	1.46	23.4	97.2	35.8	95.7	
			MIBC (30)	Rougher Conc 1-3	23.0	15.2	1.17	16.6	98.3	41.1	97.7	
		Actual 125		Rougher Conc 1-4	28.1	12.5	1.05	13.7	98.6	45.0	98.3	
				Rougher Conc 1-5	32.0	11.0	0.98	12.1	**98.9**	**48.0**	**98.6**	
				Rougher Tailing	68.0	0.06	< 0.5	0.08	1.1	52.0	1.4	
				Calc Head		3.56	0.65	3.91				
F2	Omega	Target 75	A208 (25)	Rougher Conc 1	10.2	32.0	4.90	31.8	93.9	51.8	86.5	**99.2**
			PAX (100)	Rougher Conc 1+2	15.6	21.8	3.49	23.0	97.6	56.3	95.4	
			MIBC (30)	Rougher Conc 1-3	22.5	15.2	2.57	16.4	98.6	59.8	98.0	
		Actual 85		Rougher Conc 1-4	28.2	12.2	2.15	13.1	99.0	62.8	98.7	
				Rougher Conc 1-5	33.3	10.3	1.90	11.2	**99.1**	**65.5**	**99.1**	
				Rougher Tail.	66.7	0.05	< 0.5	0.05	0.9	34.5	0.9	
				Calc Head		3.48	0.96	3.75				
F3	Omega	Target 50	A208 (25)	Rougher Conc 1	10.4	35.7	5.90	29.9	90.7	56.9	71.7	**99.4**
			PAX (100)	Rougher Conc 1+2	18.9	21.3	3.56	22.0	98.3	62.4	95.7	
			MIBC (35)	Rougher Conc 1-3	28.3	14.4	2.55	15.1	99.1	66.7	98.1	
		Actual 52		Rougher Conc 1-4	32.6	12.4	2.27	13.1	99.3	68.8	98.8	
				Rougher Conc 1-5	35.2	11.5	2.14	12.2	**99.4**	**70.0**	**99.1**	
				Rougher Tail.	64.8	0.04	< 0.5	0.06	0.6	30.0	0.9	
				Calc Head		4.09	1.08	4.34				

2.5. Diagnostic Leach Testing

Included in the metallurgical test program was the testing of a selected leach tailing residue sample to a diagnostic leach program. The tailing residue from the whole ore cyanidation test CN-3 performed on the Omega Composite sample was subjected to a three stage diagnostic leach program to determine the gold deportment of the sample.

The diagnostic leach procedure was developed mainly to characterise certain aspects of the deportment of gold. The first stage involved leaching the residue in a hot sodium hydroxide solution in an attempt to dissolve minerals such as iron-arsenic compounds. A cyanidation test was then performed to leach any gold which may have been liberated during the first stage. The residue was then forwarded to a second stage of hot hydrochloric acid leaching, where acid soluble minerals such as calcite and dolomite among others may be dissolved. Another cyanidation test was performed on the hydrochloric acid leach residue to recover any liberated gold. The final stage applied consisted of taking the residue and leaching it in a hot aqua regia solution. The aqua regia leach was done to determine any gold locked in sulphides such as pyrite and arsenopyrite. Any remaining gold in the residue is assumed to be gold mainly associated with silicates or fine sulphides locked in silicates.

The residue selected for the diagnostic leach program was the leach tailing residue from the whole ore leaching, test CN-3 which was the cyanidation performed at the finer size at a P_{80} of 57 microns. The test

CN-3 leach showed a 84.2% gold recovery of readily leachable gold. During the first stage hot sodium hydroxide leach and subsequent cyanidation it was observed that 3.2% of the gold was further extracted from possible gold associations with iron-arsenic compounds or bismuth minerals. In the second stage hot hydrochloric acid leach and subsequent cyanidation it was observed that 2.6% of the gold was further extracted from possible gold associations with the weak acid soluble compounds (i.e. calcite and dolomite among others such as pyrrhotite and ferriites. In the final third stage hot aqua regia leach it was observed that 7.4% of the gold was further extracted from possible gold associations with or occluded by sulphide minerals, pyrite and arsenopyrite. The remaining 2.5% of the gold left in the final diagnostic leach residue was deemed to be the gold mainly associated with silicates or fine sulphides locked in silicates. The results from the diagnostic leach program should be viewed as an indication of general trends and possibilities only. The metallurgical overall balance summary of the test details are shown in Tables 7 and 8, and the full test details can be viewed in Appendix B.

Table 7 – Diagnostic Leach Metallurgical Balance Summary

Stage	Product	Amount g, mL	Assays, mg/L, g/t Au	% Distribution Au	% Extraction Au
Cyanide Leach	CN-3a PLS	527.1	2.06	84.2	84.2
NaOH Leach	CL-3 PLS	793.2	<0.01	0.6	84.9
Cyanide Leach	CN-3b PLS	561.3	0.06	2.6	87.5
HCl Leach	HCl-3 PLS	1290.3	0.01	1.0	88.5
Cyanide Leach	CN-3c PLS	410.3	0.05	1.6	90.1
Aqua Regia Leach	AR-3 PLS	958.2	0.10	7.4	97.5
Aqua Regia Leach	Final Residue	230.0	0.14	2.5	
	Head (calc.)	350	3.68	100.0	
	Head (direct)	350	3.58		

Table 8 – Diagnostic Leach Gold Distribution Summary

Process	Distribution (%) Au
Cyanide Leach: Extraction of readily leachable gold and silver	84.2
Sodium Hydroxide Leach & Cyanide Leach: Extraction of gold and silver associated with iron arsenate, arsenic oxides or bismuth minerals	3.2
Hydrochloric Acid Leach & Cyanide Leach: Extraction of gold and silver associated with weak acid soluble compounds	2.6
Aqua Regia Leach: Extraction of gold and silver associated with, or occluded by sulphide minerals, pyrite, arsenopyrite, etc	7.4
Remaining Material: Gold and silver locked in silcates, or associated with fine sulphides locked in silcates	2.5
	100

Conclusions and Recommendations

Initial direct head assaying of the Omega Composite sample yielded a gold grade of 3.58 g/t and a silver grade of < 2 g/t. The composite sample also yielded a sulphide sulphur grade of 3.54%.

Whole ore cyanidation testing of the composite yielded final gold recovery results ranging from 76% to 86% for all three grind sizes tested. An increase in gold recovery was seen going from the coarser grind to the finer grind. An increase was observed in the NaCN consumption, 0.53 kg/t to 1.38 kg/t NaCN from a P_{80} of 112 to 57 microns. The residue tailing gold grades for the three tests were reported at 0.55 g/t to 0.81 g/t for the composite.

Gravity separation testing on the Omega composite at a P_{80} size of 125 microns showed a very poor result of gold recovery of only 3.1%.

Gravity tailing cyanidation testing of the composite yielded final gold recovery results ranging from 75% to 84% for all three grind sizes tested. An increase in gold recovery was seen going from the coarser grind to the finer grind. An increase was observed in the NaCN consumption, 0.54 kg/t to 1.39 kg/t NaCN from a P_{80} of 125 to 52 microns. The residue tailing gold grades for the three tests were reported at 0.58 g/t to 0.86 g/t for the composite. The combined gravity plus gravity tailing cyanidation gold recoveries for the composite ranged from 75% to 84% showing no real increase due to the very low gravity recovery of gold.

Gravity tailing flotation testing of the composite yielded very good gold recovery results at all the grind sizes tested. The composite gold recoveries were 99% for all three grind sizes with only a very small incremental increase observed in going to the finer grind size from the coarser grind. The composite silver recoveries ranged from 48% to 70% with a significant increase observed in going to the finer grind size from the coarser grind. The residue tailing gold grades were reported in the range of 0.04 g/t to 0.06 g/t for the composite. The combined gravity concentrate plus gravity tailing flotation concentrate gold recoveries for the composite were reported at 99% for all three tests showing no real increase due to the very low gravity recovery of gold.

The diagnostic leach program showed an initial 84.2% gold recovery of readily leachable gold. 3.2% of the gold was further extracted from possible gold associations with iron-arsenic compounds and 2.6% of the gold was further extracted from possible gold associations with weak acid soluble minerals. 7.4% of the gold was observed to be from possible gold associations with or occluded by sulphide minerals, pyrite and arsenopyrite. The remaining 2.5% of the gold remianing in the final leach residue was deemed to be the gold mainly associated with silicates or fine sulphides locked in silicates. The results from the diagnostic leach program should be viewed as an indication of general trends and possibilities only.

Appendix A – Sample Information

SGS

SAMPLE RECEIPT : 0379-FEB12

Receipt #:	0379-FEB12	**Project Number:**	
Received Date:	27-Feb-12	**Project Manager:**	DILAURO, PETER
Received By:	yanthal	**Attention:**	Pete DiLauro
		Department:	MetOps

Client Info

Name:	Mistango River Resources Inc
Contact:	New Contact
Address:	4 P.O Box, 4 Al Wende Kirkland lake, ON P2N 3J5, Phone: Fax:

Description:	2 pails of samples
Reference:	
Carrier:	Purolator
Waybill #:	329584455805,813
Payment:	PrePaid
Shipping Wt(kg):	30 lbs
Actual Wt(kg):	
Hazards:	none
Geiger Count:	0 μSv/h

Sample Preparation

Pulverized : ____ Crushed : ____

Notes

Printed Date: 27-Feb-12 13:17

0379-FEB12 Inventory

07-Mar-12
Pete DiLauro

Client: **Mistango River Resources** Labelled: Omega Samples
Project number: **13634-001**
Project Manager: **Peter DiLauro**
Sample Receipt: **0379-FEB12**
Date Received: **27-Feb-12**
Details: Two pails with six samples each.
Total weight on shipping receipt: 30 lbs
Actual total sample weight: 17.0 kg
Individual sample weights include bags

Bag ID	Sample #	Weight (kg)
Pail #1	no bag id	1.0
Pail #1	no bag id	1.5
Pail #1	no bag id	0.5
Pail #1	no bag id	1.0
Pail #1	no bag id	1.5
Pail #1	no bag id	1.5
Pail #2	no bag id	3.0
Pail #2	no bag id	1.5
Pail #2	no bag id	2.0
Pail #2	no bag id	1.5
Pail #2	no bag id	1.0
Pail #2	no bag id	1.0
Total Weight		17.0

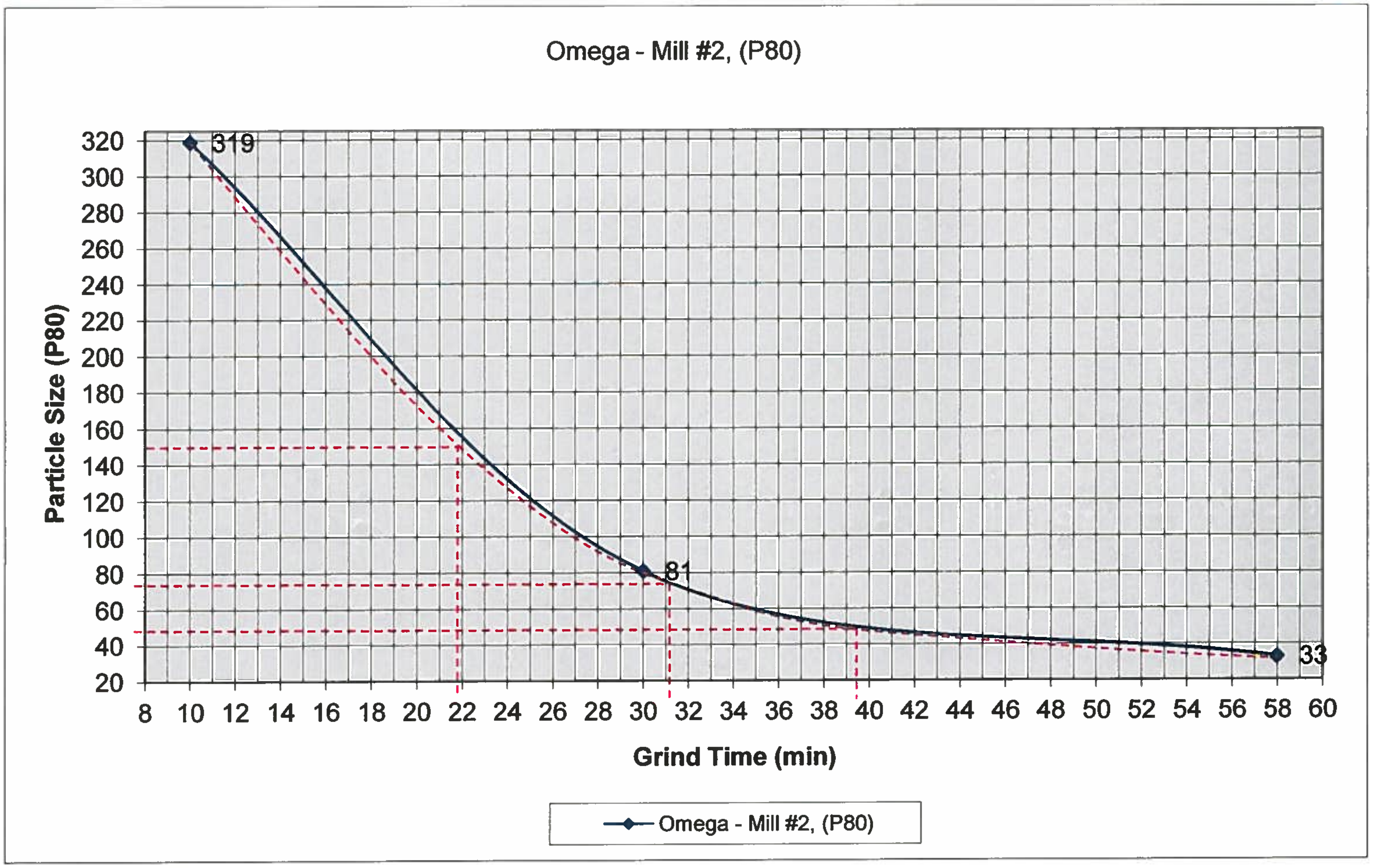
Appendix B – Metallurgical Test Details

GRIND CURVE DATA & CURVES MISTANGO RIVER - Omega, Proj.: 13634-001

120320, Michael Unger

Grind Time (min)	Omega - Mill #2, (P80)
10.00	**319**
30.00	**81**
58	**33**

Time (min)	Target (K80)	Regrind Time (min)
21.75	150	
31.25	75	9.5
39.5	50	17.75



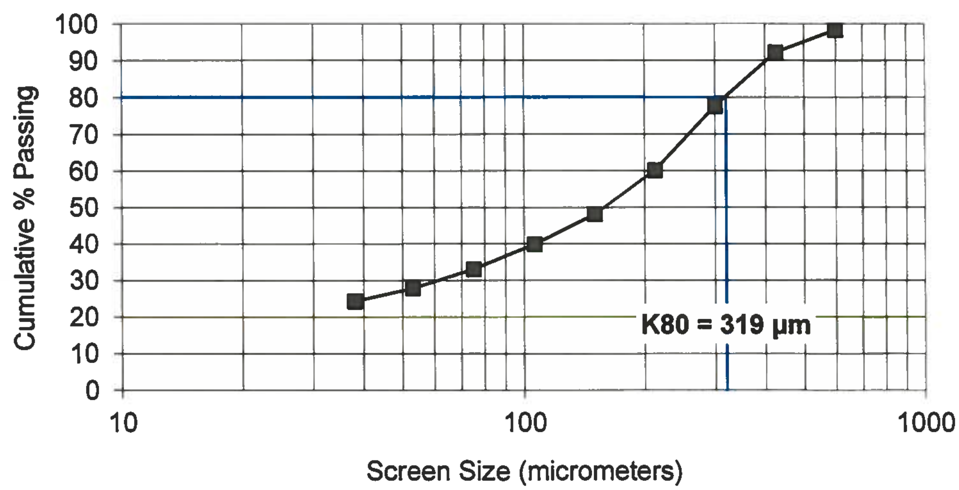
SGS Minerals Services
Size Distribution Analysis

Project No.
13634-001

Sample: **1+2 omega** Test No.: **10 min grind curve**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
28	600	2.7	1.8	1.8	98.2
35	425	9.3	6.0	7.8	92.2
48	300	22.5	14.6	22.4	77.6
65	212	26.9	17.4	39.8	60.2
100	150	18.5	12.0	51.9	48.1
150	106	12.7	8.3	60.1	39.9
200	75	10.4	6.8	66.9	33.1
270	53	8.2	5.3	72.2	27.8
400	38	5.4	3.5	75.7	24.3
Pan	-38	37.5	24.3	100.0	0.0
Total	-	**154.1**	100.0	-	-
K80	**319**				

Enter K
80



Graph
10
319
319

K80 = 319 µr

	K80	Calculations
34	24.32	732.2
33	27.80	433.9
32	33.10	387.0
31	39.87	382.8
30	48.13	329.8
29	60.16	312.7
28	77.61	319.0
27	92.20	196.6
26	98.24	#REF!
25	#REF!	#REF!
24	#REF!	#REF!
23	#REF!	#REF!

SGS Minerals Services
Size Distribution Analysis

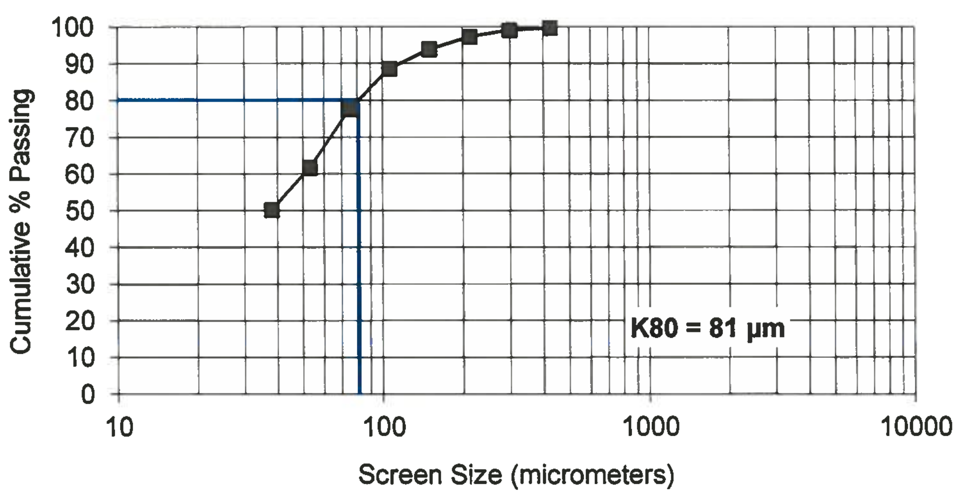
Project No.
13634-001

Sample: **1+2 omega** Test No.: **30 minute grind curve**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
35	425	0.5	0.3	0.3	99.7
48	300	0.9	0.6	0.9	99.1
65	212	2.7	1.8	2.7	97.3
100	150	5.1	3.4	6.1	93.9
150	106	8.0	5.3	11.4	88.6
200	75	16.6	11.0	22.3	77.7
270	53	24.3	16.0	38.4	61.6
400	38	17.4	11.5	49.9	50.1
Pan	-38	75.9	50.1	100.0	0.0
Total	-	**151.3**	100.0	-	-
K80	**81**				

Enter K
80

Particle Size Distribution



Graph
10
81
81

K80 = 81 µm

	K80	Calculations
34	50.14	80.7
33	61.63	78.4
32	77.65	81.1
31	88.62	57.4
30	93.90	31.4
29	97.28	5.3
28	99.09	0.0
27	99.66	#REF!
26	#REF!	#REF!
25	#REF!	#REF!
24	#REF!	#REF!
23	#REF!	#REF!
22	#REF!	#REF!

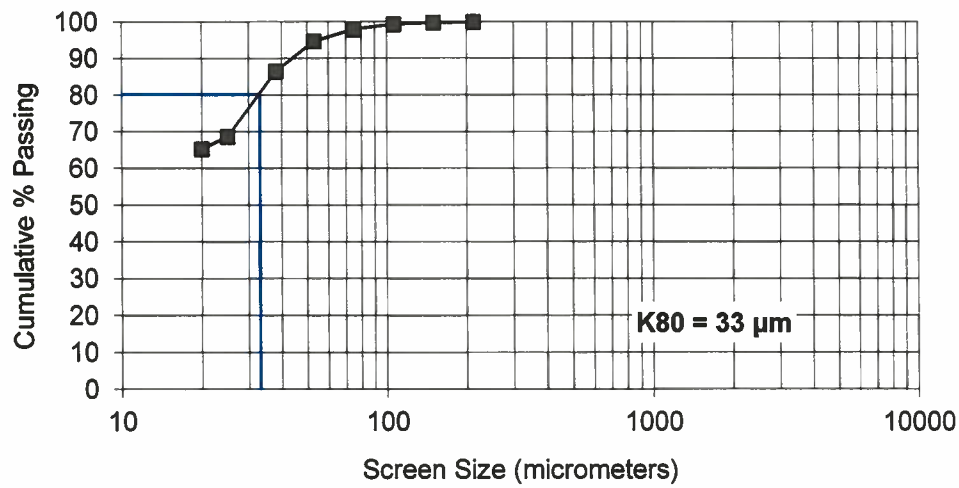
SGS Minerals Services
Size Distribution Analysis

Project No.
13634-001

Sample: **58 min**   Test No.: **Omega grind curve**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.1	0.1	0.1	99.9
100	150	0.2	0.1	0.2	99.8
150	106	0.7	0.4	0.6	99.4
200	75	2.2	1.4	2.1	97.9
270	53	5.1	3.3	5.3	94.7
400	38	12.9	8.3	13.6	86.4
500	25	27.8	17.8	31.4	68.6
635	20	5.2	3.3	34.8	65.2
Pan	-20	101.7	65.2	100.0	0.0
Total	-	**155.9**	100.0	-	-
K80	**33**				

Enter K
80



Graph
10
33
33

K80 = 33 µm

	K80	Calculations
34	65.23	49.8
33	68.57	33.1
32	86.40	28.7
31	94.68	9.5
30	97.95	0.6
29	99.36	0.0
28	99.81	0.0
27	99.94	#REF!
26	#REF!	#REF!
25	#REF!	#REF!
24	#REF!	#REF!
23	#REF!	#REF!
22	#REF!	#REF!

MALVERN INSTRUMENTS

MASTERSIZER 2000

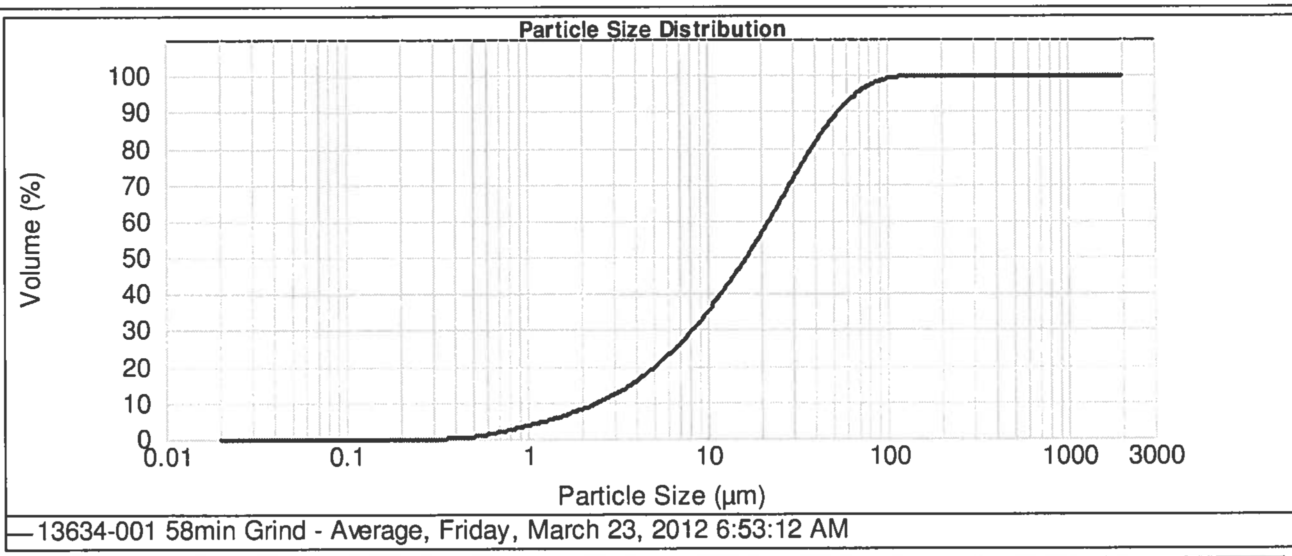
Result Analysis Report

Sample Name: 13634-001 58min Grind - Average	**SOP Name:** default	**Measured:** Friday, March 23, 2012 6:53:12 AM
Sample Source & type: Factory	**Measured by:** lr_hydro1	**Analysed:** Friday, March 23, 2012 6:53:27 AM
Sample bulk lot ref: ar	**Result Source:** Averaged	

Particle Name: Default	**Accessory Name:** Hydro 2000G (A)	**Analysis model:** General purpose	**Sensitivity:** Normal
Particle RI: 1.520	**Absorption:** 0.1	**Size range:** 0.020 to 2000.000 um	**Obscuration:** 12.79 %
Dispersant Name: Water	**Dispersant RI:** 1.330	**Weighted Residual:** 1.180 %	**Result Emulation:** Off

Concentration: 0.0122 %Vol	**Span :** 3.058	**Uniformity:** 0.968	**Result units:** Volume
Specific Surface Area: 1.02 m^2/g	**Surface Weighted Mean D[3,2]:** 5.881 um	**Vol. Weighted Mean D[4,3]:** 23.394 um	

d(0.1): 2.506 um   d(0.5): 16.688 um   **D(0.80) : 38.16 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	4.15	11.482	38.61	120.226	99.66	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	4.93	13.183	42.58	138.038	99.89	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	5.75	15.136	46.83	158.489	99.99	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	6.63	17.378	51.36	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	7.62	19.953	56.16	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	8.74	22.909	61.19	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	10.02	26.303	66.37	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	11.48	30.200	71.57	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	13.15	34.674	76.65	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	15.04	39.811	81.43	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.08	4.365	17.16	45.709	85.75	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.33	5.012	19.53	52.481	89.51	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.75	5.754	22.14	60.256	92.63	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	1.29	6.607	24.99	69.183	95.10	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	1.94	7.586	28.06	79.433	96.96	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	2.65	8.710	31.36	91.201	98.28	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	3.39	10.000	34.87	104.713	99.14	1096.478	100.00		

Operator notes:

CN-1 **13634-001** **Omega-150** **Michael Unger** **April 9, 2012**

Purpose: To access the amenability to the Cyanide leach process on approximately 1 kg of an Omega Sample at P80 particle size target of 150 µm.

Procedure: Bottle roll cyanidation test were completed on ~1 kg charge of Omega Sample at conditions applied below. CN Tests are completed after grinding to 150 µm target. Pregnant solution sub-samples were removed & assayed for Au and Ag at ~7, 24 & 48 hrs. Leach residue samples were dried, weighed and sampled in duplicate for Au and Ag analysis and a separate sample was taken for confirmatory size analysis. The conditions applied are outlined below.

Feed: 1,000 g Omega sample at ~150 µm

Solution Volume: 1,500 mL

Pulp Density: 40 % solids

Sol'n Composition: 0.5 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Grind: G.G.2kg Mill#2 21.75 min. P_{80} 112

Reagent Addition (kg/t of cyanide feed) NaCN: 1.24 CaO: 0.465

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.53 CaO: 0.453

Time	Added, Grams				Residual		Consumed		pH	D.O
	Actual		Equivalent		Grams		Grams			mg/L
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									8.1	
0-1	0.789	0.397	0.750	0.294	0.495		0.255		10.8 - 10.9	
1-3	0.268	0.000	0.255	0.000	0.585		0.165		10.9 - 10.8	6.1
3-7	0.174	0.000	0.165	0.000	0.735		0.015		10.8 - 10.6	7.6
7-24	0.016	0.035	0.015	0.026	0.735		0.015		10.9 - 10.0	
24-31	0.016	0.165	0.015	0.122	0.705		0.045		10.9 - 10.6	
31-48	0.047	0.032	0.045	0.024	0.710	0.012	0.040	0.012	10.8 - 10.4	
Total	1.31	0.63	1.25	0.47	0.71	0.01	0.53	0.45		

Cyanidation Results:

Product	Amount g, mL	Assays, mg/L, g/t, % Au	% Distribution Au
7 h PLS	1,522	1.71	77.0
24 h PLS	1,500	1.73	76.7
48 h PLS	1,479	1.71	76.0
Final Residue	1,001	0.81	24.0
Head (calc.)	1,001	3.38	100
Head (direct)	1,000	3.58	

Final Residue Assays

	Au
A	0.82
B	0.80
Avg	**0.81**

Other Assays;

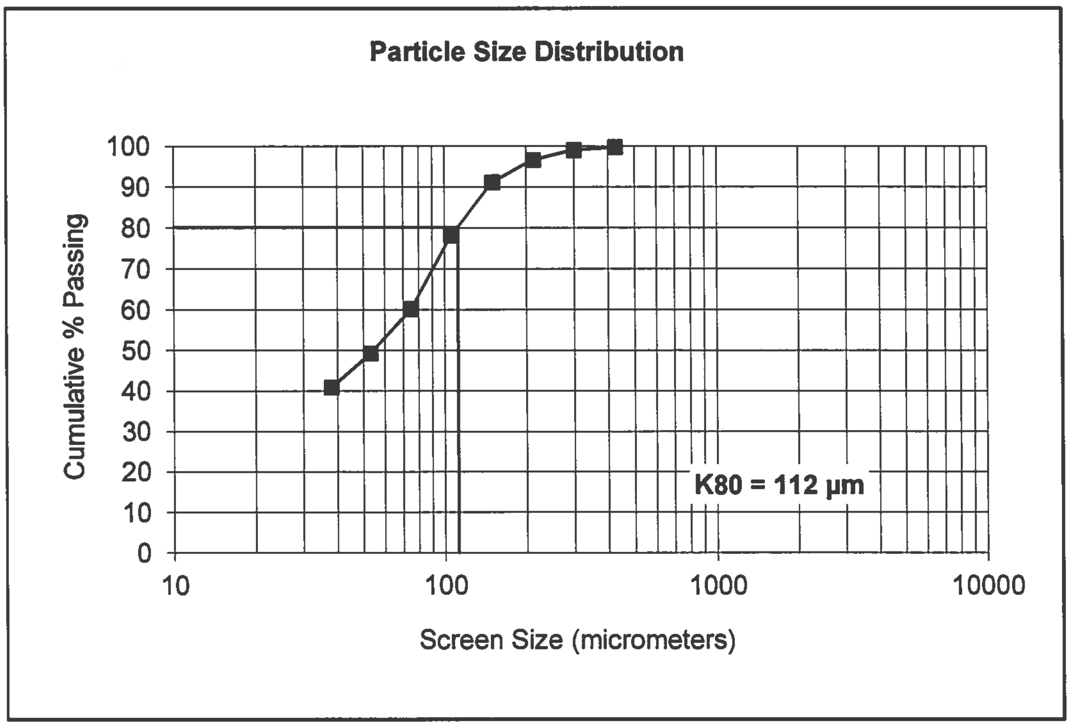
Assay, mg/L, g/t PLS's	Ag
7hr	0.41
24hr	0.43
48hr	0.45
Residue	<10

SGS Minerals Services
Size Distribution Analysis

Project No.
13634-001

Sample: **WOCN Res** Test No.: **CN 1 150**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
35	425	0.4	0.3	0.3	99.7
48	300	1.0	0.7	0.9	99.1
65	212	3.7	2.4	3.3	96.7
100	150	8.4	5.5	8.9	91.1
150	106	19.7	12.9	21.8	78.2
200	75	27.5	18.0	39.8	60.2
270	53	16.6	10.9	50.7	49.3
400	38	12.8	8.4	59.1	40.9
Pan	-38	62.3	40.9	100.0	0.0
Total	**-**	**152.4**	100.0	-	-
K80	**112**				



	K80	Calculations
34	40.88	125.6
33	49.28	123.1
32	60.17	109.2
31	78.22	111.6
30	91.14	69.6
29	96.65	15.0
28	99.08	0.0
27	99.74	#REF!

CN-2 | **13634-001** | **Omega-75** | **Michael Unger** | **April 9, 2012**

Purpose: To access the amenability to the Cyanide leach process on approximately 1 kg of an Omega Sample at P80 particle size target of 75 µm.

Procedure: Bottle roll cyanidation test were completed on ~1 kg charge of an Omega Sample at conditions applied below. CN Tests are completed after grinding to 75 µm target. Pregnant solution sub-samples were removed & assayed for Au and Ag at ~7, 24 & 48 hrs. Leach residue samples were dried, weighed and sampled in duplicate for Au and Ag analysis and a separate sample was taken for confirmatory size analysis. The conditions applied are outlined below.

Feed: 1,000 g Omega sample at ~75 µm

Solution Volume: 1,500 mL

Pulp Density: 40 % solids

Sol'n Composition: 0.5 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Grind: G.G.2kg Mill#2 31.25 min. P_{80} 68

Reagent Addition (kg/t of cyanide feed) NaCN: 1.67 CaO: 0.437
Reagent Consumption (kg/t of cyanide feed) NaCN: 1.04 CaO: 0.425

Time	Added, Grams				Residual		Consumed		pH	D.O
	Actual		Equivalent		Grams		Grams			mg/L
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									8.3	
0-1	0.789	0.412	0.750	0.305	0.345		0.405		10.8 - 11.1	
1-3	0.426	0.000	0.405	0.000	0.270		0.480		11.1 - 11.3	2.4
3-7	0.505	0.000	0.480	0.000	0.750		0.000		11.3 - 11.1	7.0
7-24	0.000	0.000	0.000	0.000	0.705		0.045		11.1 - 10.1	
24-31	0.047	0.141	0.045	0.104	0.750		0.000		10.8 - 10.5	
31-48	0.000	0.040	0.000	0.030	0.631	0.012	0.119	0.018	10.8 - 10.4	
Total	1.77	0.59	1.68	0.44	0.63	0.01	1.0490	0.43		

Cyanidation Results:

Product	Amount g, mL	Assays, mg/L, g/t, % Au
7 h PLS	1,517	1.96
24 h PLS	1,491	1.97
48 h PLS	1,469	1.95
Final Residue	1,004	0.59
Head (calc.)	1,004	3.49
Head (direct)	1,000	3.58

% Distribution Au
84.9
83.9
83.2
16.8
100

Final Residue Assays

	Au
A	0.59
B	0.58
Avg	**0.59**

Other Assays;

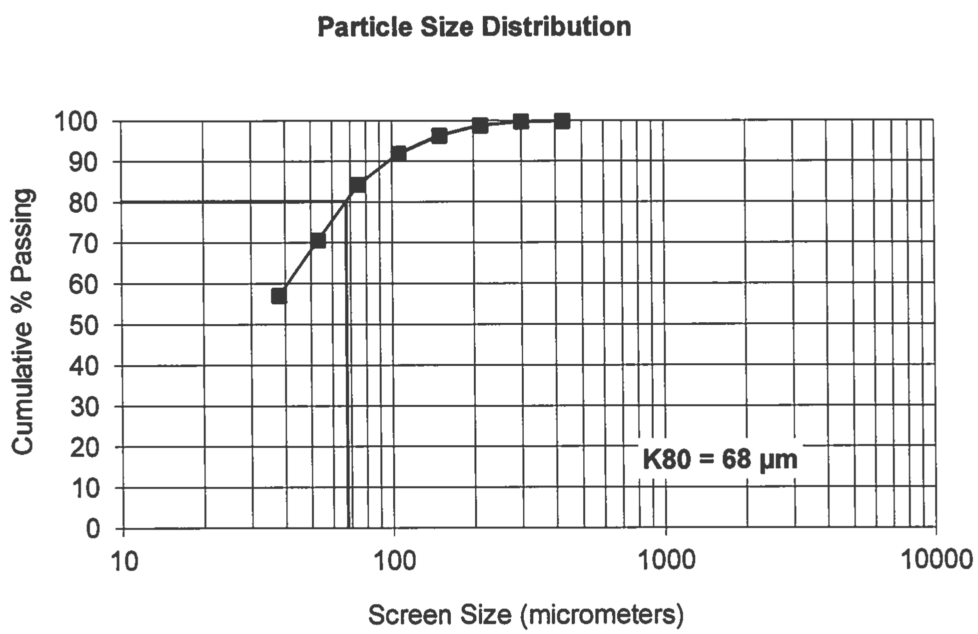
Assay, mg/L, g/t PLS's	Ag
7hr	0.47
24hr	0.54
48hr	0.49
Residue	<10

SGS Minerals Services
Size Distribution Analysis

Project No.
13634-001

Sample: **WOCN Res** Test No.: **CN 2 75**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
35	425	0.2	0.1	0.1	99.9
48	300	0.2	0.1	0.2	99.8
65	212	1.5	0.9	1.2	98.8
100	150	4.1	2.5	3.7	96.3
150	106	7.2	4.4	8.1	91.9
200	75	12.4	7.6	15.7	84.3
270	53	22.3	13.7	29.4	70.6
400	38	22.1	13.5	42.9	57.1
Pan	-38	93.1	57.1	100.0	0.0
Total	-	**163.1**	100.0	-	-
K80	**68**				



	K80	Calculations
34	57.08	64.4
33	70.63	67.7
32	84.30	60.8
31	91.91	38.0
30	96.32	12.4
29	98.84	0.1
28	99.75	0.0
27	99.88	#REF!

CN-3 | **13634-001** | **Omega-50** | **Michael Unger** | **April 9, 2012**

Purpose: To access the amenability to the Cyanide leach process on approximately 1 kg of an Omega Sample at P80 particle size target of 50 μm.

Procedure: Bottle roll cyanidation test were completed on ~1 kg charge of an Omega Sample at conditions applied below. CN Tests are completed after grinding to 50 μm target. Pregnant solution sub-samples were removed & assayed for Au and Ag at ~7, 24 & 48 hrs. Leach residue samples were dried, weighed and sampled in duplicate for Au and Ag analysis and a separate sample was taken for confirmatory size analysis. The conditions applied are outlined below.

Feed: 1,000 g Omega sample at ~50 μm

Solution Volume: 1,500 mL

Pulp Density: 40 % solids

Sol'n Composition: 0.5 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Grind: G.G.2kg Mill#2 39.5 min. P_{80} 57

Reagent Addition (kg/t of cyanide feed) NaCN: 2.10 CaO: 0.433
Reagent Consumption (kg/t of cyanide feed) NaCN: 1.38 CaO: 0.402

Time	Added, Grams				Residual		Consumed		pH	D.O
	Actual		Equivalent		Grams		Grams			mg/L
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									8.4	
0-1	0.789	0.393	0.750	0.291	0.195		0.555		10.8 - 11.1	
1-3	0.584	0.000	0.555	0.000	0.255		0.495		11.1 - 11.4	0.7
3-7	0.521	0.000	0.495	0.000	0.495		0.255		11.4 - 11.2	5.9
7-24	0.268	0.000	0.255	0.000	0.720		0.030		11.2 - 10.2	
24-31	0.032	0.134	0.030	0.099	0.720		0.030		10.8 - 10.5	
31-48	0.032	0.061	0.030	0.045	0.728	0.031	0.022	0.014	10.8 - 10.3	
Total	2.23	0.59	2.12	0.44	0.73	0.03	1.39	0.40		

Cyanidation Results:

Product	**Amount**	**Assays, mg/L, g/t, %**	**% Distribution**
	g, mL	**Au**	**Au**
7 h PLS	1,563	1.96	80.2
24 h PLS	1,544	2.08	84.1
48 h PLS	1,562	2.06	85.5
Final Residue	1,006	0.55	14.5
Head (calc.)	1,006	3.80	100
Head (direct)	1,000	3.58	

Final Residue Assays

	Au	
A	0.55	
B	0.55	
Avg	**0.55**	

Other Assays;

Assay, mg/L, g/t	
PLS's	**Ag**
7hr	0.46
24hr	0.53
48hr	0.53
Residue	<10

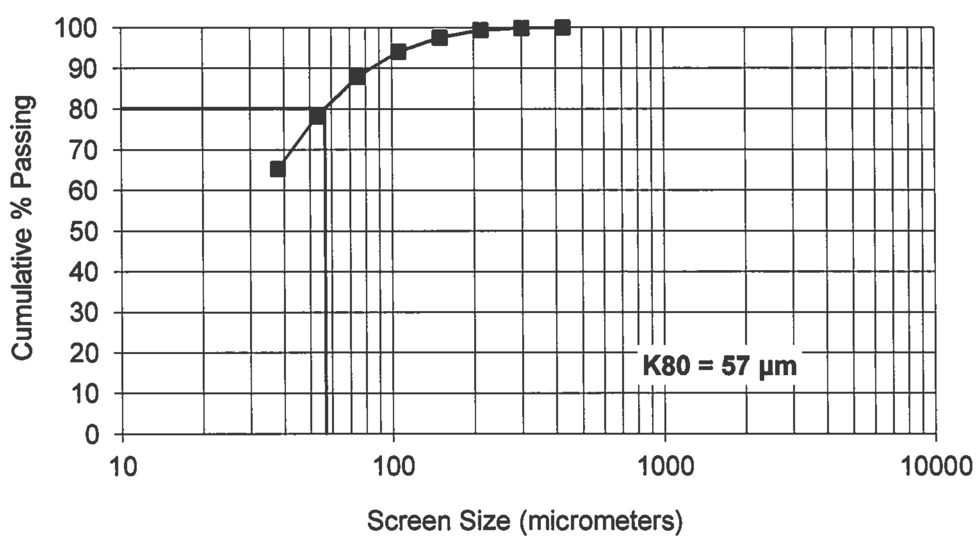
SGS Minerals Services
Size Distribution Analysis

Project No.
13364-001

Sample: **WOCN Residue** CN-3, 50

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
35	425	0.0	0.0	0.0	100.0
48	300	0.2	0.1	0.1	99.9
65	212	0.7	0.5	0.6	99.4
100	150	2.6	1.8	2.5	97.5
150	106	4.9	3.5	5.9	94.1
200	75	8.6	6.1	12.0	88.0
270	53	13.8	9.7	21.7	78.3
400	38	18.3	12.9	34.7	65.3
Pan	-38	92.6	65.3	100.0	0.0
Total	-	**141.7**	100.0	-	-
K80	**57**				

Particle Size Distribution



	K80	Calculations
34	65.35	55.2
33	78.26	56.6
32	88.00	45.7
31	94.07	22.3
30	97.53	3.8
29	99.36	0.0
28	99.86	0.0
27	100.00	#REF!

Test No.: G-1 | Project: **13634-001, Mistango** | **Operator:** Michael Unger | **April 9, 2012**

Purpose: To generate a gravity concentrate for assay and a gravity tailing feed for downstream cyanidation testwork in comparison to whole ore CN testing also evaluating the impact of grind size. Additional Tailing samples will be used for Flotation test work.

Procedure: The ore was ground in a 2 kg batch rod mill, 1kg at a time as per results from 1kg grind curve data, and passed together through the Knelson MD-3 concentrator. The concentrate was recovered and further upgraded on the Mozley Laboratory Separator (MLS). The final gravity concentrate was submitted for Au and Ag assay (to extinction), the Mozley tailing was combined with the Knelson tailing, for subsequent splitting and testwork including cyanidation.

Feed: 10-kg of Omega Sample to target 150 μm grind.

Grind: 21.75 mins. @ 65% solids P_{80} = 125 μm

Conditions:

Metallurgical Balance

Product		Weight		Assays, g/t	% Distribution
		g	%	Au	Au
Mozley Concentrate		10.248	0.10	109	3.1
Knelson + Mozley Tailing		9,990	99.9	*3.48	96.9
Head	Calc	10,000	100.0	3.58	100.0
	Direct			3.58	

* = average of the below G.T. Heads Assays.

Gravity Tails Head Assay	Au (g/t)
Cut A	3.57
Cut B	3.40
Cut C	3.46
Avg. =	3.48

Other Assays;

PLS's	Assay, mg/L, g/t Ag
Mozley Concentrate	31.0

CN-4 **13634-001** **Omega-150** **Michael Unger** **April 9, 2012**

Purpose: To access the amenability to the Cyanide leach process on approximately 1 kg of an Omega Sample at P80 particle size of 150 µm.

Procedure: Bottle roll cyanidation test were completed on ~1 kg charge of Omega Sample at conditions applied below. CN Tests are completed after grinding to 150 µm target. Pregnant solution sub-samples were removed & assayed for Au and Ag at ~7, 24 & 48 hrs. Leach residue samples were dried, weighed and sampled in duplicate for Au and Ag analysis and a separate sample was taken for confirmatory size analysis. The conditions applied are outlined below.

Feed: 1,000 g Omega sample at ~150 µm

Solution Volume: 1,500 mL

Pulp Density: 40 % solids

Sol'n Composition: 0.5 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Grind: G.G.2kg Mill#2 21.75 min. P_{80} 125

Reagent Addition (kg/t of cyanide feed) NaCN: 1.27 CaO: 0.473

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.54 CaO: 0.461

Time	Added, Grams				Residual		Consumed		pH	D.O
	Actual		Equivalent		Grams		Grams			mg/L
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									8.1	
0-1	0.789	0.397	0.750	0.294	0.495		0.255		10.8 - 10.9	
1-3	0.268	0.000	0.255	0.000	0.585		0.165		10.9 - 10.8	6.1
3-7	0.174	0.000	0.165	0.000	0.735		0.015		10.8 - 10.6	7.6
7-24	0.016	0.035	0.015	0.026	0.735		0.015		10.9 - 10.0	
24-31	0.016	0.165	0.015	0.122	0.705		0.045		10.9 - 10.6	
31-48	0.047	0.032	0.045	0.024	0.719	0.012	0.031	0.012	10.8 - 10.4	
Total	1.31	0.63	1.25	0.47	0.72	0.01	0.53	0.45		

Cyanidation Results:

Product	Amount	Assays, mg/L, g/t, %
	g, mL	Au
7 h PLS	1,540	1.58
24 h PLS	1,518	1.61
48 h PLS	1,497	1.59
Final Residue	983	0.86
Head (calc.)	983	3.32
Head (direct)	1,000	3.48

% Distribution
Au
74.6
75.0
74.2
25.8
100

Final Residue Assays

	Au
A	0.85
B	0.86
Avg	**0.86**

Other Assays;

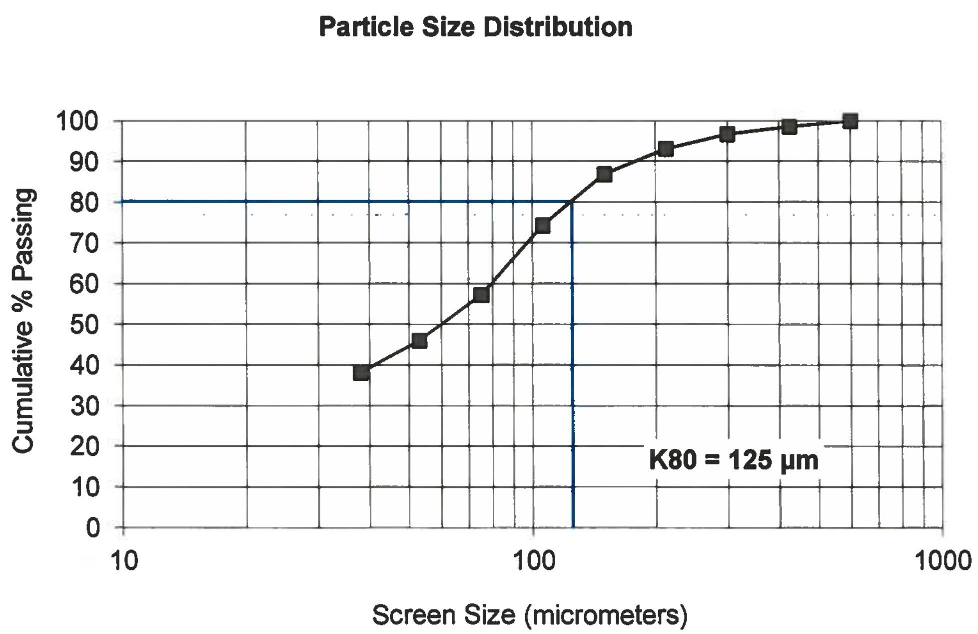
Assay, mg/L, g/t	
PLS's	Ag
7hr	0.36
24hr	0.38
48hr	0.39
Residue	<0.5

SGS Minerals Services
Size Distribution Analysis

Project No.
13634-001

Sample: **GT CN Leach Res** Test No.: **CN 4, 150**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
28	600	0.0	0.0	0.0	100.0
35	425	2.2	1.4	1.4	98.6
48	300	2.9	1.9	3.3	96.7
65	212	5.6	3.6	6.9	93.1
100	150	9.6	6.2	13.2	86.8
150	106	19.3	12.5	25.7	74.3
200	75	26.5	17.2	42.8	57.2
270	53	17.2	11.1	54.0	46.0
400	38	12.2	7.9	61.9	38.1
Pan	-38	58.8	38.1	100.0	0.0
Total	-	**154.3**	100.0	-	-
K80	**125**				



CN-5 **13634-001** **Omega-75** **Michael Unger** **April 9, 2012**

Purpose: To access the amenability to the Cyanide leach process on approximately 1 kg of an Omega Sample at P80 particle size of 75 µm.

Procedure: Bottle roll cyanidation test were completed on ~1 kg charge of an Omega Sample at conditions applied below. CN Tests are completed after grinding to 75 µm target. Pregnant solution sub-samples were removed & assayed for Au and Ag at ~7, 24 & 48 hrs. Leach residue samples were dried, weighed and sampled in duplicate for Au and Ag analysis and a separate sample was taken for confirmatory size analysis. The conditions applied are outlined below.

Feed: 1,000 g Omega sample at ~75 µm

Solution Volume: 1,500 mL

Pulp Density: 40 % solids

Sol'n Composition: 0.5 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Grind: G.G.2kg Mill#2 31.25 min. P_{80} 85

Reagent Addition (kg/t of cyanide feed) NaCN: 1.71 CaO: 0.447

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.06 CaO: 0.435

Time	Added, Grams				Residual		Consumed		pH	D.O mg/L
	Actual		Equivalent		Grams		Grams			
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									8.3	
0-1	0.789	0.412	0.750	0.305	0.345		0.405		10.8 - 11.1	
1-3	0.426	0.000	0.405	0.000	0.270		0.480		11.1 - 11.3	2.4
3-7	0.505	0.000	0.480	0.000	0.750		0.000		11.3 - 11.1	7.0
7-24	0.000	0.000	0.000	0.000	0.705		0.045		11.1 - 10.1	
24-31	0.047	0.141	0.045	0.104	0.750		0.000		10.8 - 10.5	
31-48	0.000	0.040	0.000	0.030	0.641	0.012	0.109	0.018	10.8 - 10.4	
Total	1.77	0.59	1.68	0.44	0.64	0.01	1.0394	0.43		

Cyanidation Results:

Product	Amount g, mL	Assays, mg/L, g/t, % Au
7 h PLS	1,539	1.77
24 h PLS	1,513	1.76
48 h PLS	1,491	1.74
Final Residue	982	0.65
Head (calc.)	982	3.34
Head (direct)	1,000	3.48

% Distribution Au
83.1
81.3
80.5
19.5
100

Final Residue Assays

	Au
A	0.66
B	0.64
Avg	**0.65**

Other Assays;

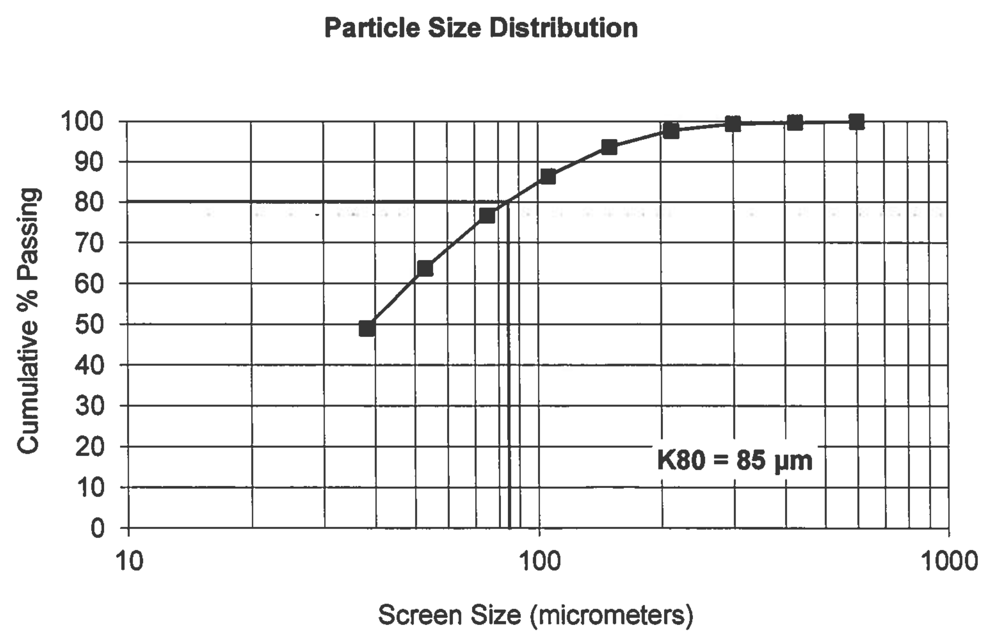
Assay, mg/L, g/t PLS's	Ag
7hr	0.42
24hr	0.43
48hr	0.40
Residue	<0.5

SGS Minerals Services
Size Distribution Analysis

Project No.
13634-001

Sample: **GT CN Leach Res** Test No.: **CN 5, 75**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
28	600	0.0	0.0	0.0	100.0
35	425	0.4	0.3	0.3	99.7
48	300	0.5	0.3	0.6	99.4
65	212	2.6	1.7	2.3	97.7
100	150	6.2	4.0	6.3	93.7
150	106	11.2	7.2	13.5	86.5
200	75	15.0	9.7	23.2	76.8
270	53	20.1	13.0	36.2	63.8
400	38	22.8	14.7	51.0	49.0
Pan	-38	75.8	49.0	100.0	0.0
Total	**-**	**154.6**	100.0	-	-
K80	**85**				



CN-6 **13634-001** **Omega-50** **Michael Unger** **April 9, 2012**

Purpose: To access the amenability to the Cyanide leach process on approximately 1 kg of an Omega Sample at P80 particle size of 50 µm.

Procedure: Bottle roll cyanidation test were completed on ~1 kg charge of an Omega Sample at conditions applied below. CN Tests are completed after grinding to 50 µm target. Pregnant solution sub-samples were removed & assayed for Au and Ag at ~7, 24 & 48 hrs. Leach residue samples were dried, weighed and sampled in duplicate for Au and Ag analysis and a separate sample was taken for confirmatory size analysis. The conditions applied are outlined below.

Feed: 1,000 g Omega sample at ~50 µm

Solution Volume: 1,500 mL

Pulp Density: 40 % solids

Sol'n Composition: 0.5 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Grind: G.G.2kg Mill#2 39.5 min. P_{80} 52

Reagent Addition (kg/t of cyanide feed) NaCN: 2.12 CaO: 0.437

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.39 CaO: 0.405

Time	Added, Grams				Residual		Consumed		pH	D.O
	Actual		Equivalent		Grams		Grams			mg/L
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									8.4	
0-1	0.789	0.393	0.750	0.291	0.195		0.555		10.8 - 11.1	
1-3	0.584	0.000	0.555	0.000	0.255		0.495		11.1 - 11.4	0.7
3-7	0.521	0.000	0.495	0.000	0.495		0.255		11.4 - 11.2	5.9
7-24	0.268	0.000	0.255	0.000	0.720		0.030		11.2 - 10.2	
24-31	0.032	0.134	0.030	0.099	0.720		0.030		10.8 - 10.5	
31-48	0.032	0.061	0.030	0.045	0.732	0.031	0.018	0.014	10.8 - 10.3	
Total	2.23	0.59	2.12	0.44	0.73	0.03	1.38	0.40		

Cyanidation Results:

Product	Amount	Assays, mg/L, g/t, %	% Distribution
	g, mL	Au	Au
7 h PLS	1,573	1.83	82.0
24 h PLS	1,554	1.87	82.8
48 h PLS	1,572	1.84	83.7
Final Residue	996	0.58	16.3
Head (calc.)	996	3.52	100
Head (direct)	1,000	3.48	

Final Residue Assays

	Au
A	0.55
B	0.60
Avg	**0.58**

Other Assays;

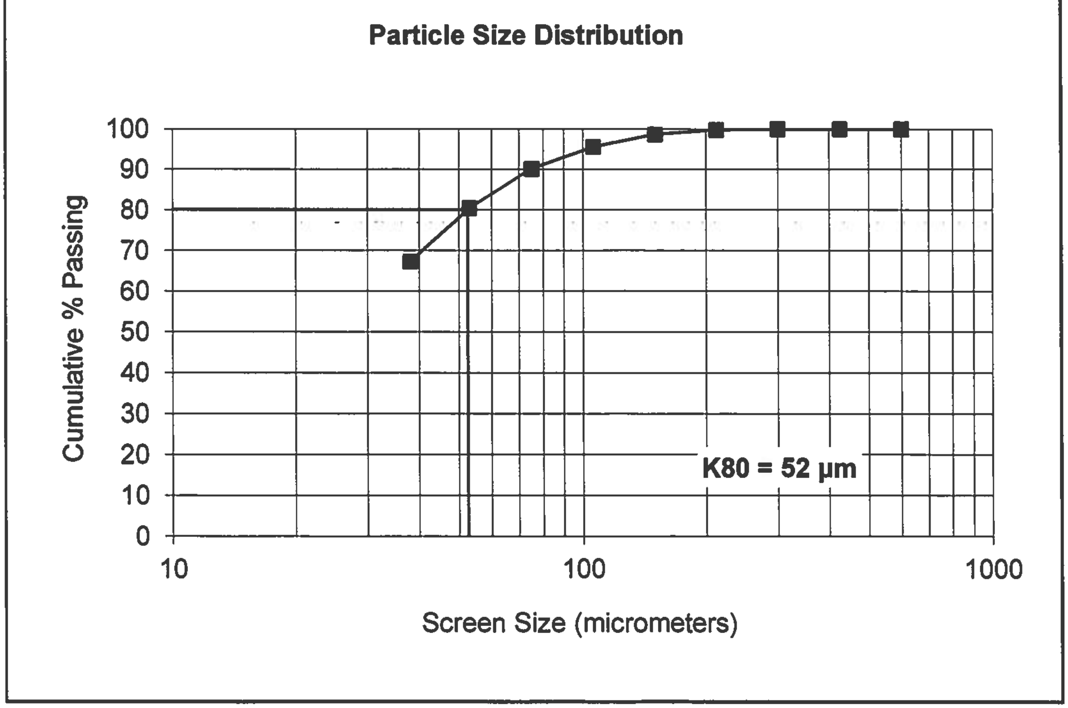
Assay, mg/L, g/t	
PLS's	Ag
7hr	0.43
24hr	0.42
48hr	0.44
Residue	<0.5

SGS Minerals Services
Size Distribution Analysis

Project No.
13634-001

Sample: **GT CN Leach Res** Test No.: **CN 6, 50**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
28	600	0.0	0.0	0.0	100.0
35	425	0.0	0.0	0.0	100.0
48	300	0.0	0.0	0.0	100.0
65	212	0.3	0.2	0.2	99.8
100	150	1.7	1.1	1.3	98.7
150	106	4.7	3.1	4.4	95.6
200	75	8.2	5.4	9.8	90.2
270	53	14.8	9.7	19.6	80.4
400	38	20.0	13.2	32.7	67.3
Pan	-38	102.1	67.3	100.0	0.0
Total	-	**151.8**	100.0	-	-
K80	**52**				



Test: F1 **13634-001** Operator:GT April 4, 2012

Purpose: To scope the flotation of sulphide and gold in the gravity tailing.

Procedure: Flotation was conducted as described below on a 2-kg gravity tailing.
The products were sent for Au, Ag, and S= assays.

Feed: 2 kg Omega Gravity Tails

Grind: as received

Target P_{80} = 150 µm.
Actual P_{80} = 125 µm.

Conditions:

Stage	Reagents added, grams per tonne			Time, minutes			pH	EMF (mv)
	A208	PAX	MIBC	Grind	Cond.	Froth		
Grind:				0			8.1	23
Rougher 1	5	20	10		1	2.0	8.3	154
Rougher 2	5	20	5		1	3.0	8.4	106
Rougher 3	5	20	5		1	5.0	8.3	77
Rougher 4	5	20	5		1	5.0	8.2	62
Rougher 5	5	20	5		1	5.0	8.4	43
Total	25	100	30			20.0		

Stage	Rougher
Flotation Cell	2000 g D-12
Speed: r.p.m.	1500

Observation:

Metallurgical Balance (Flotation)

Product	Mass		Assays, g/t, %			% Distribution		
	g	%	Au	Ag	$S^=$	Au	Ag	$S^=$
Ro Conc 1	187.4	9.2	34.80	1.8	34.6	89.9	25.3	81.4
Ro Conc 2	138.7	6.8	3.85	1.0	8.23	7.4	10.4	14.3
Ro Conc 3	142.9	7.0	0.52	<0.5	1.12	1.0	5.4	2.0
Ro Conc 4	104.3	5.1	0.25	<0.5	0.48	0.4	3.9	0.6
Ro Conc 5	78.9	3.9	0.21	<0.5	0.30	0.2	3.0	0.3
Ro Tail	1384.7	68.0	0.06	<0.5	0.08	1.15	52.0	1.4
Head (calc)	2,037	100.00	3.56	0.65	3.91	100.0	100.0	100.0
Head (direct)								

Combined Products (Flotation)

Product	Mass		Assays, g/t, %			% Distribution		
	g	%	Au	Ag	$S^=$	Au	Ag	$S^=$
Ro Conc 1	187.4	9.2	34.80	1.80	34.60	89.9	25.3	81.4
Ro Conc 1+2	326.1	16.0	21.64	1.46	23.38	97.2	35.8	95.7
Ro Conc 1-3	469.0	23.0	15.20	1.17	16.60	98.3	41.1	97.7
Ro Conc 1-4	573.3	28.1	12.48	1.05	13.67	98.6	45.0	98.3
Ro Conc 1-5	652.2	32.0	11.00	0.98	12.05	98.9	48.0	98.6

% Recovery Au Gravity = 3.1
Overall % Recovery Au (Grav+Flot) = 98.9

Test: F2 **13634-001** Operator:GT April 4, 2012

Purpose: To scope the flotation of sulphide and gold in the gravity tailing.

Procedure: Flotation was conducted as described below on a 2-kg gravity tailing.
The products were sent for Au, Ag, and S= assays.

Feed: 2 kg Omega Gravity Tails

Grind: 9.5 min/1 kg in 2 kg Mill #2 Target P_{80} = 75 µm.
Actual P_{80} = 85 µm.

Conditions:

Stage	Reagents added, grams per tonne			Time, minutes			pH	EMF (mv)
	A208	PAX	MIBC	Grind	Cond.	Froth		
Grind:				9.5			8.1	-12
Rougher 1	5	20	10		1	2.0	8.2	-20
Rougher 2	5	20	5		1	3.0	8.3	-20
Rougher 3	5	20	5		1	5.0	8.3	-19
Rougher 4	5	20	5		1	5.0	8.2	-17
Rougher 5	5	20	5		1	5.0	8.3	-18
Total	25	100	30			20.0		

Stage	Rougher
Flotation Cell	2000 g D-12
Speed: r.p.m.	1500

Observation:

Metallurgical Balance (Flotation)

Product	Mass		Assays, g/t, %			% Distribution		
	g	%	Au	Ag	$S^=$	Au	Ag	$S^=$
Ro Conc 1	203.4	10.2	32.00	4.9	31.8	93.9	51.8	86.5
Ro Conc 2	107.1	5.4	2.40	0.8	6.20	3.7	4.5	8.9
Ro Conc 3	137.8	6.9	0.48	<0.5	1.42	1.0	3.6	2.6
Ro Conc 4	114.6	5.7	0.23	<0.5	0.48	0.4	3.0	0.7
Ro Conc 5	101.6	5.1	0.12	<0.5	0.30	0.2	2.6	0.4
Ro Tail	1329.4	66.7	0.05	<0.5	0.05	0.86	34.5	0.9
Head (calc)	1,994	100.0	3.48	0.96	3.75	100.0	100.0	100.0
Head (direct)								

Combined Products (Flotation)

Product	Mass		Assays, g/t, %			% Distribution		
	g	%	Au	Ag	$S^=$	Au	Ag	$S^=$
Ro Conc 1	203.4	10.2	32.00	4.90	31.80	93.9	51.8	86.5
Ro Conc 1+2	310.5	15.6	21.79	3.49	22.97	97.6	56.3	95.4
Ro Conc 1-3	448.3	22.5	15.24	2.57	16.35	98.6	59.8	98.0
Ro Conc 1-4	562.9	28.2	12.18	2.15	13.12	99.0	62.8	98.7
Ro Conc 1-5	664.5	33.3	10.34	1.90	11.16	99.1	65.5	99.1

% Recovery Au Gravity = 3.1
Overall % Recovery Au (Grav+Flot) = 99.2

Test: F3 **13634-001** Operator:GT April 4, 2012

Purpose: To scope the flotation of sulphide and gold in the gravity tailing.

Procedure: Flotation was conducted as described below on a 2-kg gravity tailing.
The products were sent for Au, Ag, and S= assays.

Feed: 2 kg Omega Gravity Tails

Grind: 17.75 min/1 kg in 2 kg Mill #2

Target P_{80} = 50 µm.
Actual P_{80} = 52 µm.

Conditions:

Stage	Reagents added, grams per tonne			Time, minutes			pH	EMF (mv)
	A208	PAX	MIBC	Grind	Cond.	Froth		
Grind:				17.75			8.2	-19
Rougher 1	5	20	15		1	2.0	8.4	-19
Rougher 2	5	20	5		1	3.0	8.4	-19
Rougher 3	5	20	5		1	5.0	8.4	-17
Rougher 4	5	20	5		1	5.0	8.4	-13
Rougher 5	5	20	5		1	5.0	8.2	-8
Total	25	100	35			20.0		

Stage	Rougher
Flotation Cell	2000 g D-12
Speed: r.p.m.	1500

Observation:

Metallurgical Balance (Flotation)

Product	Mass		Assays, g/t, %			% Distribution		
	g	%	Au	Ag	$S^=$	Au	Ag	$S^=$
Ro Conc 1	191.5	10.4	35.70	5.9	29.9	90.7	56.9	71.7
Ro Conc 2	156.5	8.5	3.69	0.7	12.30	7.7	5.5	24.1
Ro Conc 3	172.6	9.4	0.32	<0.5	1.08	0.7	4.3	2.3
Ro Conc 4	80.9	4.4	0.18	<0.5	0.67	0.2	2.0	0.7
Ro Conc 5	47.9	2.6	0.18	<0.5	0.59	0.1	1.2	0.4
Ro Tail	1193.1	64.8	0.04	<0.5	0.06	0.63	30.0	0.9
Head (calc)	1,843	100.0	4.09	1.08	4.34	100.0	100.0	100.0
Head (direct)								

Combined Products (Flotation)

Product	Mass		Assays, g/t, %			% Distribution		
	g	%	Au	Ag	$S^=$	Au	Ag	$S^=$
Ro Conc 1	191.5	10.4	35.70	5.90	29.90	90.7	56.9	71.7
Ro Conc 1+2	348.0	18.9	21.30	3.56	21.99	98.3	62.4	95.7
Ro Conc 1-3	520.6	28.3	14.35	2.55	15.05	99.1	66.7	98.1
Ro Conc 1-4	601.5	32.6	12.44	2.27	13.12	99.3	68.8	98.8
Ro Conc 1-5	649.4	35.2	11.54	2.14	12.20	99.4	70.0	99.1

% Recovery Au Gravity = 3.1
Overall % Recovery Au (Grav+Flot) = 99.4

CN-3a | 13634-001 | Omega-50 | Michael Unger | April 9, 2012

Purpose: To access the amenability to the Direct Cyanide leach process on approximately 1 kg (Adjusted to 350g) of an Omega Sample at P80 particle size target of 50 µm.

Procedure: Bottle roll cyanidation test were completed on ~1 kg charge of an Omega Sample at conditions applied below. CN Tests are completed after grinding to 50 µm target. Pregnant solution sub-samples were removed & assayed for Au and Ag at ~7, 24 & 48 hrs. Leach residue samples were dried, weighed and sampled in duplicate for Au and Ag analysis and a separate sample was taken for confirmatory size analysis. The conditions applied are outlined below and adjusted by calculation to 350 gram of sample feed inplace of the actual 1000kg for comparison through the following Diagnostic Leach Tests..

Adjusted to 350 gram

Feed: 350 g Omega sample at ~50 µm

Solution Volume: 525 mL

Pulp Density: 40 % solids

Sol'n Composition: 0.5 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Grind: G.G.2kg Mill#2 39.5 min. P_{80} 57

Reagent Addition (kg/t of cyanide feed) NaCN: 2.12 CaO: 0.975
Reagent Consumption (kg/t of cyanide feed) NaCN: 1.41 CaO: 0.945

Time	Added, Grams				Residual		Consumed		pH	D.O
	Actual		Equivalent		Grams		Grams			mg/L
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									8.4	
0-1	0.276	0.393	0.263	0.291	0.068		0.194		10.8 - 11.1	
1-3	0.204	0.000	0.194	0.000	0.089		0.173		11.1 - 11.4	0.7
3-7	0.182	0.000	0.173	0.000	0.173		0.089		11.4 - 11.2	5.9
7-24	0.094		0.089		0.252		0.011		11.2 - 10.2	
24-31	0.011	0.047	0.011	0.035	0.252		0.011		10.8 - 10.5	
31-48	0.011	0.021	0.011	0.016	0.246	0.011	0.017	0.005	10.8 - 10.3	
Total	0.78	0.46	0.74	0.34	0.25	0.01	0.49	0.33		

Cyanidation Results:

Product	Amount g, mL	Assays, mg/L, g/t, % Au	% Distribution Au
7 h PLS	507	1.96	79
24 h PLS	500	2.08	84
48 h PLS	527	2.06	86
Final Residue	350	0.55	14
Head (calc.)	350	3.85	100
Head (direct)	350	3.58	

Final Residue Assays

	Au	
A	0.55	
B	0.55	
Avg	**0.55**	

Other Assays;

Assay, mg/L, g/t PLS's	Ag
7hr	0.46
24hr	0.53
48hr	0.53
Residue	<10

Test: CL-3 **Project: 13634-001** Operator: Michael Unger Date: April 16 2012

Purpose: To perform a caustic leach on a cyanidation leach residue in an attempt to dissolve any iron arsenate, arsenic oxides or bismuth minerals and to liberate the associated gold.

Procedure: The sodium hydroxide solution was placed in a 3 L reaction kettle.
The reaction kettle was placed in a heating bath. Mechanical agitation was applied via a mixer.
The heating bath was turned on and the solution was heated to setpoint temperature, 82,5 °C.
The sample was slowly added to the hot NaOH solution.
The sample was maintained at temperature for the specified amount of time. Upon completion the pulp weight was obtained and the pulp filtered. The pregnant leach solution was collected and the leach residue was washed well with water. The wash water was discarded. The pregnant leach solution was submitted for Au and ICP scan analysis. The washed residue was forwarded to a subsequent cyanidation leach test.

Caustic Leach Feed: 349.64 g of CN-3 Residue

Solution Volume: 816 mL NaOH Sol'n

Pulp Density: 30 w/v % solids

Leach Setpoint Temp.: 82.5 °C

Sol'n Composition: 100 g/L NaOH

Time at Temp.: 3 hours

Kettle Tare Wt.: 1380 grams

Results:

Total Weight in Kettle: 2605.7 g
Pulp Weight.: 1225.7 g
Leach Residue (dry) Weight: 349.3 g
Pregnant Solution Weight: 876.3 g
Pregnant Solution Volume: 793.2 mL

PLS wt.: 55.239 g
PLS vol.: 50 mL

PLS density: 1.10478 g/mL

Product	Amount g, mL	Assays,mg/L,g/t,% Au	% Distribution Au
Pregnant Solution	793.2	0.01	3.8
Residue	349.3	^ 0.57	96.2
Head (calc.)	295.8	0.71	100.0

^ assay back calculated from cyanide leach (CN-3b)

Test: CL-3 **Project: 13634-001** Operator: Michael Unger Date: April 16 2012

Additional Assays

Pregnant Leach Solution, 3hr		
Element		ICP-Scan
Ag	mg/L	< 0.08
Al	mg/L	5.6
As	mg/L	23
Ba	mg/L	0.024
Be	mg/L	< 0.002
Bi	mg/L	< 1
Ca	mg/L	8.3
Cd	mg/L	< 0.3
Co	mg/L	<0.3
Cr	mg/L	<0.1
Cu	mg/L	<0.1
Fe	mg/L	1.9
K	mg/L	107
Li	mg/L	< 2
Mg	mg/L	< 0.07
Mn	mg/L	0.47
Mo	mg/L	< 0.6
Na	mg/L	59800
Ni	mg/L	< 0.6
P	mg/L	< 5
Pb	mg/L	< 2
Sb	mg/L	< 1
Se	mg/L	< 3
Sn	mg/L	< 2
Sr	mg/L	0.166
Ti	mg/L	< 0.02
Tl	mg/L	< 3
U	mg/L	< 1
V	mg/L	< 0.2
W	mg/L	< 2
Y	mg/L	< 0.02
Zn	mg/L	1.1

CN-3b **13634-001** **Omega** **Michael Unger** **April 18, 2012**

Purpose: To examine the recovery of gold by intensive cyanidation.

Procedure: The sample was pulped with water in a 2.5L bottle. The pH was adjusted and NaCN was then added and the leach was carried out over 24 hours on the rolls. NaCN concentration and pH were maintained throughout the leach period, as described below. At the end of the leach period, the pulp was filtered and washed several times with water. The pregnant solution was submitted for chemical analysis.

Feed: 349 g Omega Sample after CN-3 Residue & CL-3 at ~57 µm

Solution Volume: 524 mL

Pulp Density: 40 % solids

Sol'n Composition: 2.0 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Reagent Addition (kg/t of cyanide feed) NaCN: 3.68 CaO: 0.748
Reagent Consumption (kg/t of cyanide feed) NaCN: 0.72 CaO: 0.748

Time	Added, Grams				Residual		Consumed		pH	D.O
	Actual		Equivalent		Grams		Grams			mg/L
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									8.6	
0-1	1.103	0.225	1.048	0.167	0.954		0.094		10.7 - 10.3	9.3
1-6	0.149	0.076	0.142	0.056	1.034		0.061		10.7 - 10.5	
6-24	0.066	0.043	0.063	0.032	1.008	0.000	0.089	0.032	10.7 - 10.5	
Total	1.32	0.34	1.25	0.25	1.01	0.00	0.24	0.25		

Cyanidation Results:

Product	Amount g, mL	Assays, mg/L, g/t, % Au		% Distribution Au	
24 h PLS	561	0.06		17.2	
Final Residue	340	^ 0.48		82.8	
Head (calc.)	340	0.57	0.00	100	0

^ assay back calculated from hydrochloric leach (HCl-3)

Test: HCL-15 **Project: 13634-001** Operator: Michael Unger Date: April 20 2012

Purpose: To conduct a standard Hydrochloric acid leach test on a received cyanidation residue sample.

Sample: CN-3b Residue

Procedure: A volume of 1169 mL of concentrated Hydrochloric acid was used. The acid solution was placed into a 3 L reactor kettle with a lid. The reactor was placed into a water bath and allowed to reach 85°C. Mixing was provided by an overhead stirrer equipped with a teflon impellor.
The forwarded CN residue sample was very slowly added to the acid solution. The temperature was monitored and maintained for 3 hours from the time all of the sample had been added. The pulp kettle was removed from the waterbath and allowed to cool to room temperature.
The pulp sample was then filtered. The filtrate (PLS) was submitted for assays. The residue was first washed with one displacement of 10g/L HCl acid. The residue was then displacement washed several times with water. The wash solutions were discarded. The residue was forwarded to further Cyanidation testing.

Conditions:

Feed Weight (dry) (g):	**340.3**	
Concentrated HCl Acid (mL):	**1361**	(36% HCl)
Pulp Density (% solids w/v):	**20.000**	(w/v)
Temperature (°C):	**85**	
Time (at temperature) (min):	**180**	

Leach Data:

Elapsed Time min	Temp °C	Remarks
0	83	Start sample addition, Tan colour dry turned dark Green.
30	85	Finish sample addition. Added 50 mL Conc.HCL, clean.
		Note: Kettle sol'n temp 85 when bath temp reading 90
0	85	Sample All In Start Leach Time
60	85	
120	85	
180	85	End leach at temp.
240	40	Cooling

Results:

Final Pulp Weight (g):	1776.1
PLS sg (mg/mL):	1.179

PLS wt.: 58.948 g
PLS vol.: 50 mL

PLS density: 1.179 g/mL

Assays

Product	Amount (mL,g)	Assays (mg/L, g/t, %) Au	Distribution (%) Au
PLS	1290	<0.01	8.0
Residue	254.9	^ 0.58	92.0
Head (calc)	340.3	0.48	100

^ assay back calculated from cyanide leach (CN-3c)

Test: HCL-15 **Project: 13634-001** Operator: Michael Unger Date: April 20 2012

Additional Assays

Leach Solution		
Element		Assay
Ag	mg/L	<0.08
Al	mg/L	2280
As	mg/L	16
Ba	mg/L	2.58
Be	mg/L	0.025
Bi	mg/L	< 1
Ca	mg/L	12100
Cd	mg/L	<0.3
Co	mg/L	4.3
Cr	mg/L	253
Cu	mg/L	7.9
Fe	mg/L	12700
K	mg/L	59
Li	mg/L	< 2
Mg	mg/L	5330
Mn	mg/L	369
Mo	mg/L	<2
Na	mg/L	81
Ni	mg/L	115
P	mg/L	193
Pb	mg/L	< 4
Sb	mg/L	< 1
Se	mg/L	< 3
Sn	mg/L	< 2
Sr	mg/L	25.9
Ti	mg/L	4.45
Tl	mg/L	< 3
U	mg/L	< 1
V	mg/L	8.3
W	mg/L	< 4
Y	mg/L	1.72
Zn	mg/L	9

CN-3c **13634-001** **Omega** **Michael Unger** **April 23, 2012**

Purpose: To examine the recovery of gold by intensive cyanidation.

Procedure: The sample was pulped with water in a 2.5L bottle. The pH was adjusted and NaCN was then added and the leach was carried out over 24 hours on the rolls. NaCN concentration and pH were maintained throughout the leach period, as described below. At the end of the leach period, the pulp was filtered and washed several times with water. The pregnant solution was submitted for chemical analysis.

Feed: 255 g Omega Sample HCL-3 residue.

Solution Volume: 382 mL

Pulp Density: 40 % solids

Sol'n Composition: 2.0 g/L NaCN maintained

pH Range: 10.5 - 11 maintained with lime as required.

Reagent Addition (kg/t of cyanide feed) NaCN: 3.13 CaO: 4.698

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.17 CaO: 4.672

Time	Added, Grams				Residual		Consumed		pH	D.O
	Actual		Equivalent		Grams		Grams			mg/L
hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Cyanidation:									5.7	
0-1	0.837	1.037	0.795	0.767	0.795		0.000		10.7 - 10.1	8.7
1-6	0.000	0.267	0.000	0.198	0.795		0.000		10.7 - 10.1	
6-24	0.000	0.310	0.000	0.229	0.752	0.007	0.043	0.223	11.0 - 10.1	
Total	0.84	1.61	0.80	1.19	0.75	0.01	0.043	1.19		

Cyanidation Results:

Product	Amount g, mL	Assays, mg/L, g/t, % Au		% Distribution Au	
24 h PLS	410	0.05		13.8	
Final Residue	254	^ 0.50		86.2	
Head (calc.)	254	0.58	0.00	100	0

^ assay back calculated from Aqua Regia leach (AR-3)

Test: AR-15 **Project:** **13634-001** Operator: Michael Unger Date: April 24 2012

Purpose: To conduct a standard Aqua Regia acid leach test on a received cyanidation residue sample.

Sample: CN-3c Residue

Procedure: A volume of 1020 mL of Aqua Regia acid was made-up consisting of 25% Nitric Acid and 75% HCl acid. The acid solution was placed into a 3 L reactor with a lid. The reactor was placed into a heating mantle. Mixing was provided by an overhead stirrer equipped with a teflon impellor. The acid solution was heated to 90°C.

The sample was very slowly added to the acid solution. The temperature was monitored. The sample was maintained at 90°C for 3 hours.

At the end of the test sample was filtered. The filtrate (PLS) was submitted for assays. The residue was displacement washed several times with water and then submitted for assays. The wash solutions were discarded.

Conditions:

Feed Weight (dry) (g):	**254**	CN-3c Residue
Concentrated HCl Acid (mL):	**765**	(36% HCl)
Concentrated HNO_3 Acid (mL):	**255**	(70% HNO_3)
Pulp Density (% solids w/v):	**20.0**	(w/v)
Temperature (°C):	**90**	(set temp.)
Time (at temperature) (min):	**180**	

Leach Data:

Elapsed Time (min)	Temp °C	Remarks
0	77	Start sample addition, temp Reading
30	89	Finish sample addition.
0	90	Start Test - add \ 75mL HNO_3
60	90	add ~50mL HNO_3
120	90	add ~50mL HNO_3
180	90	End Test
240	35	Cooling

Results:

Final Pulp Weight (g):	1393
PLS sg (mg/mL):	1.214

Assays

Product	Amount (mL,g)	Assays (mg/L, g/t, %) Au		Distribution (%) Au	
PLS, 3hr	958	0.10		74.9	
Residue	230.0	0.14		25.1	
Head (calc)	254.2	0.50	0	100	0

A	0.05
B	0.05
Avg	**0.05**

Test: AR-15 **Project:** **13634-001** Operator: Michael Unger Date: April 24 2012

Additional Assays

Leach Residue		
Element		Assay
S	%	2.82
$S^=$	%	<0.05

Leach Residue		
Element		Assay
Ag	mg/L	< 2
Al	mg/L	28500
As	mg/L	< 30
Ba	mg/L	106
Be	mg/L	0.2
Bi	mg/L	< 20
Ca	mg/L	322
Cd	mg/L	< 2
Co	mg/L	9.0
Cr	mg/L	72
Cu	mg/L	< 0.5
Fe	mg/L	732
K	mg/L	7890
Li	mg/L	< 5
Mg	mg/L	142
Mn	mg/L	3.9
Mo	mg/L	<5
Na	mg/L	55200
Ni	mg/L	< 20
P	mg/L	< 30
Pb	mg/L	< 20
Sb	mg/L	< 10
Se	mg/L	< 30
Sn	mg/L	< 20
Sr	mg/L	24.3
Ti	mg/L	5190
Tl	mg/L	< 30
U	mg/L	< 20
V	mg/L	34
W	mg/L	
Y	mg/L	3.4
Zn	mg/L	15

Leach Solution		
Element		Assay
Ag	mg/L	<0.08
Al	mg/L	91.1
As	mg/L	302
Ba	mg/L	0.888
Be	mg/L	0.003
Bi	mg/L	< 1
Ca	mg/L	782
Cd	mg/L	0.4
Co	mg/L	5.7
Cr	mg/L	17.6
Cu	mg/L	2.6
Fe	mg/L	10200
K	mg/L	49
Li	mg/L	< 2
Mg	mg/L	14.0
Mn	mg/L	1.26
Mo	mg/L	<0.6
Na	mg/L	40
Ni	mg/L	14.9
P	mg/L	<5
Pb	mg/L	< 2
Sb	mg/L	< 1
Se	mg/L	< 3
Sn	mg/L	< 2
Sr	mg/L	0.416
Ti	mg/L	0.96
Tl	mg/L	< 3
U	mg/L	< 2
V	mg/L	< 0.2
W	mg/L	< 2
Y	mg/L	0.10
Zn	mg/L	3