Report to:

STRATAGOLD CORPORATION

Technical Report on the Dublin Gulch Property, Yukon Territory, Canada

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TECHNICAL REPORT ON THE DUBLIN GULCH PROPERTY, YUKON TERRITORY, CANADA

JANUARY 2009

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REVISION HISTORY

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1.0 SUMMARY

The Dublin Gulch Property (Property) is located in the central Yukon Territory about 400 kilometres (km) north of the capital, Whitehorse.

The Property is comprised of 1,856 quartz claims with an aggregate area of 33, 904 hectares (ha) and is owned by StrataGold Corporation.

The Dublin Gulch Property contains vein-hosted gold mineralization that occurs proximal to the contact of a Cretaceous-age granitoid intrusive. One of these gold occurrences, the Eagle Zone, is in an advanced stage of exploration. The Property has been extensively explored by a series of owners and operators, initially for tungsten and subsequently for gold. StrataGold acquired the Property in 2004 and in 2005 carried out about 8,100 metres (m) of core drilling, primarily to delineate and further expand the Eagle Zone. A second program in 2006-2007 was conducted to increase the confidence in the resource estimate. The most recent drill program was carried out during 2008 with the completion of nine diamond drilled holes. This report is comprised of a resource estimate for that zone.

In 1997, Mineral Resources Development Inc. (MRDI) carried out a resource estimate of the Eagle Zone resulting in a Measured and Indicated resource of 88.8 million tonnes at an average grade of 0.698 grams per tonne (g/t) gold, and an Inferred resource of 106 million tonnes at an average grade of 0.345 g/t gold. While this estimate is considered reliable and relevant, and the estimate uses categories that are compatible with the National Instrument 43-101 (NI 43-101) this estimate pre-dates the implementation of the NI 43-101 and is only included here for purposes of historical reference.

In 2004, Snowden Mining Industry Consultants (Snowden) reviewed the 1997 MRDI resource estimate and concluded that the estimation methodology of MRDI was in accordance with the guidelines of NI 43-101, with the exception of the classification of the resource. Snowden re-estimated the resource, using the MRDI data and parameters. This resource estimate defined 55.2 million tonnes grading 0.934 g/t Au of Indicated resources and 17.3 million tonnes grading 0.743 g/t Au of Inferred resources at a cut off 0.50 g/t gold.

Wardrop was commissioned in 2006 to prepare a NI 43-101 compliant resource estimate subsequent to a drill program completed in 2005. That estimate reported that the Eagle Zone contained 66.5 million tonnes at 0.92 g/t of Indicated resource, and 14.4 million tonnes of 0.80 g/t of Inferred resource. This report updates that resource estimate, based on the drilling that occurred in 2006, 2007 and 2008 to 98.5 million tonnes grading 0.849 g/t gold Indicated and 2.0 million tonnes grading 0.671 g/t gold Inferred. A cut off 0.50 g/t Au was used in the last resource estimate.

The Dublin Gulch Property is underlain by Proterozoic to Lower Cambrian-age Hyland Group meta-sedimentary rocks and the intrusive Dublin Gulch granodioritic stock. The stock has been dated at approximately 93 million years (ma), and is therefore a member of the Tombstone Plutonic Suite.

The Hyland Group is comprised of interbedded quartzite and phyllite. The quartzite is variably gritty, micaceous, and massive. The phyllite is composed of muscovite-sericite and chlorite. Limestone is a relatively minor constituent of this stratigraphic sequence.

The Dublin Gulch stock is comprised of four phases, the most significant of which is granodiorite. Quartz diorite, quartz monzonite, leucogranite and aplite comprise younger intrusive phases that occur predominantly as dikes and sills and cut both the granodiorite and surrounding country rocks. The stock has intruded the Hyland Group meta-sedimentary rocks near their contact with the underlying Upper Schist.

The Eagle Zone gold occurrence is localized at the narrowest portion of the stock, near its known western limit. The intrusive-meta-sedimentary contact is sharp but irregular and varies between steep attitudes that crosscut meta-sedimentary foliation, to shallow southwest dips parallel to foliation.

The zone is comprised of gold-bearing sub-parallel extensional quartz veins that are best developed within the granodiorite proximal to both the hanging wall and footwall intrusive-meta-sedimentary contacts. Veining is apparently best-developed on the hanging wall contact, but this may be more apparent than real as more drilling has taken place on the hanging wall side.

Veins are typically composed of white or grey quartz with subordinate potassium feldspar, and strike 060 degrees (°) to 085°, typically dip at about 60° to the south, and range in width from one millimetre to about 10 centimetres (cm); contacts are typically sharp. Vein densities range from less than one per metre to more than 15 per metre, and average about three to five per metre. The greatest concentration of veins appears to coincide with both the narrowest constriction as well as the local apex of the intrusion.

Subordinate quantities of gold mineralization occur in quartz veins within the adjacent meta-sedimentary rocks. Veins strike at about 060° azimuth, sub-parallel to the intrusive contact and are commonly fractured by repeated movement along the host fractures.

In 2005 StrataGold drilled 34 holes with an aggregate length of 8,105 m. The program had two main purposes: the first was to test the west margin of the area for which a resource had been estimated and an open pit had been designed by MRDI in 1997, and the second was to provide fill-in data in several areas of the designed pit.

In 2006, Wardrop completed an estimation of the mineral resource of the gold mineralization on the basis of drill hole data. This resulted in an Indicated resource of 66.5 Mt grading 0.92 g/t gold and an Inferred resource of 14.4 Mt grading 0.80 g/t gold at a cut-off of 0.50 g/t. As a result of this estimation, an additional diamond drill program was recommended to increase the confidence in the resource. An additional drill program was completed in 2006, 2007 and 2008. The updated resource estimate contains an Indicated resource of 98.5 million tonnes grading 0.849 g/t gold and an Inferred resource of 2.0 million tonnes grading 0.671 g/t gold above a cut-off of 0.50 g/t.

Verification of the drill hole database indicates that the information is reliable and is believed to be accurate. Measurements obtained from bulk density samples taken from the 1995 and 1996 drill core appear to be consistent with values for intrusive and meta-sedimentary rock types.

The geological interpretation of the mineralization was based on an approximate cut-off of 0.20 g/t gold. The resource estimation was interpolated by nearest neighbour (NN), inverse distance squared (ID) and ordinary kriging (OK) methodologies. No significant discrepancies exist between the methods, and values obtained by ordinary kriging have been used for the resource tabulation.

Additional drilling down depth of the mineralization following the extension intercepted with drill holes DG08357C and DG7334C is recommended. Three relatively deep drill holes are recommended between sections 459650E and 459750E and between 460100E to 460200E.

Three additional drill holes are recommended to test the extension of the northern hanging wall zone between drill holes DG06310C and DG06315C1. A total of 550 m strike length need to be tested with the initial three drill holes, which would help to define the potential of the zone. Further drilling would be outlined pending the success of the initial holes.

Supplementary bulk density determinations of specific mineralization types should be undertaken and the resultant values used to refine the resource estimate model.

As a result of the most recent resource estimate completion and following the recommendations from this report the next phase of the project should be a NI43-101 compliant pre-feasibility study.

1.1 RESOURCE STATEMENT

Wardrop completed an updated estimation of the gold mineral resource on the Dublin Gulch Property in the Eagle Zone for StrataGold. This has resulted in an Indicated resource of 98,584,000 tonnes grading 0.849 g/t gold (Au), plus an Inferred resource of 2,023,000 tonnes grading 0.671 g/t Au at a cut-off of 0.50 g/t Au.

This resource estimation resulted with an increase in total resources. The mineral resource update was based on an additional 39 drill holes completed during the 2006, 2007 and 2008 drill programs. Ten drill holes were abandoned during these exploration projects. This resource update includes a total of 14,058 m of new drill core data.

Drill core recovered from the 2008 exploration program was sent to the laboratory to be assayed on gold values.

Wardrop carried out data verification on 5% of the drill hole database from drill logs and assay values. The database verification conducted by Wardrop found no discrepancies with the original information. Wardrop concludes that the database meets industry standards for resource estimation. This data verification was completed after the 2008 drilling program.

Estimation of the resource included the interpolation methods of nearest neighbour, inverse distance squared and ordinary kriging. The methods were validated by comparison of global mean grades and a visual review of coded block grades. No significant discrepancies exist between the methods. Ordinary kriging methodology was selected for grade estimation on the deposit.

1.2 RECOMMENDATIONS

Additional drilling down depth of the mineralization following the extension intercepted with drill hole DG08357C and DG7334C is recommended.

Three deep drill holes (approximately 550 m each) are recommended between sections 459650E and 459750E in the vicinity of DG08357 and within the sections 460100E to 460200E near DG7334C (see Appendices I and J)

Three additional drill holes are recommended to test the extension of the northern hanging wall zone between drill holes DG06310C and DG06315C1. A total of 550 m strike length needs to be tested with the initial two drill holes, which would help to define the potential of the zone. Further drilling would be outlined pending the success of the initial holes.

A comprehensive program of metallurgical sampling, including spatial variance across the ore body to obtain the standards of accuracy, is recommended and Wardrop can provide assistance in identifying representative ore zones.

Additional bulk density determinations of the mineralized material should be carried out, so that specific gravity data can be incorporated into the resource block model for resource estimation.

Wardrop believes that the current resource block model honours the high grade values well and that local grade variations are reasonably well represented.

However, more work is required to improve the local grade estimation within the block model estimation, possibly using multiple indicator kriging.

As the project advances to prefeasibility phase, where selective mining methods or detailed engineering design work will be required, Wardrop recommends updating the current resource block model. This exercise will be beneficial if the new drilling data becomes available within the lower part of the deposit.

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 INTRODUCTION

The Dublin Gulch Property (Property) is located in the central part of Yukon Territory and is owned by StrataGold Corporation. The Property contains vein-hosted gold mineralization that occurs proximal to the contact of a Cretaceous-age granitoid intrusive. One of these gold occurrences, the Eagle Zone, is in an advanced stage of exploration. The Property has been extensively explored by a series of owners and operators, initially for tungsten and subsequently for gold. StrataGold acquired the Property in 2004 and in 2005 carried out about 8,100 m of core drilling, primarily to delineate and further expand the Eagle Zone. An additional 9,809 m of drilling was completed in 2006-2007. The most recent drilling program was carried out during 2008 with a total of 4,249 m of drilling completed. This report updates the resource estimate for the Eagle Zone based on the 2006-2008 drill programs.

2.2 TERMS OF REFERENCE

StrataGold Corporation (StrataGold) has retained Wardrop Engineering Inc. (Wardrop) to update the 2006 resource estimate of the Eagle Zone. The estimate that is the subject of this report incorporates drill data acquired since the completion of the last estimate.

3.0 RELIANCE ON OTHER EXPERTS

3.1 RELIANCE ON OTHER EXPERTS

Wardrop has followed standard professional procedures in preparing the contents of this resource estimation report. The data used in this report has been verified where possible, and Wardrop has no reason to believe that the data was not collected in a professional manner.

Wardrop has relied on others for information on this report. Information from third-party sources is footnoted, quoted as a report in the text or referenced.

3.2 DATA PROVIDED TO WARDROP

Data provided to Wardrop is comprised of electronic files. The files included drill logs, survey, collar data and assay results for the 2006-2007 and 2008 drill programs.

A site visit was conducted by Cliff Duke, P.Eng. from Wardrop on February 13th, 2008 and by S. Bob Jankovic, P.Geol, one of the authors of this report, on November 4th, 2008. Drill core from the 2008 drilling program was completely sampled and sent to the laboratory to be assayed. During the site visit, only the core from the 2006-2007 drill programs was available for visual examination.

Three out of nine drill collars indicated in the 2008 drilling program were located by Wardrop and StrataGold geologists and the exact collar location was determined by a Global Positioning System (GPS) (see Appendix F for a photo of the drill collar location). Wardrop coordinates matched StrataGold's survey coordinates.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Dublin Gulch Property is located in the central part of Yukon Territory, about 400 km north of Whitehorse, and is accessible by road about 85 km north of the village of Mayo, the closest community with significant commercial services. The centre of the Property is situated at the confluence of Haggart Creek and Dublin Gulch, at about 64° 02' 13.409″ North Latitude, and 135° 44' 32.616″ West Longitude (UTM Coordinates 7100950N / 453750E, Zone 8, NAD 83 Datum). The Property is located within NTS areas 106D/04 and 105M/114, 106D03, 106A01 and 115P16.

StrataGold acquired the Dublin Gulch Property from Sterlite Gold Ltd. in 2004 in exchange for US$6 million and 5,000,000 common shares of StrataGold.

The Property is comprised of 1,906 quartz claims and lease and one crown grant with an aggregate area of 34,557 ha. Figure 4.1 and Figure 4.2 show the location of the claims. A list of claims is included in Appendix E. The claim expiry dates, taken from the Yukon Mining Recorders website, range from 2009-2023. **Wardrop has not conducted a title search and has not verified the legal description of the Property; instead we have relied on information provided by StrataGold.**

A portion of the Dublin Gulch property, historically known as the Mar Gold Property is subject to a royalty agreement with a minimum annual royalty payment of $20,000 to a maximum of $1,000,000, after which, the royalty reverts to 1% with no end price. There are no other known encumbrances on the property associated to the Eagle Zone Deposit.

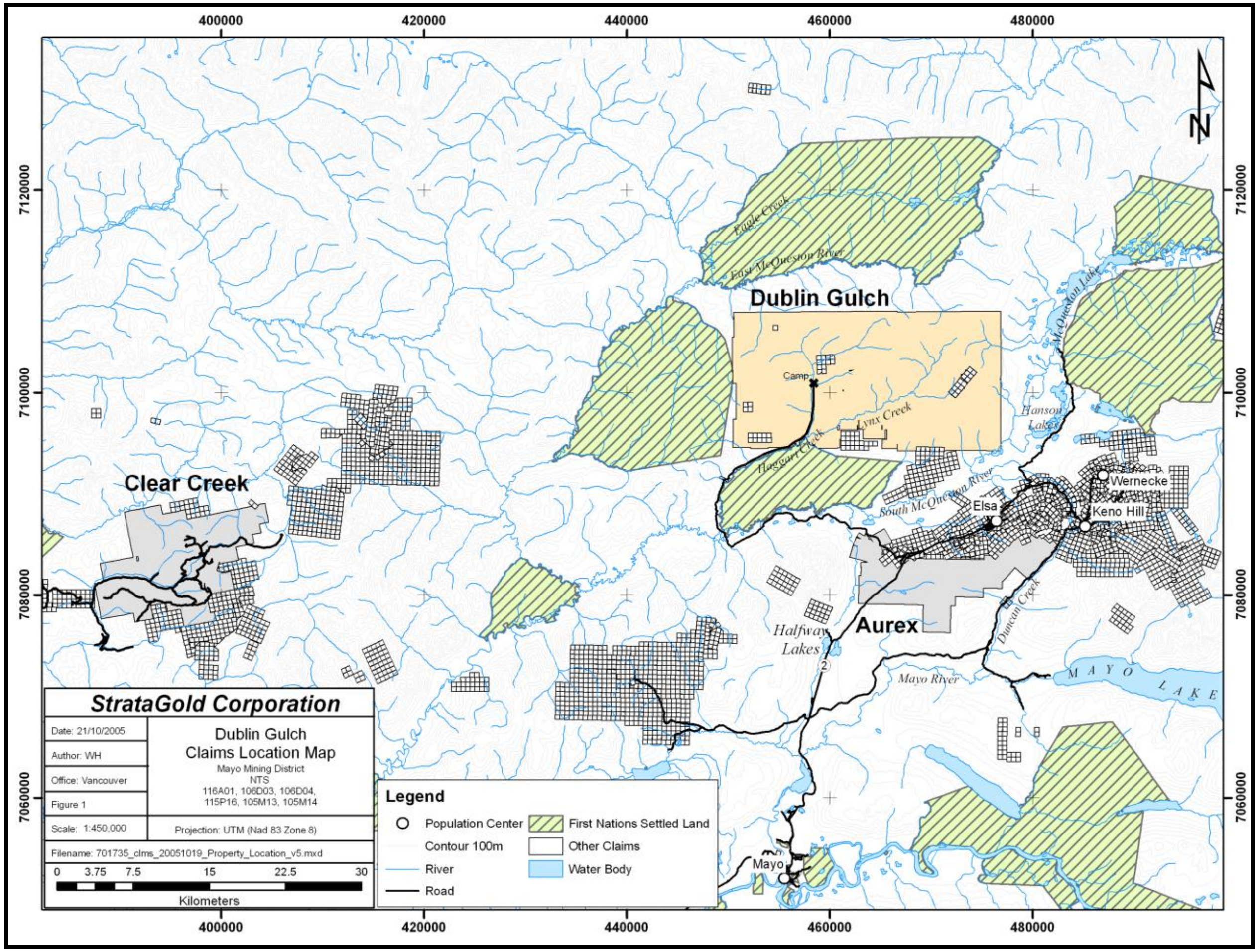
The Property contains extensive placer workings, principally within the drainages of Dublin Gulch and Haggart Creek. The Eagle Zone is situated north of Haggart Creek and east of Dublin Gulch. A number of other mineral occurrences and old prospects are located within the Property, but are not considered in this report.

The Property is not subject to any known environmental liabilities, but this assumption has not been verified by Wardrop.

Permits necessary for proposed work are in place.

Figure 4.1 Location Map – Dublin Gulch Property

Figure 4.2 Claim Location Map – Dublin Gulch Property



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PHYSIOGRAPHY, CLIMATE, ACCESSIBILITY, AND LOCAL RESOURCES

The topography of the Property area is characterized by rolling hills and plateaus that range from about 800 m to a local maximum of 1,650 m above sea level at the summit of Potato Hills, and are drained by deeply incised creek canyons. The ground surface is covered by residual soil and felsenmeer. Outcrop is rare, generally less than two percent of the surface area, and is limited to ridge tops and creek walls. Lower elevations are vegetated by black spruce, willow, alder and moss, and higher elevations by sub-alpine vegetation.

The central Yukon has a northern continental climate with average minimum and maximum temperatures of about -10 degrees Celcius (°C) and +3°C. January is the coldest month, and July is the warmest. Annual precipitation ranges from about 375 to 600 millimetres (mm), about half of which falls as snow. Patchy permafrost occurs on north-facing slopes.

The Property is accessible by road during non-winter months. A paved highway extends to Mayo, about 40 km directly to the south, and from there access is by Highway 2 (the Silver Trail) for 35 km and then by the South McQuesten Road for 46 km, for the last 25 of which the road is unpaved and not maintained, but is largely in good repair. A network of four-wheel drive roads provides access to most parts of the Property.

The Property has sufficient appropriate sites to accommodate mining and processing facilities as well as disposal of waste rock and tailings. Local water resources are assumed to be adequate for the likely scale of mining that may take place. It is probable that electrical power would have to be generated on-site, and it is probable that workforce requirements could be filled from within the Yukon.

6.0 HISTORY

The Dublin Gulch Property has a lengthy history of exploration and placer mining dating back to 1895. The following synopsis is derived principally from Goodwin and Burns, 2004 (Goodwin, 2004).

Placer gold mining began in 1895, and in 1904, tungsten was identified in placer concentrates. In 1916, the Geological Survey of Canada discovered bedrock sources of scheelite in Dublin Gulch. Since 1970, there has been essentially continuous exploration on the Property, initially for tungsten and subsequently for gold. Since 1978 when documentation of production was initiated, about 110,000 ounces of placer gold have been recovered from the Dublin Gulch area.

The chain of tenure leading to the current circumstances of ownership began in 1977, at which time Queenstake Resources Ltd. staked the Mar Claims to cover tungsten–bearing skarns in the Ray Gulch area. Canada Tungsten Mining Corp. optioned the ground and carried out exploration for both tungsten and gold between 1977 and 1986. The Eagle Zone, the most significant of the known gold occurrences, is located about three kilometres to the southeast of the tungsten occurrences and became the subject of significant exploration interest during this period.

In 1991, Ivanhoe Goldfields acquired the Dublin Gulch claims from Queenstake and commenced exploration for “Fort Knox Type” intrusive-hosted gold mineralization.

No fieldwork was conducted in 1994.

In 1995, First Dynasty Mines Ltd. acquired the property through the acquisition of Ivanhoe Goldfields. In 1996, First Dynasty transferred the Dublin Gulch Property to New Millennium Mining Ltd., a wholly owned subsidiary.

In 1993, Ivanhoe estimated inferred and potential resources within the Eagle Zone of 98.6 Mt with an average grade of 1.19 g/t gold. This estimate is considered not relevant and of unknown reliability, it is only stated here for historical completeness, and the estimate should not be relied upon.

In 2002, First Dynasty changed its name to Sterlite Gold Ltd. In 2004 StrataGold acquired the Dublin Gulch Property from Sterlite as part of a larger transaction that included the Clear Creek Property, in exchange for US$6 million and 5,000,000 common shares of StrataGold.

The most extensive phase of exploration of the Eagle Zone took place between 1991 and 1996, during which time 48 core holes with an aggregate length of about

9,000 m, and 118 reverse-circulation holes with an aggregate length of 21,300 m were drilled within the presently-defined Eagle Zone and surrounding area.

In 1997, Mineral Resources Development Inc. (MRDI) carried out a resource estimate of the Eagle Zone resulting in a Measured and Indicated resource of 88.8 million tonnes (Mt) at an average grade of 0.698 grams per tonne (g/t) gold, and an Inferred resource of 106 Mt at an average grade of 0.345 g/t gold. This estimate is considered reliable and relevant, the estimate uses categories that are compatible with National Instrument 43-101 (NI 43-101), but the estimate pre-dates the implementation of NI 43-101 and it is only included here for purposes of historical reference.

In 2004, Snowden Mining Industry Consultants (Snowden) reviewed the 1997 MRDI resource estimate and concluded that the estimation methodology of MRDI was in accordance with the guidelines of NI 43-101, with the exception of the classification of the resource. Snowden re-estimated the resource, using the MRDI data and parameters.

The Snowden estimate was superseded by Wardrop's 2006 estimate on the basis of additional drill data acquired by StrataGold during 2005. Wardrop estimated the Eagle Zone to contain an Indicated Resource of 66.5 Mt grading 0.916 g/t Au, and an Inferred Resource of 14.4 Mt grading 0.803 g/t Au.

7.0 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Dublin Gulch Property is located in the central part of the Selwyn Basin, the stratigraphy of which is divisible into four main lithological units, from lowermost to uppermost, the Lower Schist, Keno Hill Quartzite, Upper Schist, and Hyland Group, formerly the Grit Unit. The Lower Schist and Keno Hill Quartzite are of probable Mesozoic-age, the Upper Schist is of Paleozoic-age and the Hyland Group of Proterozoic to Lower Cambrian age. These units have been juxtaposed by laterally extensive, northward-directed thrust sheets that formed in the early Cretaceous period.

There are three principal thrust sheets, from east to west, the Dawson, Tombstone, and Robert Service. The Robert Service thrust is proximal to the Dublin Gulch area and is inferred to have superimposed the Hyland Group on the Mississippian-age Keno Hill Quartzite.

Four phases of deformation have been documented, of which only the first two resulted in the generation of prominent structures. Thrusting during the first phase resulted in the widespread development of foliation that was subsequently deformed by gentle, regional-scale folding during the second phase of deformation. Several east-trending, south-plunging anticlines in the Dublin Gulch area are attributed to this second deformational event.

During the Cretaceous period, there were three events of granitoid intrusion: the Selwyn Suite dated between 98 and 104 million years (ma), the Tombstone Suite between 92 and 94 ma, and the McQueston Suite at 64 ma. It is probable that the Selwyn and Tombstone intrusive events were synchronous with the second, regional folding event. Intrusives commonly are emplaced within the Grit Unit and less commonly within the Upper Schist.

Cretaceous-age deformation and intrusion are possibly related to north-northeast directed subduction and related arc-trench magmatism of the oceanic Farallon Plate beneath continental North America (Stephens, 2003).

Numerous mineral deposits are associated with the Cretaceous-age intrusives and are generally vein, shear and skarn related. Gold, silver, lead, zinc and tungsten are the principal elements of economic interest. The Tombstone Suite is the primary source of intrusion-hosted gold deposits.

7.2 PROPERTY GEOLOGY

The Dublin Gulch Property is underlain by Proterozoic to Lower Cambrian-age Hyland Group meta-sedimentary rocks and the intrusive Dublin Gulch granodioritic stock. The stock has been dated at approximately 93 ma, and is therefore a member of the Tombstone Plutonic Suite.

The Hyland Group is comprised of interbedded quartzite and phyllite. The quartzite is variably gritty, micaceous, and massive. The phyllite is composed of muscovite-sericite and chlorite. Limestone is a relatively minor constituent of this stratigraphic sequence.

The Dublin Gulch anticline, located approximately midway between Dublin Gulch and Lynx Creek to the south, has folded the meta-sedimentary rocks about an axis that trends 070° azimuth and plunges gently to the southwest.

The meta-sedimentary rocks are the product of greenschist-grade regional metamorphism, and proximal to the Dublin Creek stock have undergone metasomatism and contact metamorphism. A hornfels thermal aureole surrounds the stock and within which the coarse clastic components of the Hyland Group have been altered to quartz-biotite, the argillaceous components to sericite-biotite-chlorite schist, and the carbonates to marble, wollastonite-quartz skarn and pyroxenite skarn. The aureole extends from 800 to 2,000 m outward from the intrusive.

The Dublin Gulch stock is comprised of four phases, the most significant of which is granodiorite. Quartz diorite, quartz monzonite, leucogranite and aplite comprise younger intrusive phases that occur predominantly as dikes and sills and cut both the granodiorite and surrounding country rocks. The stock has intruded the Hyland Group meta-sedimentary rocks near their contact with the underlying Upper Schist.

The granodiorite stock is elongate, measures about 5 km in length in a northeast direction, and has a maximum width of about 2 km. The long axis of the stock coincides with the axis of the Dublin Gulch anticline. Sheet-like sills of granodiorite extend from the stock and cut the meta-sedimentary strata at low angles.

The intrusive-meta-sedimentary contact dips shallowly to steeply to the north on the northern side of the intrusive, and steeply to the north or south along its southern margin. No chilled margin is apparent at the contact.

7.3 EAGLE ZONE

The Eagle Zone gold occurrence is localized at the narrowest portion of the stock, near its known western limit. The intrusive-meta-sedimentary contact is sharp but irregular and varies between steep attitudes that crosscut meta-sedimentary foliation to shallow southwest dips parallel to foliation.

The zone is comprised of sub-parallel extensional quartz veins that are best developed within the granodiorite proximal to both the hanging wall and footwall intrusive-meta-sedimentary contacts. Veining is apparently best developed on the hanging wall contact, but this may be more apparent than real as more drilling has taken place on the hanging wall side.

Veins are typically composed of white or grey quartz with subordinate potassium feldspar and strike 060° to 085° azimuth, typically dipping at about 60° to the south, and ranging in width from less than one millimetre to about 10 cm. Contacts are typically sharp. Vein densities range from less than one per metre to more than 15 per metre, and average about three to five per metre. The greatest concentration of veins appears to coincide with both the narrowest constriction as well as the local apex of the intrusion (Rescan, 1997).

Sulphides account for less than five percent of vein material and occur in the center, on the margin, and disseminated throughout. The most common sulphide minerals are pyrrhotite, pyrite scheelite, arsenopyrite, sphalerite, bismuthinite and galena.

Secondary potassium feldspar is the dominant mineral in alteration envelopes. Sericite-carbonate is generally restricted to narrow vein selvedges, although alteration zones of this type also occur with no obvious relation to veins.

Vein formation can be attributed to contrasts in cohesion and tensile strength between the intrusion and enclosing meta-sedimentary rocks. Embayments and narrow portions of the stock represent stress shadows that constitute favourable areas for rheological failure and therefore for the formation of extensional quartz veins. Protrusions in the stock created favourable areas for the development of extensional shear-veining in the adjacent country rocks (Sieb, 1996).

8.0 DEPOSIT TYPE

The Dublin Gulch Eagle Zone gold mineralization is closely similar to the Fort Knox deposit north of Fairbanks, Alaska (Bakke) and both deposits may occupy portions of the same belt that were offset along the Tintina Fault (Stephens, 2003). At Fort Knox, gold occurs as distinct grains or in cracks in sulphides in structurally-controlled pegmatite dikes, grey quartz veins, quartz vein stockworks, and quartz-filled shears that cut a granite stock that has been dated at 90 to 92 ma.

The stock intrudes fine-grained muscovite-quartz schist and micaceous quartzite of the Fairbanks Schist and is comprised of several phases with an overall granitic composition.

The structures occupied by the dikes, veins and shears are believed to have formed as a result of doming and subsequent subsidence of the granite intrusive. Veins are most abundant in the apical portion of the stock.

Gold is most commonly associated with various native, sulphide and oxide bismuth minerals as well as tellurides. Associated sulphide minerals include pyrite, marcasite, pyrrhotite, arsenopyrite, and molybdenum. Sulphides represent less than one percent of the rock by volume. Oxide minerals include scheelite and rutile.

9.0 MINERALIZATION

Within the Eagle Zone, gold occurs in extensional quartz veins that are most abundant on the hanging and footwall contacts of the narrowest portion of the Dublin Gulch granodiorite near its known western limits. Subordinate quantities of gold mineralization occur in quartz veins within the adjacent meta-sedimentary rocks. Veins strike at about 060° azimuth, sub-parallel to the intrusive contact and are commonly fractured by repeated movement along the host fractures.

A mineralogical study conducted in 1992 (MRDI, 1997) identified 142 gold grains that ranged in size from 4 to 1,400 microns, with an average size of 155 microns. This average size was considered by MRDI to be sufficiently coarse to cause a “nugget effect” in resource estimation (MRDI, 1997). A second mineralogical study in 1993, which examined 13 gold grains that ranged from 30 to 322 microns in size with an average size of 118 microns, was also sufficiently large to be considered coarse-grained.

Mineralization occurs as elemental gold, both as isolated grains and most commonly in association with arsenopyrite, and less commonly with pyrite and chalcopyrite. In descending abundance, the principal sulphides present are pyrrhotite, pyrite, arsenopyrite and chalcopyrite. Minor sphalerite and galena are also present. Scorodite and limonite are common weathering products.

10.0 EXPLORATION

Exploration on the Dublin Gulch property dates back to 1895 (see Section 6). Until 1991, most of the exploration was focused on the placer mining and the tungsten skarn deposit of the Mar Zone (2004, November, R Goodwin, N. Burns, Snowden Mining industry consulting, Technical Report for the Eagle Zone of the Dublin Gulch Project).

- During 1991, two kilometres of trenching was completed in five trenches and 921 channel samples were cut. During the same year, 16 HQ-NQ diamond drill holes were completed totalling 2,410 m of core produced from the program (Amax).
- An additional 46 reverse circulation (RC) holes were completed in 1992 totalling 5,651 m. Furthermore an extensive sampling, mapping and property evaluation program was conducted.
- In 1993, Ivanhoe Mines conducted detailed mapping, 250 of trenching, soil sampling, geophysics, baseline environmental monitoring, mineralogical and metallurgical studies and completed a drilling of ten RC holes with total of 2,078 meters.
- Aurum Geological Consulting Ltd. conducted a soil auger sampling in 1994.
- First Dynasty Mines Ltd. conducted an extensive drilling program in 1995. A total of 8,347 m of RC drilling was completed in 40 holes. In addition, 4,480 m of HQ core was drilled in 34 diamond drill holes and 1,038 m of PQ core was drilled in five holes for metallurgical testing.
- New Millenium Mining Ltd. drilled an additional 5,399 m of HQ core in 54 diamond drill holes. The purpose of this additional drilling was for exploratory purposes, additional resources definition, resources metallurgical studies and geotechnical orientated and non-orientated core. An additional 37 RC holes were completed of total 5,271 m. During the same year, a total of 460 m of trenching, 299 geotechnical test pits, 700 soil samples and a legal survey of the critical claims on the property was completed.
- StrataGold conducted an extensive drill program in 2005 completing 34 diamond drill holes with total of 8,105 m of drilling. Since then, the company carried out a continuous exploration program.
- In 2006-2007 a drilling program consisted of a total of 9,809 m of core drilling from 30 diamond drill holes.

- The most recent program was completed in 2008 which consisted of 15 holes totalling 4,249 m.

11.0 DRILLING

11.1 DRILLING

In 2005, StrataGold drilled 34 holes with an aggregate length of 8,105 m. The program had two main purposes: the first was to test the west margin of the area for which a resource had been estimated, and an open pit had been designed by MRDI in 1997; and the second was to provide fill-in data in several areas of the designed pit. Holes were numbered starting with #276 (DG05-276C) to maintain the historic sequence. The "C" designation stands for core.

All holes were drilled on a bearing of 0° at dips of 50° to 55°. Holes were located by GPS, and collar locations were checked a second time at the completion of the hole after the drill had been removed. Collars of completed holes were marked by posts or tripods with an aluminum tag bearing the hole number, UTM coordinates (NAD 83, NAD 27, historic NAD 27*), collar elevation and completed hole length. Casing was not left in the holes, and the holes were neither cemented nor plugged.

Each hole was surveyed using the Reflex EZ-Shot optical system. Three to four surveys were made at the completion of each hole at approximately equal intervals down the hole. Readings were documented on the corresponding drill log cover sheet, and the field tabulations are filed with other data for each hole.

The 2006-2007 drill program consisted of 30 holes totalling 9,809 m. The program was targeted to increase the confidence in the resource estimate previously outlined, and to test the gap in the resource model near the western edge. StrataGold commissioned Underhill Geomatics to survey the locations of all the drill holes subsequent to the 2007 drill program. Underhill performed the survey using the NAD83 datum, and the resource calculations have been based on this survey.

Historic (pre-2005) location data, including drill collar locations, were found by StrataGold to approximate NAD 27 coordinates when checked in the field, but were consistently offset by 30 m to the west and 10 m to the north. To make the current and historic location data compatible, StrataGold GPS readings were taken using the NAD 27 datum, and then 30 m was subtracted from the easting and 10 m was added to the northing. Although this introduced a deliberate error, the actual reading is known as is the introduced error. If corrections had been applied to the historical data, the resultant locations would have been presumed but unconformable, and the possibility of ambiguity would have been increased by the introduction of new coordinates different from those published in existing, extensive historical documentation.

The 2008 drill program consisted of 15 holes totalling 4,249 m. The objective of the program was to test the extension of the mineralized zone and consequently increase mineral resources. StrataGold commissioned Underhill Geomatics to survey the locations of all the drill holes subsequent to the 2008 drill program. Underhill performed the survey using the NAD83 datum, and the resource calculations have been based on this survey.

12.0 SAMPLING METHOD AND APPROACH

12.1 SAMPLING METHODOLOGY

Drill core was delivered to the core processing facility by the drill crew at the end of each drill shift. The core was first processed for geotechnical characteristics. Drill-run lengths and depth measurements were checked for errors. Recovery and rock quality were then measured together with fracture angle, abundance and type, weathering (hardness), abundance of fault gouge and breccia, and percentage of broken core. The latter two characteristics were recorded graphically; all other measurements were quantitative. Geotechnical data was recorded on paper and archived with the other paper documentation for each hole; this information was entered into an electronic format.

Prior to logging, core was photographed using a color digital camera. Photos for each hole are archived as electronic files together with other data for each hole.

Core was then logged for lithology, alteration (iron oxide, sericite, chlorite, clay, carbonate or other), veining (number per metre, type, maximum thickness, aggregate thickness, primary and secondary angles), selvedge (width, nature and intensity of alteration, intensity), percent of sulphides, type and degree of oxidation. With the exception of selvedge alteration type, all this data was numeric. This information was recorded in paper format and was then entered into an electronic spreadsheet file of the same format, together with a cover sheet that contains relevant statistics including start and completion dates of the hole, overburden thickness, casing length, collar elevation and coordinates, bearing and dip, core size, sample sequence, sample shipping date, date of receipt of analyses, field duplicate, preparation duplicate, standard and blank sample numbers, and survey data.

After the core was logged it was marked up for sampling. Holes DG05-276C to 297C were sampled in their entirety. With rare exceptions, all samples were 1.524 m (five feet, to coincide with the drill run) in length. For holes DG05-298C to DG05-309C, all intervals of granodiorite intercepts were sampled in their entirety and meta-sediments were sampled in those intervals with quartz vein densities of three or more veins per metre. These intervals were bracketed above and below by one sample length.

Each sample was documented by a four-part, bar-coded sample tag. One part remained in the sample book and three parts were stapled to the core box at the beginning of the sample interval. The stapled portion remains affixed to the core box and the other two parts were placed in the sample bag after the core was cut. One of these two parts was intended to remain with the sample during the preparation process at the analytical laboratory.

Core was then transferred to the core splitting building where it was sawed in half. One half was placed in a heavy-gauge plastic sample bag, and the other was returned to the core box for archival reference. The two-part sample tag was placed in the sample bag and the sample number was written in indelible ink on the outside of the sample bag. Bags were closed with plastic “zip straps” and placed in rice bags for shipping.

The samples were taken to Mayo by the company expeditor and then transhipped via Whitehorse to the ALS Chemex laboratory in North Vancouver.

During the 2008 drilling program, an exception was made to the sample procedure. After the core logging procedure using Gemcom Logger was carried out, the complete core was sent to ALS Chemex laboratory in North Vancouver to be assayed. Consequently there is no split core left in the core boxes for archival reference.

12.2 Quality Control Measures

Quality control protocols for core samples included one duplicate, two laboratory repeats, one standard and one blank sample for each of the 20 samples.

The duplicate was obtained by sawing the designated half-core into quarters. Each quarter was submitted for assay as a separate sample with a unique sample number. The laboratory repeat samples were prepared by ALS Chemex during the sample preparation process. After the designated sample was pulverized, three samples were prepared; two were retained by ALS Chemex for assaying and the third was sent for analysis to ACME Analytical Laboratories in Vancouver.

In the 2008 drilling program, the duplicates were obtained in such a way that the empty bag was put into the sample stream. A sample tag with the label “Prep Duplicate” was inserted into the empty sample bag. These samples were derived from the same core material as the previous sample. ALS submitted this sample to ACME Analytical Laboratories as a prepared duplicate. ALS also assayed this sample and marked it as “Check Assay”.

Seven standards were employed, two obtained from Canmet and five from Ore Research and Exploration (Pty) Limited, Australia (OREAS). The OREAS standards were pre-packaged in sealed envelopes; the Canmet standards were obtained in bulk and were weighed into 30-gram packages at site. One Canmet standard and four OREAS standards were submitted for each 100 core samples.

Commercially-available crushed dolomite was purchased for use as blanks.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 SUMMARY AND PROCEDURES

The thirty-two drill holes from the 2005 drilling program conducted by StrataGold were technically reviewed for quality control by Lynda Bloom, P.Geo. (Bloom, 2005). The thirty drill holes from the 2006-2007 drilling program carried out by StrataGold were technically reviewed for quality control by Wardrop. These technical reviews form the basis of this section of the report.

13.1.1 SAMPLE PREPARATION AND ANALYSES

Approximately 7,583 samples from the 32 drill holes drilled in the 2005 drill program, the 7,408 samples from the 30 drill holes drilled during 2006-2007 and the 2,538 samples from the 15 drill holes drilled in 2008 were prepared and assayed at ALS-Chemex, Vancouver with check assays performed at ACME Analytical Laboratories (ACME) in Vancouver. The sample preparation and assay procedures for each drill program are outlined below:

2005

- 1- 5 kg split drill core samples were shipped to ALS-Chemex.
- Samples were crushed to 70% passing 2 mm.
- 250 g was riffle split from the crushed sample.
- Of the 250 g crushed sample, a sub-sample was pulverized to better than 85% passing 75 microns
- Pulps were analyzed for gold by fire assay on a 30 g aliquot with an ICP finish (Au-AA23 code).
- Pulps were analyzed for a multi-element suite by four-acid digestion and ICP-OES finish (ME-ICP61 code).

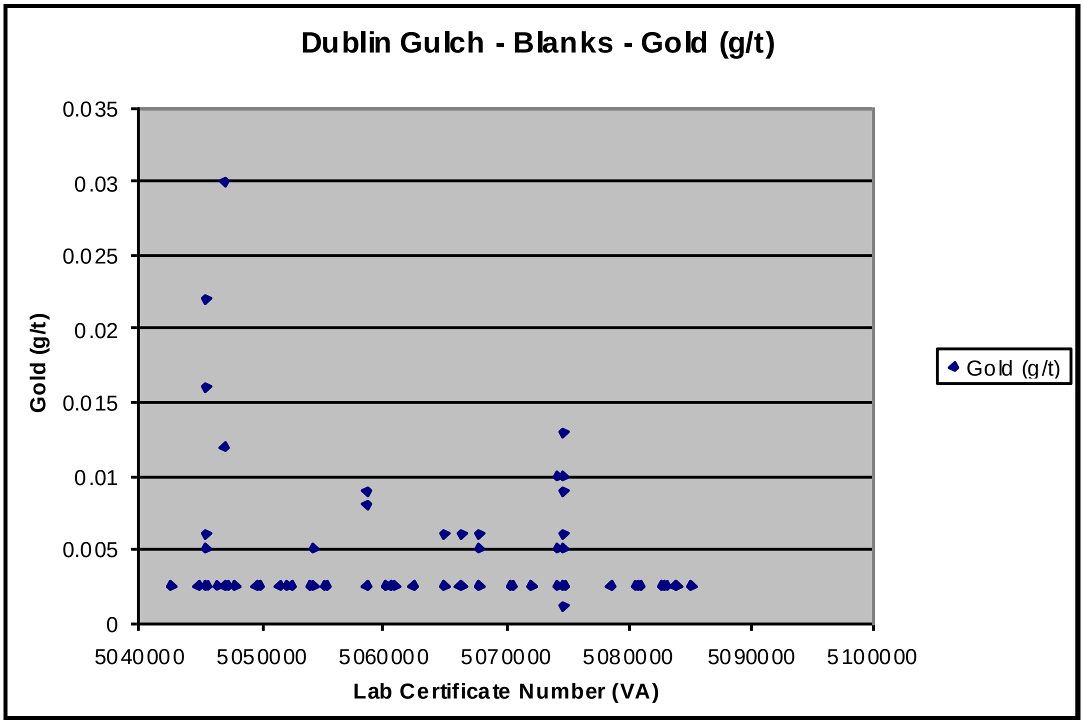
2006-2007

- 1-5 kg split core samples were crushed to 85% passing 2 mm.
- 1,000 g was riffle split from the crushed sample.
- Of the 1,000 g crushed sample, a 150 g sub-sample was collected by taking a number of scoops at right angles through the homogenized 1,000 g split sample; the remaining 850 g were stored in Terrace, British Columbia.
- The 150 g sample was put in an envelope and sent to ALS-Chemex in Vancouver for analysis.
- Of the 150 g sample, a 50 g sub-sample was collected by taking 2-3 scoops from different places in the envelope.
- The 50 g sample was then used for gold fire assay and AAS (Au-AAS24) with 27 element ICP (ME-ICP61), consisting of 4-acid "near total" digestion by HF-HNO3-HclO4 acid digestion, HCl leach and ICPAES.
- All samples with gold exceeding ten parts per million (ppm) were assayed using fire assay and gravimetric finish using Au-GRA22 if 50 g samples were available and if not, the Au-GRA21 for 30 g samples.

13.1.2 2005 Quality Assurance and Control

In addition to the assayed core samples, a coarse blank sample was submitted for analysis with every 20 samples. The blank material was commercially-available crushed dolomite. A total of 359 coarse blank samples were submitted and gold values for 94% of the blanks reported less than the detection limit of 5 ppb gold. The data was plotted and tabulated by Bloom and verified by Wardrop (see Figure 13.1, which does not include four samples that were outside acceptable limits).

Figure 13.1 Blank Sample Gold Assays



There appears to be no systematic gold contamination from the assaying of blank material at the laboratory.

A reference sample was submitted for analysis with every 20 drill core samples. A total of 368 reference samples from CANMET and OREAS were submitted to identify possible bias within any specific batch of samples, or within the overall data. Results were tabulated by Bloom and were verified by Wardrop as in Table 13.1. The average of all 368 reference material assays is within ±7% of the expected values.

Table 13.1 Reference Standards Assayed During the 2005 Drill Program

Reference Material	No. of Insertions	Expected Au (ppm)		Observed Au (ppm)	
		Mean	Std. Dev.	Mean	Std. Dev.
Canmet CH-4	38	0.88	0.04	0.945	0.400
Canmet Ma-2c	47	3.02	0.13	3.062	0.263
OREAS 42 P	67	0.091	N/A	0.114	0.126
OREAS 51 P	65	0.43	0.023	0.453	0.145
OREAS 6 Pb	59	1.425	0.052	1.421	0.098
OREAS 7 Pb	50	2.77	0.05	2.731	0.285
OREAS 61 Pa	42	4.46	0.13	4.495	0.140
Total	**368**				

ALS-Chemex routinely conducts analysis of pulp repeats for each batch of samples that are assayed to provide a measure of reproducibility related to the uncertainties of the analytical method and the homogeneity of the pulps. ALS-Chemex analyses gold samples in batches of 84 samples. Each batch includes one blank, two internal standards and three repeats.

Variability between pairs of repeat samples, by specific percentage ranges, is shown in Table 13.2. The reproducibility of 87% of the pairs, with values greater than five times the detection limit, falls within ±20%.

Table 13.2 Laboratory Pulp Repeat Pairs: Variability by Percent Ranges

Criteria	N	5%	10%	20%	25%	50%	> 50%
All Samples	562	287	377	451	468	515	47
		51%	67%	80%	83%	92%	8%
> 5 Times Detection Limit	306	146	221	267	277	296	10
		48%	72%	87%	91%	97%	3%

StrataGold submitted two bags of samples having consecutive sample numbers to ALS-Chemex with a request to prepare 250-gram duplicate samples with consecutive numbers. It is noted in Bloom (2005) that these duplicates are not defined in industry as preparation duplicates, since preparation duplicates should be created by splitting a second cut of the crushed sample in the same manner and of the same weight as the original sample. A total of 362 of these pulp duplicates were assayed by ALS-Chemex and the relative percentage differences (RPD), the difference between the original and duplicate assay values relative to the average, were calculated and are summarized in Table 13.3. For sample values greater than five times the detection limit, there are nearly equal numbers of positive and negative RPDs, indicating a lack of analytical bias.

Table 13.3 Duplicate Samples: Relative Percentage Differences

Criteria	Number of Samples	Positive	Negative	Zero
All Samples	362	153	173	36
		42%	48%	10%
> 5 Times Detection Limit	292	137	140	6
		47%	48%	2%

The distribution of duplicate pairs by percentage difference is summarized in Table 13.4. These duplicate pairs are essentially repeat assays of the same 250 g pulp, with the exception that the pulp was split into two 125 g portions and then two 30 g aliquots were assayed. Approximately 61% of the assay values fall within ±20%, and 14% of the assays differ by more than ±50%. The reproducibility of these pulp duplicates is much lower than that of internal laboratory repeats. As suggested and addressed by Bloom, "it is possible that some uncertainty is added when the 250 g pulp is split prior to taking 30 g aliquots for fire assay. Any time that materials are sub-sampled, uncertainty and sampling errors are introduced" (Bloom, 2005).

Table 13.4 Duplicate Samples: Variability by Percent Ranges

Total Samples	Number of Samples > 5 Times Detection Limit	Samples Falling Within Percentage Ranges					
		5%	10%	20%	25%	50%	>50%
362	292	61	118	179	197	250	42
		21%	40%	61%	67%	86%	14%

StrataGold took drill core duplicates, by quartering half-split core. The 364 duplicate sample pair values are plotted in Figure 13.2. Drill core duplicates were summarized in Table 13.5 by determining the RPD. It is evident that there are equal samples in positive and negative territory. There are 273 cases where the mean of the two values is greater than 25 ppb, which is five times the detection limit (see Table 13.6). However, there were only 22% of the assays from core duplicates falling within the ±20% agreement. This low agreement of core duplicates does not necessarily suggest that there is bias evident between the original sample and duplicate halves of the drill core. However, the gold variation may be a function of the rock type mineralization.

Figure 13.2 Scatterplot: Drill Core Duplicate Gold Assay Values

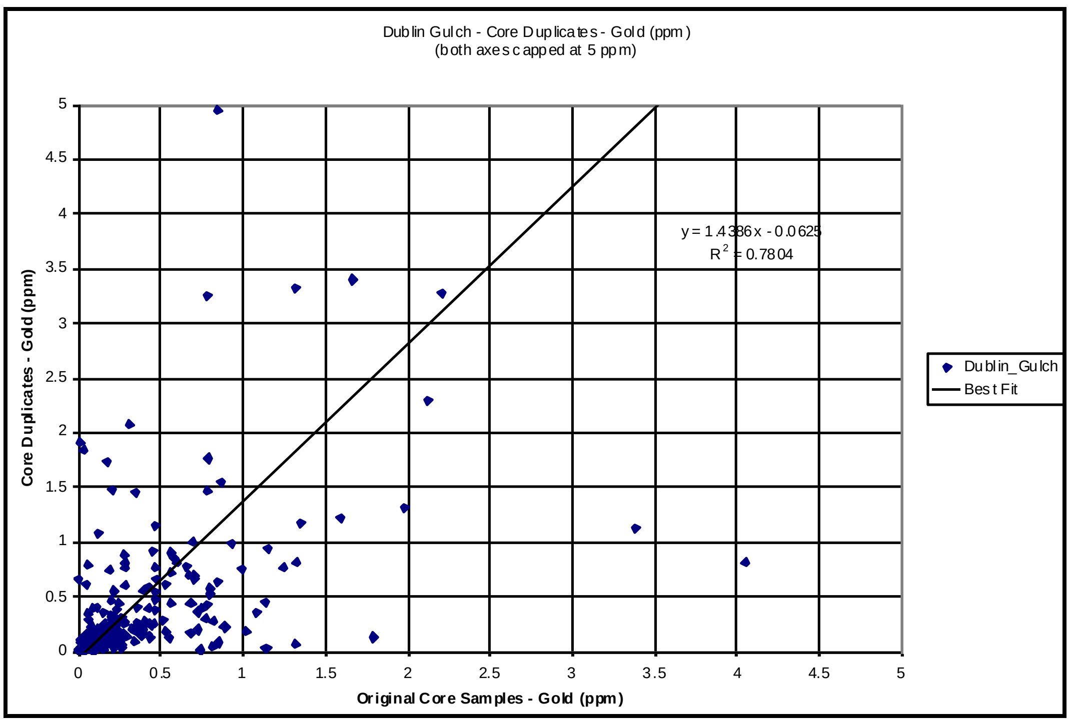


Table 13.5 Summary of Drill Core Duplicate Gold Assay Values

Criteria	N	Positive	Negative	Zero
All samples	364	157	176	31
		43%	48%	9%
> 5 Times Detection Limit	273	136	136	1
		50%	50%	0%

Table 13.6 Drill Core Duplicate Pairs: Variability by Percent Ranges

Element	Total Samples	Samples > 5 Times Dection Limt	Samples Falling Within Percentage Ranges					
			5%	10%	20%	25%	50%	>50%
Au (ppm)	364	273	18	39	60	81	102	123
			7%	14%	22%	30%	37%	45%

A total of 359 check assays were submitted to a referee laboratory (ACME analytical in Vancouver). RPDs between the original and check assays is summarized in Table 13.7. An equal number of positive and negative differences between the values obtained by the two laboratories indicate a lack of analytical bias. Of the 294 samples for which values are greater than five times the detection limit, 157 ACME assays exceed ALS-Chemex values and 130 are lower. Bloom states "the variance in the data could be caused by the cumulative uncertainties in preparation and sample splitting, it is difficult to determine if the bias is statistically significant and possible causes" (Bloom, 2005).

Table 13.7 Summary of Check Assay Laboratory Comparisons

Criteria	Number of Samples	ALS-CHEMEX > ACME	ALS-CHEMEX < ACME	ALS-CHEMEX = ACME
All Samples	363	153	198	12
		42%	55%	3%
> 5 Times Detection Limit	294	130	157	7
		44%	53%	2%

In reviewing and reproducing the 2005 drill program quality control measures, Wardrop agrees with the findings by Bloom (2005).

13.1.3 2006-2007 Quality Assurance and Control

In addition to the assayed core samples, a coarse blank sample was submitted for analysis with every 20 samples. The blank material was commercially-available crushed dolomite. A total of 147 coarse blank samples were analyzed with samples from the eight drillholes included in the resource update (DG06-314C, DG07-327C, DG07-330C, DG07-331C, DG07-332C, DG07-333C, DG07-335C, DG07-338C). Gold values fell below detection limits of 5 ppb gold for 91% of the blanks, exceeded detection limits for 6% of the blanks, and were equal to detection limits for 3% of the blanks. Table 13.8 displays gold values for those blanks which exceeded detection limits.

Table 13.8 Blank Samples Exceeding Detection Limits

Drillhole ID	Sample ID	Au (ppm)
DG06-314C	C082396	0.006
DG07-332C	G017156	0.008
DG06-314C	C082016	0.009
DG06-314C	C082156	0.009
DG07-338C	G019756	0.01
DG07-338C	G019536	0.011
DG07-330C	E688916	0.012
DG06-314C	C082116	0.019
DG06-314C	C082176	0.095

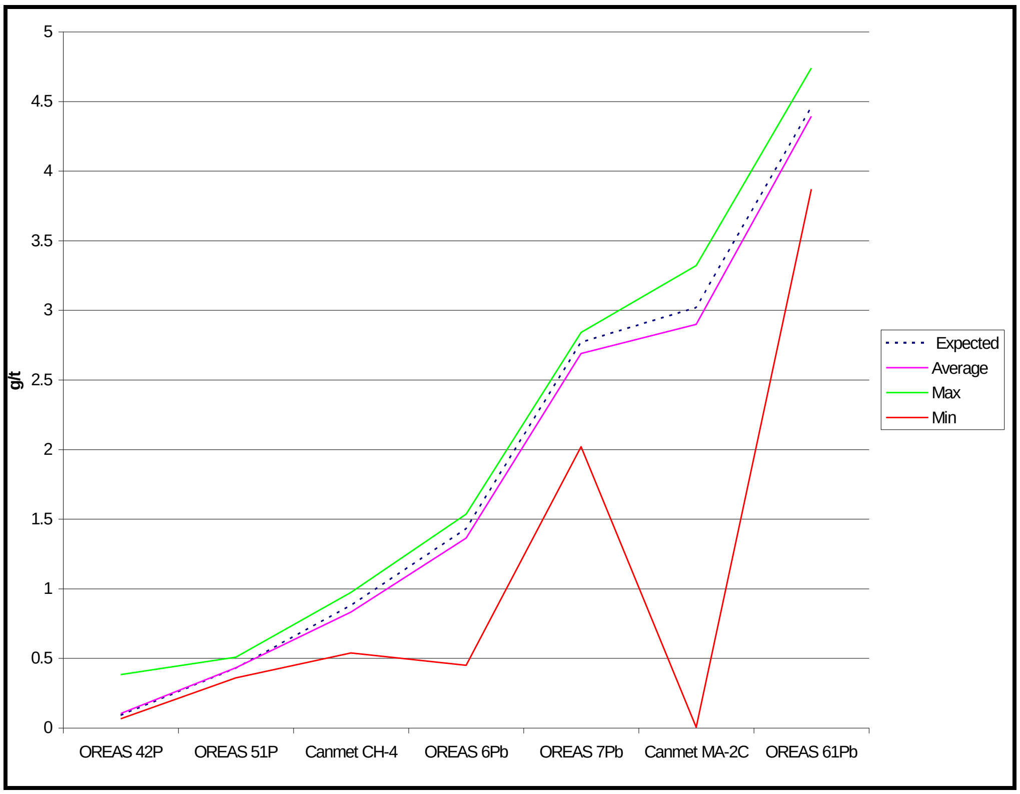
Sample C082176 is regarded as anomalously high for a sample blank, but is still well below the lower cut-off limit for the resource estimate. Adjacent samples from the original lab certificate were reviewed to check for sample contamination. The previous and subsequent samples had gold values of 0.026 ppm and 0.131 ppm, respectively. Overall the blank samples show that there has been no systematic cross contamination between samples at the assay lab.

Reference samples obtained from CANMET and OREAS were submitted for analysis with every 20 drill core samples. A total of 148 reference samples were analyzed with samples from the eight drillholes used to update the resource estimate. There was good overall correlation between the expected standard values and the assayed standard values. The only anomaly consisted of a Canmet MA-2C standard assaying below detection limit (see Table 13.9). A summary of the standard assays is presented in Figure 13.3.

Table 13.9 Reference Standards Assayed in 2006-2007 Program

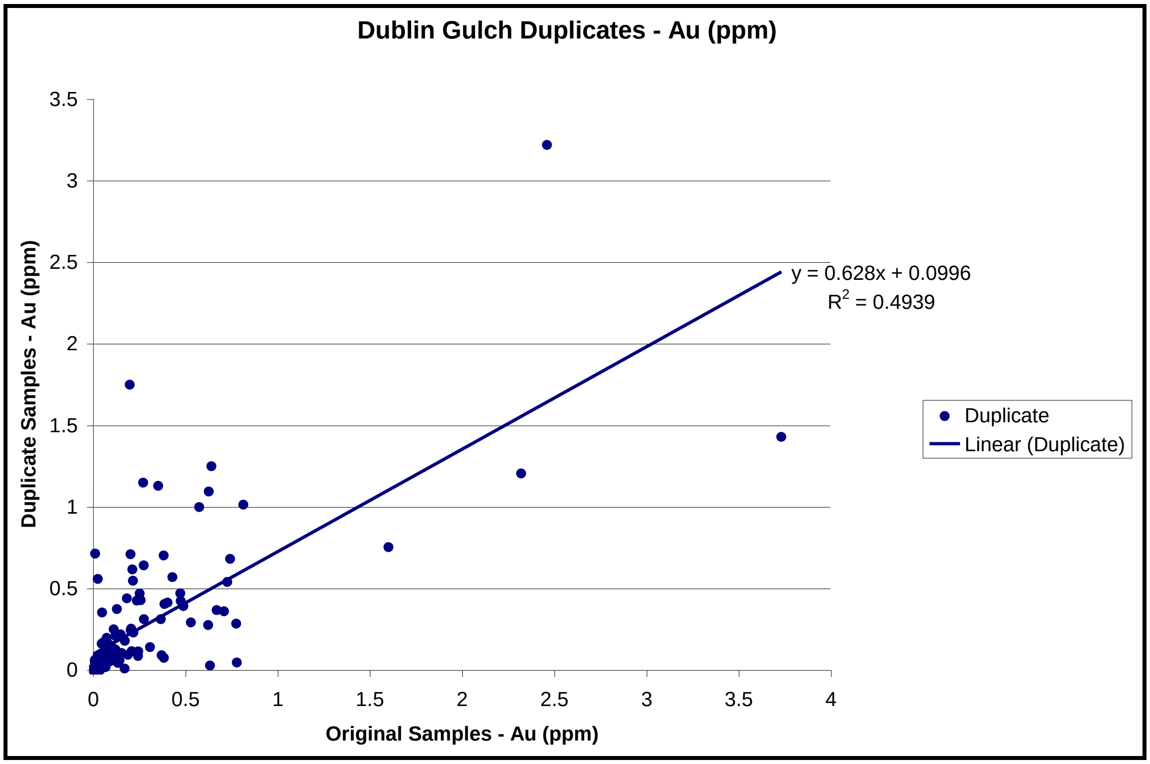
Reference Material		Number	Expected		Observed	
Standard Name	**Logging Code**	**of samples**	**Mean**	**Std Dev**	**Average**	**Std Dev**
Canmet CH-4	1A	22	0.88	0.04	0.83	0.10
Canmet MA-2C	2B	20	3.02	0.13	2.90	0.70
OREAS 42P	3C	23	0.091	n.a.	0.10	0.06
OREAS 51P	4D	22	0.43	0.023	0.43	0.03
OREAS 6Pb	5E	21	1.43	0.052	1.36	0.22
OREAS 7Pb	6F	21	2.77	0.05	2.69	0.19
OREAS 61Pb	7G	19	4.46	0.13	4.39	0.22
Total		**148**				

Figure 13.3 Comparison of Standard Samples Submitted for Assay



During the 2006-2007 drilling program, StrataGold submitted 149 pairs of duplicate samples for assay. The sample duplicates were obtained from quarter sawed drill core, and submitted using different sample numbers than the original samples. The wide scatter and relatively poor correlation between the original and the duplicate core assays suggests that coarse gold is present in the Dublin Gulch deposit (see Figure 13.4).

Figure 13.4 Scatterplot of Duplicate vs Original Gold Values



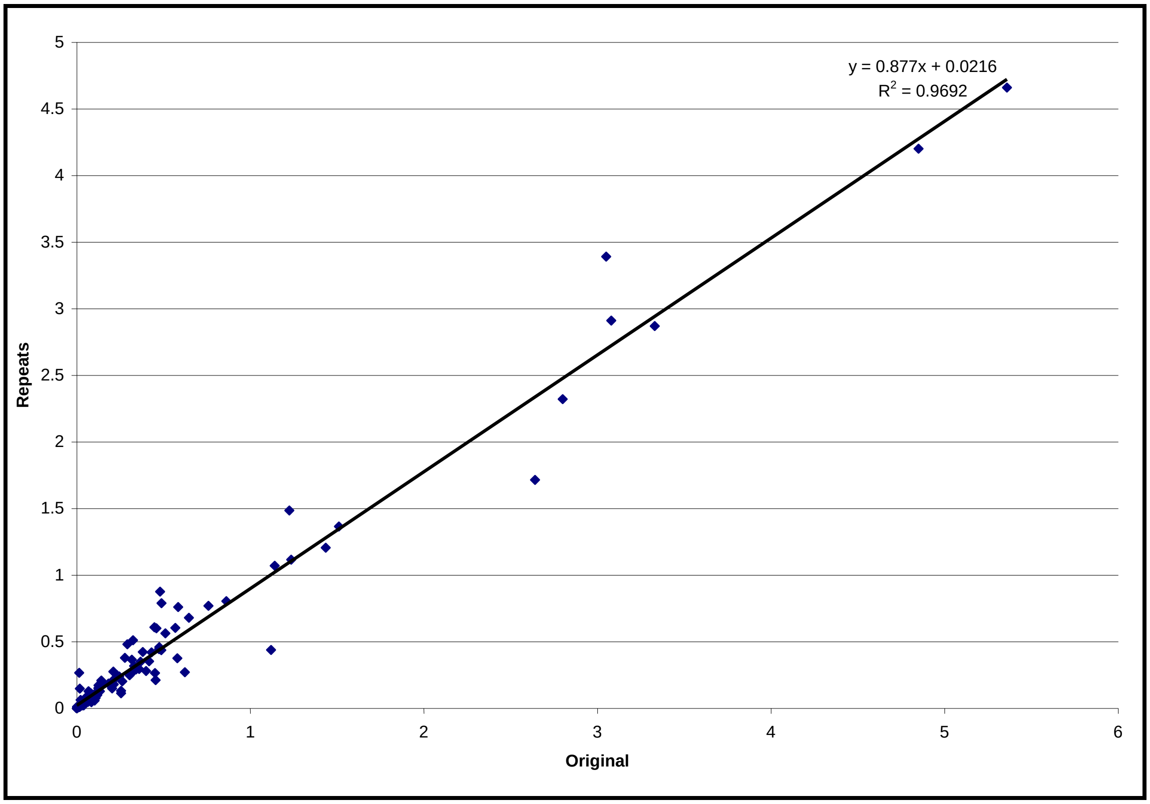
In addition to the sample duplicates, 147 sample repeats were assayed by ALS-Chemex. These sample repeats were taken from samples crushed by ALS-Chemex. This is typically done by assay labs as an internal Quality Control (QC) check. Figure 13.5 illustrates that, with the removal of one anomalous sample from the population, there was good correlation between the two sample sets. The sample repeats removed from the population (see Table 13.10).

Table 13.10 Sample Repeats Removed from the Population

	Sample	Au g/t
Original	G017228	9.02
Repeat	G017229	3.93

The high values in both these samples relative to the overall resource grade are typically seen in deposits that carry coarse gold.

Figure 13.5 Scatterplot of ALS-Chemex Internal Repeat Assays



There were 58 sample repeats from the 2006-2007 drill program that were assayed by both ACME labs and ALS-Chemex that were available at the time of this report. Again, when one anomalous sample in the population was removed, there was very good correlation between the two laboratories.

The anomalous sample that was removed is reported in Table 13.11.

Table 13.11 Anomalous Sample Removed from the Population

Sample	Assay Au g/t	
Number	ACME	ALS
G019568	2.434	0.368

As in previous duplicate and repeat assay programs, the large difference and high grade suggests the presence of coarse gold in the samples.

13.1.4 2008 Quality Assurance and Control

The following Protocols and Procedures were used for every 20 sample sequences i.e. samples 1-20, samples 21-40, samples 41-60 and so on.

PREPARED ORIGINAL

For every 8th, 28th, 48th, etc. sample, three 500 g samples are prepared by ALS. One sample is sent to ACME for the 'Prepared Duplicate'.

PREPARED DUPLICATE

For every 9th, 29th, 49th etc. sample, an EMPTY bag was put into the sample stream. A sample tag labelled "Prep Duplicate" was inserted into the empty sample bag. These samples are derived from the same core material as sample #8. ALS submits this sample to ACME as a prepared duplicate. ALS also assays this sample and marks it as a 'Check Assay'.

STANDARDS

For every 10th, 30th, 50th etc. sample, a prepared STANDARD was entered into the sample stream. Standard types are randomly chosen when inserting into the sample bag. The Standard type was carefully recorded on sample tags in order to verify the lab assay result. The following standard reference materials outlined in Table 13.12 were used.

Table 13.12 Reference Standards Assayed in the 2008 Program

Code	Au (ppm)	Cu (ppm)	Cert. Ref. Number	Producer	Comments
STANDARD	0.88	0.2	CH-4	Canmet	30.5 gram packets
STANDARD	3.02	0.095	Ma-2C	Canmet	30.5 gram packets
STANDARD	0.091 (± 3ppb)		42P	Oreas	Greywacke, 60 gram packet
STANDARD	0.43 (±		51P	Oreas	60 gram package
STANDARD	1.43 (±		6Pb	Oreas	Greywacke, 60 gram packet
STANDARD	2.77 (±		7Pb	Oreas	Greywacke, 60 gram packet
STANDARD	4.46 (±		61Pb	Oreas	Meta-Andesite, 60 gram
STANDARD	1.02		15Pa	Oreas	
STANDARD	1.61		15Pc	Oreas	
STANDARD	0.841		50Pb	Oreas	
STANDARD	0.623		53Pb	Oreas	

BLANK

For every 16th, 36th, 56th etc. sample, a blank sample of dolomite rock was placed into a sample bag along with a regular sample tag. This BLANK material was purchased in bulk bags and sourced from Home Hardware in Whitehorse.

Note: No field duplicate samples were sent as whole core was sent for analysis.

During the 2008 drilling program, StrataGold requested the screen fire assay methodology (SCR), which was applied on 1,564 samples. A total of 437 samples were assayed using both, fire assay (AA) and SCR methodologists. Consequently, a total of 1,127 samples were assayed using only SCR methodology which reflects 44% of the total number of assay results obtained for the 2008 drilling.

Analyzing the complete dataset and the impact of the results obtained using SCR method indicates only 3.21% on a total of 35,112 assay results belongs to this group of data (see Figure 13.7).

Figure 13.6 shows the correlation between AA and SCR methodologist in 2008 drill program.

Figure 13.6 Scatterplot of ALS-Chemex vs. ACME Labs

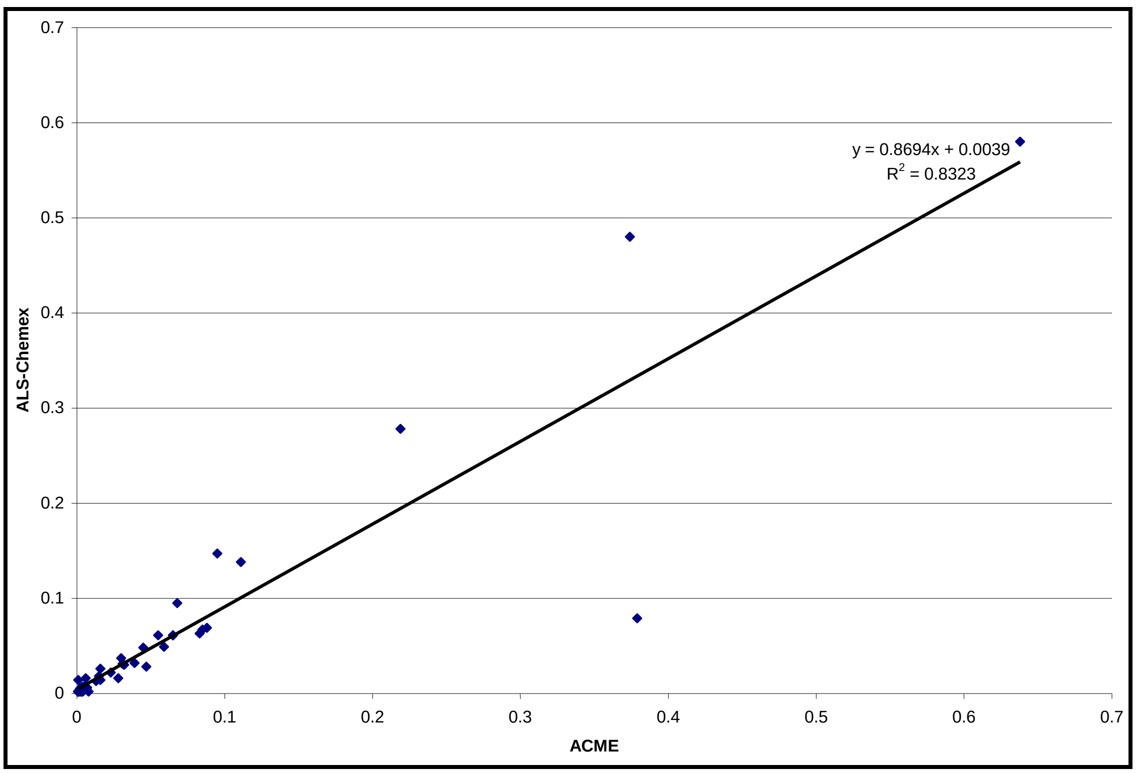
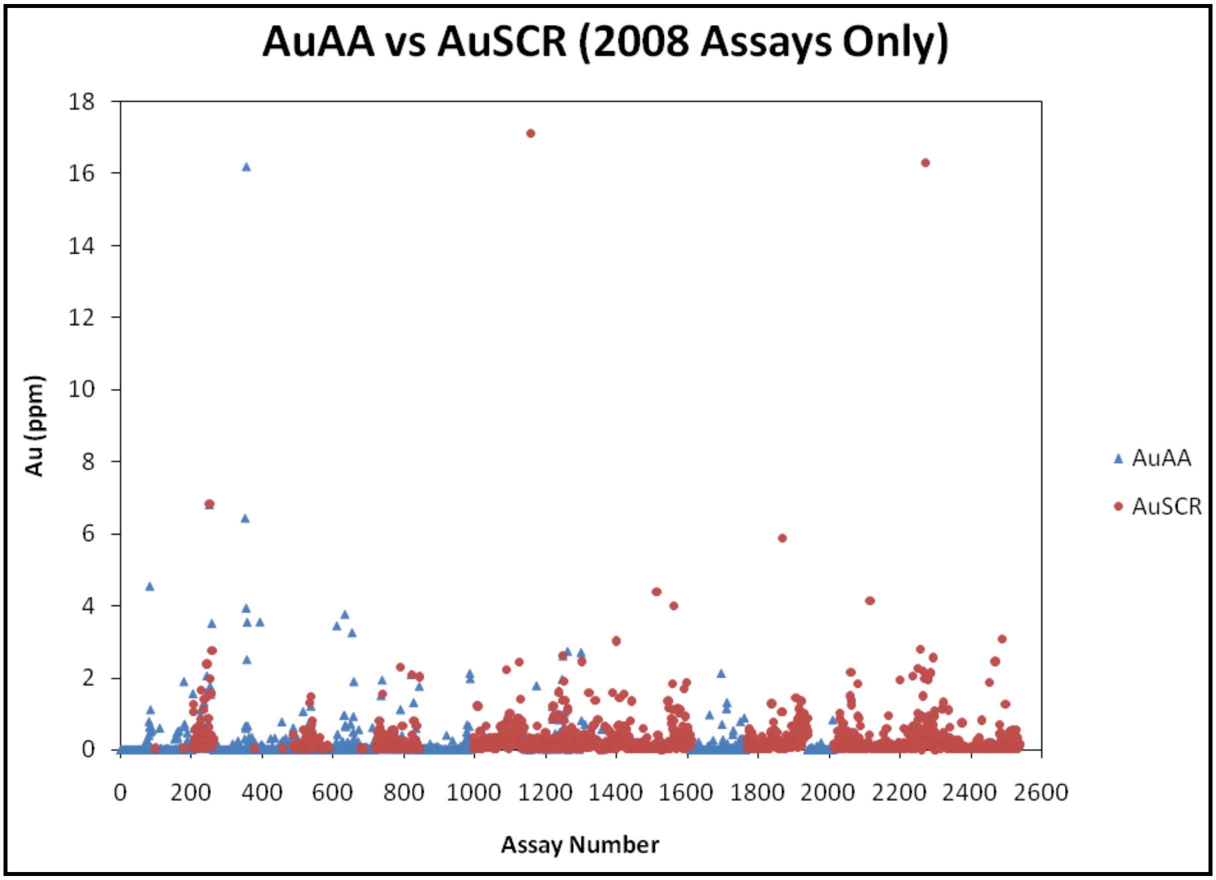


Figure 13.7 Scatterplot of AA Assay and SRC Assay of Gold Results from the 2008 Drilling



13.2 Historic Sampling Procedures

The historic drill holes were reviewed in detail for quality control by MRDI (MRDI, 1997) and Snowden (Goodwin, 2004). These technical reviews form the basis of this section of the report.

13.2.1 *Historic Sampling Method and Approach*

During the 1991 and 1992 drilling programs, no sampling methodology was made available for scrutiny.

Samples collected during the 1993 drilling program, which consisted of ten reverse circulation holes, were quarter-splits of drill chips obtained at the drill site using a two-tier riffle splitter for dry samples and a rotary splitter for wet samples (MRDI, 1997). When high water flow was encountered, samples were collected by use of three tiered buckets with plastic baffles. Overflow of the third bucket was not sampled. The sample interval used was 1.5 meters (m).

The 1995 drilling program consisted of both core and reverse circulation drilling with sample preparation and analysis performed at Chemex Laboratory in Vancouver and coarse-reject duplicate assays by Min-En of Vancouver. It was reported by MRDI that sample preparation and analytical procedures were similar to those used in 1993 (MRDI, 1997).

With the exception of dry samples that were split with a Jones riffle splitter in camp, the intermediate grinding was stipulated to be 60% passing –60 mesh screen (utilizing a modified Bico rotary pulverizer), and screen fire assays were completed on one kilogram splits (rather than 4 kg) (MRDI, 1997).

The 1996 drilling program consisted of both core and reverse circulation drilling. The description of sampling methodology is extracted from Snowden (Goodwin, 2004). The entire core was sampled in sample lengths of 1.5 m or 5 feet (ft). Fines from the box were collected and included with the last sample.

Holes drilled by reverse circulation were cased to 20 ft utilizing an Odex hammer. If the hole was collared in bedrock only a 10 ft casing was inserted. Drilling was conducted with a face-centered 5-3/8” drill bit using 20 ft drill rods. Holes were sampled at five foot intervals. Drilling was conducted as dry as possible and 100% of the sample was collected using a cyclone mounted on the side of the drill.

A hopper gate was attached at the bottom of the cyclone to feed one of two bags, mounted on each side. Each bag could contain 2.5 ft of sample. The hopper gate was removed for wet samples and a rotary splitter was attached to the bottom of the cyclone. This rotary splitter had 16 equal-sized troughs, eight at discharge and another eight to the side of which four were blocked.

A cascading bucket system collected the wet sample. A plastic garbage can was placed under the side discharge to collect the 25% fraction. A v-notch at the top of the can allowed for overflow and was collected in a five gallon pail. This overflow from the five gallon pail was not collected. It was assumed the potential gold loss from this overflow was minimal due to the low clay content.

13.2.2 *Historic Sample Preparation, Analyses*

No information was made available regarding the sample preparation, analyses or security of the 1991 program.

The 1992 drilling samples were prepared by Bondar-Clegg and the following were the procedures used for analyses of the samples (MRDI, 1997):

- 10 to 20 kg of drill chips from the reverse circulation were dried, weighed and crushed to –10 mesh.
- From this crushed sample, a riffle split of approximately 5 kg was extracted and pulverized to –80 mesh in a ring and puck pulverizer.

- A 250 g split was then pulverized to 90% passing a 150 mesh screen.
- Atomic absorption spectroscopy with a detection limit of 0.034 g/t was the mode of final analysis (Snowden, 2004). Any samples with one assay-ton greater than 2 g/t were analyzed by fire assay.

The 1993 sample preparation was performed by ALS-Chemex laboratory in Vancouver with procedures similar to those used by Bondar-Clegg in 1992, with the exception that Bondar-Clegg used a ring-and-puck pulverizer to reduce material from –10 mesh to –80 mesh and ALS-Chemex used a disc-type grinder.

Sample preparation from the 1995 drill program was also performed by ALS-Chemex laboratory in Vancouver, with coarse-reject duplicate assays by Min-En of Vancouver. The procedures were similar to those used in 1993, except that intermediate grinding was stipulated to be 60% passing a -60 mesh screen, using a modified Bico rotary pulverizer.

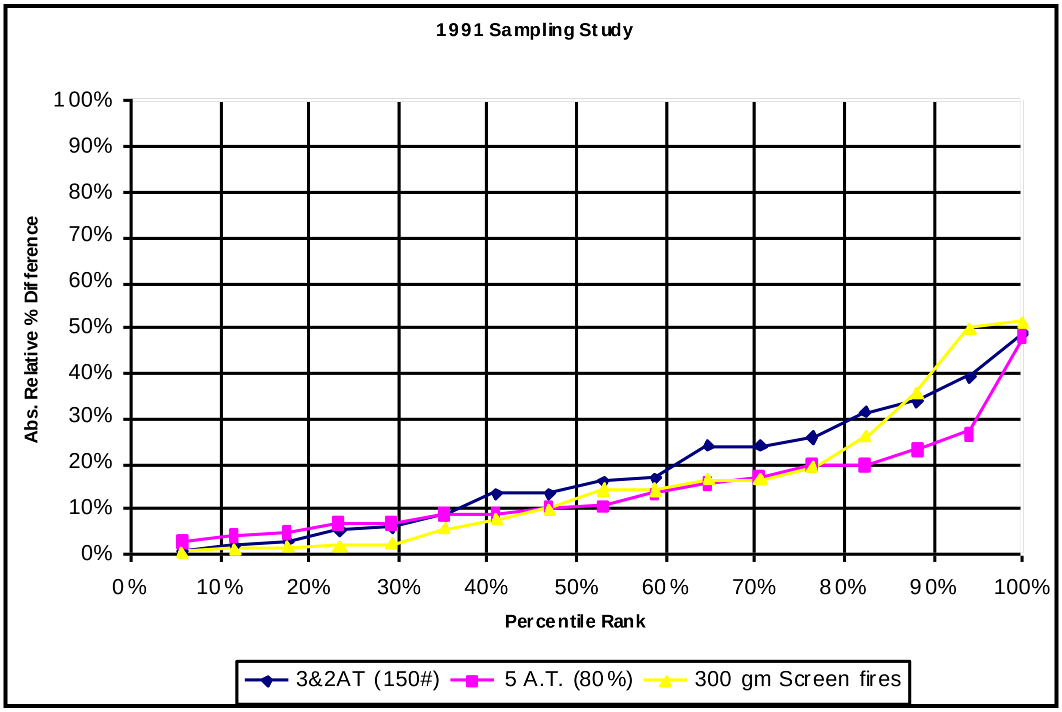
The 1996 samples of meta-sediments were analyzed by means of one assay-ton aliquots with an atomic absorption finish. The granodiorite, or intrusive sample intervals, were assayed by the following procedures (Goodwin, 2004):

- The entire sample was crushed to –10 mesh (60% as per ALS-Chemex normal procedure.
- A two kilogram split was extracted using a riffle splitter, and the reject was saved,
- The split was pulverized to 90% of the material passing through a 100 mesh screen.
- The split was further sub-split into two one kilogram samples using a riffle splitter (one for assay and one archive).
- The one kilogram sub-split was screened, and the oversize assayed in entirety.
- The undersize material was riffle split to approximately 200 to 250 g using a quarter riffle splitter (the reject was saved).
- Samples were placed in an envelope and sent to the assay laboratory.
- The samples sent to the laboratory were re-blended after screening by repeatedly passing them through a riffle splitter that used a 1/8th micro splitter on a vibrating base and recombining. Then they were sub-sampled by repeatedly riffle splitting the material until a weight of between 25 and 50 g was obtained.
- The final split was assayed entirely allowing the weight to vary (variable fusion weight).

13.2.3 *HISTORIC QUALITY ASSURANCE AND CONTROL*

The sampling of the 1991, 16 core drill program consisted of 300 g screen fire assays, five assay-ton fire assays on splits of –80 mesh material, and two or three assay-ton determinations on splits of 150 mesh material. Figure 13.8, reproduced from MRDI, displays similar levels of reproducibility between the methods of assaying (MRDI 1997).

Figure 13.8 Sampling Study Results (1991)



In 1992, 46 reverse circulation holes were completed on the Property as well as a mineralogical study on the gold size distribution. The study indicated that gold is on average 155 microns in size which was considered sufficiently coarse to cause possible lack of reproducibility of assay results.

Bondar-Clegg conducted fire assays on the 1992 drilling samples. Quality assurance procedures included:

- Collecting a second sample on every tenth at the drill site as duplicates
- Crushed gravel was submitted as a blank with each job
- Standard reference materials from Canmet, Nevada Bureau of Mines and Amax Gold were inserted with every 20th sample
- Bondar-Clegg analyzed the pulp of every tenth sample twice
- A second pulp was prepared from every fortieth sample as a coarse reject.

No quality control assay data information was made available from the 1992 drilling to conduct analyses.

The 1993 drilling of reverse circulation holes used the following quality assurance procedures:

- Collecting a second sample for every tenth sample at the drill site as duplicates
- Crushed gravel was submitted as a blank into the sample stream following suspected high-grade intervals and randomly
- Standard reference material from Canmet, Nevada Bureau of Mines and Amax Gold were inserted with every 20th sample.

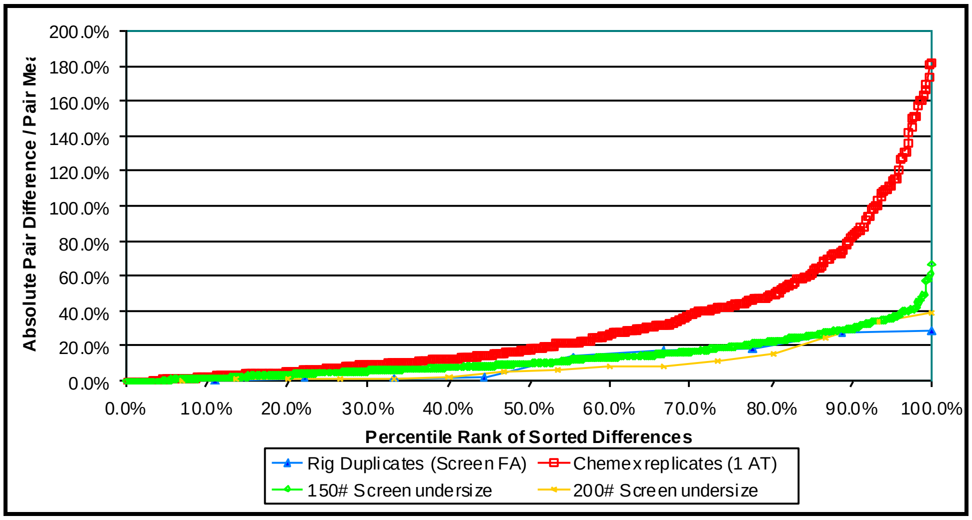
No quality control assay data information was made available from the 1993 drilling to conduct analyses.

The 1995 program of both core and reverse circulation drilling used the following quality assurance procedures:

- ALS-Chemex conducted internal repeats for their own quality assurance
- All one-assay-ton samples that assayed greater than 2 g/t were fire re-assayed using the –60 mesh coarse reject; in addition, other screen fire assays were performed on samples below the 2 g/t limit
- Coarse reject duplicate assays (second split of the –60 mesh reject) were submitted to Min-En for a one assay-ton fire assay check
- Drill rig duplicates that were prepared from a second quarter-split of the reverse circulation drilling samples were submitted to ALS-Chemex
- Standard reference material was inserted into the drill sample stream.

Figure 13.9, reproduced from MRDI, compares the duplicate samples from the 1995 drill program (MRDI, 1997). Fifty-five percent of the Chemex one assay-ton replicates differ by no more than ±20%, and 10% of the results differ by ±80%.

Figure 13.9 Drill Program Duplicate Samples (1995)



The 1996 program, consisting of both core and reverse circulation drilling, used the following quality assurance procedures:

- ALS-Chemex selected an internal same-pulp duplicate for their quality assurance
- Drill rig duplicates that were prepared from a second quarter-split of the reverse circulation drilling samples were submitted to ALS-Chemex
- Reference material standards were inserted for every 20th sample
- Cone Geochemical performed check assays from the undersized fraction on the same pulp using a one-assay-ton fire assay
- Screen fire assay at the –10 mesh was performed on the duplicate one kilogram samples and the minus fraction at the 250 g stage.

No quality control assay data information was made available from the 1993 drilling.

Wardrop reviewed the documentation, graphs and charts prepared by MRDI from the historic drilling and reproduced information where possible (MRDI, 1997). The sampling procedures from 1991 to 1996 drill programs do not appear to identify any systematic bias of the sampling. Quality assurance measures are consistent with industry standards. The low reproducibility of the duplicate and check samples may be due in part by the coarse-grained nature of the gold on the Property (MRDI, 1997).

14.0 DATA VERIFICATION

14.1 HISTORIC

MRDI conducted an extensive statistical review of the Eagle Zone drill data used in their 1997 resource estimate (MRDI, 1997). This data was generated during the period 1991 through 1996 inclusive, and was comprised of 5,305 core samples (30%) and 12,241 reverse circulation samples (70%). Approximately 75% of this data was generated during the 1995 and 1996 drill campaigns. All drill program sand core and reverse circulation employed quality assurance measures and the details of these measures, as well as the analytical procedures known to and evaluated by MRDI.

MRDI checked about five percent of the database against corresponding geological logs and assay certificates. For the portions checked, error rates were 0.53% for the 1991 data, 0.02% for 1992, 0.23% for 1993, 0.9% for 1995 and zero for 1996 (MRDI, 1997).

In 2004, Snowden reviewed the MRDI resource estimate, including data verification, and concluded that the data had been verified to a level compatible with the requirements of NI 43-101 (Goodwin, 2004).

Wardrop completed checks on the historic data verification by reproducing histogram and probability plots on the various core and reverse circulation drilling campaigns, and compared results of the twinned core and reverse circulation drill holes. The historic verification check did not reveal any discrepancies with MRDI findings (MRDI, 1997).

14.2 2006

Wardrop completed an initial data validation check on 24 drill holes from the 1991, 1992, 1995, 1996 and 2005 drill programs from copies of original drill logs and assay certificates with the database. Validation was conducted on the assay information by checking the sample intervals and corresponding gold values and sample identification numbers (see Table 14.1). Survey depth information was compared to corresponding dip and azimuth with no errors reported.

Table 14.1 Database Validation Errors

	Samples	Records	Error Records	% of Records with Errors
Assay	3,559	10,352	154	1.49%

The percentage of errors was larger than the 1% threshold required by Wardrop, so further database verification was necessary. In reviewing the errors, 60 of 154 were found to relate to two drill holes from the 1991 drill program. Wardrop decided to complete a 100% check on the 14 remaining drill holes from the 1991 drill program. Of the 1,266 records in the 14 drill holes, only six errors were identified.

14.3 2008

Wardrop completed an initial data validation check on the 45 drill holes from the 2006-2007and 2008 drill program. The database was manually compared to copies of original drill logs and assay certificates. Validation was conducted on the assay information by checking the sample intervals and corresponding gold values and sample identification numbers. Because data transfer was electronic, there seemed to be little chance for error in the nominal assay values, so checking was limited to samples that assayed above or below detection limits. This also filtered the data set down to near the 5% of data check level prescribed for this report. From the 2006-2007 drill programs, 361 of the 7,220 samples were checked, and there were four inconsistencies. The inconsistencies appeared to be rounding errors that occurred in the last two decimal places (typically .005 was imported as .01). Survey depth information was compared to corresponding dip and azimuth. Overall, the percentage of errors was below than the 1% threshold required by Wardrop, so the database was accepted for resource estimation purposes.

During the site visit prior to the 2008 drilling program, a handheld GPS was used to record the locations of five drill collars. Four of the collars were from the 2007 drill program, and one was from the 2005 program. The aluminum tags on the tripods were photographed, and the locations recorded by the handheld GPS were compared to the database. Diamond drill hole locations had previously been recorded in NAD27, modified NAD27, and NAD83 datums, and some co-ordinate system confusion existed. Underhill Geomatics was commissioned to survey the drill collar locations in 2007. The collars were surveyed by GPS with differential correction. Collar locations were recorded in the NAD83 datum, and the database was adjusted to reflect this survey.

An additional site visit was completed in November 2008 during which three drill collars from 2008 drill program were recorded. The location was recorded with a handheld GPS and compared with the database. The diamond drill location was recorded in the NAD83 datum. The aluminum tags on the tripods and casing that remained in one of the collars were photographed.

The drill core from the 2006-2007 drill program was cross piled in the Dublin Gulch camp, and covered with tarps. The core that could be viewed corresponded to geological descriptions of the property. The camp was not in operation at the time of the visit, so the author was unable to view the sampling procedure, although descriptions of the sampling procedure agreed with observations made at the site.

15.0 ADJACENT PROPERTIES

There are no adjacent properties that are relevant to the understanding, exploration or evaluation of the Eagle Zone of the Dublin Gulch Property.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 METALLURGY

In late 2006, Process Research Associates Ltd. (PRA) carried out laboratory metallurgical test work on of four gold composites from the Eagle Zone on the Dublin Gulch Property(26). Mr. Jim Smolik from TJS Mining-Met Services directed the work program and identified the composite ore rock samples to be retrieved from drill core stored on the Dublin Gulch Property. Sixty-two bags of drill core from the 2005 and 2006 drill programs, conducted by StrataGold, were sent to PRA. The objective of the program was to investigate the response of four mineral types to gravity floatation and gravity-cyanidation processing, and to assess the mineralogical characteristics and nugget gold occurrences.

Based on the two processes tested, gravity-floatation versus gravity-cyanidation, the composites responded better to the gravity-floatation process for all but the silicified composite. The higher grade silicified composite had readily accessible gold and apparently a lower refractory content. However, both processes displayed good overall recoveries with an average gravity-floatation recovery of 94.63% and the gravity-cyanidation process recovering 90.63% of the gold. Under the conditions tested, the feed particle size did not seem to affect gold extraction. A low cyanide consumption for all the composites averaged 0.27 kilogram per tonne (kg/t).

In 1980, Bacon Donaldson and Associates Ltd. conducted metallurgical tests on a 200-pound “representative” sample from several veins from the Eagle Zone (Rescan, 1997). This material was subjected to three recovery tests: gravity separation (jig concentration), sulphide flotation and cyanidation. Gold recoveries were 22.96% for gravity separation, 63.3% for flotation and 82.3% for 72-hour cyanidation. Gold recoveries by ore type were determined by Rescan and are listed in Table 16.1.

Table 16.1 Eagle Zone Weighted Average Gold Recovery (Rescan, 1997)

Ore Type	Weighted Average Gold Recovery			
	Intrusive		Mixed	
	Au < 1.3 g/t	Au > 1.3 g/t	Au < 1.3 g/t	Au > 1.3 g/t
A – Weathered (Weathered granodiorite high iron oxide intensity)	80%	85%	72%	77%
B – Unaltered (Fresh to weakly altered granodiorite, low iron oxide intensity)	72%	77%	72%	77%
C – Sericitic (Sericite altered granodiorite)	80%	85%	72%	77%

In 1997, Kappes, Cassidy and Associates conducted metallurgical tests on approximately 13 tonnes (t) of drill core to provide design criteria for a planned heap leach process (Goodwin, 2004). Column-leach and bottle-roll cyanidation tests were performed on three types of ore: weathered granodiorite, unaltered granodiorite, and sericite-altered granodiorite. These tests indicated that gold recoveries of 78% ±5% could be achieved on material, of which at least 80% passed through a 5 mm screen, over a leach period of 150 to 180 days. Gold grade and ore type appear to have an insignificant impact on gold recovery.

In 1996 a series of environmental and metallurgical tests were conducted on core samples from Dublin Gulch. The test program consisted of:

- Head characterization to determine both elemental as well as environmental considerations.
- Cyanide bottle roll leach tests.
- Tests to determine agglomeration parameters.
- Compaction tests were conducted to determine the affect of high loading conditions on the permeability of the bottom section of a multiple lift heap leach.
- Column leach tests to determine the amenability of the material to coarse and fine crushed heap leaching, two of which were conducted between +2˚C and -2˚C and parameters under which heap leach solutions could be neutralized.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

A mineral resource estimate of the gold content of mineralized zone at Dublin Gulch was based on a previous estimate conducted by Wardrop in 2006. This estimate has been made using data from both historic drilling and that generated by the 2006, 2007 and 2008 drill programs. A database was compiled using data from 293 core holes, 158 reverse circulation holes and four trenches, and includes collar, survey, geological and assay information. Table 17.1 summarizes the records in the database.

Table 17.1 Summary of Database

Drill Holes or Trenches	Collar	Survey	Geology	Assay
Records	455	793	1,884	35,306

In assembling the database for the 2006 estimate, a systematic approach similar to that employed by MRDI was used to select assay data for core and reverse circulation drill samples. The indicated recoveries were less than 50-60% or greater than 120% of the sample interval (MRDI, 1997). Intervals noted in the drill logs as being subject to caving, drilling problems, high water inflow or some form of contamination were also removed from the database. A total of 1,750 sample intervals from 41 cored and 24 reverse circulation holes were removed from the resource estimation database. Table 17.2 summarizes the samples removed from the database by drill hole and intervals. All the diamond drill holes data from the 2006, 2007 and 2008 drill programs were appended to the database for the 2009 resource estimate. Drill hole collar locations for all the drill holes were adjusted as reported by the Underhill Geomatics collar survey.

Table 17.2 Samples Removed from Resource Estimation Database

Drill Hole	From	To	Reason
91-003C	27.43	83.82	Low core recovery, generally <50%
91-004C	19.81	54.86	Low core recovery, generally <60%
91-004C	79.25	91.44	Low core recovery, generally <60%
91-005C	0.00	25.91	Low core recovery, generally <60%
91-009C	0.00	42.67	Low or no recovery, generally <60
95-074C	0.00	243.20	Not sampled, PQ metallurgical hole
95-075C	0.00	234.60	Not sampled, PQ metallurgical hole
95-076C	0.00	260.00	Not sampled, PQ metallurgical hole
95-077C	0.00	199.60	Not sampled, PQ metallurgical hole
95-078C	0.00	100.50	Not sampled, PQ metallurgical hole
95-103C	0.00	165.20	Not sampled, Geotechnical hole
95-104C	0.00	35.05	Overburden
95-106C	0.00	82.30	No recovery
95-108C	0.00	19.81	Not sampled, water well hole
95-108C	21.34	117.35	Not sampled, water well hole
95-109C	0.00	33.52	Low core recovery
95-123C	0.00	36.57	Overburden
95-124C	13.74	25.91	Low core recovery
96-153C	0.00	89.92	Low core recovery
96-194C	0.00	60.66	Not sampled, Geotechnical hole
96-195C	0.00	79.86	No recovery
96-196C	0.00	42.37	Not sampled, Geotechnical hole
96-197C	0.00	179.83	Not sampled, Geotechnical hole
96-198C	0.00	161.54	Not sampled, Geotechnical hole
96-208C	0.00	299.62	No recovery
DG05-276C	15.24	24.38	Low core recovery, generally <50%
DG05-277C	19.81	35.05	Low core recovery, generally <50%
DG05-277C	48.77	72.90	Low or no recovery, generally <60%
DG05-277C	86.87	91.44	Low core recovery, generally <40%
DG05-278C	19.81	42.67	Low or no recovery, generally <60%
DG05-279C	9.14	41.20	Low or no recovery, generally <60%
DG05-280C	7.62	18.90	Low or no recovery, generally <60%
DG05-281C	50.29	53.34	Low or no recovery, generally <60%
DG05-282C	10.67	12.19	Low or no recovery, generally <30%
DG05-282C	24.38	30.48	Low core recovery, generally <50%
DG05-282C	33.53	37.68	Low core recovery, generally <50%
DG05-282C	45.72	51.82	Low core recovery, generally <50%
DG05-282C	88.39	91.44	Low core recovery, generally <10%
DG05-282C	108.20	114.30	Low or no recovery, generally <50%
DG05-283C	38.10	51.82	Low or no recovery, generally <60%
DG05-283C	60.96	62.48	Low core recovery, generally <40%
DG05-283C	67.06	68.58	Low or no recovery, generally <30%
DG05-287C	149.35	150.88	Low core recovery, generally <40%
DG05-288C	41.15	45.72	Low or no recovery, generally <60%
DG05-288C	65.53	70.10	Low core recovery, generally <40%
DG05-288C	82.30	85.34	Low or no recovery, generally <60%
DG05-289C	15.69	16.76	Low or no recovery, generally <60%
DG05-289C	42.67	57.91	Low or no recovery, generally <60%
DG05-290C	25.91	30.50	Low core recovery, generally <50%
DG05-291C	5.18	15.24	Low or no recovery, generally <60%
DG05-291C	29.49	31.32	Low core recovery, generally <50%
DG05-291C	35.05	48.77	Low core recovery, generally <50%
DG05-300C	62.58	64.01	Low core recovery, generally <10%
DG05-302C	4.57	7.62	Low core recovery, generally <50%
DG05-304C	5.79	15.24	Low core recovery, generally <40%
DG05-306C	21.34	24.38	Low or no recovery, generally <30%
DG05-307C	32.00	33.53	Low core recovery, generally <40%
DG05-308C	32.00	36.58	Low core recovery, generally <40%
92-029R	7.62	16.76	Not sampled, shut down due to caving of drill hole
92-031R	0.00	91.44	Remove entire drill hole. Large quartz vein plus contamination and high weights noted in log
92-054R	0.00	7.62	Overburden
92-054R	118.87	120.40	Caving in bottom of hole
93-066R	153.92	167.64	Contamination, high sample weights
93-066R	233.17	241.10	High sample weights
95-080R	233.17	236.22	Drilling problems
95-081R	275.84	297.18	Splitter problems, high sample weights
95-083R	129.54	147.83	Drilling problems, high sample weights, high water (20-30 gpm)
95-085R	91.44	106.68	Bad drilling conditions, high sample weights
95-085R	129.54	134.11	Bad drilling conditions, high sample weights
95-086R	94.49	112.78	Drilling problems, some high sample weights
95-087R	109.73	112.78	High sample weights
95-087R	115.82	120.40	High sample weights
95-089R	144.78	169.16	Drilling problems, high sample weights

17.1 Exploratory Data Analysis

Exploratory data analysis was completed on raw assay and composite drill data within the interpreted zones of mineralization.

17.1.1 *Assays*

There were 35,306 assay intervals in the database from 293 cored holes, 159 reverse circulation holes, and four trenches. In the current resource estimate, a total of 19,916 assay intervals from 80 cored holes and 102 reverse circulation holes that intersected the zone of mineralization. Zero assay values and intervals with no assay value were not used in the resource estimate. Table 17.3 displays the range of gold values from the selected assay data.

Table 17.3 Range of Gold Assay Data for Resource Estimation

Au g/t				
	Records	**Minimum**	**Maximum**	**Mean**
Total Available	35,306	0	52.36	0.472
Resource Estimate	19,922	0	52.36	0.694

17.1.2 *Capping*

The 2009 resource estimate was based on 12.65 g/t capped grade. On the basis of a review of probability plots of the assays and a decile analysis of the gold assay grades (see Appendix B). The gold content (assay value multiplied by sample length) of the top decile (10% of the sample population) contains 36% of the total gold, and the 100 percentile contains about 7% of gold content. The cumulative probability plot degenerates to a ragged tail and a possible break in the trend of gold grade continuity at the 99.9% cumulative percentile of the assay data. The ragged tail of the cumulative probability plot suggests a cap of 12.65 g/t gold is appropriate in order to diminish the possible distorting effect of erratic and non-representative high gold values. The cap used in the 2006 resource estimate was 10.5 g/t. Table 17.4 displays the capping limitations on the resource estimation.

Table 17.4 Capping of Gold Grades and Affected Number of Records

Au g/t				
Total Records	**Avg. Grade**	**Capped Records**	**Capping Value**	**Avg. Capped Grade**
19,922	0.694	49	12.65	0.688

17.1.3 Composites

The selected assays were composited into five-metre down-hole lengths within the mineralized shell. There were 5,825 composites created within the mineralized shell. The minimum compositing length of down-hole assays was 2.5 m. Of the 135 composites less than 5 m in length, an average grade of 0.524 g/t was obtained.

Table 17.5 Average Composite Grades

Composite Length Range	Number of Records	Au g/t		
		Minimum	Maximum	Mean
All records	5,825	0.00	7.749	0.688
= 5 m	5,690	0.00	7.749	0.689
5m< and >2.5m	135	0.01	4.809	0.524

17.2 Bulk Density

Bulk density determinations of 111 core drillhole samples were completed in 1995. Table 17.6 summarizes these bulk density determinations. In 1997 MRDI established bulk densities for their resource block model as a function of intensity of oxidation of the host rock as recorded for the nearest drillhole (MRDI, 1997). The higher the iron oxide intensity value, the lower the bulk density applied to the resource model.

Table 17.6 Iron Oxide Intensity and Bulk Density Assignments by MRDI (1997)

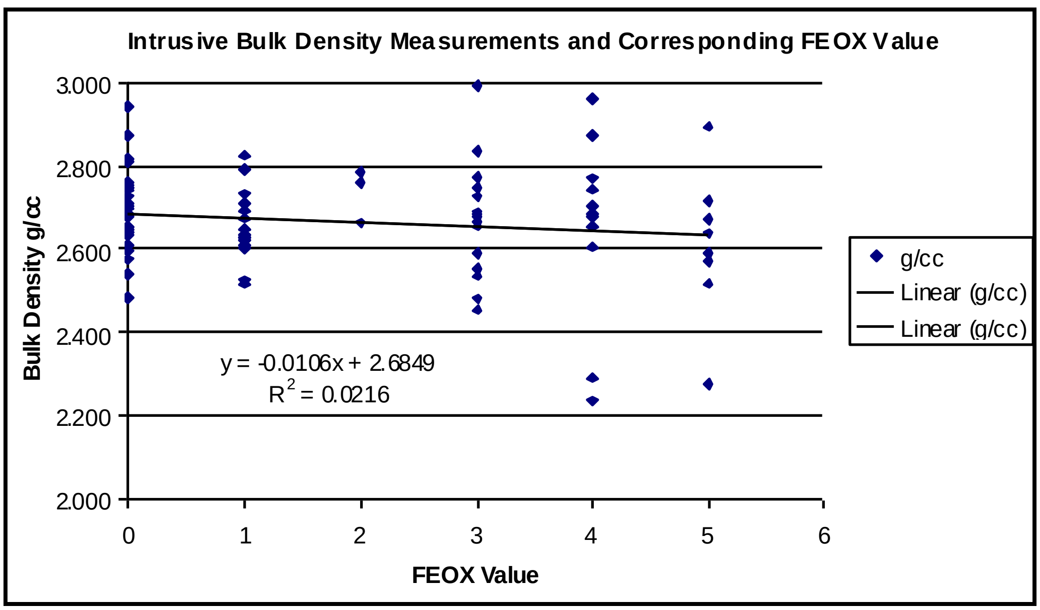
Rock Type	Iron Oxide Intensity	Bulk Density (tonnes/m^3)
Granodiorite	0,1	2.67
	2	2.60
	3	2.54
	4	2.40
	5	2.27

Wardrop graphed the bulk density measurements and corresponding iron oxide intensity value for intrusive rock type, see Figure 17.1. There does not appear to be a direct relationship between bulk density and iron oxide intensity, however there were some lower bulk density determinations for iron oxide intensity greater than three. Since no noticeable diminution of bulk density appears to exist, the average bulk density value of 2.66 t/m^3 was used in the resource block model as shown in Table 17.7.

Table 17.7 Summary of Bulk Density Determinations

Rock Type	Drill Hole	Records	Average Bulk Density (tonnes/m^3)
Intrusive	95-074C	25	2.62
	95-075C	30	2.70
	95-076C	10	2.69
	95-077C	10	2.69
	95-078C	10	2.63
	Subtotal	85	2.66
Meta-Sediments	95-075C	20	2.68
	96-194C	3	2.55
	96-197C	3	2.67
	Subtotal	26	2.66
	Total	**111**	**2.66**

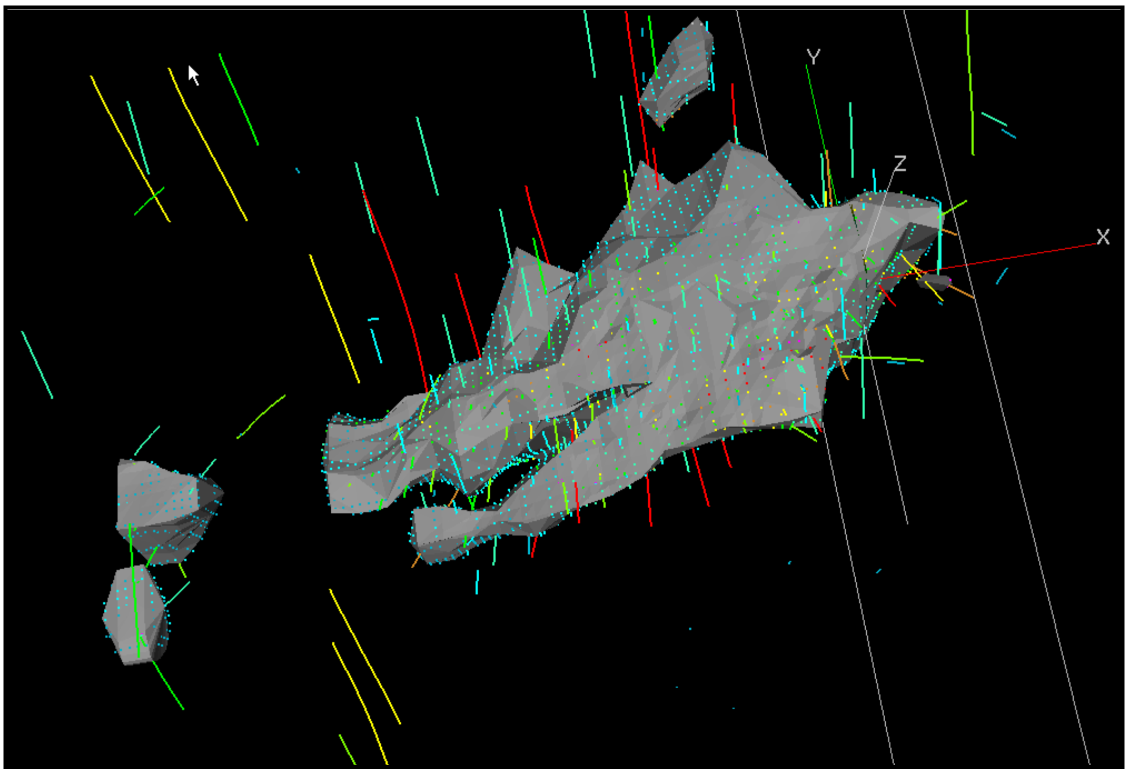
Figure 17.1 Bulk Density Measurements for Intrusive Rock Type and Corresponding Iron Oxide Intensity Values



17.3 GEOLOGICAL INTERPRETATION

Existing wireframe models from the 2006 Wardrop resource estimate were updated using Gemcom modeling software, based on 2006-2007 and 2008 drilling database. Updated wireframe was imported into Datamine 3.0.18 modeling software used for the final block modeling. The topography model created in 2006 report was converted from NAD27 to NAD83 using an average offset of 175 m North and 88 m East. A Wireframe solid of the mineralization, above a cut-off grade of 0.2 g/t, was created by digitizing polylines on east-west sections at 50 m intervals through the deposit. Polyline interpretations were verified in North looking sections for consistency. These polylines were linked and triangulated in order to create three-dimensional wireframe solid. Figure 17.2 displays the interpreted mineralization. A topography model from the 2006 interpretation required an update due to the elevation discrepancies with surveyed drill collars.

Figure 17.2 Interpreted Mineralization Looking North



17.4 Spatial Analysis

Based on the data available for the resource estimate, variography, using Datamine 3.0.18 software, was completed on the gold values. Three directional variograms were used to determine the nugget effect, and then semi-variograms were modelled to determine spatial continuity of the composited gold grades. Semi-variograms of gold distributions were calculated for all composites. The range of the variogram was, not surprisingly, longest in 270˚ orientation representing the signature along the strike of the mineralized zone. Table 17.8 summarizes the results of the variography and Appendix D contains the full variography documentation.

The search parameters and rotation angle was determine with the search ellipsoid. (see Appendix K).

Table 17.8 Dublin Gulch Variography

	Rotation			1st Structure				2nd Structure			
	Z	**Y**	**X**	**Sill**	**X**	**Y**	**Z**	**Sill**	**X**	**Y**	**Z**
Nugget	Angle1	Angle2	Angle3	Parameter	ST1PAR1	ST1PAR2	ST1PAR3	ST2PAR1	ST2PAR1	ST2PAR2	ST2PAR1
0.273	-15	12	30	0.395	60	30	75	0.597	265	155	90

17.5 RESOURCE BLOCK MODEL

Drillhole spacing across the deposit is generally on the order of 50 m in the east-west direction. Drillhole spacing by elevation and northing ranges from 40 to 70 m, with the 40 m spacing being characteristic of the near-surface portion of the mineralized zones. A block size of 15 x 15 x 15 m was selected in order to accommodate the drill hole spacing and width of the mineralization.

The total volume of the wireframe compares very well with the block model volume. The difference between the geological solids and the block model is less than 1%. Table 17.9 tabulates the wireframe and the associated volume.

Table 17.9 Summary of Wireframe Volumes on the Eagle Zone

Volume (m^3)			
Wireframe	**Model within Block**	**Volume Difference**	**Percent Difference**
79,161,127	78,824,446	336,681	0.43%

Table 17.10 summarizes the block limits co-ordinates for the Dublin Gulch block model.

Table 17.10 Resource Block Model Parameters

Coordinate	Number of Blocks	Minimum	Maximum
Easting	130	458885	460835
Northing	100	7098800	7100300
Elevation	60	540	1440

17.6 INTERPOLATION PLAN

The interpolation plan of the Dublin Gulch resource model was completed using nearest neighbour (NN), inverse distance squared (ID^2) and ordinary kriging (OK) methodologies.

Search volume dimensions are defined from the variogram models, based on the 80% of the difference between the first and second structure sill values. The obtained value was used for the second pass and a half of it for the first pass. Limits are set for the minimum and maximum number of samples used per estimate and as a restriction on the maximum number of samples used from each hole.

The estimation was designed as a three pass system, as outlined in Table 17.11, in order to differentiate indicated and inferred resource categories. In the first pass, the search ellipse distance was 70 m X x 40 m Y x 40 m Z, a minimum of seven composites was required and a total of 15 composites were allowed, with a maximum of three composites from any one hole. Further, a minimum of three holes was required in order to estimate a block. The search distance in the second pass was two times the search distance of the first pass. All blocks that had grades assigned by the first and second passes were classified as Indicated resource. For the third pass, the search distance was three times that of the first pass. However, a minimum of only five composites from three different drill holes were required on the second and third pass. Blocks that were estimated by the third pass were classified as Inferred.

Table 17.11 Block Model Estimation Pass Parameters

	Search Distance (meters)			Number of Composites		
Pass	**X**	**Y**	**Z**	**Minimum**	**Maximum**	**Max per Hole**
First	70	40	40	7	15	3
Second	140	80	80	5	15	3
Third	210	120	120	5	15	3

17.7 MINERAL RESOURCE CLASSIFICATION

Several factors were used in the determination of the mineral resource classification as follows:

- CIM requirements and guidelines.
- Experience with similar deposits.
- Spatial continuity of the mineralization.

No known environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to the authors that may affect the estimate of a mineral resource. Mineral reserves can only be estimated on the basis of an economic evaluation that is used in a preliminary feasibility or a feasibility study on a mineral project, thus no reserves have been estimated. As per NI 43-101 guidelines, the mineral resources that do not demonstrate economic viability are not mineral reserves. A summary of the mineral resource classification is outlined below:

In order to classify a block as part of an Indicated resource, there had to be at least three sample composites from two diamond drill holes within a search ellipse of 140 m X, 80 m Y, and 80 m Z. Blocks that were beyond the Indicated criteria but had three sample composites from two different drill holes, within a search ellipse of 210 m X, 120 m Y and 120 m Z, were classified as Inferred.

Mineral Resource Tabulation

The mineral resource estimation for Dublin Gulch is tabulated in Table 17.12 and 17.13 for the Indicated and Inferred resources. This was done to provide a more detailed view of the grade distribution, particularly in the lower grade ranges.

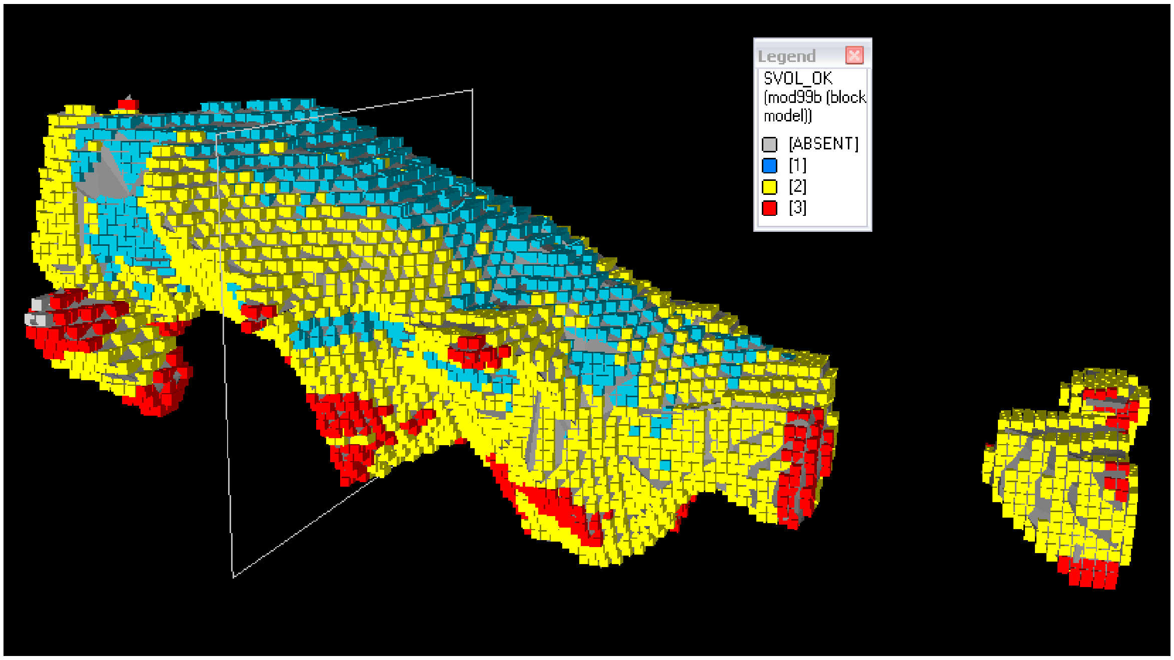
Table 17.12 Dublin Gulch Cumulative Indicated Resources

Cut-Off Au (g/t)	Tonnes x 1000	Au (g/t)
0.2	192,537	0.607
0.3	164,913	0.666
0.4	129,231	0.754
0.5	**98,584**	**0.849**
0.6	75,937	0.938
0.7	56,374	1.039
0.8	41,587	1.142
0.9	31,071	1.242
1.0	23,318	1.340

Table 17.13 Dublin Gulch Cumulative Inferred Resources

Cut-Off Au (g/t)	Tonnes x 1000	Au (g/t)
0.2	5,762	0.484
0.3	4,873	0.526
0.4	4,038	0.561
0.5	**2,023**	**0.671**
0.6	852	0.849
0.7	520	0.979
0.8	315	1.128
0.9	230	1.240
1.0	222	1.250

Figure 17.3 Resource Classification, Looking South East



Blocks in yellow and blue, numbered 1 and 2, represent Indicated Resources. Blocks in red, numbered 3, represents Inferred Mineral Resources

17.8 Block Model Validation

The Dublin Gulch resource estimation grade model was validated by the following methods:

- Comparison of the global mean based on NN, ID^2 and OK estimation methods.
- Visual comparison of colour-coded block grades for the three estimation methods of NN, ID^2 and OK.

17.9 Global Comparison

The global block model estimation for the OK method was compared to that of the global estimation of the NN and ID^2 model values. Table 17.4 shows the comparisons of the two estimation methods using all the blocks. In general, there is reasonable agreement between all the methods with Kriging having the lowest grade and Nearest Neighbour the highest grade.

Appendix F shows the Global Statistics comparison on all three estimation methods.

There is small discrepancy in the mean of each of the methods and the Variance, Standard Deviation and Skewness values suggested that the Ordinary Kriging was the most suitable method chosen for resource estimation.

The mineral resource estimation reported in the preceding tables is based on a minimum cut-off grade of 0.50 g/t gold.

Table 17.14 Global Resources by Estimation Method

Category	Kriging Method		Inverse Distance	
	T X 1000	Au g/t	T X 1000	Au g/t
Indicated	98,584	0.849	97,121	0.869
Inferred	2,023	0.671	2,134	0.724

Lower drillhole densities in some areas of the deposit have resulted in larger discrepancies between the method values. There is a degree of smoothing apparent in the ordinary kriging interpolation which is an artefact of the estimation method.

17.10 Visual Comparison

The visual comparisons of block model grades with composite grades of gold show a reasonable correlation between the values. No significant discrepancies were apparent between section and plan views (see Appendices G and H).

17.11 Resource Estimation Comparison

In 2006, Wardrop completed a NI 43-101 compliant resource estimate of the property. This estimate reflects the additional drilling done on the property since that estimate was completed. There has been a nominal change in both tonnes and contained gold in this estimate. The principle change has been the shift of classification of the resources from Inferred to Indicated. Table 17.15 compares the 2008 Wardrop estimate with the 2006 estimate. The current Wardrop estimate includes additional data from the 2006-2007 and 2008 drilling program. Both estimates use a cut-off of 0.50 g/t gold. Generally, there is agreement between the two; however the 2008 Wardrop estimation resulted in an addition of 32 Mt of Indicated resources and a reduction of 12 Mt of Inferred resources. These changes are attributed to the impact of additional data from the 2006-2007 and the 2008 drill program. The additional drill holes permit a higher level of confidence in the spatial continuity of the mineralization. The Wardrop estimate was constrained by mineralized envelopes with an approximate cut-off grade of 0.20 g/t gold.

Table 17.15 Wardrop 2006 and 2008 Resource Estimation Comparison

Estimation	Classification	Cut-Off g/t	Tonnes	Grade	Grams Au	Troy Oz Au
2006	Indicated	0.5	66,540,000	0.916	60,959,000	1,960,000
2009	Indicated	0.5	98,584,000	0.849	83,697,800	2,690,400
Change			**32,044,000**		**22,738,800**	**730,400**
2006	Inferred	0.5	14,390,000	0.803	11,549,000	371,000
2009	Inferred	0.5	2,023,000	0.671	1,357,400	43,630
Change			**-12,367,000**		**-10,191,600**	**-327,370**

18.0 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data to report.

19.0 INTERPRETATION AND CONCLUSIONS

19.1 CONCLUSIONS

Wardrop has conducted a mineral resource estimate on the Eagle Zone of the Dublin Gulch Project. Gold mineralization occurs in conjunction with sub-parallel extensional quartz veins that are best developed within the granodiorite, proximal to the hanging wall and footwall intrusive-meta-sedimentary contacts. The resource was based on composited gold assay data derived from core and reverse circulation drillholes and was estimated on the basis of interpreted mineralized envelope with a nominal cut-off of approximately 0.20 g/t gold. The resource was estimated by three interpolation methods: nearest neighbour, inverse distance squared and ordinary kriging. No significant discrepancies exist between the methods and ordinary kriging is used for the resource tabulation.

The Eagle zone has been drilled by both north and south dipping holes. Quartz veins are reported to strike 060º to 085º and dip about 60º to the south. Variography of the samples indicates that the direction of greatest continuity is indeed east-north-east, along the strike of the deposit, but dips slightly to the north-west, along the intrusive-sedimentary contact.

Data verification of the drillhole database suggested that the information is reliable and is believed to be accurate. The bulk density samples taken from the 1995 and 1996 drilling appear to be consistent with expected values for intrusive and meta-sedimentary rock types.

At a gold cut-off of 0.50 g/t the Eagle Zone Indicated Resource is estimated to be 98.5 million tonnes grading 0.849 g/t of gold, and the Inferred Resource is estimated to be 2.0 million tonnes grading 0.671 g/t gold. The deposit has been largely defined down to a depth of about 400 m. Mineralization is noted in the bottom of the zone and additional drilling is required to define the extension of the most significant results. Two major zones were encountered within the envelope. They are distinguished by lower grade inclusion between them generally parallel to the plunge. The accessibility of the lower footwall zone by open pit mining methods would be determined by the next stage of the project leading to a prefeasibility study.

20.0 RECOMMENDATIONS

20.1 RECOMMENDATIONS

Based on the completed analyses, interpretation and model resource estimate by Wardrop Engineering the following are respectful recommendations for further development of the Eagle Zone:

1. Additional drilling within the specific areas is recommended to determine the extension of the mineralization open at depth. Two deep drill holes (approximately 500 m each) are recommended between sections 459650E and 459750E in the vicinity of DG08357 and within the sections of 460100E and 459750E near DG7334C (see Appendices I and J). A low priority target would be the extension of the northern hanging wall zone between drill holes DG06310C and DG06315C1. Two additional drill holes were recommended to test this area. As reported in Wardrop's 2006 resource estimate, supplementary bulk density determinations on specific mineralization type should be investigated further and assigned to resource block model data for estimating tonnage. Of the 35,306 assays, only 111 (0.03%) have been measured for SG. There is a significant variation in SG especially in areas of higher FeOx intensity. At least 5% of the assays (1,765), including all the local lithologies, should be measured for SG. The cost estimates of the budget for this additional drilling are outlined in Table 20.1 and a drill plan is outlined in Appendix L.

Table 20.1 Additional Drilling Cost Estimate Budget

Item	Cost
Accommodation and Meals	$30,504
Assays	$124,195
Geology and Geophysics	$281,287
Drilling and Supplies	$979,430
Engineering	$58,399
Environment	$56,636
Field Costs	$59,853
Field Equipment and Rentals	$178,235
Air Transport	$77,764
Labour	$74,354
Land and Recording Fees	$110
Travel, Freight and Warehouse	$59,231
Total	**$1,980,000**

2. Additional bulk density determinations of the mineralized material should be carried out, so that specific gravity data can be incorporated into the resource block model for resource estimation.

   A comprehensive program of metallurgical sampling, including spatial variance across the ore body to obtain the standards of accuracy, is recommended, and Wardrop can provide assistance in identifying representative ore zones. A positive Feasibility Study was completed in 1997 by Rescan Engineering, which is not NI43-101 compliant. It is recommended that StrataGold undertake a NI43-101 compliant Pre-feasibility Study that will include the following work and estimated costs outlined below in Table 20.2.

Table 20.2 Pre-feasibility Study Tasks and Estimated Costs

Activity	Estimated Costs ($CDN)
Project Management and Controls	$134,000
Site Visits	
Mine Engineering	$152,000
Process and Infrastructure	$230,000
Tailings and Geotechnical Drilling	$485,000
CAPEXand OPEX Estimate	$37,000
Expenses	$70,000
Administration, Reports and Lists	$100,000
Project Administration Services	
Subtotal	***$1,208,000***

3. A full Environmental Base Line Study and Assessment is also recommended. This study has been estimated to cost $1,000,000 and will include environmental application requirements for an open pit mine and metallurgical process facilities.

4. Bulk Sample and Metallurgical Testing is also recommended, as outlined in Table 20.3, which includes a representative sample with the possibility of streamlining and optimizing the process flow sheet. A sample of representative mineralization will need to be collected, including oxidized near surface material.

Table 20.3 Bulk Sample and Metallurgical Testing Cost Estimate

	Work Scope	Dollar per Unit	Cost ($CDN)
Metallurgical Sampling and Testing (200 kg)	200 kg	$150	$110,000
Near Surface Oxidized sample and Metallurgical Evaluation			250,000
Flotation Evaluation Test			$15,000
Bulk Density			$26,000
Alternative Drilling Program	2,000 m	$175	$350,000
		Total Estimate	**$751,000**

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22.0 CERTIFICATE OF QUALIFIED PERSON

I, Slobodan (Bob) Jankovic, P.Geo., of Toronto, Ontario, do hereby certify that as an author of this report titled “Technical Report on the Dublin Gulch Property, Yukon Territory, Canada”, dated February 4, 2009, I hereby make the following statements:

- I am a Senior Geologist with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- I am a graduate of the University of Belgrade, Serbia (B.Sc. Geological Science, 1986).
- I am a member in good standing of the Association of Professional Geoscientists of Ontario (Registration #1388) and The Association of Professional Engineers, Geologists and Geophysicists of Alberta (Member Number 57413).
- I have practiced my profession in mineral exploration and mining geology continuously since graduation.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to this report includes 12 years experience in the mining sector covering database, mine geology, grade control and resource modeling. I was involved in numerous projects around the world in both base metals and precious metals deposits
- I am responsible for the preparation of all sections of this technical report titled “Technical Report on the Dublin Gulch Property, Yukon Territory, Canada “, dated February 4, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1

Signed and dated this 4th day of February, 2009 at Toronto, Ontario.

"Original document, revision 01, signed and stamped by Slobodan (Bob) Jankovic, P.Geo."

Signature