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National Instrument 43-101 Technical Report for the Dawson Property, Colorado, USA

Submitted to:

Zephyr Minerals Ltd

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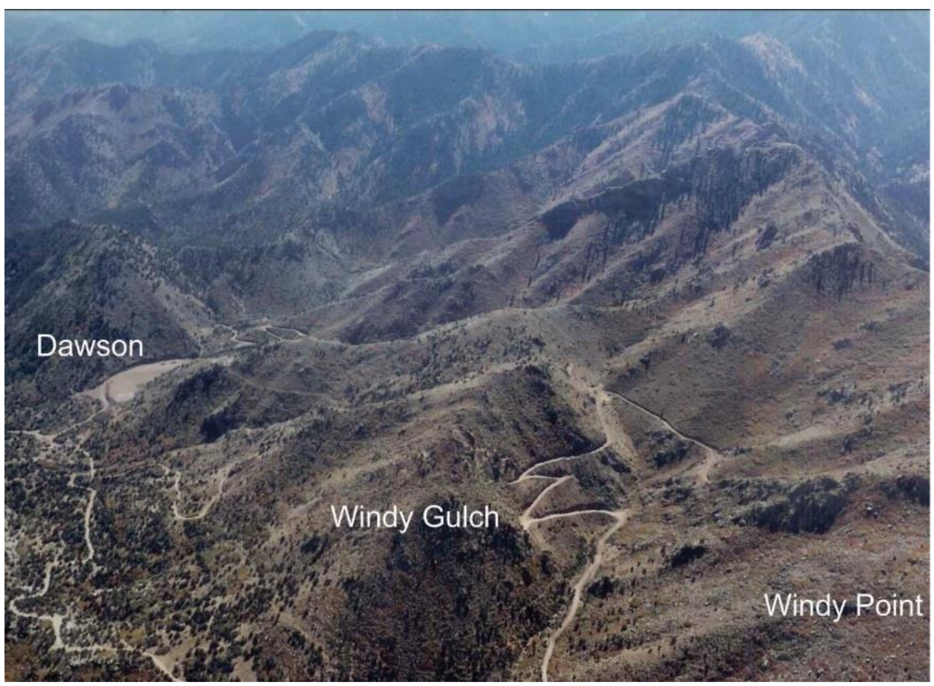

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REPORT

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Bolu Minerals Engineering & Consulting Inc.

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1.0 SUMMARY

1.1 Scope of Work

Golder Associates Ltd. (Golder) was retained by Zephyr Minerals Ltd. (Zephyr) to complete an independent Preliminary Economic Assessment (PEA) NI 43-101 technical report for the Dawson Project and provide an updated mineral resource estimate for the Windy Gulch Segment. This report represents the initial public disclosure of a PEA for the Dawson Project.

The mineral resource estimates for Windy Gulch were completed by Brian Thomas, P.Geo. Mr. Thomas completed a site visit to the project site on August 2 and 3, 2016. Mr. Thomas is a qualified person (QP) as defined by NI 43-101.

The 2016 Zephyr diamond drill program at Windy Gulch was managed by Mark Graves, P.Geo., who was on-site from May 26, 2016, to September 1, 2016.

A summary of the metallurgical work completed on the Dawson Property (the Property) has been completed by Matt Bolu, P.Eng., of Bolu Minerals Engineering & Consulting Inc. (BOMENCO), who is the QP responsible for Sections 1.7, 1.10.3, 13, 17, 18.1, 18.2 18.4, 18.5,18.6, and 21.3, 21.4, 21.5, 25.3, and 26.3 of this report.

The Dawson underground mine plan was prepared by Doug Roy, P.Eng., of MineTech International Ltd. (MinTech), who is the QP responsible for Sections 16.1 and 21.1.

The tailings management facility design was prepared by Brett Byler, P.E. of Amec Foster Wheeler Environment & Infrastructure Inc. (Amec Foster Wheeler) who is the QP responsible for Sections 1.10.4, 18.3, 18.9 to 18.16, 25.4, and 26.4.

A preliminary economic assessment of the Property has been completed by Danny Tolmer, P.Eng., of Golder. Mr. Tolmer is a QP as defined by NI 43-101.

1.2 Location and Ownership

The Dawson Property is in south-central Colorado (CO), about 5.9 mi southwest of Cañon City in Fremont County, within United States Geological Survey (USGS) Map Sheet Royal Gorge Quadrangle (7.5-minute series) and centred at about 473,215 mE and 4,249,147 mN UTM Projection (Zone 13, NAD 83 Datum). The rugged terrain is situated in the foothills of the northern Wet Mountains; elevations range from about 6,235 to 7,874 ft above sea level. Access to the Property can be gained from Cañon City on Temple Canyon Rd., a maintained county road. Four-wheel-drive trails that meander throughout the property provide access for drilling and trenching programs. Additional access trails are constructed from these trails as required.

The Property consists of 45 contiguous unpatented lode mining claims plus 8 patented mining lode claims and one patented placer claim which cover a total of 1.52 mi^2. All patented mining lode claims and 21 of the unpatented claims are subject to a mining lease agreement with several private individuals that covers yearly payments and a sliding scale royalty. The patented placer claim is subject to a royalty interest of 3% of the value of the concentrates produced from ore on the claim.

1.3 Geology and Mineralization

The Dawson gold system occurs along a major, syntectonic structural discontinuity which has significant strike-slip motion. In such a strike-slip regime, compression and dilational zones develop and result in pinch and swell of the relatively incompetent mineralized package. Intrusive into these dilational zones are two major gold-bearing peraluminous map units: 1) a banded biotite-granodiorite aplite having accessory garnet and attendant copper-gold mineralization associated with greisenous amphibole, muscovite and biotite. This unit is considered a magmato-hydrothermalite; and 2) a late quartzoid-limonite fracture filling event that locally contains Bonanza gold grades overprinting unit above in tensile structures. This bonanza gold is inferred to be an extreme magmatic fractionate. A pink peraluminous granite and allied pegmatite cross-cut the mineralization described above and are interpreted to be the source of the gold-bearing hydrothermal solutions.

Gold mineralization occurs in all the shear zone hosted rock types with the highest grades—up to 225 g/t gold (DDH GC 44), occurring predominantly in late stage hydrothermalite and limonite veins ranging in thickness from 2 mm to 5 cm. The high grade gold mineralized hydrothermalites are hosted in biotite aplite and leucogranite that host lower grade gold—0.5 to 10 g/t. Gold mineralized zones attain thickest widths of up to 15 m, in dilational or "buckle" zones within the Dawson Shear Zone and show a spatial relationship with northeast trending faults. Gold grains vary from 50 to 200 µm in size.

1.4 Exploration Programs

Modern exploration activities began in 1976, when a base-metal reconnaissance program by US Borax identified a laterally extensive sulphide horizon; the company's property ranged from the Sentinel Segment in the east for about 3.4 mi to the west-southwest to Marsh Gulch. From 1976 to 1986, the company's field work included surface diamond drilling (65 holes; 50,082 ft), detailed surface mapping, minor geochemical sampling, trenching and sampling and an airborne electromagnetic and magnetic survey. Two mineralized zones (Dawson and Windy Gulch) were outlined, with historical "resources" (non-NI 43-101 compliant) totaling about 387,000 short tons (tn) at 0.31 ounces per short ton (oz/tn) gold using a 0.15 oz/tn cut-off. US Borax sold the Property in 1987 to the joint venture of Jascan Resources Inc. and Atlantic Goldfields Inc.

From 1987 to 1993, Jascan Resources Inc. (Jascan), with several joint venture partners, completed additional diamond drilling (69 holes, 33,740 ft), geological mapping, and geochemical surveys and several ground magnetic and very low frequency electromagnetic (VLF-EM) surveys. The latest historical "reserve" estimate was carried out by joint venture partner Uranerz USA Inc. (Uranerz) in 1991 (non-NI 43-101 compliant), in which the Dawson Segment comprised 263,000 tn at 0.46 oz/tn gold; the Windy Gulch Segment was estimated to have about 33,000 tn at 0.408 oz/tn gold (Mettler, 1992). These historical estimates used categories other than the ones set out in Sections 1.2 and 1.3 of NI 43-101 and should not be relied upon. By the end of 1993, Jascan had a 70% interest in the Property; Uranerz had 30% interest.

In 1995, Celtic Minerals Ltd. (Celtic) acquired Jascan's 70% interest in the Property; the 30% owned by Uranerz remained with that company after it merged with Cameco Gold Inc. in 1998 (and renamed to UUS Inc.). From 1995, the original land position of Jascan had decreased through attrition to the current 45 unpatented and 8 patented lode mining claims and 1 patented placer mining claim. The unpatented lode mining claims are owned outright by Celtic-UUS Inc.; the patented lode and placer mining claims are 50% owned by Celtic-UUS Inc., with the remaining 50% leased from various private stakeholders. Since Celtic's acquisition in 1995, several reviews and compilations of the Property have been written, but no active exploration was conducted.

On October 31, 2012, Zephyr announced that it had closed a purchase and sale agreement with Celtic to purchase a 100% interest in the Property. The transaction also included the completion of a purchase agreement between Celtic and UUS for the minority interest held by UUS. The acquisition was done by way of a share purchase agreement, whereby Zephyr acquired 100% of Celtic Gold Ltd., a Colorado company and wholly owned subsidiary of Celtic which held title to the Property at that time, and renamed it Zephyr Gold USA Inc. Zephyr holds a 100% interest in the 45 contiguous unpatented claims, 50% interest in the eight patented claims and a 50% interest in one patented placer claim. The 50% of the eight patented lode mining claims not held by Zephyr is leased by Zephyr through a “Mining Lease and Agreement” which effectively gives Zephyr 100% control of the these claims. Twenty-one of the 45 unpatented claims, the eight patented lode mining claims, and the 50% interest in the one patented placer claim are subject to a sliding scale Net Smelter Return (NSR) whereby Zephyr agrees to pay up to a 3% NSR on the aforementioned claims. The entire claim package covers an area of about 950 acres.

In 2013, Mercator Geological Services Limited (Mercator) calculated a mineral resource estimate on the Dawson Segment and Windy Gulch Segment deposits for Zephyr. It is based on validated results of 76,559.6 ft of historical diamond drilling in 118 holes, 100.5 ft of sampling in two historical trenches completed between 1981 and 1991, plus 1,928 ft of diamond drilling in 13 holes and 87 ft of chip sampling in one trench and two previously excavated pits completed by Zephyr during the spring of 2013. Mercator reported an inferred mineral resource of 392,000 tonnes grading 10.50 g/t gold and containing 132,300 ounces of gold using a 5 g/t gold cut-off and capped at 40 g/t gold.

The 2016 exploration program comprised an airborne survey, grid geophysics, and diamond drilling. A LiDAR survey was competed by Eagle Mapping Ltd. (2016) of Port Coquitlam, BC, which generated a detailed topographic contour map and orthophoto utilized for accurate surface mapping, sampling control, and plant site and mine engineering. RDF Consulting Ltd. of St. John's, NL, performed ground magnetic and induced polarization (IP) surveys to assist in geological mapping and to direct the locating of diamond drill holes. Godbe Drilling LLC of Montrose, CO, was contracted by Zephyr to drill 17 holes totalling 2,539.5 ft of NQ drill core. All drilling was performed in the Windy Gulch Zone Segment of the Dawson Property. The drill program was supervised by Mark Graves, P.Geo., of Wolfville, NS. It was the gold assay information collected during this drilling that provided the data for the resource calculation, pit model, and mine plan design performed by Golder in this report.

1.5 Sample Preparation, QA/QC, and Chain of Custody

Drill core sample preparation was conducted in Zephyr's secure core facility in Cañon City, CO. The logging of the NQ core, cutting of the core by electric diamond saw blade, and sampling of the core for assay were performed here. Samples collected for analysis were placed in plastic bags, numbered using a three-tag system and stored in sealable five-gallon heavy plastic buckets for delivery with standards, blanks and replicates for quality assurance / quality control (QA/QC) purposes to the laboratory. Core samples were first analyzed for gold by standard one-assay tonne fire assay. All samples exceeding 1 g/t were then analyzed by the screen metallic method.

The core was retrieved at the drill site by the QP and delivered by truck to Zephyr's coreshed in Cañon City. After sample preparation, the core samples were shipped via Fedex in Cañon City to the analytical laboratories in Vancouver, BC, and Ancaster, ON.

A QA/QC program was implemented by Zephyr for all holes in drilling program. Standard samples were inserted at a rate of 1:13 samples, blank samples inserted at a rate of 1:22 samples, and replicates at rate of 1:16 samples. Upon analyzing all procedures and results gathered under the QA/QC program undertaken by Zephyr, it can be concluded the data are of sufficient quality to support a resource estimate.

1.6 Data Validation

1.6.1 Dawson

Drill core sample records, lithological logs, available laboratory assay results and associated drill hole information for all historical drill programs completed by US Borax, Jascan, and Uranerz were made available to Mercator in both digital database and supporting hard copy format. After initial spot checking of the database against original source documents it was determined that a comprehensive review of the digital data set should be completed. Mercator completed a detailed review, which consisted of checking all assay database entries against source documents and checking drill hole collar coordinates and downhole surveys against summary tables also found in hard copy reports. Database entries were corrected where mistakes were noted, and data added where missing. Zephyr provided 2013 Zephyr drilling program results in digital format and Mercator compiled digital assay results from laboratory spreadsheets and checked 10% of entries to original values found in corresponding laboratory certificates supplied by ALS.

Manual review and checking of compiled database logging and sample records showed generally good agreement between original records and digital database values. No historical or current drill holes were excluded from the resource drilling database due to lack of supporting records but several items attributed to data entry errors were identified and appropriate changes were made in the database. Drilling and sampling database records were further assessed through digital error identification methods available through the deposit modelling software. The digital review and import of manually checked data sets through Gemcom Surpac® provided a validated Microsoft Access database that Mercator deemed acceptable with respect to support of the Dawson Segment resource estimation program described in this report.

Co-author Graves carried-out a site visit and an independent drill core check sampling program for historical drill core from the Dawson Segment deposit between March 25 and March 31, 2013, to support the resource estimation program by Mercator. General observations regarding the character of the landscape, vegetation, site elevations, surface drainage and road access were carried out as well as a review of historical drill core and completion of a drill collar coordinate check program. Quarter core check samples for 17 historical sample intervals from the Dawson Segment deposit were collected from eight historical drill holes and checks of logging integrity and sample record errors were carried out. Samples were analyzed using screen metallic gold analysis methods at ALS Minerals in Reno, Nevada, an ISO 9001:2008 Certified, accredited, independent, commercial laboratory.

Based on observations made during the 2013 site visit, analysis of check sampling program results, and review of other project data, author Graves and Mercator have determined that, to the extent reviewed during the Dawson Segment site visit, ample evidence exists of the historical exploration programs carried out on the deposit as well as those carried out to that date by Zephyr. Results of site visit check sampling, identification of historical workings and collars, core review, and checking of historical logging information against archived core and records for 2013 site visit purposes are considered acceptable and no substantive inconsistencies have been recognized with respect to records of past work or associated results.

Co-author Graves and Mercator are not qualified to provide professional opinions with respect to environmental conditions, potential hazards or liabilities that may be present on the Dawson Property. However, during the course of site visit and drilling supervision programs, co-author Graves observed that various historical underground workings areas are present on the Property and that some of these are characterized by rusty-weathering waste rock piles. These piles may represent potential sources of acid rock drainage. Additionally, hazards appear to exist locally with respect to open historical mine workings and pits.

Zephyr management has warranted that no additional technical work material to the Dawson Segment deposit has been has been carried out by the company since the 2013 mineral resource estimate for the deposit by Mercator. On this basis, Mercator determined that a new site visit was not required to support assessment of currency of the 2013 Dawson resource estimate.

1.6.2 Windy Gulch

Golder compared selected assay data from the Zephyr database to the original assay certificates from Activation Laboratories (Actlabs), Bureau Veritas and ALS Chemex for the 2013 and 2016 Windy Gulch drill programs and no significant errors were identified during this review.

During the QP site visit, Brian Thomas of Golder surveyed three drill hole collars and then compared the coordinates to those provided by Zephyr. All collars were found to be consistent with the Zephyr surveyed collar coordinates, within the accuracy of the handheld GPS that was used.

Golder conducted verification sampling of drill core from the Windy Gulch and Windy Point segments. A total of nine samples were taken along with an additional certified reference material (CRM) sample. Assay values from the verification sample program were reasonably consistent with results obtained by Zephyr, with a couple of notable exceptions possibly due to naturally occurring variability (nugget effect) or differences in analytical procedures used.

Golder has concluded that the Zephyr drill hole database is of suitable quality to support the resource estimate disclosed in this technical report.

1.7 Mineral Process and Metallurgical Testing

1.7.1 Dawson

Dawson zone is the more significant of the two gold zones identified within the Dawson Project in Colorado, USA as it constitutes potentially greater than 80% of the total estimated gold content and resource tonnage. In 1991, a metallurgical test program by Hazen Research Inc. (HRI) was conducted on two composite samples (Altered and Unaltered) from the Dawson zone to investigate their response to selected gold beneficiation techniques including gravity and flotation recovery techniques.

The results for the "altered" material showed that approximately 50% of the gold values were recovered to the gravity concentrate with grind size having no significant effect, in the size ranges tested (47 to 129 µm P80). Comparably, results for the "unaltered" sample showed gold recoveries to the gravity concentrate to be about 60%.

Preliminary flotation tests showed that nearly 86% of the total gold was recovered to a flotation concentrate assaying 169 g/t and at 5.4 weight percent of the feed for the "altered" sample. Similarly and in test 2, for the unaltered sample, over 94% of the gold was recovered to a flotation concentrate assaying 135 g/t and at 7.8 weight percent of the feed.

1.7.2 Windy Gulch

In 2014, a test program was conducted on a composite sample from the Windy Gulch zone at the laboratories of Bureau Veritas - Inspectorate Metallurgical Division (Inspectorate). This program focused on gravity and flotation response of the Windy Gulch composite sample.

Gravity concentration tests indicated that approximately 24% to 30% of the gold could be recovered into a concentrate assaying 30 to 35 opt. When combined with flotation concentrate, recovery increases to over 78% however, at a cumulative gold grade of 56 g/t.

It was cleare from these results that the Windy Gulch samples respond less favourably to flotation in particular and gravity separation in general than the Dawson samples that were tested at Hazen. This is highly likely due to the altered (oxidized) nature of the samples.

In conclusion, the results of gravity and flotation tests summarized above offer a technically viable process option for the recovery of precious metals from the Dawson and Windy Gulch resource. And based on these results a process flowsheet encompassing comminution followed by gravity and flotation processes was developed for the recovery of gold and silver from the Dawson ores. The plant flowsheet and design further includes concentrate and tailings dewatering and all necessary ancillary facilities to enable the recovery of gravity and flotation concentrates for further refining and marketing.

1.8 Mineral Resource Estimate

1.8.1 Dawson

The Dawson Segment mineral resource estimate appears in Table 1-1 below and is based on a 3D block models developed by Mercator in 2013 using Gemcom Surpac 6.1.4 modelling software. It is based on validated results of the project drill hole database, which was inclusive of both the Dawson and Windy Gulch segments at the time of resource estimate preparation. This included 76,559.6 ft (23,335.37 m) of historical diamond drilling in 118 holes and 100.5 ft (30.63 m) of sampling in two trenches completed between 1981 and 1991 by US Borax, Jascan, and Uranerz, plus 1,928 ft (587.65 m) of diamond drilling in 13 holes and 87 ft (26.52 m) of chip sampling in 1 trench and 2 small pits (previously excavated) completed by Zephyr during the winter of 2013 on the Windy Gulch Segment. The deposit was assessed and modelled with respect to underground development potential.

Block model grades were interpolated using inverse distance squared (ID^2) methodology. ID^2 interpolations used 5 ft downhole gold assay composites that were capped at 1.17 oz/tn (40 g/t) Au. Block size for both models was 5 ft (1.52 m) (Y) by 16.5 ft (5.03 m) (X) by 16.5 ft (5.03 m) (Z) with no sub-blocking. The resource statement cut-off grade of 0.15 oz/tn (5.00 g/t) Au reflects reasonable potential for underground development based on a gold price of $US 1,200/oz.

Table 1-1: Dawson Segment Mineral Resource Estimate Effective July 19, 2013

Resource Category	Au Cut-Off	Tonnes (Rounded)	Tons (Rounded)	Au Grade	Ounces**
Inferred	0.12 oz/tn (4 g/t)	371,000	409,000	0.29 oz/tn (10.09 g/t)	120,400
Inferred	***0.15 oz/tn (5 g/t)**	**343,000**	**378,000**	**0.31 oz/tn (10.55 g/t)**	**116,300**
Inferred	0.18 oz/tn (6 g/t)	310,000	342,000	0.32 oz/tn (11.08 g/t)	110,400

*Resource statement cut-off value of 0.15 oz/tn (5 g/t) Au is highlighted in **bold**
**Ounces may not sum due to rounding
Notes:

(1) Tonnes and tons have been rounded to the nearest 1,000.
(2) Ounces have been calculated from reported tonnes and g/t Au grade and are rounded to the nearest 100 ounces
(3) Contributing 5 ft (1.5 m) assay composites were capped at 1.17 oz/tn (40 g/t) Au
(4) The resource statement cut-off grade of 0.15 oz/tn (5.00 g/t) Au is highlighted in Table 1-1 above through bolding and reflects underground development potential based on a Au price of $US 1,200/ounce.
(5) A density value of 0.082 tn/ft^3 (2.63 g/cm^3) was used for the Dawson Segment
(6) Mineral resources were estimated in conformance with the Canadian Institute of Mining, Metallurgy and Petroleum – Standards on Mineral Resources and Reserves – Definitions and Guidelines, as referenced in NI 43-101.
(7) The rounding of tonnes as required by NI 43-101 reporting guidelines may result in apparent differences between tonnes, grade and contained ounces.
(8) Mineral resources are not mineral reserves and do not have demonstrated economic viability. This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
(9) The quantities and grades of reported Inferred Mineral Resources are uncertain in nature and further exploration may not result in their upgrading to Indicated or Measured status

1.8.2 Windy Gulch

The mineral resource estimate for the Windy Gulch Segment has been prepared by Brian Thomas (B.Sc, P.Geo), Senior Resource Geologist at Golder, and is summarized in Table 1-2 below. The effective date of the mineral resource estimate is January 17, 2017. Mr. Thomas is an independent QP pursuant to NI 43-101.

The mineral resource estimate represents an open pit mining scenario and is derived from a Datamine constructed block model (block size = 10 ft long by 5 ft wide by 5 ft high) consisting of five mineral domains (Zones 1 to 5). Open pit constrained resources are reported within an economic pit shell outline at a Au cut-off of 0.035 oz/tn, and underground resources are reported outside of the pit shell at a 0.093 oz/t Au cut-off. Zones 1 and 2 were “unfolded” and had top cuts of 1.167 oz/tn (40 g/t) applied to restrict outlier values. The five domains (see Figures 14.13 and 14.14) utilized Ordinary Kriging (OK) and Inverse Distance (ID) methodology to interpolate Au grades from 3 ft composite drill hole samples. Inverse Distance Cubed (ID^3) estimates were chosen as the final grades due to lower overall grade smoothing. Density values were estimated from specific gravity measurements taken from half core samples and, where absent, were assigned average default values. The resources reported are based on a “blocks above cut-off” basis and were then examined visually by Golder and found to have reasonable continuity.

The resource estimate for the Windy Gulch Segment is summarized below.

Table 1-2: Windy Gulch Open Pit and Underground Mineral Resource Estimate

Resource Classification	Tons	Au (oz/tn)	Au Ounces
Pit Constrained Resources (0.035 oz/tn cut-off)			
Indicated	67,000	0.11	7,300
Inferred	6,000	0.09	500
Underground Resources (0.093 oz/tn cut-off)			
Indicated	11,000	0.18	2,000
Inferred	14,000	0.19	2,700
Total Indicated	**78,000**	**0.12**	**9,300**
Total Inferred	**20,000**	**0.16**	**3,200**

Notes:
1) All pit constrined resources constrained to a pit shell and reported at a 0.035 oz/tn Au cut-off.
2) All underground resources reported outside and below the pit shell at a 0.093 oz/tn Au cut-off.
3) Resource tonnages have been rounded to the nearest 1,000 tons.
4) Grade estimates have been rounded to the nearest one hundredth of an ounce of gold.
5) Calculated Au ounces are rounded to the nearest 100 ounces.
6) Resource estimates do not include mining recovery or dilution factors.
7) Resource estimates have accounted for metallurgical recovery.
8) Calculated Au ounces may not add up correctly due to rounding.

1.9 Conclusions

1.9.1 Dawson

This mineral resource estimate presented in this technical report for the Dawson Segment deposit was prepared in accordance with NI 43-101 and the CIM Definition Standards and was originally prepared by Mercator on behalf of Zephyr in 2013. Since Zephyr management has confirmed that no new work material to the 2013 resource estimate has been completed on the Dawson Segment deposit since 2013, Mercator has deemed the 2013 resource estimate to be current for the purposes of NI 43-101 and this report.

The Dawson Segment deposit was assessed for resource modelling purposes as being representative of an intrusion related gold deposit hosted by Early-Proterozoic faulted, siliceous and peraluminous felsic gneisses with lesser semi-massive sulphide/sulphide-rich zones. Gold occurs in multiple mineralized horizons, is most commonly associated with sillimanite, sericite or biotite, and is sited in fractures in quartz and garnet grains and at quartz grain boundaries. The main mineralized horizon has been structurally modified to form pinch-and-swell features, thickened zones and offsets as well as relatively planar zones of mineralization that locally show southwest plunging grade shoots. Gold mineralization zones are relatively discrete and range in true thickness from approximately 2 ft (0.61 m) to as wide as 50 ft (15 m). Gold is locally nuggety and is commonly accompanied by 1% to 5% disseminated pyrite +/- chalcopyrite (Theye, 1989). While gold is the only metal considered to have economic significance at this time, copper also occurs in association with the hosting sulphide-bearing unit and could be of future economic interest. Potential for development through underground mining using sublevel stoping and shrinkage stoping has been assumed for resource assessment of the deposit.

Continuity of gold mineralization as defined by current core drilling results is generally good, but the deposit has been locally disrupted by faulting and pegmatitic intrusions. In the Windy Gulch area, potential exists for interpretation of tight fold closures but this is not represented in the current resource model. Rugged terrain has historically limited surface drilling access points in some areas and both downhole survey control and core recoveries for some historical diamond drill holes are incomplete.

Based upon results of the current resource estimation program and associated geological interpretation, Mercator is of the opinion that good potential exists for delineation of down-plunge (southwest) and up-plunge (northeast) extensions to the currently defined Dawson Segment deposit at the 0.15 oz/tn (5.0 g/t) Au cut-off grade level. Both of these areas warrant further evaluation through additional core drilling. Additionally, infill core drilling at closer spacing within the current resource limits will be required to upgrade Inferred resources to Indicated or Measured status.

1.9.2 Windy Gulch

This mineral resource estimate represents an open pit mining scenario for the Windy Gulch Segment of the Dawson Project. It has been prepared in accordance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) best practice guidelines and definitions as referred to in NI 43-101 regulations.

Due to the low grade cut-off grade used to outline mineralization and the relatively large smallest mining unit (SMU) size for open pit blocks relative to narrower underground mining units, a degree of grade smoothing will have been introduced into the model and resource estimates. When dealing with lower grades and larger block sizes, it is common that grades will be smoothed (averaged) due to the combination of high and low grade samples and greater volume differences between samples and blocks (volume variance). As a result, the resource tonnage may appear higher and the grade lower than reported from historical polygonal estimates, but the total ounces contained within the deposit are likely to be similar. Differences in contained metal may occur at higher cut-offs due to the smoothing effect, so the reader is cautioned that this resource estimate and block model is intended to evaluate an open pit mining scenario and may not accurately reflect conditions attainable from underground mining (i.e. lower tonnage at a higher grade).

Brian Thomas, P.Geo., is the QP of the resource and has visited the site, collected samples for metal verification, and reviewed the Windy Gulch Segment data, including geological reports, maps, technical papers, digital data including lab results, sample analyses, and other miscellaneous information. The QP believes that the data presented are an accurate and reasonable representation of Windy Gulch and concludes that the database is of suitable quality to provide the basis of the conclusions and recommendations reached in this report.

The Dawson Project is considered a project of merit and has the potential for mining and increased resources through additional exploration.

1.9.2.1 Risks

Many drill holes within the outlined resource area are historical and may have data quality risks related to a lack of established QA/QC procedures along with some accuracy concerns regarding the collar locations. These issues may have the potential to affect the accuracy of the modelled volumes and the estimated grades in the block model.

The relatively low number of bulk density measurements and the use of average density values in some zones could affect the accuracy of the resource tonnage and contained metal. Density values are based on pycnometer volume measurements of pulp samples from drill core and may not accurately reflect the porosity or fracture spacing present in the rock mass, which could result in an over-estimation of resource tonnage. In some instances, core recovery from diamond drilling is relatively poor locally which could result in non-representative sample data.

Golder accounted for the above risks by being conservative with projected contacts and by assigning appropriate resource classifications to each domain. The resource classification provides a reasonable evaluation of the risks associated with the mineral resource estimates.

1.9.2.2 Opportunities

Based on the information collected to date, there is an opportunity to increase the size and confidence (resource classification) of the resource with future infill and exploration drilling. Inferred open pit mineral resources in Zone 1 at Windy Gulch have a reasonable probability of being upgraded to an Indicated Mineral Resource with the completion of adequate infill drilling. Indicated resources have potential to be upgraded to Measured Resource with adequate infill drilling with 30 ft centres or less and the completion of a twinning program to confirm the accuracy and quality of the historical drill hole data.

Zone 1 has potential to be extended down dip with additional step-out drilling. There is currently very little drill hole information in the down-dip direction beyond what has been captured in this model.

Windy Gulch also has reasonable Greenfield and Brownfield exploration potential due to the continuous nature of the host lithology which has been traced over to the Windy Point Segment and is likely continuous with the Dawson mineralization as well.

1.9.3 Preliminary Economic Assessment

A preliminary economic assessment (PEA) of the project was completed based on inputs from various consultants and estimates from similar sized operations. The assessment was done in US dollars. The base case assumes an owner-operated, underground mine project with gold price of US$ 1,250/oz. The following are some key highlights and inputs from the PEA:

- Pre-tax IRR and $NPV_{5\%}$ of 66% and $35.5 million and a 2.4 year payback of initial capital
- After-tax IRR and $NPV_{5\%}$ of 46% and $22.1 million and a payback of 2.7 years
- Initial capital estimate of $33.2 million including contingency
- Life of mine (LOM) cash cost of $563/oz[1]
  - Unit site operating costs (US$ per tn processed):
    - Mining costs ($91/tn)

  - Processing costs ($31/tn)
  - G&A ($7/tn)
  - Smelting and Refining Cost ($22/tn)
  - LOM Average Cash Cost ($151/tn)
  - LOM Cash Cost Including Sustaining Cost ($186/tn)
- Royalties (1.5%)
- Corporate Income Tax/ Colorado Mining Tax (35%/4.6%)
- LOM all-in sustaining cost, including cash cost, of $692/oz[2]
- Total Resource Mined/Milled (450,000 tn)
- LOM diluted head grade 0.27oz/tn (ounces per short ton)
- LOM gold combined gravity and float recovery of 92%
- Total gold ounces recovered 111,300 oz

[1]Cash cost includes mining cost, G&A, mill and refining costs

[2]Sustaining capital cost includes underground equipment costs after the mill has been commissioned.

The PEA included a five-year underground mine project encompassing the Dawson Segment of the Dawson Project. The Dawson Segment provides 450,000 tons of mill feed (which includes Inferred Material) at an average diluted grade of 0.27 oz/tn gold. The Windy Gulch resource, including Inferred Resource, was evaluated assuming an open pit mining operation, however, due to the limited amount of easily accessible material, the relatively high strip ratio, and the limited economic benefit the open pit provided to the overall PEA, Zephyr decided to omit the Windy Gulch mill feed, from the PEA. Additional drilling and expansion of the mining licence could increase the mining potential of the Windy Gulch resource.

The model was compiled using the information available at the time. Due to risks and uncertainty related to global factors, government regulations, environmental considerations and other inherent risks associated with mining projects, actual results may differ from those reflected in the model.

1.10 Recommendations

1.10.1 Dawson

No material work has been completed on the Dawson segment deposit since the 2013 resource estimate technical report prepared by Mercator. On this basis, and after review, Mercator is of the opinion that the following recommendations that are based on those originally presented in the 2013 technical report remain applicable at the effective date of the current report.

1) Additional core drilling should be carried out to evaluate the direct up-plunge and down-plunge extensions of the main gold grade trends that comprise the Dawson Segment deposit.

2) Infill core drilling should be carried out within the deposit limits to increase confidence in resource continuity and to thereby support future conversion of Inferred mineral resources to Indicated or Measured status.

3) Zephyr should continue to review and update internal QA/QC protocols to include analysis of duplicate pulp and reject splits, core duplicate splits and drill core check sampling with analysis at a third party, independent, accredited laboratory.

4) The Dawson Segment resource estimate and associated resource files should be transformed to Colorado State Plane coordinate system (NAD83) to conform to the coordination system of the adjunct Windy Gulch Segment deposit

5) Drill several holes at the Dawson Segment to compare screen metallic assaying to standard one-assay ton gold analyses performed during the 1980s drill programs. The core could then also be utilized for geotechnical and metallurgical testing.

6) Re-log archived core from Dawson and Copper King Segments to update and bring into compatibility and context with the new intrusion related and shear zone hosted geological model.

7) Drill several holes beneath the near-surface copper prospect at Copper King to explore for buried gold mineralization analogous to that occurring beneath the copper zone at Dawson.

Estimated costs for the recommended work program are summarized in Table 1-3 below.

Table 1-3: Summary of Dawson Segment Recommendations and Cost Estimates

Recommendations	Estimated Cost ($)
Diamond drilling (deposit infill and extension – 1500m)	$375,000
Core re-logging, sampling and mapping for Dawson and Copper King Segments	$50,000
Dawson Segment comparative drilling for screen metallics assessment (300m)	$75,000
Updated resource estimate and deposit modelling review	$60,000
Exploration drilling at Copper King Segment (300 m)	$75,000
Administration and reporting	$65,000
Total Cost of Program	$700,000

1.10.2 Windy Gulch

The Windy Gulch deposit requires further diamond drilling to 1) expand the deposit foot print and 2) increase confidence in the current resource estimates. The following work program is recommended as an initial phase towards achieving these goals.

- Develop a new road cut between the existing upper and middle road cuts to expose mineralization for mapping and sampling as well as providing an additional location for diamond drilling.
- Complete an HQ diamond drill program from the new road cut to infill gaps in existing drilling and test mineralization down dip and down plunge to the west, as well as to determine if the larger core size reduces nugget variance.
- Complete a twinning drill hole program of two to five historical holes in order to verify the quality and accuracy of the historical drill hole data.
- Re-evaluate the mineral resource on successful completion of the drill program.

Estimated costs for the recommended work program are summarized in Table 1-3 below.

Table 1-4 Summary of Windy Gulch Recommendations and Cost Estimates

Recommendations	**Estimated Cost ($CDN)**
New road cut	50,000
Diamond drilling	120,000
Twinning of historical holes	60,000
Updated resource estimate	35,000
Total Cost of Program	**265,000**

1.10.3 Metallurgical Recommendations

The following are recommended before the next phase of project development.

- Representative samples of the mineable areas including earlier months/years of mining as well as varying rock and mineral types should be taken for a comprehensive metallurgical test program including gravity and cleaner flotation in locked cycle tests. . The cost estimate for a comprehensive metallurgical test program described above is $50,000.
- Grade-recovery relationship should be established to aid in determining economically and metallurgically optimum parameters. The cost to prepare the relationship is included in the above metallurgical test program.
- Rod mill, ball mill, crushing and abrasion work index tests should be conducted to aid in design, sizing and selection of major comminution equipment. The cost estimate to conduct the tests is $10,000.
- Dewatering tests for thickening and filtration must be carried to optimize sizing of these unit operations. The cost estimate to conduct the dewatering tests is $15,000.
- Marketing studies should be conducted to establish economically optimum smelting options for the concentrates. The cost estimate to conduct marketing studies for this project is $10,000.

1.10.4 Tailings Management Recommendations

The following are recommended for the filtered tailings storage facility for the next phase of project development:

- Bench-scale pressure filtration testwork on representative tailings samples is recommended in the next study phase. The estimated cost for this test work is US$15,000.
- Additional testwork is recommended to characterize the geochemistry of the tailings. The geochemical testing should include static and kinetic testing on each ore type from the Windy Gulch and Dawson segments. The cost for such geochemical characterization will depend on the number of ore types, but is estimated to be US$30,000 to $50,000.
- Additional geotechnical investigation is recommended for the next study phase to better characterize the subsurface conditions at the FTSF. The estimated cost for these geotechnical investigations is US$50,000.

2.0 INTRODUCTION

Golder was retained by Zephyr to provide an independent PEA NI 43-101 technical report for the Dawson Project and an updated mineral resource estimate for the Windy Gulch Segment. This report provides details for a new geological interpretation based on the recent and historical work of MagmaChem Exploration, Inc (MagmaChem).

The mineral resource estimates for Windy Gulch were completed by Brian Thomas, P.Geo with an effective date of January 17, 2017. Mr. Thomas completed a site visit to the project site on August 2 and 3, 2016. Mr. Thomas is a QP as defined by NI 43-101.

Mineral resource estimates for the Dawson Segment were completed by Andrew C. Hilchey, P.Geo., of Mercator, with an effective date of July 19, 2013.

The tailings management deisgn was prepared by Brett Byler, P.E. of Amec Foster Wheeler. Brett visited the property on June 16, 2015, and inspected site surface conditions and potential locations for a tailings management facility.

This report was prepared as a NI 43-101 PEA technical report for Zephyr. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Golder's services, based upon:

- Information available at the time of preparation
- Data supplied by outside sources
- The assumptions, conditions, and qualifications set forth in this report

This report is intended to be used by Zephyr, subject to the terms and conditions of its contract with Golder. That contract permits Golder and Zephyr to file this report as a technical report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, use of this report by any third party is at that party's sole risk.

During the site visit, Mr. Thomas reviewed the site conditions, reviewed logging, sampling, QA/QC and chain of custody procedures, confirmed collar locations, and confirmed metal mineralization through the inspection of core and independent check assays. Logging and storage facilities were also visited. A full summary of the site visit is included in Section 12.

2.1 Source of Information

The sources of information that were provided in the preparation of this technical report and mineral resource estimate were provided by Zephyr, under the direction of Loren Komperdo, and Mark Graves. This report and mineral resource estimate is based on the following data and pre-existing reports:

- 2016 geological mapping and interpretation from MagmaChem
- Updated NI 43-101 technical report for the Dawson Property, CO, 2015
- Uranerz progress and summary reports, 1990
- Jascan progress and summary reports, 1988–1989
- Zephyr drill hole database including
  - Au assays
  - Lithological logging and sample details
  - Bulk density data
  - Collar and down-hole survey data
  - QA/QC data
- Recent assay certificates from Actlabs, Bureau Veritas, ALS Chemex
- Historical assay certificates from Bondar Clegg
- Historical geological reports and maps
- Dawson underground mine plan from MineTech International Ltd.
- Mineral processing design from BOMENCO Inc.
- Tailings management facility design from Amec Foster Wheeler
- Environmental study from Environmental Alternatives Inc.

2.2 Units of Measure and Abbreviations

All units of measure used in this report are in the imperial system, unless stated otherwise. Currencies outlined in the report are in US dollars unless otherwise stated.

Abbreviations used in this report are defined in Table 2-1.

Table 2-1: Abbreviations and Units of Measure

Abbreviation	Term or Unit
3D	three-dimensional
°	Degrees
°C	degrees Celsius
<	greater than
>	less than
%	percent
AA	atomic absorption
ACA Howe	ACA Howe International Pacific Ltd.
ACS	American Chemical Society
Actlabs	Activation Laboratories
Ag	silver
ALS	ALS Minerals
AMSE	American Mine Services Engineering Inc.
Au	gold
BD	bulk density
BLM	Bureau of Land Management
BOMENCO	Bolu Minerals Engineering & Consulting Inc.
CAPEX	capital expenditure
Celtic	Celtic Minerals Ltd.
CFM	cubic feet per minute
CGC	centrifugal gravity concentrator
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CRM	certified reference material
Css	Closed side setting
Cu	copper
CUP	Conditional Use Permit
DRMS	Division of Reclamation, Mining and Safety
DTH	down-the-hole hammer
DTM	digital terrain model
Dynatec	Dynatec Mining Corporation
EM	electromagnetic
ft	foot

Abbreviation	Term or Unit
FTSF	filtered tailings storage facility
ft^3	cubic foot
ft^3/tn	cubic feet per ton
FA	fire assay
FC	Fremont County Department of Planning and Zoning
G	gram
g/t (or gpt)	grams per tonne
G&A	general and administration
Ga	giga-annum
Gbu	grey banded unit
GG	granular gneiss
Golder	Golder Associates Ltd.
gpm	US gallons per minute
ICP	inductively coupled plasma
ID	Inverse Distance
ID^3	Inverse Distance Cubed
IP	induced polarization
Jascan	Jascan Resources Inc.
Kb	kilobar
Kg	kilogram
kV	kilovolt
kVA	kilovolt ampere
kW	kilowatt
LHD	load-haul-dump
m	metre
Ma	mega-annum
MagnaChem	MagmaChem Exploration, Inc.
Mercator	Mercator Geological Services Limited
mi	mile
mi^2	square miles
MIBC	methyl isobutyl carbinol
MineTech	MineTech International Limited
mm	millimetre

Abbreviation	Term or Unit
µg	microgram
µm	micron
MRDS	Mineral Resources Data System
MVA	megavolt ampere
MWMP	Meteoric Water Mobility Procedure
NN	Nearest Neighbour
NOI	Notice of Intent
NSR	Net Smelter Return
OK	Ordinary Kriging
OPEX	operating expenditure
oz	ounce
oz/tn	ounces per ton
oz t	troy ounce
PAX	potassium amyl xanthate
Pbu	pink banded unit
PEA	Preliminary Economic Assessment
Pmu	pink massive unit
QA/QC	quality assurance / quality control
QB	quartz-biotite
QBG	quartz-biotite-garnet
QBGS	quartz-biotite-garnet-sillimanite
QP	qualified person
PAG	potentially acid generating
PGE	platinum group element
REE	rare earth elements
ROM	Run-of-Mine
SG	specific gravity
SMU	smallest mining unit
SPLP	Synthetic Precipitation Leaching Procedure
t	metric tonne
t/m^3	tonnes per cubic metre
tn	US short ton
tn/ft^3	tons per cubic foot

Abbreviation	Term or Unit
U-1	Unit-1 granodiorite
UCS	unfolded coordinate system
Uranerz	Uranerz USA
USBRC	US Borax Research Centre
USGS	United States Geological Survey
V	volt
VLF-EM	very low frequency electromagnetic
Wbu	well-banded unit
WQCC	Water Quality Control Commission
WRSA	waste rock storage area
yd^2	square yard
yd^3	cubic yard
Zephyr	Zephyr Gold USA

3.0 RELIANCE ON OTHER EXPERTS

In Section 4 the QPs have relied upon, and believe there is a reasonable basis for this reliance on, information provided by Zephyr regarding mineral tenure, surface rights, ownership details, royalties, environmental obligations, permitting requirements and applicable legislation relevant to the Dawson Project. The QPs have not independently verified the information in these sections and have fully relied upon, and disclaim responsibility for, information provided by Zephyr in these sections.

4.0 PROPERTY DESCRIPTION AND LOCATION

Sections 4 through 6 and 10 through 12 have been updated from the previous 2015 Dawson technical report completed by Hilchey et al. (2015). Golder and Zephyr have reviewed and updated these sections for completeness and accuracy, but are not the primary authors with some minor exceptions. Some text may appear unchanged from the 2015 technical report as authorized for use by Zephyr.

4.1 General

The Property is in south-central Colorado, about 5.9 mi southwest of Cañon City, in Fremont County, CO, USA, and is approximately 162 mi south-southwest of the state capital of Denver (Figure 4.1). The approximate centre of the Property is located at 473,215 mE and 4,249,147 mN (UTM Zone 13 NAD 83).

In October 2012, Celtic Gold Ltd., a Colorado company and subsidiary of Celtic Minerals Ltd. that controlled the Property was purchased by Zephyr Minerals Ltd. and renamed Zephyr Gold USA Inc. Zephyr now holds a 100% interest in a total of 45 contiguous unpatented mining claims and a 50% interest in 8 separate patented mining lode claims and 1 patented mining placer claim. The other 50% interest in the 8 separate mining lode claims is controlled by Zephyr through an agreement with the Allen Group that runs to 2041.

The surface rights on the un-patented mining claims are owned by the Bureau of Land Management (BLM), and Zephyr owns the surface rights for the patented mining claims.

All claims that comprise the Property are found on USGS topographic map sheet Royal Gorge Quadrangle (7.5-Minute Series), and the entire holding covers an area of approximately 950 acres. Claim descriptions and ownership percentages are summarized in Table 4-1 and Table 4-2.

The approximate locations of the claims are shown in Figure 4.2. The locations are approximate because the claims were staked on rugged topography at a time when accurate surveys were not available. Where claims overlap, the guiding principle with the federal BLM has been to give precedence to the earliest-filed claim. The maps (or “plats”) in the county assessor’s office show positions of patented claims and housing lots and the BLM has information on unpatented claims. There does not appear to be a mapping system showing both sets of claims which can be easily accessed digitally and this presents a degree of uncertainty with respect to their graphic representations in this report.

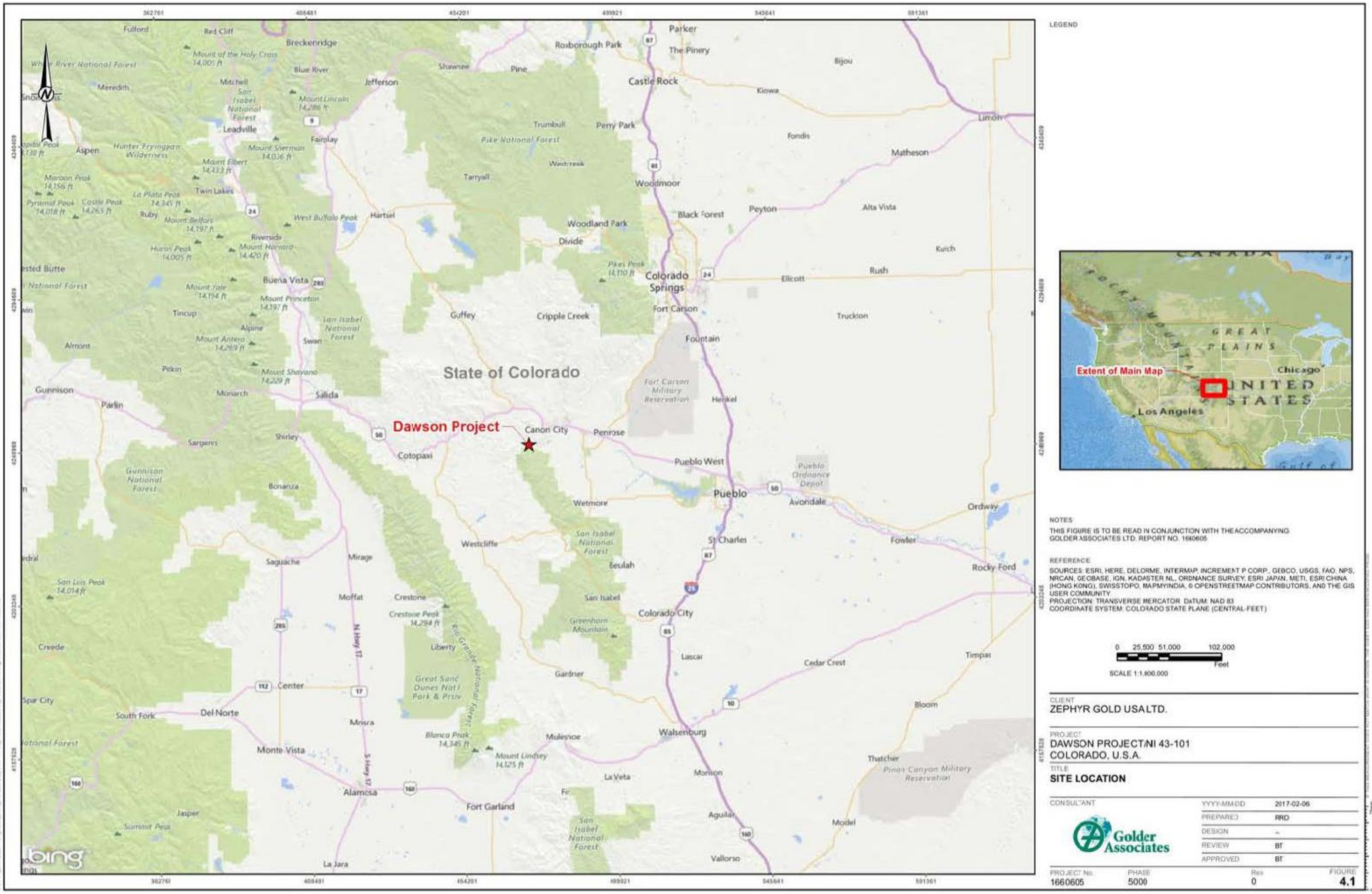


Figure 4.1: General Property Location Map, Colorado, USA

Details of the various agreements between Zephyr and the other claim owners appear in Section 4.2.

Table 4-1: List of Unpatented Lode Mining Claims

Claim Name	BLM Serial Number	Meridian/Township/ Range/Section	Subdivision	Zephyr Ownership (%)
SAM 4	CMC 2404	06 / 0190S / 0710W / 013 06 / 0190S / 0710W / 014	SW SE	100
SAM 5	CMC 2405	06 / 0190S / 0710W / 014	SE	100
SAM 6	CMC 2406	06 / 0190S / 0710W / 014	SE	100
SAM 7	CMC 2407	06 / 0190S / 0710W / 014	SE	100
SAM 8	CMC 2408	06 / 0190S / 0710W / 014	SW / SE	100
SAM 9	CMC 2409	06 / 0190S / 0710W / 014	SW	100
SAM 10	CMC 2410	06 / 0190S / 0710W / 014	SW	100
SAM 11	CMC 2411	06 / 0190S / 0710W / 014	SW	100
SAM 12	CMC 2412	06 / 0190S / 0710W / 015	SE	100
SAM 13	CMC 2413	06 / 0190S / 0710W / 014	SW	100
SAM 14	CMC 2414	06 / 0190S / 0710W / 014	SW	100
SAM 15	CMC 2415	06 / 0190S / 0710W / 013 06 / 0190S / 0710W / 014	SW SE	100
SAM 30	CMC 248398	06 / 0190S / 0710W / 015	NW / SW	100
SAM 31	CMC 248399	06 / 0190S / 0710W / 015	NW / SW	100
SAM 32	CMC 248400	06 / 0190S / 0710W / 015	NW / SW	100
SAM 33	CMC 248401	06 / 0190S / 0710W / 015	NE / NW / SW / SE	100
SAM 34	CMC 248402	06 / 0190S / 0710W / 015	NE / SE	100
SAM 35	CMC 190656	06 / 0190S / 0710W / 015	NE / SE	100
SAM 36	CMC 190657	06 / 0190S / 0710W / 015	NE / SE	100
SAM 37	CMC 190658	06 / 0190S / 0710W / 015	NE / SE	100
SAM 38	CMC 248403	06 / 0190S / 0710W / 015	SW	100s
SAM 39	CMC 248404	06 / 0190S / 0710W / 015	SW	100
SAM 40	CMC 248405	06 / 0190S / 0710W / 015	SW	100
SAM 41	CMC 248406	06 / 0190S / 0710W / 015	SW / SE	100
SAM 42	CMC 248407	06 / 0190S / 0710W / 015	SE	100
SAM 43	CMC 190664	06 / 0190S / 0710W / 015	SE	100
SAM 44	CMC 190665	06 / 0190S / 0710W / 015	SE	100
SAM 45	CMC 190666	06 / 0190S / 0710W / 015	SE	100
SAM 46	CMC 190667	06 / 0190S / 0710W / 022	NE	100
SAM 47	CMC 190668	06 / 0190S / 0710W / 022	NE	100
SAM 48	CMC 190669	06 / 0190S / 0710W / 022	NE	100
SAM 49	CMC 248408	06 / 0190S / 0710W / 022	NE	100
SAM 50	CMC 248409	06 / 0190S / 0710W / 022	NE / NW	100

Claim Name	BLM Serial Number	Meridian/Township/ Range/Section	Subdivision	Zephyr Ownership (%)
SAM 51	CMC 248410	06 / 0190S / 0710W / 022	NW	100
SAM 52	CMC 248411	06 / 0190S / 0710W / 022	NW	100
SAM 53	CMC 248412	06 / 0190S / 0710W / 022	NW	100
SAM 81	CMC 203642	06 / 0190S / 0710W / 015 06 / 0190S / 0710W / 022	SE NE	100
SAM 82	CMC 203643	06 / 0190S / 0710W / 014 06 / 0190S / 0710W / 023	SW NW	100
SAM 83	CMC 203644	06 / 0190S / 0710W / 014 06 / 0190S / 0710W / 023	SW NW	100
SAM 84	CMC 203645	06 / 0190S / 0710W / 014 06 / 0190S / 0710W / 023	SW NW	100
SAM 85	CMC 203646	06 / 0190S / 0710W / 014 06 / 0190S / 0710W / 023	SW, SE NE, NW	100
SAM 86	CMC 203647	06 / 0190S / 0710W / 014 06 / 0190S / 0710W / 023	SE NE	100
SAM 87	CMC 203648	06 / 0190S / 0710W / 014 06 / 0190S / 0710W / 023	SE NE	100
SAM 88	CMC 203649	06 / 0190S / 0710W / 014 06 / 0190S / 0710W / 023	SE NE	100
SAM 100	CMC 206786	06 / 0190S / 0710W / 014	NW	100

Notes:

1) All the unpatented claims listed in Table 4-1 are Lode Mining claims held in the County of Fremont, in the State of Colorado.
2) All unpatented claims require annual rental fees of $155 per claim, payable to the Colorado State Office of the Bureau of Land Management before August 31 of each year. Affidavit and Notice of Intent to hold claims must be filed on or before December 30 of each year.
3) The shaded cells refer to those claims that are subject to the terms and conditions listed in the 1992 Mining Lease and Agreement with respect to Earned Royalty Payments (see Section 4.2.1).

Table 4-2: List of Patented Claims

Patented Claim Name	Claim Type	Patent Number	Mineral Survey Number	Date of Patent (mm/dd/yyyy)	Percent Ownership		
					Zephyr	Allen Group	Mary Louise Adamic
Copper Boy	Mining Lode	33216	13077	11/20/1900	50	50	
Copper King	Mining Lode	31618	12986	10/23/1899	50	50	
Copperopolis	Mining Lode	37311	14991	10/28/1903	50	50	
Last Show	Mining Lode	37963	14992	02/09/1904	50	50	
Mike Sutton	Mining Lode	38038	14993	02/18/1904	50	50	
Rose Bud	Mining Lode	32066	13170	02/03/1900	50	50	
Sentinel	Mining Lode	33126	13015	11/12/1900	50	50	

Patented Claim Name	Claim Type	Patent Number	Mineral Survey Number	Date of Patent (mm/dd/yyyy)	Percent Ownership		
					Zephyr	Allen Group	Mary Louise Adamic
Windy Point	Mining Lode	33067	13127	10/13/1900	50	50	
Fremont	Mining Placer	32379	COCOAA 060787	04/18/1900	50		50

Notes:

1) The shaded cells refer to those claims that are subject to the terms and conditions listed in the 1992 Mining Lease and Agreement with respect to Earned Royalty Payments (see Section 4.2.1).

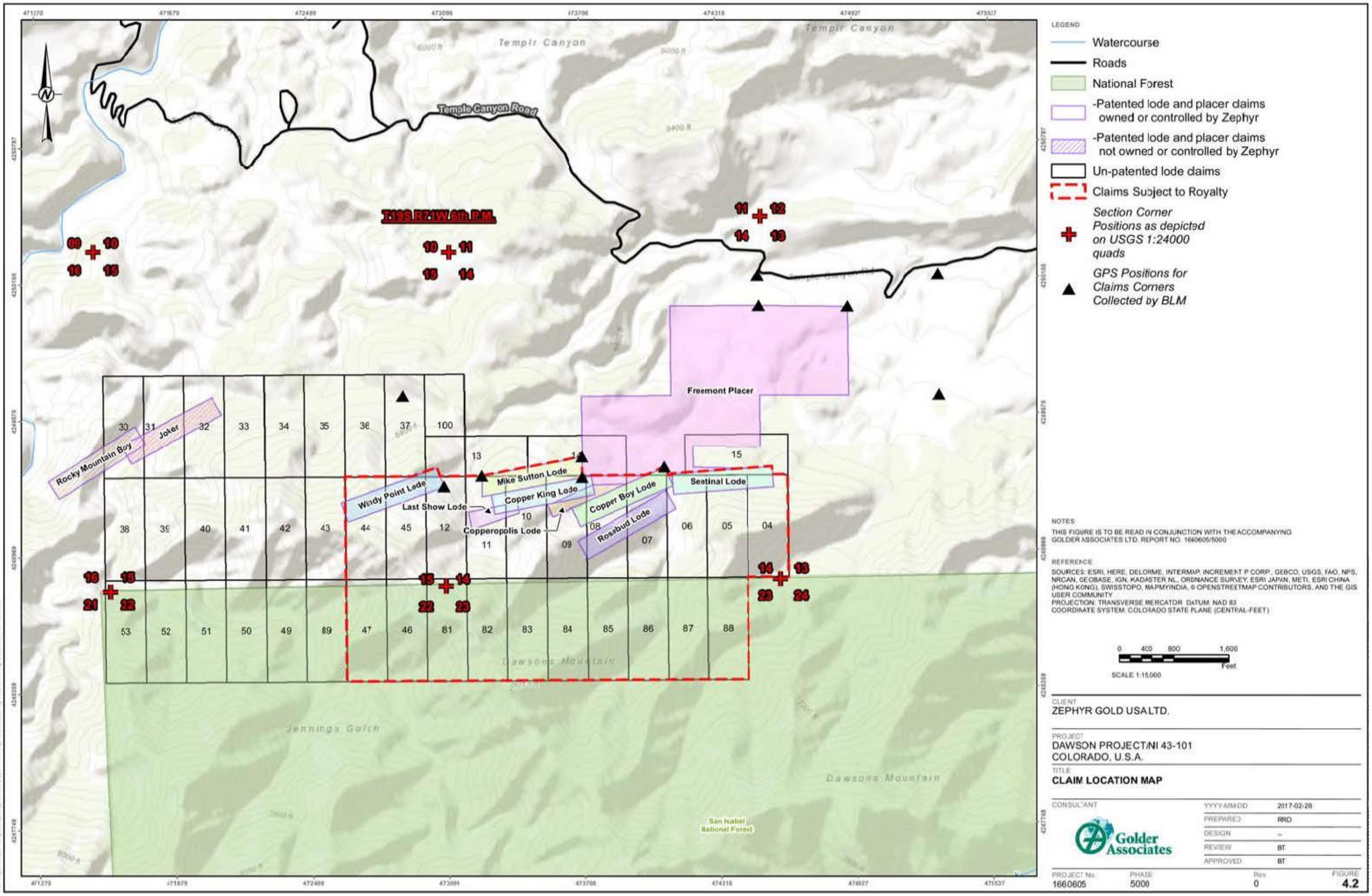


Figure 4.2: Dawson Project Claim Location Map

4.2 Agreements and Royalties

The Dawson Project consists of 45 contiguous unpatented lode mining claims, eight patented lode mining claims, and one patented placer claim, covering approximately 988 acres (1.54 mi^2). Zephyr holds a 100% interest in the 45 contiguous unpatented claims, a 50% interest in the eight patented claims, and a 50% interest in one patented placer claim. The 50% of the eight patented lode mining claims not held by Zephyr is leased by Zephyr through a "Mining Lease and Agreement" which effectively gives Zephyr 100% control of the these claims. Twenty-one of the 45 unpatented claims, the eight patented lode mining claims, and the 50% interest in the one patented placer claim are subject to a sliding scale NSR whereby Zephyr agrees to pay up to a 3% NSR on the aforementioned claims as discussed further in Section 4.2.1.

Where unpatented claims impinge or overlap predating patented claims, the earlier claims take precedence as to exploration, mining, and any agreements made.

4.2.1 1992 Mining Lease and Agreement

The 1992 Mining Lease and Agreement ("the Agreement") is an agreement with the seven owners ("the Allen Group") of the remaining one-half (50%) interest in the eight patented mining lode claims (known in the Agreement as the Dawson Claims). The Agreement is now in its second consecutive term and is due to expire either in 2042 or when the lessors have not received any earned production royalty payment, whichever comes first. If the owners were to abandon all or part of the SAM unpatented claims, the Allen Group is to be given 90 days' notice and has the option to take a conveyance of interest in said claims.

According to the Agreement, Zephyr leases the Allen Group's 50% interest in the 8 patented mining lode claims and in return provides remuneration as outlined in the section of the Agreement dealing with Earned Production Royalties for the 8 patented mining lode claims and 21 unpatented lode mining claims. A sliding-scale NSR of from 1.5% to a maximum of 3.0% (when the price of gold is over $550 per ounce) is to be paid to the Allen Group for these areas. Both patented and un-patented claims are subject to the sliding-scale NSR. The Copper King Lodes occur at the Windy Gulch Segment and the Copper Boy Lode at the Dawson Segment.

An annual minimum royalty payment of $25,000 is due on the anniversary of the Effective Date (January 1, 1992) of the Agreement. The Agreement also states that if an underground program to explore and/or mine starts on the claims listed in the Agreement, the lessors are to be notified and a payment of $90,000 made to them (see the Agreement for details).

Zephyr has advised that at the effective date of this report, all payments required under terms of the Agreement had been made and that it was current and in good standing. The authors have relied upon Zephyr's assertions with respect to validity and currency of mineral titles associated with the Property and have not independently verified any aspects of such titles.

4.2.2 Grant of Royalty Interest

Zephyr has an agreement pertaining to the Freemont patented placer claim for which a Grant of Royalty Interest (3%), dated April 11, 1986, appears in the Fremont County Records.

4.3 Summary of Claim Requirements

Information in this section of the report was taken from a summary of the procedures required to stake and maintain mining claims on federal lands (Rohling, 2008).

In the State of Colorado, staking (or “locating”) a claim is allowed on federally administered lands that are open to mineral entry. The Bureau of Land Management (BLM) manages the surface of public lands and is responsible for the subsurface on public and National Forest System lands. Claims are recorded with the BLM State Office and the local county recorder’s office (in this case, the Fremont County Recorder’s Office, located in Cañon City, CO).

For each unpatented mining claim, an annual maintenance fee of US$155 is required to be paid to the BLM by the 1st of September of each year. For 2016, Zephyr has paid $6,975 to the BLM for maintenance of claims that occur on BLM lands. An unpatented mining claim is a parcel of federal land, valuable for a specific mineral deposit or deposits. It is a lease from the federal government for the right to extract minerals. The rights granted by an unpatented mineral claim are restricted to the development and extraction of a mineral deposit; no land ownership is conveyed. It is possible for the surface and subsurface rights for the same area of land to be owned by different entities; this is known as a “split estate” and each owner has rights and responsibilities for each portion of the land.

There are two types of unpatented mining claims: lode and placer. Lode mining claims cover veins and zones of mineralized rock. Federal statute limits a lode claim to a maximum of 1,500 ft in length and 600 ft in width. Placer mining claims cover all those deposits not subject to lode claims and can include mineral bearing sands and non-metallic layered deposits such as limestone.

Two types of mineral entries, mill sites and tunnel sites, are located to provide support facilities for lode and placer mining claims. The maximum size of a mill site claim is 5 acres, but multiple mill site claims may be consolidated to support a mining operation. A tunnel site claim is used to excavate and develop a known vein or lode or to discover an unknown (or “blind”) vein or lode.

A patented mining claim or mill site is one where the federal government has conveyed title to a specific owner or group of owners, making it private land. A mining patent gives the owner(s) exclusive title to the locatable minerals and, in most cases, also grants title to the surface rights. Mineral patents can be issued for lode and placer claims and for mill sites, but not for tunnel sites. Since late 1994, the BLM has been prohibited by Acts of Congress from accepting any new mineral patent applications. Where mining claims overlap, the guiding principle has been to grant precedence to the claim that was patented first.

Landowners have the responsibility to ensure that any abandoned or inactive mine openings located on their property are adequately safeguarded and are required to maintain the mine closure after the work has been completed (Colorado Revised Statutes: 34-24-110[1]). As of 2010, the Division of Reclamation, Mining and Safety of the Department of Natural Resources for the State of Colorado had placed barricades to most of the entrances of adits and shafts of the historical workings on the Property. Several do not permit access but the rest consist of locked grates with keys held by the BLM in Cañon City.

4.4 Environmental Liabilities, Permitting, and Easements

The authors of this technical report are not qualified to provide an opinion with respect to environmental conditions, potential hazards or liabilities that may be present on the Property. However, during the site visit and drilling supervision programs, co-author Mark Graves observed that various historical underground workings areas are present on the Property and that some of these are characterized by rusty-weathering waste rock piles. These piles may represent potential sources of acid rock drainage under certain surface drainage conditions. Additionally, hazards appear to exist locally with respect to open historical mine workings and pits. Zephyr informed Mercator that it has posted a bond for the future reclamation of drilling related work which includes roads, lease, and drill hole sites, but it is not subject to any other known environmental liabilities.

Zephyr must have a "Notice of Intent" (NOI) to work on the Property. If Zephyr applies and obtains a mining permit in the future, it will not need an NOI for exploration and development work. Patented claims are subject to permitting by the State of Colorado, whereas the unpatented claims are subject to permitting by the Bureau of Land Management.

Zephyr has received a legal opinion stating it has an un-contested right to access the Property without needing to receive legal easement from other patented claim holders. Zephyr did not experience any issues accessing the Property during the 2013 and 2016 drill programs.

The status of environmental studies, permitting and social or community impact is discussed in Section 20 of this report.

Federal mining law protects Zephyr's right to develop minerals on the Property. There aren't any known risks to the title or legitimacy to perform work.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility, Climate and Physiography

The Property is situated in south-central Colorado, approximately 5.8 mi southwest of the community of Cañon City. The town is some 162 mi along paved US Interstate Highway 25 and State Highway 50 south-southwest of Denver International Airport (see previous Figure 4.1). Access to the eastern part of the claim block from Cañon City is by a maintained gravel road (County Road 3, also known as Temple Canyon Road) for about 5 mi. From that point, and after passing through a locked gate, it is about another mile via gravel and dirt track to the historical Copper Boy mine workings. Several trails meander throughout the hilly topography and can be navigated by an experienced operator of a four-wheel-drive or all-terrain vehicle. An abandoned rail bed along Grape Creek provides level foot access west of the Property. Access to the Property is year-round, although wet winter to spring conditions may limit it to the less hilly terrain.

This area of Colorado sits in the high desert, with a moderate semi-arid climate and average temperatures ranging from about -19°F in January (with a record low of about -24°F) to about 90°F in July (with a record high of about 108°F). Precipitation totals about 13.3 inches during the summer, precipitation occurs as showers with scattered thunderstorms; snow occurs from November to March (Wheeler et al., 1995). The area experienced drought conditions during 2013. The working season for this area is 12 months of the year.

The easterly flowing Arkansas River and its tributaries drain Fremont County. Grape Creek, just west of the claim block, is a perennial stream that drains most of the Wet Mountains in south-central Colorado. Water flow through the area is controlled upstream by the DeWeese Reservoir to the south.

At higher elevation the ground is open, with minor undergrowth of cactus, sagebrush, grass, and yucca. Lower elevations contain pinon-juniper, gambel oak shrubs, aspen, dwarf cedar, cottonwood, and jack pine (Pearson and Fielder, 2001; Wild Connections, 2008).

The Property lies in the Central Cordillera, on the northern edge of the Wet Mountains, just south of the Front Range of the Southern Rocky Mountains (Figure 5.1), an area characterized by rugged topography ranging in elevation from 6,135 to 7,874 ft. Most of the Property contains exposed bedrock interspersed with areas of residual granitic and gneissic soils. The lower areas of gulches tend to have relatively thick (up to about 75 ft) accumulations of fine- to coarse-grained and poorly sorted alluvium and colluvium.

About 6 mi to the northeast, an embayment in these mountains is occupied by a small portion of the Interior or Great Plains physiographic province (the Cañon City–Florence Embayment or Basin), defining a less rugged topography. The embayment is surrounded by the foothills of the Precambrian granites, gneisses and schists of the Wet Mountains to the south and southwest and the Front Range Precambrian granodiorites to the north and northwest. The foothills are characterized by hogbacks and cuestas in areas of sedimentary bedrock and by rugged hills of gneiss and granite (Campbell, 1910; Kruse, et al., 1988). The Property sits within BLM-administered land where exploration and mining activities are under the purview of the regulations of the Secretary of the Interior contained in the Code of Federal Regulations (CFR), Title 43 (Public Lands: Interior).

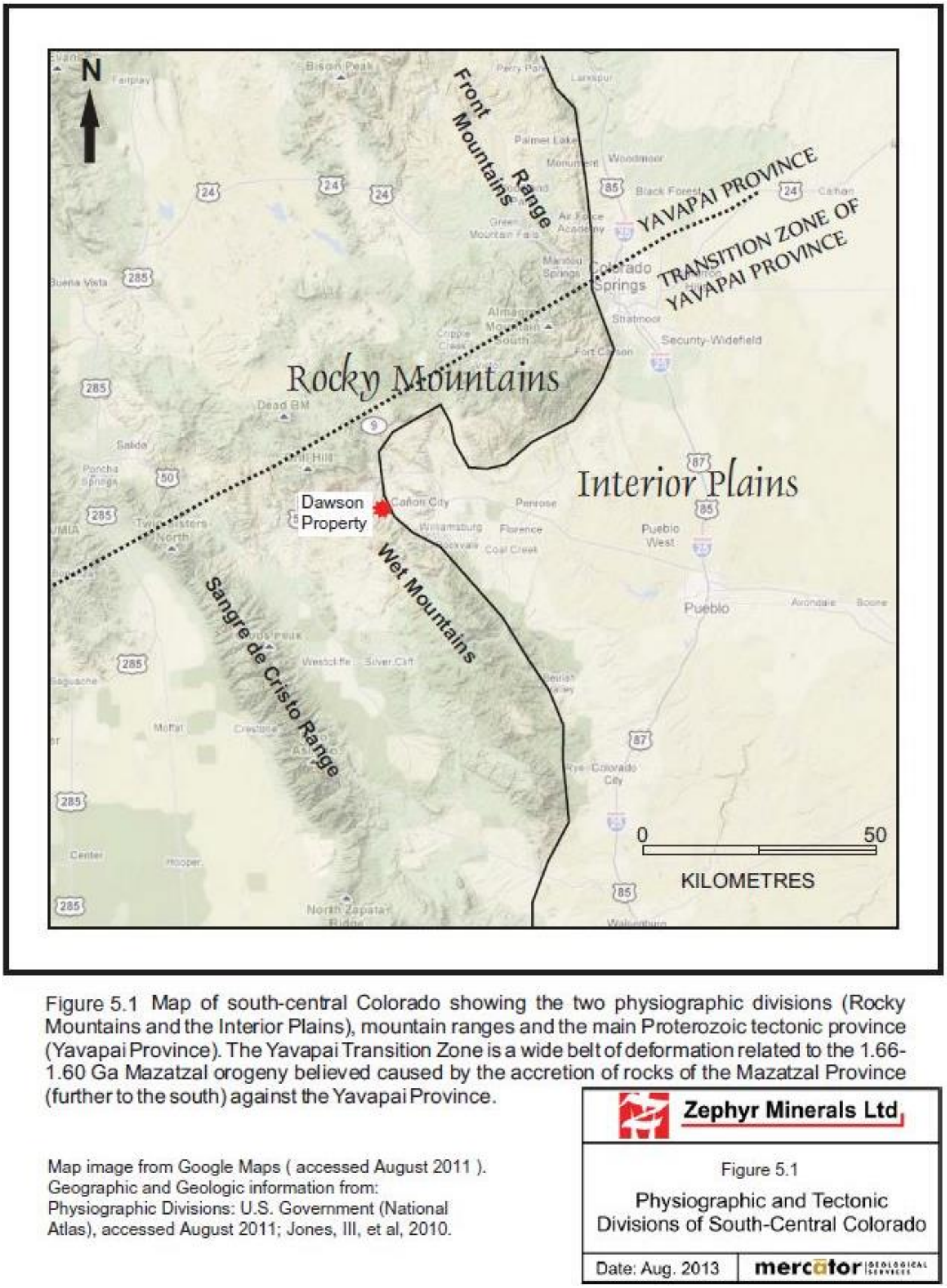


Figure 5.1: Physiographic and Tectonic Divisions of South-Central Colorado

5.2 Local Resources and Infrastructure

Cañon City is the county seat of Fremont County with a population of about 22,000 in the city and surrounding area. The economy is based mainly on the tourist trade, government services (for the 13 state and federal correctional systems), and retail, health, and construction services. Cañon City offers all the amenities of a small community that caters to government workers, retirees and tourists with the addition of minor manufacturing. Cañon City is about 162 mi from the Denver International Airport by paved interstate and state highways. The route includes 124 mi of travel from the Denver airport south along Interstate 25 to the city of Pueblo (population about 104,000) and then west from Pueblo along US Highway 50 for another 37 mi. The county-managed Fremont County Airport, 6 mi east of town, provides unscheduled charter services. A charter helicopter service is also available for pick-up and drop-off of passengers with about one weeks' notice to the Federal Aviation Administration. A 4 to 5 mi power line would need to be installed to bring power to the Property, and electricity prices are around $0.08 per kilowatt.

Water is an important resource in Colorado. Water to the town of Cañon City is provided by groundwater sources near major streams and by the Arkansas River through a series of diversion ditches and canals. Water stored in the Deweese Reservoir in Custer County (to the south) is released into and diverted from Grape Creek and is used for irrigation in the Cañon City area (Wheeler, et al., 1995; Trimble, 1980). Some wells that were drilled in search of oil instead hit artesian water, which in places supplies parts of Cañon City (Henry, et al., 1996). An important source of surface water is snowmelt from the adjacent mountains.

Zephyr informed Mercator that for the scale of mining operations it has planned, it currently has sufficient surface rights to accommodate potential tailings storage areas, waste storage areas, and a processing plant site. Zephyr does not currently anticipate conducting heap leaching on the Property.

Fremont County was founded on the mining industry which included metals and coal which included development back to the mid-1800s. Currently, Fremont County is a major construction aggregate producer in the State of Colorado. Cripple Creek, Teller County is 18.6 mi north of Cañon City and employs Fremont County residents at the gold mine.

6.0 HISTORY

6.1 Introduction

Cañon City was settled in 1860 and during the late 1800s became a supply centre for the gold mining industry in the Cripple Creek mining district approximately 21 mi to the north. The town also housed several smelters for Cripple Creek's gold "ore."

During the mid-1800s, poor prospecting results and a gold hoax in the Cripple Creek area discouraged mineral exploration in the region, with sporadic prospecting during the late 1870s and 1880s. By the 1890s, most prospectors' attention was drawn back to the genuine gold rush of Cripple Creek.

In late 1898, the entrepreneurial Dawson family reported a small pocket of gold "ore" on its copper-bearing Copper King claim (located on the Property). Early inflated accounts of a rich, 14.9 mi long vein of gold "ore" near the Property started a short-lived staking rush. This led to the rapid growth (and equally rapid decline) of nearby Dawson City (Akron Weekly Pioneer Press, 1898-1899; Retzler and Burke, 2011).

Information regarding early exploration and mining is sparse; most of it has been gleaned from the United States Geological Survey's Mineral Resources Data System (MRDS), digital copies of local newspapers of the day (accessed through the Colorado Historic Newspapers Collection, 2011), and industry notes taken from historical Cañon City newspapers. Exploration conducted since the 1970s is reported from internal industry reports.

Summary descriptions of US Borax, Jascan, and Uranerz work programs reflect information obtained from the following reports: Aiken (1977), Aiken (1982), Aiken et al. (1983), Boyko (1978), Freiberg (1979), Ganderup and Woods (1986), Hambrick and Theye (1985), Theye (1986), Theye and Aiken (1986), American Mine Services Engineering Inc. (AMSE 1991), Barnett (1989a), Barnett (1989b), Kettner et al. (1991), Lendrum (1988), Mettler (1991), Mettler (1992), Mettler (1993), Priesmeyer (1990), Theye (1988), Theye (1989), and Wolfson (2011).

6.2 Summary of Past Exploration

This section provides a brief history of past exploration on the Property. The reader is advised that all resources and reserves referenced in this section of the report are historical in nature and not compliant with NI 43-101. A QP as defined under NI 43-101 has not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and Zephyr is not treating the estimates as current mineral resources or mineral reserves as defined in Sections 1.2 and 1.3 of NI 43-101. These historical estimates should not be relied upon.

The Copper King claim had been explored as a copper prospect for several years before late 1898, when a 3 ft wide zone of gold-bearing quartz was exposed. Various optimistic newspaper articles reported that the main vein had been traced for some 15 mi and that the find was the biggest news in Colorado. A massive staking rush ensued, during which time the Dawson City town site was surveyed and built on a nearby placer claim owned by EF Jewett. The city was noted to have several houses, a general store, and a school in the early 1900s.

Although one lone reporter believed that the gold "ore" came from a small isolated pocket within a copper prospect, the ensuing rush led to the development of several small shafts and adits and numerous pits both on and to the west of the Property (archived newspapers of the late 1890s via Colorado Historic Newspapers Collection: Aspen Daily; Aspen Tribune, Chaffee County Republican, Colorado Transcript; Pagosa Springs, Yuma Pioneer). The small pod of gold "ore" at the Copper King claim was mined out in three months and minor scattered pods of similar gold-bearing rock were also described in the area.

From 1900 to about 1908, the Copper King Mine was owned by several consortiums and sporadically produced an estimated 550 metric tonnes of ore from two timbered shafts (one to about 367 ft depth), three adits and four lateral development levels (USGS MRDS, accessed August 2011, Cañon City Recorder, 1900 to 1907). By the 1930s, the claim contained only derelict foundations and some abandoned boilers and mining machinery.

The nearby Copper Boy Mine had reported production (1902 only) of 442.9 metric tonnes of copper-bearing "ore" from three pits, three adits, and a 100 ft winze (USGS MRDS, accessed August 2011). Newspapers of the day reported one shaft at 175 ft by 1902, with another down about 56 ft (March 1901).

Farther east, the Sentinel Mine had a shaft down to about 90 ft (reported March 1901), with copper mineralization in a body 6 to 8 ft wide.

The Mike Sutton claim to the west had several shallow shafts and one haulage adit.

The Windy Point claim contained several shafts, one of which was 110 ft deep (reported March 1901), with 150 ft of lateral development.

Farther to the west, on the Joker claim (which overlaps several unpatented claims), the Joker tunnel was reported to be about 600 ft in length as of December 1901, and was working on a 4 ft wide "vein" of copper-bearing mineralization.

Other small shafts, adits, and prospect pits throughout area were mined as prospectors followed copper/iron gossans and/or copper-bearing iron sulphide zones.

Minor copper production from the area was noted in the early 1910 to 1920 period and during the 1940s. The Horseshoe Mine to the west of the Property worked the same mineralized horizon as that at the Copper King Mine and produced several thousand short tons of zinc-lead ore during this period. Continuing west, the El Plomo Mine was noted to contain lead-silver mineralization and was worked from two shafts and one tunnel. The Columbine Mine farther to the west (and on the same horizon) produced copper and minor zinc during the 1940s.

As of 2010, most of the larger historical mine workings had been barricaded by temporary or permanent closures by the Division of Reclamation, Mining and Safety of the BLM (Carter, 2010–2011).

Dedicated exploration of the Property did not resume until the entrance of US Borax (now Rio Tinto Borax) in the 1970s.

6.2.1 US Borax (1976–1986)

In 1976, a reconnaissance program carried out by company geologists examined the thin, but extensive sulphide-bearing zone. A later helicopter-supported geophysical electromagnetic (EM) survey in 1978 delineated an east–west fault region. Their property was subdivided into "segments" for administrative purposes and included land from the current Property and to the west for about 1.7 mi to the Marsh Gulch.

US Borax began drilling the Property in 1979 (GC02). In 1981, eight holes were drilled from known surface expressions or old workings (GC03 to GC10) to intersect the sulphide horizon at depth. GC08 and GC09 were drilled to the west of the Property. GC05 was drilled in the Copper Boy claim area and intersected about 11 ft of almost 2% copper and about 10 ft of 0.9 oz/tn of gold (which was later re-assayed to 0.4 oz/tn).

In 1982, six holes were drilled around GC05 to test for strike and dip continuity of the gold mineralization (GC11 to GC16). Two apparently separate sulphide zones with gold mineralization below both zones were encountered. Fourteen holes were drilled in 1983 (GC17-40) to test the western portion of the Dawson Segment and the down dip extension of known mineralization.

Drilling continued in 1984 in the Dawson Segment (15 holes; 2 abandoned) and the Windy Gulch Segment (3 holes) testing deeper targets (GC41 to GC57). During the same year, the Sentinel Segment, east of the Dawson Segment, was mapped; new roads were constructed to permit better access for diamond drill rigs, and road cuts in the Windy Gulch Segment were mapped and sampled. Additional lode mining and mill site claims were either staked or bought to increase their land position and close any gaps between claims.

In 1985, five deep drill targets in the eastern and central segments were drill-tested, trying to confirm the hypothesis that the mineralization occurred as plunging grade shoots (GC58 to GC62), but no "ore-grade" mineralization of mineable width was intersected. Surface trenches were cut or dug out at the historical Copper Boy and Sentinel workings. At the Copper Boy trench, isolated gold anomalies were found with repetition of gossans and granular gneiss units, possibly due to complex folding or high-angle faulting. The previously built drill roads were reclaimed at the end of the exploration season. Company geologists discussed the need for underground mapping and bulk sampling and proposed a two-phase underground exploration program designed to test about 70% of the drill-indicated mineralization in the Dawson Segment.

Hoping to reduce drilling costs, the company in 1986 tested reverse circulation drilling adjacent previously cored diamond drill holes. It found that the holes deviated too much, with high water flows (100 to 150 US gallons per minute [gpm]) washing away drill cuttings, resulting in unreliable assays. By 1986, US Borax had drilled 62 diamond core holes totaling about 50,032 ft. The longest hole was 1,809 ft (GC62), and over the years the company had experienced drilling difficulties due to technical problems and access to the rugged terrain. In the western half of its property, only four short holes had been drilled due to access problems.

In 1986 the company purchased a half-interest in the Fremont placer claim and worked on geological interpretation and calculation of a historical “mineral reserve” (see Section 6.3 for reserve details). The mineralized zone in the Dawson Segment was interpreted to be an irregular-shaped tabular body dipping about 65° to the south. By November of that year, the planned 1987 exploration budget was reduced and the company offered its ongoing gold exploration projects to interested parties.

6.2.2 Jascan Resources Inc.–Atlantic Goldfields Inc. Joint Venture (1987–1988)

In 1987 the joint venture of Jascan (60% ownership) and Atlantic (40% ownership) bought the Property from US Borax and contracted ACA Howe International Pacific Ltd. (ACA Howe) to carry out exploration. At this date, the Property comprised 100 unpatented mining lode claims, leases on 11 patented mining claims, and a lease on Colorado state section 16. Surface rights to 35 mill-site claims and a one-half interest in the 144 acre Fremont placer property made up the rest of the land package.

A 10-year conditional-use permit was secured from Fremont County to allow mining operations on the Property.

ACA Howe proposed an underground development program for bulk sampling and diamond drilling as well as a surface drill program. The plan was to confirm grade, determine grade distribution and continuity, obtain a representative bulk sample, block out proven and probable reserves, and look for additional mineral resources.

During the year, ACA Howe compiled previous drill hole and geological data and completed 9,826 ft of surface diamond drilling in 17 holes. A new gold-bearing grade shoot was discovered about 197 to 295 ft immediately west of the Dawson Segment. In the Windy Gulch Segment, two new gold-bearing shoots were identified: one may be the down-plunge extension of a previously identified road cut exposure; the second shoot is about 400 to 449 ft west of the road cut exposure and has only been drill tested by two drill holes. Windy Gulch drilling (seven holes) tested the shallow, down-plunge extension of the gold mineralization found in 1984 channel sampling along the Windy Gulch road cut exposures.

A reconnaissance mapping and sampling program along a 2.5 mi strike length of the mineralized horizon was conducted all the way west to the Property's western-most claim boundary (Marsh Gulch) in which 44 mines, prospects, and outcrops were mapped in detail; numerous geochemical anomalies were identified for further follow-up.

About 282 ft southeast of the historical Copper Boy workings (on Dawson Segment), a work pad and some portal site work was completed in preparation for the underground program. The approximately 5,561 yd^2 work pad was sited in Dawson Gulch and a 2.1 m diameter, 200 m long culvert was placed beneath the work pad to allow its continued drainage. A 39.4 ft wide by 24.9 ft high portal face in the hanging wall rocks was excavated just west of the work pad.

A mining claim lease dispute involving the eight patented mining lode claims in the Windy Point to Sentinel segments stopped the planned underground program.

In 1988, ACA Howe conducted detailed geological mapping and sampling of the mineralized horizon along the 1 mi strike length between the Dawson Segment and Windy Point Segment. A series of sub-parallel, northeast-trending, high-angle faults were mapped.

An intensely carbonatized and sericitized zone was found in an epidote-quartz-garnet gneiss within the patented Rosebud claim, but drilling in the area failed to intersect any gold mineralization.

About 8,445 ft of core was drilled in 1988. The 12 holes at the Windy Gulch Segment (GC80 to GC91) were drilled to further delineate the gold mineralization found in the Windy Gulch road cut. Seven underground holes (in the Mike Sutton adit; MS01 to MS07) were drilled to determine near-surface eastward continuation of the road-cut mineralization, but suffered poor core recovery. The Windy Gulch Segment drilling identified three individual gold-bearing zones within three-fold limbs.

Seven holes drilled in the Dawson Segment (GC92 to GC98) defined a third distinct gold-bearing zone (named 2a), in addition to the two zones previously found by US Borax.

Ground geophysical surveys over the Dawson to Windy Gulch and Windy Point segments included magnetics and VLF-EM surveys, which showed the relationship between known gold mineralization and areas of low magnetic response, and confirmed the presence of mapped high-angle faults. The geophysical surveys also identified eight additional untested shoots within the Dawson Segment to Windy Point Segment.

6.2.3 Jascan Resources Inc. (1989–1990)

In 1989, Jascan acquired the remaining 40% interest in the Dawson Project, previously held by Atlantic, to have 100% ownership in the Property. Jascan maintained ACA Howe for its exploration work on the project.

A trenching program was carried out at the Windy Point Segment in which the trenches were mapped and 88 channel samples taken. Mineralization within the road-cut mineralization of the Windy Gulch Segment was re-sampled to resolve assay discrepancies which were found due to the nugget effect.

The geophysical grid in the Dawson Segment was extended to the east to cover the southern boundary of the Sentinel patented claim, northward to the contact with the Mesozoic sedimentary rocks. A ground proton magnetometer survey was carried out over this grid and demonstrated that the magnetic low, which is associated with gold mineralization, continues to the east and remains open.

ACA Howe continued its reconnaissance geology program and carried out a reinterpretation of hanging wall and footwall relationships of the mineralization. The mineralization model was reinterpreted to be confined to discrete southwest-plunging grade shoots controlled by F2 fold hinges.

ACA Howe also conducted project presentations and field tours to prospective joint venture partners. Potential buyer Cyprus Minerals hired a consultant (Monte Swan of MagmaChem) to evaluate the Property for its mineral potential and examine its chemistry to try to get a sense of the nature of the mineralization. The consultant stated that the known mineralized plunging grade shoots (correlative with geochemical anomalies and magnetic lows) are open at depth and along strike and that there were untested geochemical anomalies and magnetic lows present on the Property.

6.2.4 Jascan Resources Inc.–Uranerz USA Joint Venture (1990–1995)

In 1990 Uranerz signed a joint venture agreement with Jascan and earned interest in the Property by making cash payments and performing exploration work during 1990 and 1991. Prior to signing the agreement, Uranerz drilled eight holes (DA9001 to DA9008) in the Dawson Segment to test the ACA Howe plunging grade shoot theory by drilling in between the identified grade shoots. Seven out of the eight drill holes intercepted gold mineralization, which established the relatively continuous nature of mineralization between grade shoots in the Dawson Segment and the geological interpretation for this part of the Property was revised.

Uranerz drilled a fence of five short Winkie[1] drill holes (DWP9001 to DWP9005) in the most densely folded part of the Windy Point Segment to try to follow up gold mineralized surface showings and establish additional resources. Gold was found to be mainly concentrated in plunging grade shoots following southwesterly plunging fold hinges. After signing a second six-month option, Uranerz drilled an additional 10 Winkie holes in two fences on either side of the original five-hole fence (DWP9007 to DWP9016).

A six-hole program was planned for the Copper King Segment to intersect the siliceous exhalative horizon, with holes ranging from about 1,312 to 1,969 ft in length. As a cost-saving measure, four holes were pre-percussed to a depth of about 600 ft. Due to numerous drill-technical problems and some ground related conditions, only three holes were completed. Two drill holes (DCK9004 and DCK9005) were completed in the central area and one hole (DCK9003) was fully core drilled on the eastern side of the segment. Two additional holes were already pre-drilled in the western half of the segment to depths of 600 and 899 ft for completion later.

The three deep drill holes on the Copper King Segment did not intersect gold mineralization or the sulphide zone. One of the holes indicated an interpreted pinch-out of the exhalative zone to the west. It was thought that mineralization occurred to the east and southeast at relatively shallow depths between the Copper King Mine and drill hole GC71. The increase in chlorite-biotite schists within the siliceous “exhalite” and the absence of the sulphide horizon possibly indicated a more distal position with respect to a hydrothermally active vent area.

In 1990 Uranerz drilled 30 holes totalling almost 17,388 ft. The existing database of the Windy Gulch area was re-evaluated and three areas of gold mineralization were identified. Resource calculations were carried out (as described in Section 6.33) and AMSE was contracted to conduct a pre-feasibility study (which was completed January 1991).

In 1991 Uranerz contracted MagmaChem (Swan, 1991) for a geochemical study of its project area. After analyzing metal dispersions and isopachs of the target horizon, MagmaChem proposed a feeder for the mineralizing fluids in the central part of the Dawson Segment, suggesting potential for gold mineralization west of drill hole GC73

[1] Very small diameter core.

(down dip of the area tested in 1991) and extending up to the fault zone dividing the Dawson and Copper King segments. The ore-bearing horizon within the Copper King Segment had been interpreted as a structural offset from the Dawson block which has been mostly eroded.

Referring to the pre-feasibility study by AMSE, Uranerz noted that the economic safety margins regarding an underground development of the Dawson Segment were too narrow to justify proceeding with an underground exploration program. A substantial increase in reserves or in the price of gold would enhance the economic viability of the project (the average gold price in 1991 was about US$350/oz t). Based on the pre-feasibility study, Uranerz designed the drilling program to test the most favourable unexplored area of the Dawson Segment, that is, west of hole GC71. This area is the continuation of the most prominent mineral trend which contains the highest grade intercepts and was believed to be situated at relatively shallow depths which could have altered the proposed mining layout so as to be more economically viable. The three holes that were drilled in the Dawson Segment up to 150 ft west of GC72 to test the shallow western strike extent of mineralization did not intersect significant mineralization (U9101 to U9103, also known as DA9101 to DA9103).

Seven holes were drilled in the Windy Gulch Segment (WG9101 to WG9107), which confirmed the continuation of the targeted auriferous horizon to the east and down dip and encountered gold mineralization (0.1 to 0.9 oz/tn) in all holes. Drilling also indicated that the lateral continuity of mineralization is locally disrupted by faulting or by post-mineral intrusions.

In 1991 Uranerz drilled a total of 3,763 ft in 10 holes. Mapping and geochemical sampling of the western segments suggested decreasing potential for gold mineralization west of the Windy Point Segment.

In 1992 Uranerz drilled one hole to comply with annual assessment work requirements (DCK9201). It was the predrilled DCK9001 hole that was deepened to 1,187.6 ft after the attempt to deepen the pre-drilled hole DCK9006 failed. This hole was drilled in the western portion of the Copper King Segment (west of DCK9004 and DCK9005). Both footwall and hanging wall rocks (Pbu and Gbu; and Unit-1, Wbu, respectively) were intersected, with no central siliceous, gold bearing horizon, suggesting a pinch out, not only in the western parts of Copper King, but following west into the eastern part of the Windy Gulch Segment (as defined by DCK9004 and DCK9005). Based on results from these three drill holes, Uranerz concluded there was limited potential to substantially increase the mineral inventory.

In 1993 Uranerz returned operatorship of the Property back to Jascan. No major exploration work has taken place on the Property since 1992.

6.2.5 Celtic, et al. (1995 to October 2012)

6.2.5.1 General

In 1995 Celtic purchased Jascan's 70% interest in the Property and commissioned several reviews.

In 1998, Uranerz was merged into Cameco and was renamed UUS Inc. Uranerz's 30% earned interest in the Property remained with UUS Inc. until it was purchased by Zephyr in October 2012.

From 2008 to 2009, Property drill logs and surface sampling information were digitally compiled and input into a geological database. No on-site exploration was conducted on the Property by Celtic, but the company prepared a NI 43-101 compliant technical report in 2011 that provided an up-to-date view of exploration potential and associated recommendations for future Property evaluation. In October 2012, Celtic Gold Ltd., a Colorado company and subsidiary of Celtic Minerals Ltd. that controlled the Property was purchased by Zephyr Minerals Ltd. and renamed Zephyr Gold USA Inc.

6.2.5.2 NI 43-101 Report (2011)

In September 2011, an NI 43-101 report was completed for Celtic on the Property by Isobel Wolfson, P. Geo., which included independent sampling and a site visit.

Ms. Wolfson completed independent sampling of four mineralized intervals previously reported by ACA Howe in its drill logs and by Uranerz in its resource calculations. Portions of drill holes GC89 (Windy Gulch Segment, drilled 1988), GC98 and DA9003 (both in the Dawson Segment, drilled 1988, 1990, respectively) were examined on August 30, 2011, at the leased shed in Cañon City, CO.

Previous workers have noted the coarse-grained nature of some of the gold mineralization on the Property and proper sampling practice would be to either quarter core the entire selected interval or to sample the complete half of the core interval. No facilities existed on site to split or cut the core into quarters, and it was thought prudent to retain some portion of core for further work if required. Approximately every other piece of core in the selected interval was therefore sampled and it is acknowledged that although confirming the presence of gold mineralization, the resultant analysis may not show good reproducibility of gold assays. Results in Table 6-1 show good reproducibility in three of the four assays.

Four drill core samples from the above-mentioned drill holes were shipped to Actlabs in southwestern Ontario. Gold analysis was by the screened metallics fire assay (FA) method: a 500 g sample split is sieved at 150 mesh (105 µm), with assays performed on the entire +150 mesh fraction and two splits of the -150 mesh fraction. When assays have been completed on the coarse and fine portions of the sample, a final assay is calculated based on the weight of each fraction (Actlabs 2011). A summary of the comparison of the assay values for samples reported by ACA Howe and Uranerz and the values returned from the current samples are reported in Table 6-1.

Table 6-1: Comparison of Drill Core Assay Results

Drill hole	Year Drilled	From (ft)	To (ft)	Sample No.	Year of Assay	Assay Lab	Au ppm**	Au oz/tn
GC89	1988	216.8	219.8	272720	2011	Actlabs	22.3	
GC89		216.8	219.8	unknown	1988?	unknown	19.4	0.566
GC89	1988	281.9	284.9	272721	2011	Actlabs	79.6	
GC89		281.9	284.9	unknown	1988?	unknown	81.1	2.364
GC98	1988	549.5	558.1	272718	2011	Actlabs	15.9	
GC98*		549.5	558.1	unknown	1988?	unknown	14.5	0.422
DA9003	1990	1133.0	1138.0	272719	2011	Actlabs	32.3	
DA9003		1133.0	1138.0	unknown	1990?	Uranerz lab?	19.7	0.574

Based on the assessment and interpretation of results of past exploration programs, Wolfson concluded that the following prospective exploration targets should be assessed. In part, these targets are based on the recommendations of previous exploration programs and are listed in geographical order from east to west.

Sentinel Segment

A ground geophysical survey found the continuation of a low magnetic response which should be checked through trenching and possible drill-testing and downhole EM surveys.

Dawson and Copper King Segments

Swan (1991) suggested drill testing the possible “exhalative feeder system" west of drill hole GC73, down dip of the area tested by 1991 drilling, as well as between holes GC57 and GC58. The area around the fault zone separating the Dawson and Copper King segments should also be checked as gold mineralization may have been remobilized and redistributed during deformation.

Holes DA9006 and GC98 were drilled under the historical Copper Boy workings and are about 174 ft apart on Drill Section S48256E (Wolfson, 2011). Both contain gold grades which should be investigated further. DA9006 has a reported intercept of 1.51 ft of 3.326 oz/tn Au; GC98 has a reported intercept of 2.99 ft of 0.938 oz/tn Au.

Windy Gulch and Windy Point Segments

The gold-mineralized zone from the Copper King workings west to the Windy Point Segment has been traced to just west of the Mike Sutton workings and then is lost under alluvial cover in Windy Gulch. Previous workers have recommended both surface and underground diamond drilling in this area, including the favourable magnetic lows between the Mike Sutton and Copper King workings.

The area between the Last Show shafts and the eastern sections of the Windy Point Segment had some interesting copper and gold assays from grab samples which should be followed up. Most of the identified resource in the Windy Gulch Segment is at or just below surface. Since limited drilling was conducted in this area (by Uranerz in 1991), the down-plunge portions of the segment should be examined in more detail.

The eastern portion of the Windy Point Segment is mostly covered with overburden. Ground EM and magnetic geophysical surveys using current technology should be conducted here to locate areas of low magnetic response for future drill testing. Up to 10 moderately deep drill holes (492 to 804 ft) and several shallow holes were recommended by previous workers.

Work by Alers (2003) suggests that the main F2 fold has been overturned during D3 to a horizontal position and plunges to the northeast, not the southwest as in previous interpretations. He suggests drilling to the northeast in the Windy Point Segment to verify this hypothesis.

Jennings Gulch–Joker Segment

Although covered by the Joker and Rocky Mountain Boy patented claims (not owned by Zephyr), the exhalative horizon dips towards the south under the unpatented claims (SAM 30 to 32 unpatented claims). There has been limited exploration here in the past, due to the exploration focus on the eastern segments.

6.3 Historical Mineral Resource or Mineral Reserve Estimates

The reader is advised that all resources and reserves referenced in this section of the report are historical in nature and not compliant with NI 43-101. A QP as defined under NI 43-101 has not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and Zephyr is not treating the estimates as current mineral resources or mineral reserves as defined in Sections 1.2 and 1.3 of NI 43-101. These historical estimates should not be relied upon. Brief notes pertaining to each estimate follow Table 6-2.

Table 6-2: Historical Mineral Resources/Reserves

	US Borax 1985		ACA Howe 1987		ACA Howe 1988		Uranerz USA 1990	Uranerz USA 1991
Method	Polygonal method on single longitudinal section		Polygons drawn on longitudinal sections for each mineralized zone		Polygons drawn on longitudinal sections for each mineralized zone		Dawson Segment: 3 different methods. Windy Gulch Segment: Mean thickness & thickness weighted grades	Inverse distance squared method
Tonnage Factor	12.0 ft^3/tn		12.0 ft^3/tn		12.3 ft^3/tn		12.4 ft^3/tn	12.4 ft^3/tn
Block Thickness	True block thickness of ≥15 ft with no cutting of individual assays		Minimum true width of ≥15 ft					True thickness of 5 ft
Cut assays?	All assay values uncut		Assays >1.0 oz/tn Au cut to 1.0 oz/tn		Assays >1.0 oz/tn Au cut to 1.0 oz/tn		All assay values uncut	All assay values uncut
Other			Au mineralization in 2 zones		Minimum true width ≥5 ft		Minimum true width ≥5 ft	
	Short tons	Au (oz/tn)	Short tons	Au (oz/tn)	Short tons	Au (oz/tn)		
Cut-off 0.04 oz/tn	1,033,100	0.15						
Cut-off 0.07 oz/tn	720,239	0.19	366,600	0.20				
Cut-off 0.08 oz/tn					465,128	0.24		
Cut-off 0.10 oz/tn	524,940	0.25	282,600	0.27	418,140	0.25		
Cut-off 0.15 oz/tn	386,639	0.31	Minimum true width of 5 ft				Dawson Segment: 349,039 tn at 0.359 oz/tn Au	Dawson Segment: 263,000 tn at 0.460 oz/tn Au
Cut-off 0.15 oz/tn							Windy Gulch Segment: 33,600 tn at 0.405 oz/tn Au	Windy Gulch Segment: 33,000 tn at 0.408 oz/tn Au
Source:	Theye and Aiken, 1986		Theye, 1988		Theye, 1989		Mettler, 1991	Mettler, 1992

Notes:

1) A QP has not done sufficient work to classify any historical estimate as current mineral resources or mineral reserve.
2) Zephyr is not treating any historical estimate as current mineral resources or mineral reserves as defined in Sections 1.2 and 1.3 of NI 43-101; the historical estimates should not be relied upon.
3) Zephyr does not make any representation or warranties as to the accuracy of the historical resource or reserve estimates. The historical estimates of the above companies in Table 6-2 belong to categories other than the ones set out in Sections 1.2 and 1.3 of NI 43-101. Such categories cannot be compared to current mineral resources or mineral reserves as defined in Sections 1.2 and 1.3 of NI 43-101 as there has been no work to verify and classify such historical estimates.

6.3.1 US Borax

In 1985, US Borax calculated several historical "reserves" using polygonal outlines around drill hole mineralized intercepts on longitudinal sections. Gold assays were uncut, but different gold cut-offs and a tonnage factor of 12.0 (cubic feet per short ton) were used to estimate various grade and tonnage scenarios. Using a cut-off of 0.1 oz/tn Au, a "reserve" of 525,000 tn averaging 0.25 oz/tn Au (about 8.6 grams per tonne; g/t) was calculated. The mineralized zone was interpreted to be an irregular-shaped tabular body dipping about 65° to the south.

6.3.2 ACA Howe

From 1987 to 1988, ACA Howe (for Jascan) used polygonal outlines around drill hole intercepts on longitudinal sections. All individual gold assays greater than 1.0 oz/tn (which amounted to less than 5% of all assays) were cut to 1.0 oz/tn.

By 1988, using a gold cut-off of 0.10 oz/tn and a minimum true width of at least 4.99 ft, a "mineral inventory" of 465,126 tn at 0.24 oz/tn Au was calculated for the mineralized zones in the Dawson and Windy Gulch segments.

6.3.3 Uranerz USA Inc.

Uranerz used both standard polygonal methods and inverse-distance-squared routines to calculate the historical "reserves" on the separate mineralized zones in the Dawson and Windy Gulch segments.

In 1990 several methods were used to calculate the historical "reserves" in the Dawson Segment for an average of 349,038 tn at a grade of 0.359 oz/tn. The Windy Gulch historical "reserves" were calculated using the mean thickness and thickness-weighted averages of drill hole intercepts within polygonal outlines, totalling 33,600 tn of 0.405 oz/tn Au (Mettler, 1991).

In 1991 Uranerz revised its estimates by constraining the mineralized zones based on a study by MagmaChem (Swan and Keith, 1991). The Dawson Segment had a historical "gold resource" of 263,000 tn at 0.460 oz/tn Au within three separate zones with an average thickness of about 6.6 ft; a fourth zone was only intersected by two drill holes and was not included in Uranerz's calculations (Mettler, 1992).

The Windy Gulch Segment was reported to have three mineralized zones with the "best defined" zone having a historical "preliminary gold resource" of 33,000 tn at 0.408 oz/tn Au with an average true thickness of 8.07 ft. The block was defined by six surface diamond drill holes, three underground drill holes and several surface channel samples (Mettler, 1992). No description of estimation methods or assumptions were reported.

A historical "resource" calculation had been planned for Windy Point, but no documentation has been found for it.

6.4 Historical Metallurgical Studies

Several metallurgical studies were completed during the 1980s by or for US Borax.

In 1982 a preliminary examination was conducted on a 45.93 ft composite sample from drill hole GC14 with a "head grade" of about 3 ppm Au. Cyanidation gave a recovery of 84% to 86% (Ganderup and Woods, 1986).

An internal US Borax memo discussed an initial sulphide rougher flotation test, showing a 76.6% Au recovery. It produced a pyrite sulphide concentrate assaying 18 ppm Au (head assay at 3 ppm). Drill core sample rejects were used in the test (from drill holes GC18 and GC27) (Ganderup and Woods, 1986).

Preliminary metallurgical testing on Dawson drill core by the US Borax Research Centre (USBRC) was also done in 1982. A first bench test using a "standard leaching method," using two different strengths of sodium cyanide solution showed 84.1% and 85.6% Au recoveries. A second bench test used a standard flotation test with a "typical reagent regime" for gold, with 76.6% recovery. No obvious consistent arsenic occurred in the samples: a check by USBRC of 137 samples from three drill holes (GC11, GC13, and GC14) had 4 samples with arsenic content over 2 ppm (above detection limits) and a high of 15 ppm (Aiken et al., 1983).

During a 1985 metallurgical study commissioned by US Borax (Ganderup and Woods, 1986), composites were taken from holes GC12, GC18, GC27, and GC57A. Small samples of massive sulphide and biotite from the surface exposures of known mineralization were also taken. Testing determined gold recoveries of between 90% and 96%.

An appendix to the study examined preliminary equipment and capital expenditures. Ganderup and Woods (1986) assumed a 500 tn per day mill and a 90% Au recovery, using a standard crushing, grinding, and cyanide leach plant with gravity separation for coarse gold. They proposed using a "Merrill-Crowe" for gold recovery instead of the more expensive carbon/electrolytic plant.

Their rough engineering costings gave a high-end cost of the mill at about US$8 million (1985 dollars). They compared costs with the Canadian Bachelor Lake Project, which had a 500 tn per day mill and mining development costs totaling about CDN$9.25 million (calculated to US$8.1 million total, with estimated US$6 million for plant alone).

6.5 Historical Pre-feasibility Study and Review

Uranerz contracted AMSE (1991) to prepare a preliminary feasibility study of the mineralized zones in the Dawson segment. Note that the terms "pre-feasibility study" and "preliminary feasibility study" as used in AMSE's report are historical in nature and may not conform to current CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council (CIM 2014). Using the resource estimates and geological interpretation supplied by Uranerz, AMSE added 1 ft of waste rock at zero gold grade to both the hanging wall and footwall of the original 5 ft minimum width (for a 7 ft mining width). AMSE calculated a historical "diluted mineable reserve" of about 397,000 tn grading 0.308 ounces per short ton gold. No other description of estimation methods or assumptions was reported.

AMSE recommended an exploration and test mining program which included 3,500 ft of ramp (about 12 ft by 12 ft) and lateral development (5 ft by 7 ft), about 18,700 ft of definition drilling and test mining of about 2,000 tn of mineralized rock.

Their recommended mining system was conventional cut-and-fill where jacklegs and stopers would be used for drilling and slushers would be used to muck narrow stopes. In wider stopes, captive Cavo muckers would be used. An underground crusher with a conveyor system was proposed to bring crushed ore to surface.

After inspection of the drill core, AMSE assumed ground conditions to be fair, requiring routine rock bolting with occasional steel sets and shotcreting. Cemented fill would provide ground support, and no pillars or sill pillars were contemplated. Mine water inflow was assumed to be 500 gpm.

The mine design was for a 500 short-ton-per-day mill and an estimated mine life of three years at an annual production of 175,000 tn. Capital costs (1991 dollars) were estimated at US$17 million, which included an exploration program, feasibility study, mine and mill construction, and equipment costs. The total underground exploration and bulk mining costs were estimated at about US$4 million.

In 1991 Uranerz hired Dynatec Mining Corporation (Dynatec) to review AMSE's study and identify alternative conceptual designs, mining methods, and design and costing criteria that could save money (Kettner et al., 1991). Dynatec suggested the initial ramp be driven at near zero grade to intersect the mineralized zone near the 6200 level and optimize the ramp configuration down to the zone to reduce capital expenditures. The ramp would be used for truck haulage with primary crushing at the mill. Shrinkage and longhole stoping mining methods were suggested to eliminate the need for backfill (as in cut and fill stopes). Dynatec also noted that the short duration of the project and the relatively low tonnage of historical "proven reserves" made the project very sensitive to changes in either revenues or expenditures.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology, Metallogeny and Structural/Orogenic History

The Dawson Property is in the northern Wet Mountains, which is part of the southern Front Range of the Rocky Mountains (Figure 7.1). The broad regional setting of the Property is within the southern part of the middle Proterozoic age Yavapai Province, a tectonostratigraphic assemblage of volcanic, volcano-sedimentary that surround domains dominated by igneous and metamorphic crystalline rocks. In central Colorado in general and the northern Wet Mountains in particular, these rocks have yielded various radiometric ages that range from 1.8 to 1.7 billion years (Ga).

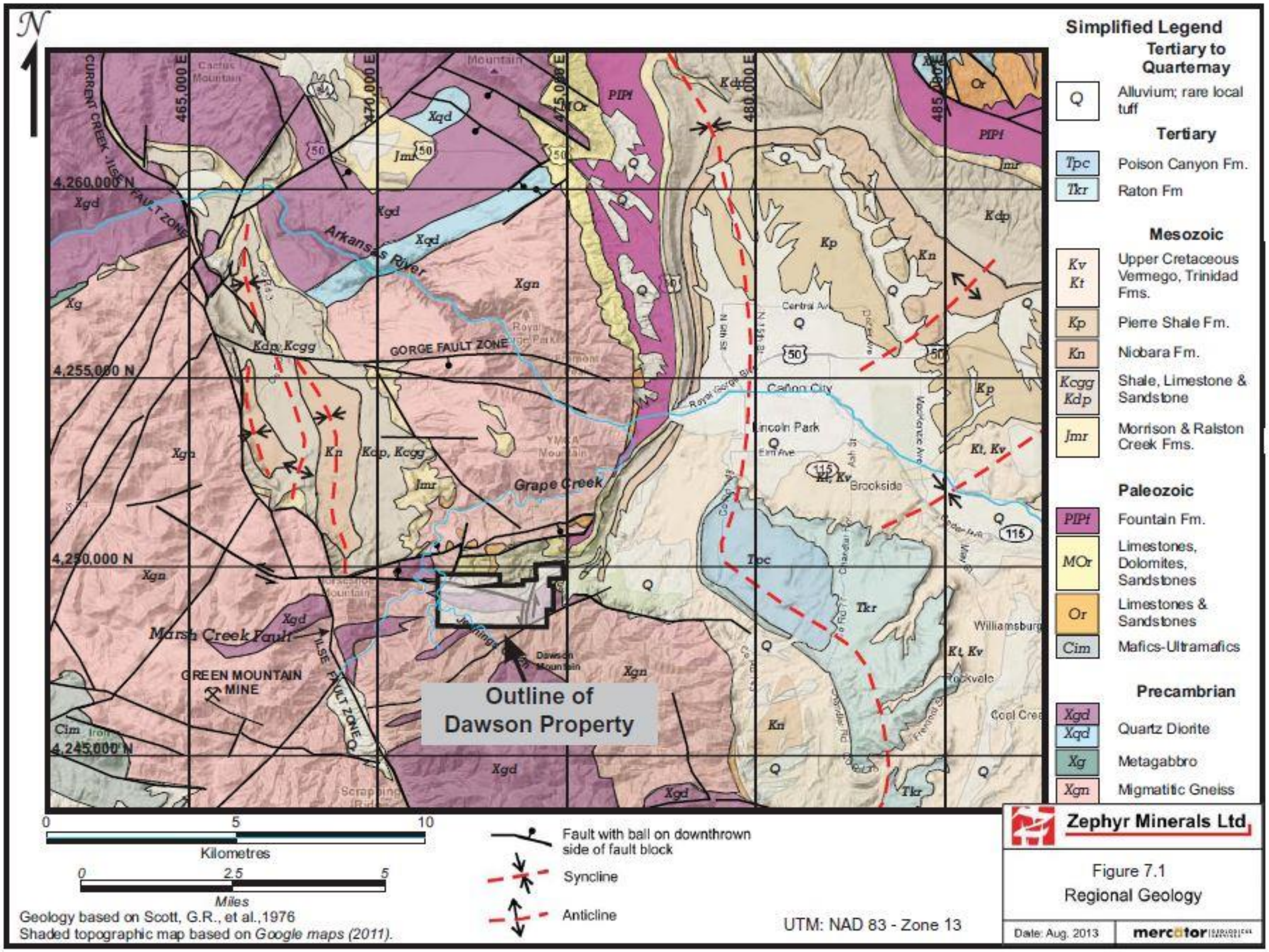


Figure 7.1: Regional Geology near the Dawson Property (modified from Taylor et al., 1975).

The northern Wet Mountains consist mainly of a gneissic crystalline complex composed mainly of aluminum-rich, peraluminous gneissic foliated plutons with an inferred minor component of metavolcanic rocks metasedimentary rocks. The gneiss complex is intruded by aluminum-poor, metaluminous quartz diorite and granodiorite plutons that have yielded a circa 1,720 million year (Ma) Rb-Sr isochron age and are correlated by Taylor et al. (1975) with the Boulder Creek plutonic suite (Figure 7.2.).

Based on the geology of Dawson and regional reconnaissance observations made in the northern Wet Mountains by Swan October–November 2016, the gneiss complex mapped by Taylor et al. (1975) is largely composed of gneissic plutons and pegmatites that have been injected into shear zones that have experienced ductile to brittle deformation. As such, the plutons are "hidden" in the gneiss geology and they may well be the main component of the gneiss complex. Mineralization is spatially related to various peraluminous plutonic phases and has been liberated from the intrusive sources by various depressurizations related to kinematic activity on the shear system referred to herein as the Dawson Shear system (Figure 7.2). The gneissic terrane also includes oceanic amphibolite-grade basaltic metavolcanics that have yielded U-Pb ages between 1,770 ±8 Ma and 1,745 ±16 Ma. These metavolcanic rocks have presumably been regionally intruded by the peraluminous intrusive suites.

The peraluminous gneissic plutonic suite is then intruded by a regional suite of metaluminous biotite/hornblende bearing plutons regionally referred to as the Boulder Creek type plutons. The Boulder Creek plutons are widespread throughout the Colorado Front Ranges and in the Boulder, Colorado area have been dated at about 1,720 Ma by both Rb-Sr and U-Pb (zircon) methods (see Peterman et al., 1968; Stern et al., 1971; Phair et al., 1971).

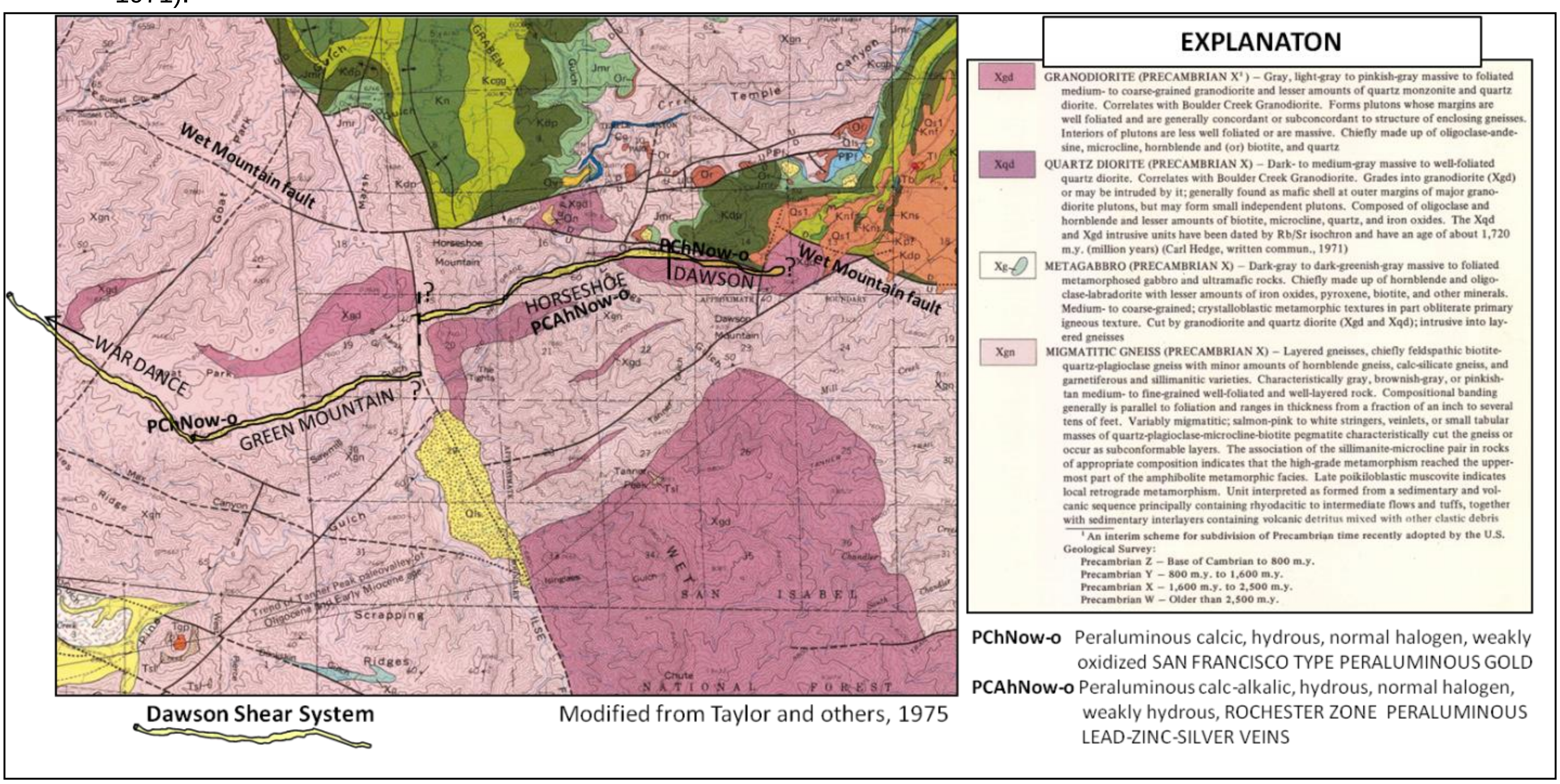


Figure 7.2: Quadrangle Geology at Dawson and Vicinity

The gold related magmato-hydrothermal depressurization is syn-kinematic with the widespread emplacement of the peraluminous intrusive suites into east-northeast- to northeast-striking tensional zones along a regional west-northwest-trending shear system referred to as the Trans-Colorado Zone (Keith et al., 2016). The Trans-Colorado Zone is expressed by prominent deflections of the central Colorado Rocky Mountains topographic grain and prominent deflections of foliation trends in the Colorado Proterozoic crystalline rocks (Figure 7.3). Based on kinematic analysis at Dawson (see below), offset of the Colorado Mineral Belt (which may have been born circa 1,750 to 1,720 Ma), and oroclinal drags of the Proterozoic foliation patterns the west-northwest- to east–west-trending elements of the Trans-Colorado Zone are inferred to have experienced initial left slip during the emplacement of the intrusion related peraluminous gold deposits between 1,745 and 1,720 Ma.

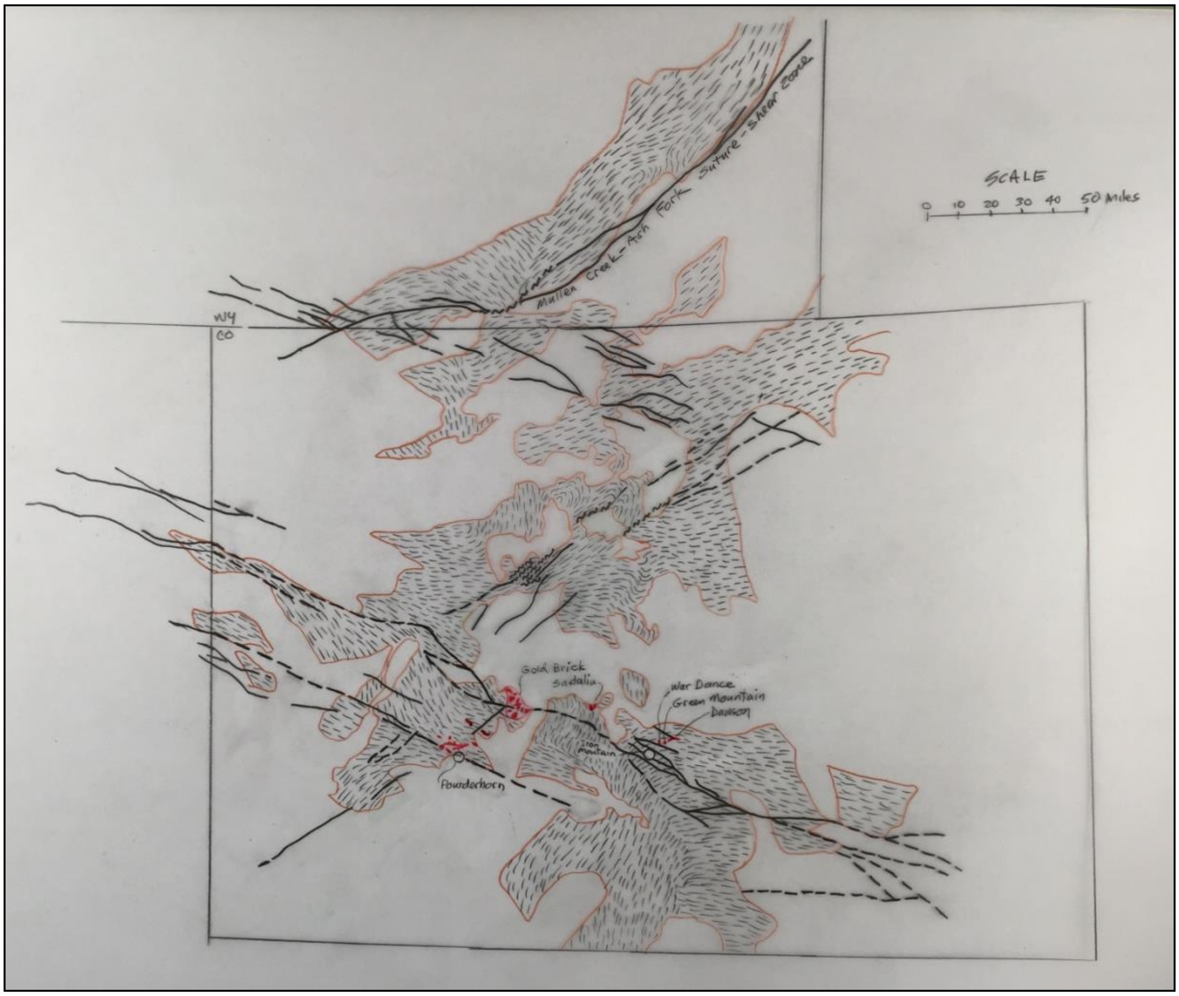


Figure 7.3: Map Showing Relationship between Regional Middle Proterozoic Foliation Trends and the Trans-Colorado Zone and the Mullen Creek-Ash Fork Shear Zone which is Widely Interpreted as a Collision Zone/suture between the Wyoming Craton to the North and the Middle Proterozoic Terranes to the South

Note: Foliation data from M. Swan, unpublished compilation.

At the Dawson Property, a major element of this shear system is referred to as the Dawson Shear Zone (Figure 7.3). The Dawson Shear Zone may also be related to an east–west structure referred to herein as the Wet Mountain fault, which forms the mountain front thrust fault between the Proterozoic crystalline basement that hosts Dawson gold deposits and the un-metamorphosed Phanerozoic sequence, manifested by the Cretaceous sedimentary sequence immediately north of Dawson (Figure 7.4). This fault also displays a left separation of various Phanerozoic sedimentary units. Hence, it is interpreted that the Trans-Colorado Zone has repeatedly experienced numerous episodes of accumulative left lateral movement (at least three) as well as reverse motions since its origin between 1,745 and 1,720 Ma.

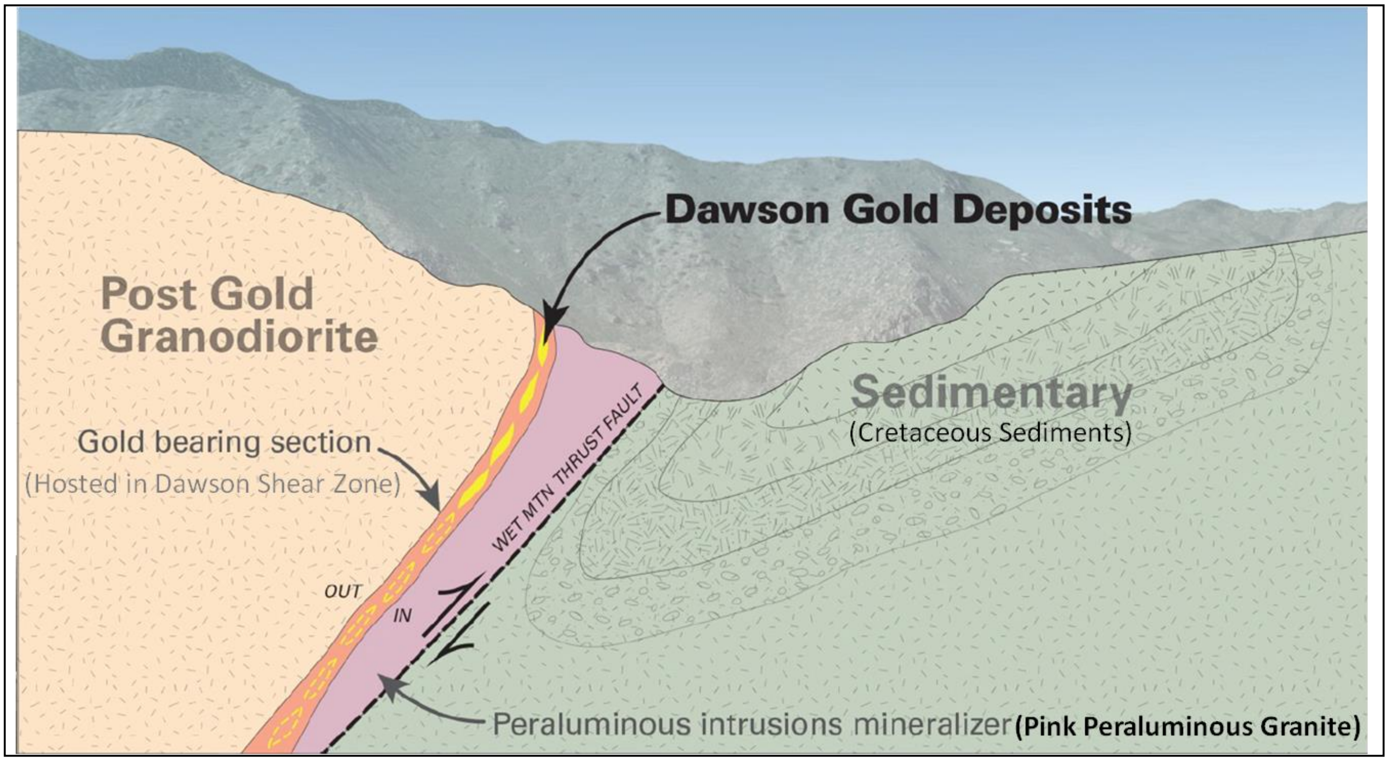


Figure 7.4: Schematic Cross-Section (looking west) showing the Relationship between the Regional Wet Mountain Fault Element of the Trans-Colorado Zone and the Dawson Shear Zone Which Hosts the Dawson Gold Deposits

The Dawson Shear Zone, while parallel to the Wet Mountain fault near the Dawson Property, veers southwestward to the west of the Windy Point Segment. The metallogenic character changes dramatically in this southwestward segment where it exhibits a strong lead-zinc-silver bias (see Table 7-1). This segment is referred to herein as the Horseshoe Segment (Figure 7.5). It is possible that the spatially associated peraluminous plutons in this segment also will show a different magma chemistry and are predicted to display a peraluminous calc-alkalic alkalinity compared to the peraluminous calcic affinity for plutons in the Dawson Segment to the east. In any case, sampling of Horseshoe Segment (mainly the Horseshoe and El Plomo prospects) indicates a strong lead-zinc-silver bias which is considered high risk for any gold-focused exploration program.

Table 7-1: Average Metal Contents of Samples Collected from Various Segments of the Dawson Shear Zone

Element	Au	Ag	Bi	Se	As	Sb	Cu	Pb	Zn	Cd	Mo	Be	Sn
Weight Unit	ppm	Ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Horseshoe-Grape Creek (ACNC) Averages (N=21)	0.096	5	4	6	2	2	762	3,076	7,477	18	4		
Horseshoe-Grape Creek (USGS) Averages (N=1-3)	0.200	17.5	23				8,367	5,690	16,772	325	15	1	30
War Dance Avg (ACNC) (N=12)	0.53	13.68	7.67	5.17	8.58	1.97	2,420.67	3,847.58	2,819.42	15.48	6.96		
Green Mountain (ACNC) Averages (N=14)	1.219	5	96.8	23	2	0.1	9,450	28	1,190	3	5		
Green Mountain (USGS) Averages (N=1-7)	2.378	11	142				22,429	1,733	20,571	25	8	5	40
Dawson Averages (N=21)	4.295	3.2	90	51	3	0.2	2,781	81	370	2	7		

Notes: ACNC = samples collected by ACNC reconnaissance program; USGS = samples collected by Raymond and Sheridan (1980).

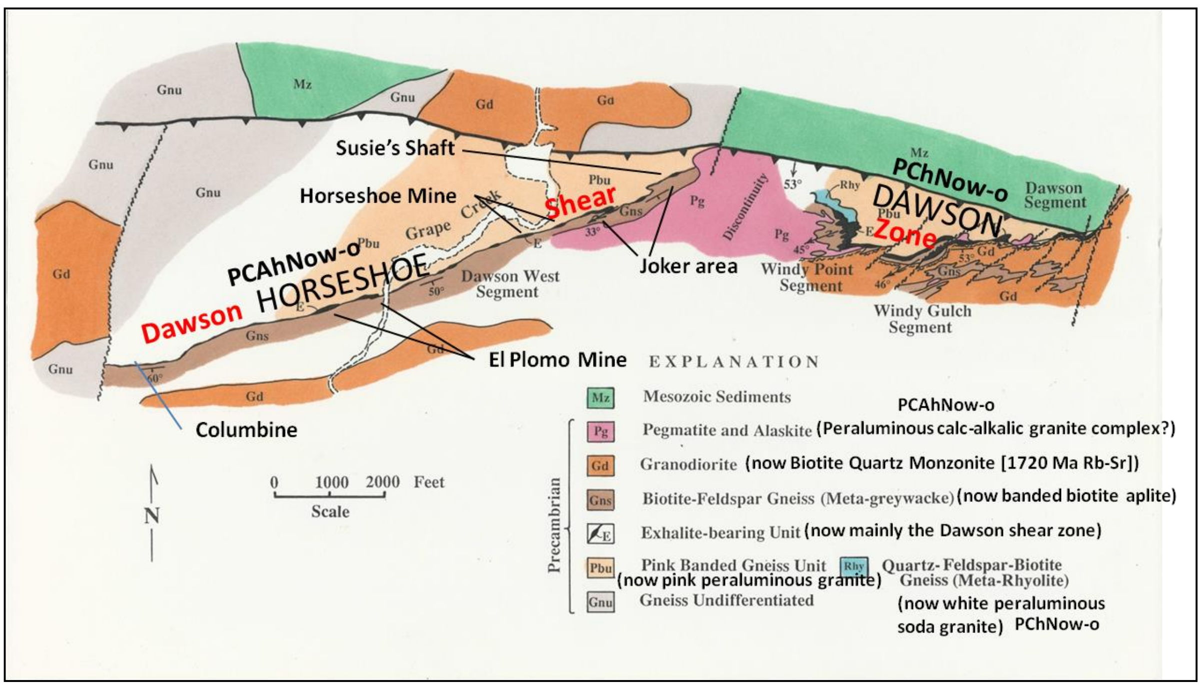


Figure 7.5: Generalized Geological Map of the Dawson Area Showing Metallogenic Segmentation and Possible Changes in Peraluminous Magma Chemistry

Note: Geology from Swan circa 1991. Note that the historical volcanogene interpretation is utilized in this map.

The lead-zinc-silver bias of mineralized samples continues along the Horseshoe Segment to the Ilse fault, a major northwest-striking, high-angle fault that bisects the entire Royal Gorge 15' Quadrangle and exhibits approximately 0.6 to 1.2 mi of apparent right lateral offset of various Proterozoic crystalline units. The Dawson Shear Zone is inferred to be offset by this fault and its apparent offset can be traced southwestward.

Metamorphic grades that accompanied the emplacement of the gold-copper and lead-zinc-silver related peraluminous magmatism in the central Colorado region range from upper greenschist to middle amphibolite facies at about 3.0 Kb (10 to 12 km). This inference is based on the occurrence of magmato-hydrothermal sillimanite in quartzo-feldspathic peraluminous rock at Dawson and garnet-biotite hydrothermalite geothermometry at other gold prospects west of Dawson (Earley and Stout, 1991), which is similar to garnet-biotite hydrothermalite greisens at Dawson (Figure 7.6).

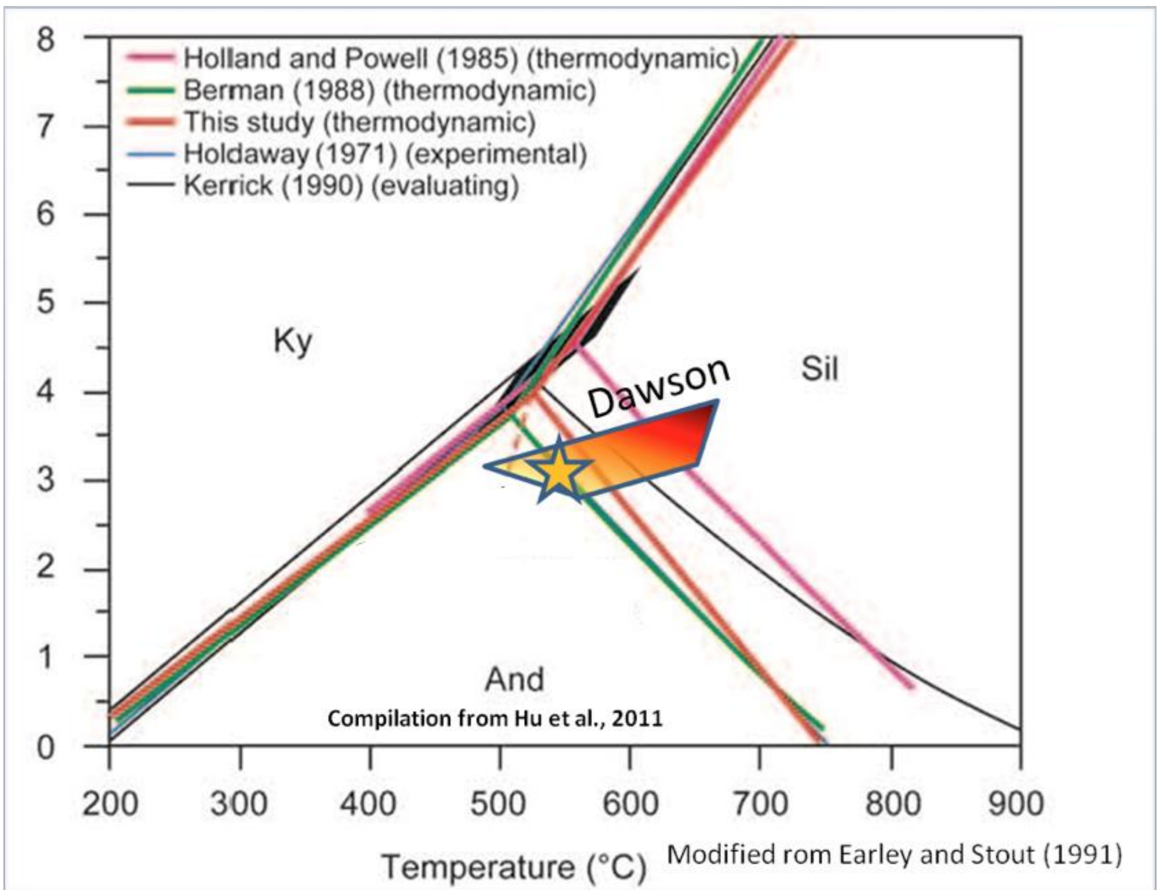


Figure 7.6: Suggested Pressure-Temperature Track for the Greater Dawson Peraluminous Gold System

The complex package of gneissoid gold and oceanic metalvolcanic rocks described above collided with the Archean Wyoming Province to the north along the Mullen Creek-Ash Fork suture between 1.745 and 1.73 Ga, forming a new continental margin (Figure 7.7). The wall rock geology to the peraluminous intrusions is inferred to be underthrust oceanic sediments and basaltic oceanic crust mixed with granitic sedimentary materials derived from the Wyoming Craton to the north. The oceanic crustal component may have hosted volcanogene Besshi

Type copper-zinc deposits like the Sedalia copper-zinc deposit near Salida, about 37 mi to the northwest of Dawson (Swan, 1989). Hydrous melting of these greywacke-dominated metasedimentary materials under amphibolite conditions in the flat subduction zone is believed to be mechanism for production of the gold-related calcic peraluminous magmatism and silver-lead-zinc calc-alkalic peraluminous magmatism. The peraluminous magmas utilized high angle transcurrent shear zones such as the Dawson shear system in the upper plate of the flat subduction zone. Decompression of the metalliferous hydrothermal component from these intrusions led to a rapid freezing of the source magma at mid-crustal levels in the mesozone. Hence, little or no volcanism accompanied the formation of the Dawson gold deposits and their associated plutons.

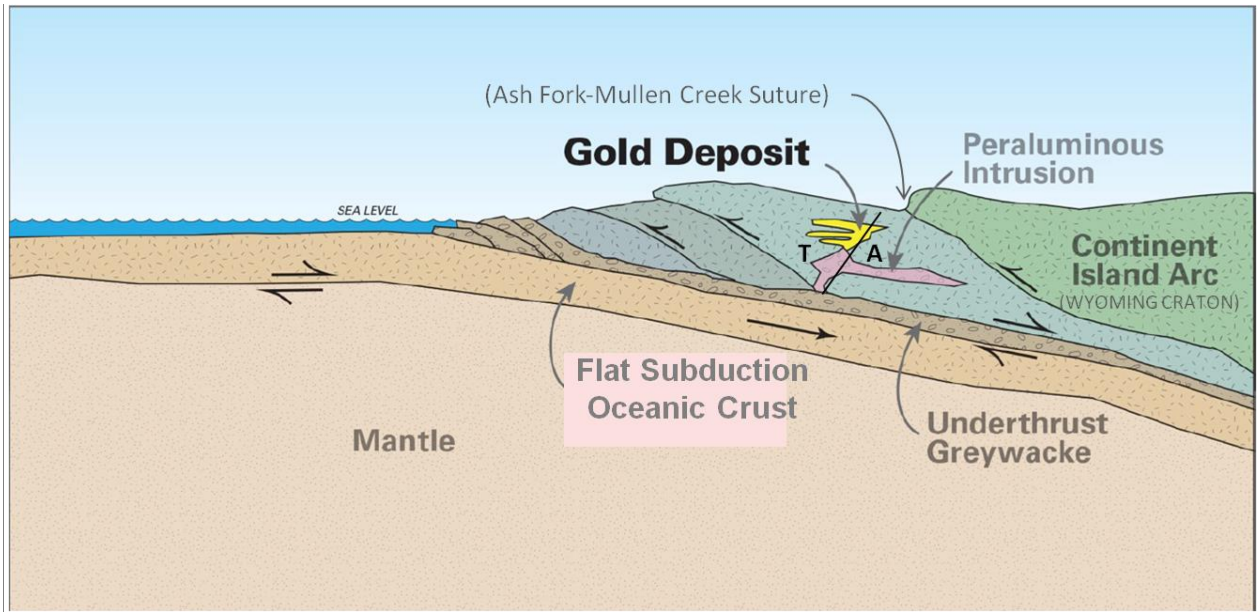


Figure 7.7: Geotectonic Setting of the Dawson Peraluminous Gold Deposit

In their broad geotectonic sense, the peraluminous intrusive suites and their accompanying metallogeny are viewed as direct anatectic products of the collision between an oceanic plate containing early-mid Proterozoic oceanic assemblages and sedimentary materials derived from the Archean aged Wyoming Craton that were deposited in a low relief trench on its southern margin. These materials were then hydrously melted under upper greenschist to lower amphibolite conditions above a flatly subducting dehydrating oceanic slab. This tectonic setting is very similar to the one that recurred during the culminate Laramide orogeny circa 70 to 40 Ma, whereby a flatly subducting Farallon oceanic slab produced gold-related calcic peraluminous magmatism beneath southwest North America. During this event, peraluminous gold deposits like Mesquite in southeast California and La Herradura in northwest Sonora were formed by melting of sodic meta-greywackes (Orocopia Schist and analogs) which became accreted to the base of a tectonically thinned North American Proterozoic crust (Keith and Swan, 1985; Smith and Graubard, 1987; Swan and Keith, 1987).

The collision between the Wyoming craton and the flatly subducting oceanic terrane to the south was followed by steeper subduction marked by the emplacement of the extensive younger suite of mildly alkaline hydrous metaluminous intrusions assigned to the Boulder Creek intrusive suite emplaced circa 1,720 to 1,710 Ma. The magma chemistry of these plutons is consistent with their origin in an underlying subduction zone. The final consolidation of the Proterozoic craton in central Colorado was marked by the emplacement of an extensive suite of 1.45 to 1.35 Ga plutions of which the Silver Plume granitoids in the central Front Range is probably the most notorious. As envisioned by Swan and Keith (1986), the Silver Plume granite is a late peraluminous member of the 1400 granite suite which, in their view, is orogenic in nature and related to a flattening slab along a subduction zone along the southeast margin of proto-Laurentia/North America. In the Dawson area, no plutonic member of the 1400 granitoid suite is apparent. However, elements of the Trans-Colorado zone may have undergone kinematic activity in the Wet Mountains (Jones et al., 2010).

At 1.08 Ga perhaps the most well-known batholith of all was intruded: the Pikes Peak peralkaline reibeckite granite and its distinctive pegmatites centred on Pikes Peak west of Colorado Springs (Noblett et al., 1998). The Pikes Peak granite is visible in the Cripple Creek area some 50 mi to the north of Dawson. Pikes Peak pegmatite phases were not emplaced in the Dawson area as is supposed in the historical literature on Dawson. Petrochemistry generated for this study shows that the extensive iron-poor pegmatite lithologies present at Dawson are not similar to the Pikes Peak suite, which shows very strong iron enrichment due to its peralkaline affinity. The pegmatites and aplogranites that are widespread at Dawson are either related to the gold-related peraluminous intrusive suite or are high K pegmatites related to the metaluminous biotite quartz monzonite Boulder Creek type granitoids in the footwall of the north-dipping Dawson Shear Zone.

In the Wet Mountains, southwest of Dawson in middle Cambrian time between 515 and 520 Ma prior to the deposition of the Paleozoic cratonic sedimentary sequence, a distinctive suite of hydrous nepheline alkalic metaluminous igneous rocks was emplaced along the west-northwest Trans-Colorado lineament and related northeast structures. Some of the syenitic phases are associated with thorium mineralization (Christman et al., 1959), and a funnel-shaped layered gabbro complex at Iron Mountain contains immiscible segregations of iron mineralization at the Iron Mountain Mine (Shaw et al., 2001) about 1.2 mi south-southwest of the Green Mountain Mine. The Cambrian alkaline complexes are broadly related to the Powderhorn titanium-rich carbonatite complex which was also emplaced in Cambrian time circa between 570 Ma (main carbonatite pyroxenite complex) and 510 Ma (large alkaline gabbro dykes) along the west-northwest striking Cimarron fault on the Powderhorn linear. The pyroxenite phase of the Powderhorn complex is best known for its titanium resource (as perovskite with minor ilmenite and titanite), which as of 2009 was established by Teck Resources in 1994 as 46 million tons of 13.2% TiO_2 with a possible additional 1.8 billion tons of resource at 10.9% TiO2 as reported by Van Gosen (2009).

During the Paleozoic within Fremont County, dolomite, limestone, and lesser quartzite were deposited with later clastics of shales, siltstones, and sandstones atop the Precambrian and Cambrian crystalline basement. No Cambrian age sedimentary rocks are present in the Dawson area (Royal Gorge Quadrangle), suggesting that much of the area was high in Cambrian time. In the middle Paleozoic, as much as 475.7 ft of cratonic sedimentary rock was deposited as Manitou Formation in the lower Ordovician (carbonate and cherty dolomite), Harding Sandstone in the middle Ordovician, and Fremont Dolomite in the middle and upper Ordovician, and Williams Canyon Limestone (dolomite and minor contorted shale) in the Mississippian (Taylor et al., 1975).

The Ancestral Rocky Mountain orogeny in middle Pennsylvanian time Paleozoic induced significant uplift in the Wet Mountain region. The principal sedimentational result of the Ancestral Rocky uplifts was the Fountain Formation, a mostly reddish brown arkosic unit with local interbeds of reddish shale and conglomerates with cobbles of Cambrian (?) syenite derived from the Wet Mountains to the south. The Royal Gorge 15' quadrangle map shows a distinct pinch out of the Fountain Formation against an east–west striking fault referred to herein as Wet Mountain fault immediately north of the Dawson Property (see Figure 7.2). The pinch out relationship suggests the Wet Mountain fault experienced a down to the north movement in middle Pennsylvanian time with Fountain Formation deposition in the downthrown block.

In the later Permian, the Dawson region underwent continuous erosion. By the end of the Permian, the land was a relative peneplain.

During the Mesozoic in upper Jurassic time, thin deposits of varicolored terrestrial dinosaur bone bearing siltstones, claystones, and conglomerates (Morrison and Ralston Creek Formations) were deposited north of the Dawson Property area. These units overlie the Precambrian gneissic basement west of Cañon City but rest on the entire Phanerozoic section (including Fountain Formation) along the Front Range in the Cañon City area. The pinch out relationship suggests some low residual ancestral Rocky topography and structural relief was present west of Cañon City and north of Dawson.

In the middle Cretaceous, the entire Colorado region became a broad down bowed synclinal foredeep to the Sevier Thrust Belt in eastern Utah to the west. Over 49,212 ft of sandstones (mainly the basal Dakota Sandstone), shales (mainly the Pierre Shale), and chalks (mainly the Niobrara Formation) accumulated in the trough during a transgressive cycle which culminated at about 89 Ma. After 89 Ma the Cretaceous seaway was pushed northeasterly off the craton by pulsed flattening subduction which marked the initial Laramide orogeny. The pulsed flattening was tracked in detailed by an oscillatory northward regression of the Cretaceous seaway.

At about 65 Ma, the main stage of the northeasterly migrating Laramide orogenic front reached the Colorado Front Range region. At this time, most of the Front Range structural relief was achieved in a geologically short 2 to 4 Ma time interval. A series of north–south to north-northwest–south-southeast-facing thrust fault fold pairs were created and comprise east-facing uplifts along the entire length of the southern Wyoming-Colorado-New Mexico Front Ranges. These uplifts were paired with basins like the Denver Basin to the north and the Raton Basin to the south.

In the Dawson–Cañon City area, the large 5.6 mi long eastward overturned fold pair of the mid to upper Cretaceous marine sedimentary sequence in the so-called Webster Park graben was formed to the northwest of Dawson. At this time, the Wet Mountain fault probably underwent minor left slip during the east–west compression that formed the main Front Range escarpment on the east side of the Wet Mountains and the Front Range.

However, the main Laramide slip on the Wet Mountain fault at Dawson may have occurred during the late Laramide when the Wet Mountain fault experienced left oblique thrust slip during a more northeasterly compression that was probably early Eocene in age circa 55 to 50 Ma (shown schematically in Figure 7.5). An important aspect of this late Laramide slip on the Wet Mountain fault was that it cut off the bottom of the auriferous Dawson Shear Zone and displaced it an unknown distance to the south and east. Based on stratigraphic throw, the offset was at least 6,561 ft, which makes the offset gold deposit an academic footwall gold target. However, much of the hanging wall down dip portions of all the gold segments (see below) at Dawson remains to be explored.

The largest Laramide structure of all was created during the late Laramide orogeny when the entire region was epeirocally uplifted several thousand yards by the flat subduction event that affected the entire Front Range region in Colorado (as well as the entire southwestern North America continent to the Mississippi River). The result of this epeirogeny in the Colorado Rockies is known as the Eocene surface as described by Epis and Chapin (1975). An important lesser feature coincident with the development of the Eocene surface was a series of northerly trending basin assigned to the Echo Park type basin of Chapin and Cather (1981). These basins were developed adjacent to north-northwest faults such as the Ilse fault, which were undergoing right slip in response to north-

northeast–south-southwest shortening of late Laramide age. It was then that the Dawson Shear Zone may have been offset to the south for an estimated 0.6 mi. Right slip on the Ilse fault may have been coordinated with oblique right slip on the Wet Mountain fault described above.

During and after the Laramide orogeny, a major plate tectonic reorganization event triggered by the assembly of the Indian subcontinent to Eurasia caused the subducting Farallon oceanic slab to gravitationally collapse (Coney and Reynolds, 1977; Keith, 1978; Keith and Wilt, 1986). Metaluminous volcanism and associated metallogeny (e.g. the metaluminous nepheline alkaline Cripple Creek volcano and its associated world class telluride gold deposit about 45 air miles to the north of Dawson) reappeared in force and lasted in the central and southwest Colorado region from about 38 to 8 Ma. Most of the volcanism and associated volcaniclastic sedimentation accumulated on the Eocene surface or in the upper portions of the aforementioned Echo Park type basins.

In between the volcanics and after most of the subduction-related isotopically spiked (with crustal volatile component) magmatism, a suite of conglomeratic fluvial sediments accumulated as the magmatic arc left Colorado and travelled westward towards the California coast (Coney and Reynolds, 1977; Keith, 1978; Dickinson and Snyder, 1980; Keith and Wilt, 1986). In the Cañon City–Dawson area, members of this largely post arc, and pre-rift (as in the Rio Grande Rift) sedimentary assemblage comprise the lower member of the Santa Fe Formation. The rock is composed of indurated conglomerate with pebble to boulder sized rounded clasts that were deposited fluvially in east-northeast aligned paleo-channels cut on the former Eocene erosion surface. Some these presumably 15 to 22 Ma early Miocene age fluvial channels occur at high elevations in the Wet Mountains south of Dawson (e.g. the Tanner Peak paleovalley shown on the Taylor et al., 1975 Royal Gorge 15' Quadrangle map).

The last major event to affect the region was the late Cenozoic trans-tensionally driven rifting that replaced subduction tectonism in the southwest Colorado region after about 8 Ma. The late Cenozoic trans-tension can be viewed as a soft transform tectonism that replaced subduction tectonics as first presented in the now classic paper by Tanya Atwater in 1970, which is now considered “settled” science. The main tectonic element of this no-slab expanding San Andreas type margin (in the sense of Dickinson and Snyder, 1980, and Keith and Dickinson, 1979) in the southern Colorado region was the northern arm of the Rio Grande Rift system and the Arkansas River graben to the west of Cañon City. Sedimentary rocks on the edge of the Cañon City–Florence Basin were apparently tilted westward toward the basin centre (Keller et al., 2000).

The rifting created dry basaltic volcanism with minor volumes of rhyolites. These magmas contain an uncontaminated mantle isotopic signature such as $^{87}Sr/^{86}Sr$ initial ratios largely between 0.703 and 0.7045. Well below the subduction related magmatism which yields $^{87}Sr/^{86}Sr$ initial ratios largely above 0.7055 and mainly between 0.7065 and 0.709. This isotopic shift to low strontium initial ratios largely appears in basaltic rocks of the region after and younger than about 8 Ma. In the Cañon City–Dawson area, the rift related volcanism is manifested by basalts of presumed Pliocene age (2 to 5 Ma) that occupy two buttes in the Cañon City–Florence Basin 1.55 mi south of Cañon City.

The late Cenozoic extension produced a dramatic lowering of the regional stream base level. Basically, the former Eocene surface base level collapsed and, like all other soft transform trans-tensional geomorphology, the stream systems reintegrated to internally drained basins or river systems (e.g. the Rio Grande) that ultimately were integrated to sea level. In the Cañon City region, the main drainage net that ultimately integrated to sea level via the Mississippi River was the Arkansas River. This integration included the Royal Gorge giant “gully cut” that is popular with local tourists. The topography of Royal Gorge suggests that it was (and is) a westward-migrating head-cut in response to a base level drop in the Cañon City–Florence Basin sometime in the last 2 Ma.

7.2 Property Geology

Rocks at the Dawson Project have been mapped and grouped into three main units: footwall pink hematitic gneisses; a fault hosted gold bearing biotite-garnet aplite with associated auriferous hydrothermalite; and a hangingwall foliated granodiorite and metasedimentary (?) rocks (Figure 7.8) A variety of thin, discontinuous pods and lenses of stratabound and cross-cutting mafic dykes have also been mapped; they are typically dark green to black, medium- to very coarse-grained amphibolites and metagabbros that are massive to weakly foliated. Minor late-stage (Tertiary?) dark green, massive and aphanitic to porphyritic dykes have been identified in outcrop and in drill core.

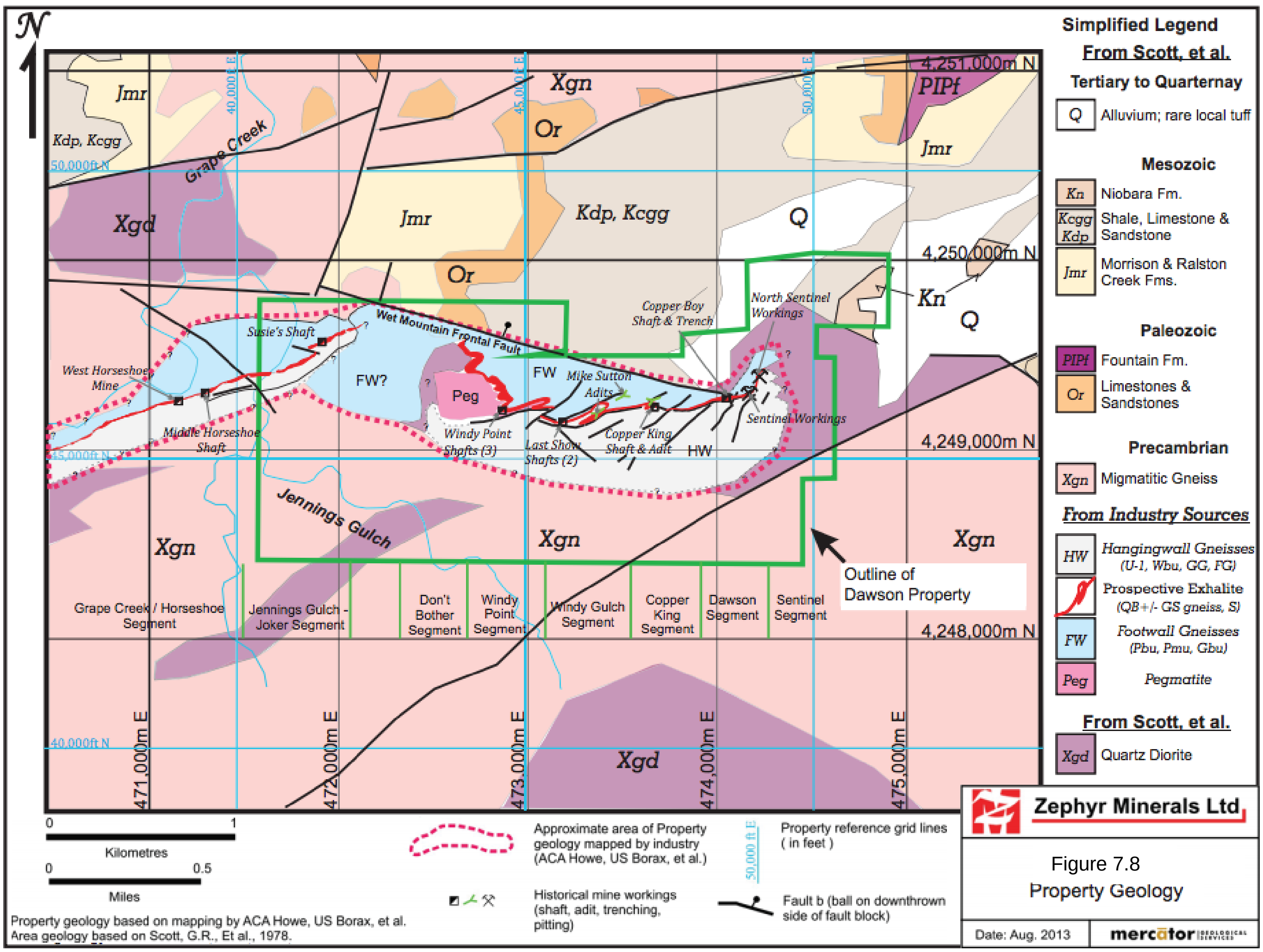


Figure 7.8: Property Geology

7.2.1 Footwall Rocks

The footwall rocks consist mainly of distinctively pink to grey, fine- to coarse-grained quartz-biotite-plagioclase feldspar gneiss with subordinate magnetite and local garnet in the upper portions. The pink banded unit (Pbu) is moderately to strongly foliated with characteristic alternating bands of pink feldspar gneiss and dark grey, quartz-feldspar-biotite gneiss. The more massive unit (pink massive unit; Pmu) is granoblastic to porphyritic; it cross cuts and intrudes the Pbu with evidence of partial melting. The distinctive pink colour is due to hematitization of the feldspars.

This stratigraphy has been extensively intruded, metasomatized and partially melted by very hydrous Pikes Peak quartz-potassium feldspar granite and pegmatite (Swan, 1991).

Potassium content decreases and garnets become more abundant west of Windy Gulch (grey banded unit, or Gbu); the unit exhibits finer grain sizes, better development of gneissic banding and more compositional variation.

7.2.2 Mineralized Horizon

The auriferous horizon on the Property is made up of interleaved mineral assemblages with numerous discontinuous sulphide lenses that are laterally continuous along strike for about a mile, but which pinch and swell along strike and range from approximately about an inch to 49 ft in true thickness. Contacts with the hanging wall and footwall gneisses have been mapped as both gradational and sheared. In more folded parts of the Property (e.g. Windy Point), the mineralized horizon comprises southwest- plunging grade shoots. The origin of the mineralized plunging grade shoots has been variously ascribed to the D1, D2, or D3 period of deformation, with migration of gold mineralization into F1, F2, or F3 fold hinges.

The stratigraphically lower part of the mineralized zone is a quartz-biotite gneissic aplite with variable amounts of garnet, sillimanite, magnetite, other silicates, and minor sulphides. It is light grey to black in colour and is typically fine- to coarse-grained, poorly to strongly foliated and gneissic to schistose in texture. Banding may be present, defined by alternating quartz-rich and biotite-rich layers. Garnet occurs as fine- to coarse-grained, pinkish-red to red-brown porphyroblasts. Plagioclase feldspar occurs in the lowest portions of the zone, increasing in abundance with depth. The upper portions contain subordinate gahnite, cordierite and 1 to 5 modal percent disseminated pyrite and chalcopyrite. Native gold occurring in this unit is typically associated with quartz, biotite, sillimanite, and coarser-grained garnet.

Stratigraphically above the biotite aplite is the sulphide portion of the horizon. Sulphides make up from 1 to 50 modal percent, consisting mainly of pyrite, with lesser chalcopyrite, pyrrhotite, sphalerite, and galena. They can form very fine-grained disseminations, clots, and discontinuous hairline veinlets parallel to foliation. Gangue mineralogy is quite variable and is made up of amphibole (mainly anthophyllite), biotite, chlorite, quartz, and feldspar with lesser gahnite, magnetite, cordierite, and sillimanite. Gold in this unit is typically associated with the amphibole-rich zones. Textures vary from foliated to schistose to fragmental, with amphiboles locally forming radiating clusters.

During D2 deformation, major shearing along the hanging wall boundary resulted in significant strike-slip motion, which formed the pinch-and-swell structures in the mineralized zones. Within the less folded areas of the Property (e.g. Dawson Segment), mineralization has essentially continuous distribution with some redistribution of grade by F2 folding and D4 faulting.

Within the more folded areas of the Property (e.g. Windy Point Segment), plunging grade shoots of gold mineralization were interpreted to have developed by pressure solution of quartz and migration of gold into lower-pressure synclinal F2 fold hinges which plunge to the southwest (e.g. Morfett, 1983). The observed repetition of mineralized horizons is thought by Morfett (1983) to be due to folding and not a primary stratigraphic feature. Other workers described the mineralized plunging grade shoots as belonging to two parallel zones, plunging 55° to 60° to the southwest, averaging 4.9 to 12.1 ft thick, 200 ft long, and with a down-plunge extension of at least 498.7 ft, with a periodicity of about 249 to 299 ft.

7.2.3 Hanging Wall Rocks

The mineralized horizon is overlain by metasedimentary (?) and intrusive rocks which average 151 ft in thickness, followed by foliated granodiorite.

Lithocodes Wbu (Well-Banded Unit), GG (Granular Gneiss), Calc-silicate Gneiss

Hanging wall rocks in direct contact with the mineralized horizon consist of a thin sliver of metasedimentary (?) rocks comprising granular gneiss (GG) and calc-silicate gneisses interlayered with or overlain by a thin well-banded unit (Wbu) of quartzo-feldspathic gneiss. They are grey to pale green, fine to medium grained, and weakly to strongly foliated. GG consists of biotite, feldspar, quartz, and magnetite. The calc-silicate gneiss consists of calcite, epidote, garnet, diopside, and idocrase with minor local garnet and actinolite.

The Wbu is believed to be the sheared equivalent of the stratigraphically higher U-1 meta-intrusives. It is medium to coarse grained and strongly banded (bands 0.4 to 2 inches thick) with local augened zones. Finely laminated biotite-magnetite is interbanded with coarse-grained potassium feldspar-quartz zones, giving the unit a distinctive black and orange striped appearance.

The stratigraphically higher Unit-1 is a foliated granodiorite (dated at 1,760 Ma; Swan, 1991) that locally contains abundant xenoliths of gneissic rocks (mapped as U-1Bx) or is intercalated with bands of well-banded felsic metasedimentary gneiss (mapped as U-1+seds). It is typically grey, coarse grained and equigranular, and composed of microcline feldspar, plagioclase, biotite, quartz and magnetite. Where mapped as U-1Bx, it contains fragments and xenoliths of Wbu. Where mapped as U-1+seds, it contains fragments and/or through-going intervals of the stratigraphically lower granular and banded gneisses. The U-1 terminology described here is a historical descriptor for a foliated felsic intrusive in the hangingwall of the Dawson Shear Zone and is distinct from the Unit 1 hydrothermalite mapped by Keith et al. (2016).

7.3 Structure

The Property, as part of the Yavapai Province, experienced multiple periods of deformation resulting in isoclinal recumbent folds overprinted by sub-vertical foliation and later local fabrics related to strike-slip zones (Noblett et al., 1998).

The area underwent at least four periods of deformation with associated multiple intrusions and upper amphibolite metamorphism. The Property sits on the south limb of a regional F2 anticline. The last period of deformation placed parts of the Dawson package of Proterozoic rocks in reverse fault contact with folded, but non-metamorphosed, Mesozoic sedimentary rocks to the north.

Imprints of the following four main periods of deformation have been identified on the Property:

1) The first period of deformation occurred between 1,700 and 1,775 Ma and was the most intense. Tight, typically recumbent, isoclinal folds (F1) were formed that are difficult to recognize due to the strong axial plane shearing (S1) that is common on the Property.

2) The second period of deformation (D2) produced upper-amphibolite facies metamorphic conditions. During or near the end of this period, U-1 granite to granodiorite intruded the region, with associated intrusive breccias. U-1 is thought to be approximately equivalent to the 1.7 Ga Boulder Creek Granite.

The D2 deformation produced a west-plunging regional F2 anticline as well as upright, tight to isoclinal F2 folds, a weak axial plane cleavage (S2), a fracture cleavage, and a weak lineation (L2) sub-parallel to the F2 fold axis and some jointing. Property mineralization occurs on the south limb of the regional anticline; the northern limb was partially removed by D4 thrust faulting. A postulated F2 fold hinge occurs in the Windy Point Segment, where the mineralized horizon is intensely folded.

Where mapped, the folds consistently plunge moderately (40° to 50°) south to southwest. Foliation strikes northeast–southwest and dips to southeast (parallel to the axial planes of folds).

The thickness of the mineralized stratigraphy is strongly controlled by F2 folding and later parallel faulting along major lithologic contacts (producing localized biotite schists). Base and precious metal mineralization appears to be concentrated in F2 fold hinges and is structurally attenuated on fold limbs.

3) The third period of deformation may have been associated with the 1.4 Ga Silver Plume plutonism and uplift, accompanied by significant thermal metamorphism and metasomatism (as shown by pegmatite bodies, local epidote-amphibole-calcite-garnet assemblages and Property-wide recrystallization textures). The long axes of the larger pegmatites and calc-silicate rocks are often parallel to F3 fold axes.

The D3 deformation produced upright, F3 folds with sub-vertical axial planes that strike 200° to 240° and plunge moderately to steeply (37° to 76°) southwest and local C3 crenulation cleavage. The folds are poorly represented on surface and appear to have little control on gold distribution.

4) A period of erosion, with deposition of Mesozoic sedimentary rocks unconformably on Proterozoic rocks, was followed by a period of uplift and brittle deformation (Laramide orogeny). It was during this period of uplift that Proterozoic rocks were thrust to the northeast over the younger Mesozoic sedimentary rocks along the Front Range Reverse Fault (also known as the Wet Mountain Frontal Fault). The fault strikes about N75°W and dips 55° to 70° to the south-southwest; drill hole intersections show its dip to shallow to the southwest.

In several historical drill holes, the Mesozoic and Precambrian lithologies interfinger and mineralized zones have been truncated by this thrust fault. Several northeast-trending tear faults related to the reverse fault locally offset the exhalative zone.

West of the Property, the western end of the mineralized trend is cut off by the north-trending Marsh Gulch Fault (part of the Ilse Fault System); the trend has not been traced farther west by previous industry workers.

7.4 Mineralization

Gold and base metal mineralization occur within an east-northeast trending, south-southwest-dipping fault in amphibolite-grade, Proterozoic-age metamorphic rocks. Strike-length of the entire mapped horizon is about 3.4 mi, of which about the eastern 1.6 mi, which is gold prospective, occurs within the Property.

East of the Windy Point Segment, mineralization is dominated by iron and copper sulphides with minor gold. The mineralized zone is apparently terminated on the east by the Front Range Reverse Thrust Fault, which juxtaposes Proterozoic rocks over Mesozoic rocks; it is terminated on the west by the Marsh Gulch Fault. The mineralized horizon is quite recognizable on the surface as a gossan, with limonite and hematite-stained siliceous rock from about 2 inches to about 10 ft wide and local malachite-azurite fracture-fill.

The mineralized host rock is predominantly a quartz-biotite gneissic aplite with variable amounts of iron-rich garnet and other silicates with local zones enriched in sulphides; it lies stratigraphically above the quartz-biotite-feldspar gneiss (Pbu/Pmu). The horizon has an average thickness of about 9.1 m, with gold occurring predominantly at several stratigraphic positions below the sulphide-bearing zones, but also at the base of the sulphide unit itself. The mineralized parts of the horizon are relatively discrete and range from approximately an inch to 49 ft in true thickness, with a sharp drop-off in metal values away from the contacts (Wilson, 1982).

The sulphide-rich zones are typically 10 to 50 modal percent base-metal sulphides in a fragmental chlorite-biotite matrix with local quartz- or anthophyllite-rich zones. Typical gangue mineralogy includes quartz, biotite, phlogopite, garnet, magnetite, amphiboles, sillimanite, cordierite, anthophyllite, gahnite, staurolite, sericite, talc, hematite, limonite, and calcite. Sulphides include pyrite and chalcopyrite with lesser amounts of pyrrhotite, sphalerite, and galena plus rare gold, molybdenite, and bismuth sulphosalts. Later-stage talc and serpentine fill fractures in both the sulphide and auriferous zones.

Historical mineralogical and metallurgical studies of core by US Borax found that native gold occurs as flakes up to 1.4 mm in size. A hole wedge-drilled from GC40 by Uranerz in 1990 (GC40-W2) intersected visible gold particles up to 3 mm in size. Uranerz also had 26 core samples from the Dawson Segment mineralogically analyzed and found gold flakes up to 0.24 mm in size. The gold is typically associated with sillimanite, sericite, or biotite; it inhabits cracks in quartz and garnet, and along grain boundaries, between quartz and sillimanite, sericite, or biotite. Gold can be associated with carbonate and/or siliceous veinlets and rarely occurs as inclusions in pyrite or chalcopyrite (Mettler, 1991). Probing of selected polished sections from drill hole GC05 showed that gold occurs freely as blebs within biotite. Various workers have noted the “nuggety” nature of the gold mineralization. During metallurgical and flotation studies, most gold was found to liberate during processing, with a smaller percentage of gold grains attached to silicate or sulphide gangue (Ganderup and Woods, 1986).

8.0 DEPOSIT TYPES

Geological mapping and sampling in the Windy Gulch Segment of the Dawson Project led to the realization that all the rocks hosting the gold mineralization could be better characterized as aluminum-rich (peraluminous) intrusions that not only host the gold but were also the direct sources of the gold mineralization (Keith et al., 2016). As such, the peraluminous intrusions comprise a portion of a staged intrusive sequence. Each stage is associated with a specific magmato-hydrothermal greisen-like hydrothermalite rock that is variously biased for a specific metal suite ranging from copper dominated to gold dominated. When compared to the literature, the gold deposits at Dawson can be broadly described as "intrusion related gold deposits" as in the recent summary of the problem by Pertzel (2013), which is a reconstitution of the phenomena as anticipated by Spurr (1906). The development of the peraluminous portion of the model was discussed by Keith and Swan (1985), Swan and Keith (1986) and at a symposium on the subject chaired by MagmaChem in Denver (Swan and Keith, 1987).

The geological epiphany summarized above completely replaces the previous volcanogene gold paradigm that has been historically utilized by previous workers at Dawson beginning with the work by US Borax in 1976 (see Section 6.2.1). One of the important geological implications of the new peraluminous intrusive model is that it allows a more predictive and specific characterization of the bonanza grade gold occurrences where gold grades generally exceed one ounce of gold per short ton. Rather than being statistical anomalies, the bonanza grade gold occurrences are herein specifically assigned to late pegmatitic differentiates of Stage 3 pink peraluminous granite and are now thought to be physically continuous bodies of high grade gold.

9.0 EXPLORATION

9.1 2013 Mapping, Sampling and Trenching

On May 1, 2013, Zephyr conducted a limited chip sampling program at Windy Gulch consisting of 29 chip samples and 2 grab samples in one road-cut trench and two small previously excavated pits (located northeast of the Windy Gulch Segment deposit). Bedrock surface was cleaned prior to sampling, and samples were collected using a hammer and chisel. All chip samples were 3 ft in length and were oriented oblique to the mineralized zone strike. The trench was established during access road construction to proposed drill hole locations, and mineralized "exhalite," footwall, and hanging wall rocks were exposed. Two small previously excavated pits to the northeast of the known Windy Gulch Segment deposit were also sampled. The Windy Gulch Segment mineralized "exhalite" horizon (Windy Gulch Zone) strikes approximately 060° and the sample line trended along an azimuth of 340°. Road-cut trench and pit details appear in Table 9-1.

Table 9-1: Zephyr Trench Sampling Results

Trench ID	Grid Easting (ft)	Grid Northing (ft)	Elevation (ft)	Length (ft)	Inclination (°)	Azimuth (°)	No. of Chip Samples
E_Trench	46303.91	45811.77	7,194.81	54	-15.3	342	18
PIT1	46288.285	45993.523	7,167.28	6	n/a	360	4*
PIT2 (East Side)	46424.252	45980.679	7,158.85	12	n/a	360	4
PIT2 (West Side)	46409.155	45980.679	7,158.85	15	n/a	360	5

**Includes two grab samples from waste rock pile*

The Windy Gulch Zone in E_Trench measures approximately 39 ft in road-cut width (36.75 ft true thickness) and was dominated by fine grained siliceous QB gneiss ± QBG gneiss hosting disseminated pyrite mineralization. Near the upper and lower contacts of the Windy Gulch Zone, the mineralization is hosted by altered micaceous rock. Thirteen 3 ft samples were collected across the Windy Gulch Zone and the weighted average gold grade was 7.57 g/t over a true thickness width of 36.73 ft. No visible gold was observed in the outcrop or in samples.

Geological mapping and trench sampling extended the known surface extent of the Windy Gulch Zone by approximately 20 ft to the east (higher up the hillside and beyond the extent of the underground drilling in the Mike Sutton Adit). Surface mapping in 2013 also confirmed the presence of QBG "exhalite" horizon approximately 130 ft to the northeast of the Windy Gulch Segment deposit where it is obscured by overburden. Historical work interpreted the QBG gneiss "exhalite" horizon found in the northeast pits to be either a faulted offset of the Windy Gulch Segment "exhalite" horizon to the south, or the northern synclinal limb of a tight west-plunging fold. A malachite stained quartz-anthophylite host rock was sampled from PIT2 (East Side) and returned a gold value of 3.14 g/t over 3 ft. A fine-grained biotite gneiss chip sample, also collected in PIT2 (West Side), returned a gold value of 1.76 g/t over 3 ft. This prospective area at Windy Gulch warrants additional follow-up drilling.

9.2 2013 Diamond Drilling

Zephyr conducted a core drilling program on the Property between April 2 and April 15, 2013. This program consisted of 13 drill holes at Windy Gulch that total 1,928 ft of drilling. These were designed to confirm and infill historical drilling results and test the eastern strike extension of the Windy Gulch deposit. Details and results of the 2013 drilling by Zephyr are presented below in report Section 10.3.

9.3 2016 Mapping, Sampling, Magnetometer and Induced Polarization / Resistivity Surveys and LiDAR

At the request of Loren Komperdo of Zephyr Minerals Ltd., MagmaChem initiated a mapping and litho-geochemical sampling program to support the drilling and exploration programs at Windy Gulch in the search for gold in the greater Dawson gold system.

In mid-April of 2016, three field days were spent by Stan Keith and Monte Swan of MagmaChem accompanied by Mark Graves on behalf of Zephyr. Two road-cuts at Windy Gulch were mapped at a scale of 1 inch = 50 feet. In addition, 30 whole rock samples were collected for petrochemical analysis and 68 mineralized rock samples for multi-element analysis.

Analysis of the data has led to the increasing realization that the greater gold system at Dawson is causally related to a suite of peraluminous granites. Thus, the entire geology of the area has been re-evaluated with respect to a new peraluminous gold paradigm. The former model was couched in a massive sulphide/exhalative paradigm.

Conclusions from this work are still being received by Zephyr at the time of writing. Sections 9.3.1 and 9.3.2 beneath were extracted from Keith et al. (2016).

9.3.1 Rock Map Units at the Dawson Trend

Geological mapping of the gold system in the Windy Gulch segment of the Dawson Gold System revealed gold is related to a widespread suite of peraluminous granitoids that are intra-mineral in nature. The peraluminous granitoid suite (referred herein to as the Dawson Perluminous granite sequence) was emplaced as a differentiation staged sequence with each stage probably related to a specific hydrothermal fluid phase (Keith et al., 2016). Petrochemical characterization of these intrusive suites is currently ongoing.

As currently understood, four major peraluminous map units are recognized:

- A banded, biotite granodiorite aplite unit with accessory garnet that is associated with copper (gold-silver[bismuth-zinc-lead-selenium]) mineralization intimately associated with coarse greisenous amphibole, muscovite, and biotite. This unit is considered a magmato-hydrothermal hydrothermalite and has been referred to in Dawson historical documents as Unit 1. This nomenclature is retained herein.
- A pink, foliated peraluminous granite and related pegmatites that cut Unit 1 mineralization and are associated with the “main stage” gold (Bi-Ag-Se-Te-Pb-Zn) mineralization that forms the bulk of the moderate to high grade (+4 g/t) gold deposits that have been delineated to date by historical drilling (at the Dawson Segment) and ongoing drilling (at the Windy Gulch segment). This gold mineralization is associated with pyrite, quartz, and finer-grained sericitic muscovite (QSP alteration) that is also sillimanite stable. Pyrrhotite, minor chalcopyrite and galena have been observed here or been described in historical documents along with accessory bismuth-selenium sulfosalts. Like Unit 1, the aggregate mineral assemblage is also considered a magmato-hydrothermal hydrothermalite that has been referred to in Dawson historical documents (Swan and Keith, 1991) as Unit 2. That nomenclature is also retained in this narrative.
- A late, brittle brown silica-quartz filled fracture event that locally contains bonanza-grade gold with a strong bismuth-selenium (tellurium-mercury) trace element suite overprints Unit 2 mineralization in tensile structural settings. As of this date, the bonanza gold occurrences are only known to be associated with Unit 2. As such, the bonanza gold is inferred to be an extreme hydrothermal fractionate of the Unit 2 hydrothermalite.
- A late highly differentiated white, flow banded foliated soda aplite intrusive occurs on the southeast edge of the map area and at Windy Point where it may be associated with a poly-metallic set of northeast tending gold veins that are currently viewed as very late-stage “dregs” derived from the soda aplo-granite that itself is a magmatic “dreg/drip” at the westernmost end of the Dawson intrusive suite.

Sometime after 1,800 Ma and prior to 1,760 Ma, the Dawson Shear Zone experienced a regional northeast-southwest directed compression. The compression induced left slip along the length of the Dawson Shear Zone coinciding with the emplacement of a suite of peraluminous intrusions that differentiated from east to west and intruded into dilatancy zones formed by crustal drag along the fault. On a mass basis, the differentiation sequence was largest in the east in the Dawson Segment and become smaller towards the west culminating in the emplacement of a small mass of white, highly differentiated peraluminous soda granite in the Windy Point Segment.

Decompression of the water-rich peraluminous intrusive in tensile zones produced a sequential exsolution of the gold-bismuth-copper hydrothermal fractionates with decompressed Stage 3 granodiorite aplite followed by gold-bismuth-selenium hydrothermal fractionates associated with pegmatoidal phases of peraluminous granite (formerly the Pbu pink banded unit). Windy Point is inferred to be the hydrous auriferous fractionates derived from decompression of the peraluminous granite.

9.3.2 Mineralization at the Dawson Trend

Because of their intimate association with peraluminous pegmatites emplaced in the mid-crust (about 3 Kb, hydrothermalites take on a more rock-like aspect and much of the hydrothermalism presents itself as a magmato-hydrothermal hydrothermalite greisen (Keith et al., 2016).

Complete textural rock types exist between the magmatic stages and the late brittle hydrothermal stages.

Much of the mineralization “hides” in the background of upper greenschist–lower amphibolite metamorphic grade facies and for this reason is commonly confused with more conventionally regarded metamorphism. In this context, the hydrothermalites at Dawson can be regarded as hydrothermal metamorphic facies and not simply as hydrothermal alteration superimposed on a “cold” hostrock protolith like that of a porphyry copper or a Carlin type gold deposit.

One of the characteristics of Dawson hydrothermalite greisen is the coarseness of the gangue mineral suite (e.g. green amphibole, muscovite, and biotite). In the earlier applied volcanogene exhalative gold model, these rocks were treated as metamorphic rock types. For example, the biotite amphibolite at Alica on the northern mountain slope of Windy Gulch was regarded as an amphibolite-grade meta-basalt. In the hydrothermalite paradigm deployed here, the coarse biotite at Alica is part of a biotite greisen hydrothermalite that is intimately associated with crystallization of the pink peraluminous granite unit at Alica and a significant gold-bismuth-copper-REE (rare earth elements) anomaly that constitutes the North Windy Gulch gold target.

Two major hydrothermalite rock types, Unit 1 and Unit 2, are present at Dawson and have been recognized over the entire Property. Units 1 and 2 were recognized during previous exploration campaigns (e.g. Swan and Keith, 1991) and interpreted in the context of a volcanogene gold model. In this report, the Unit 1 and 2 hydrothermalites are reconstructed in the context of a peraluminous magmato-hydrothermal gold model. A third coarse biotite dominant hydrothermalite formerly interpreted as a meta-basalt is now regarded as a biotite hydrothermalite greisen.

9.3.3 Magnetometer and Induced Polarization / Resistivity Surveys

9.3.3.1 General

Zephyr Minerals Ltd. commissioned RDF Consulting Limited from May 11 through to June 14, 2016, to perform a high resolution, GPS enabled, ground magnetometer and induced polarization (IP)/Resistivity survey on the Property (Fraser, 2016). The Dawson Gold Project magnetometer survey consisted of 22.02 line kilometres of data collection over eighteen 100 m spaced reconnaissance lines. Some infill and reconnaissance work was also performed. The IP survey, also reconnaissance in nature, consisted of 6.9 km of data collected on eight priority lines. The surveys were performed to gain a better understanding of the known gold mineralization and local geology of the area.

A reconnaissance grid was initially established over a portion of the Dawson area to facilitate the IP/Resistivity geophysical surveys. The magnetometer survey was performed using a high resolution GEM GSMP-35 potassium field magnetometers and a GEM GSM-19 Overhauser base station magnetometer. The base station unit was used to correct for variations in the Earth's magnetic field at the end of each survey day. The IP survey was performed using the pole-dipole array implementing dipole spacings of 25 and 50 m. The 25 m survey was performed using six dipoles, and eight dipoles were used for the 50 m. dipole survey. Maximum depths of penetration achieved by these methods are 75 m and 200 m respectively.

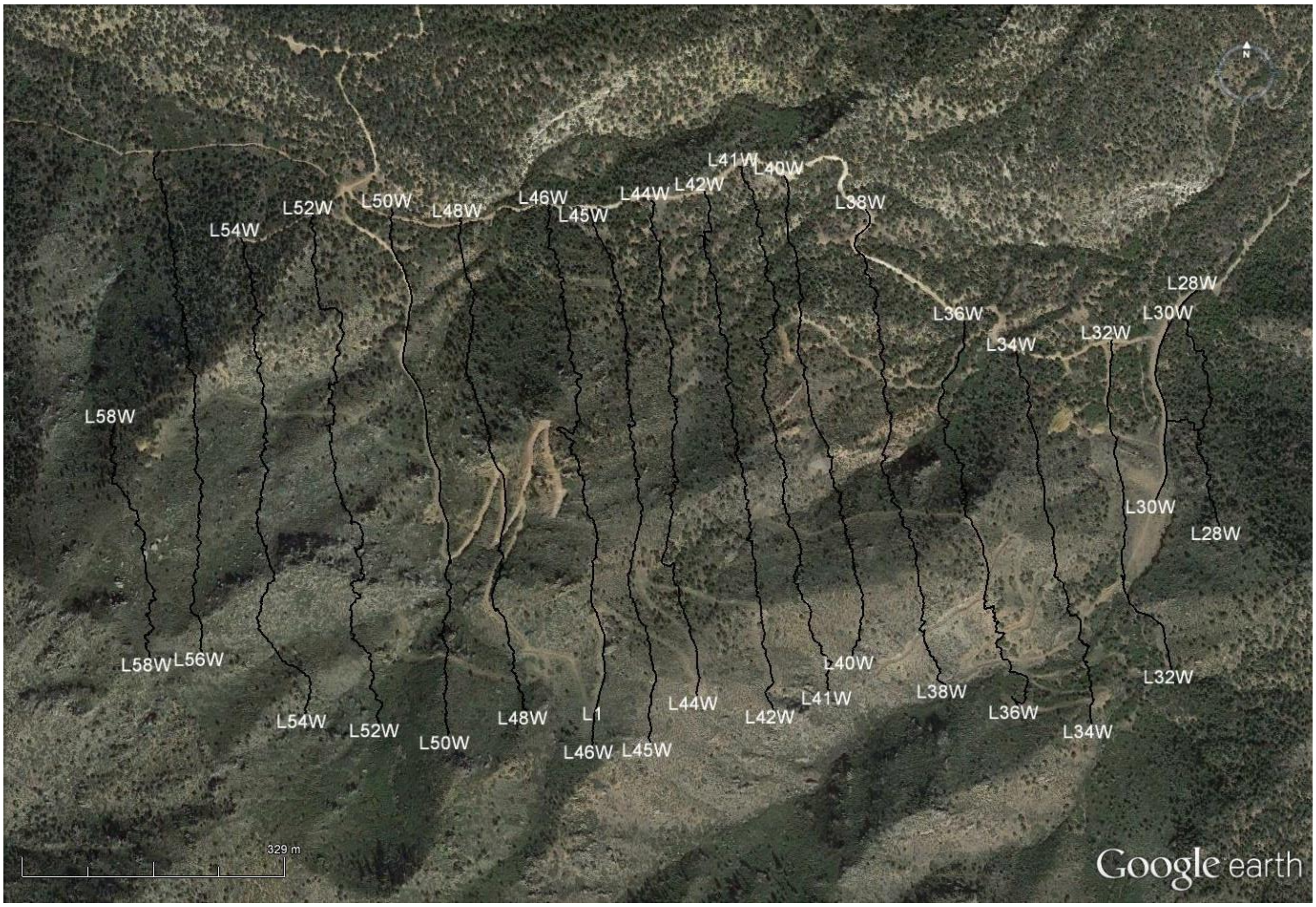


Figure 9.1: Total Magnetic Image for the Dawson Gold Project With Significant Gold Prospects

9.3.3.2 Results of the Magnetometer Survey

The following are considered the key interpretive findings obtained from the ground magnetometer survey data:

1) The main east–west structural feature (Dawson Shear Zone) interpreted from the magnetic data correlates directly with the known gold mineralization at Windy Gulch and Dawson.

2) The interpreted structural corridor delineated by the strong low magnetic signature transects the entire Property and is locally up to 328 ft wide. The actual structure is much narrower.

3) The main structure (Dawson Shear Zone), directly related to the gold mineralization, becomes difficult to trace past L41W and may be offset to the south.

4) Most known showings and historical shafts are located within the structural corridor and very close to the interpreted main structure.

5) The intrusive unit in the Windy Gulch area is magnetically complex and appears to be different in geophysical characteristic than the intrusive to the south.

6) 3D inversion modelling of the data suggests that a southerly dip of the intrusive is consistent with field observations.

7) The hanging wall intrusive related to the main mineralization has a somewhat complex shape with embayments and offsets that may affect the location of gold mineralization.

8) The line to line magnetic profiles appear to effectively map the main mineralizing structure (Dawson Shear Zone).

9) Cross structures, interpreted as secondary structures, occur in proximity to both the Windy Gulch and Dawson deposits. The exact orientation of these features is not known.

10) The interpreted contact outlining the sediments is peculiar in nature as it has a stronger magnetic signature than the intrusive it is in contact with as well as being located within and part of a major thrust fault. The response is believed to be from a deeply buried source.

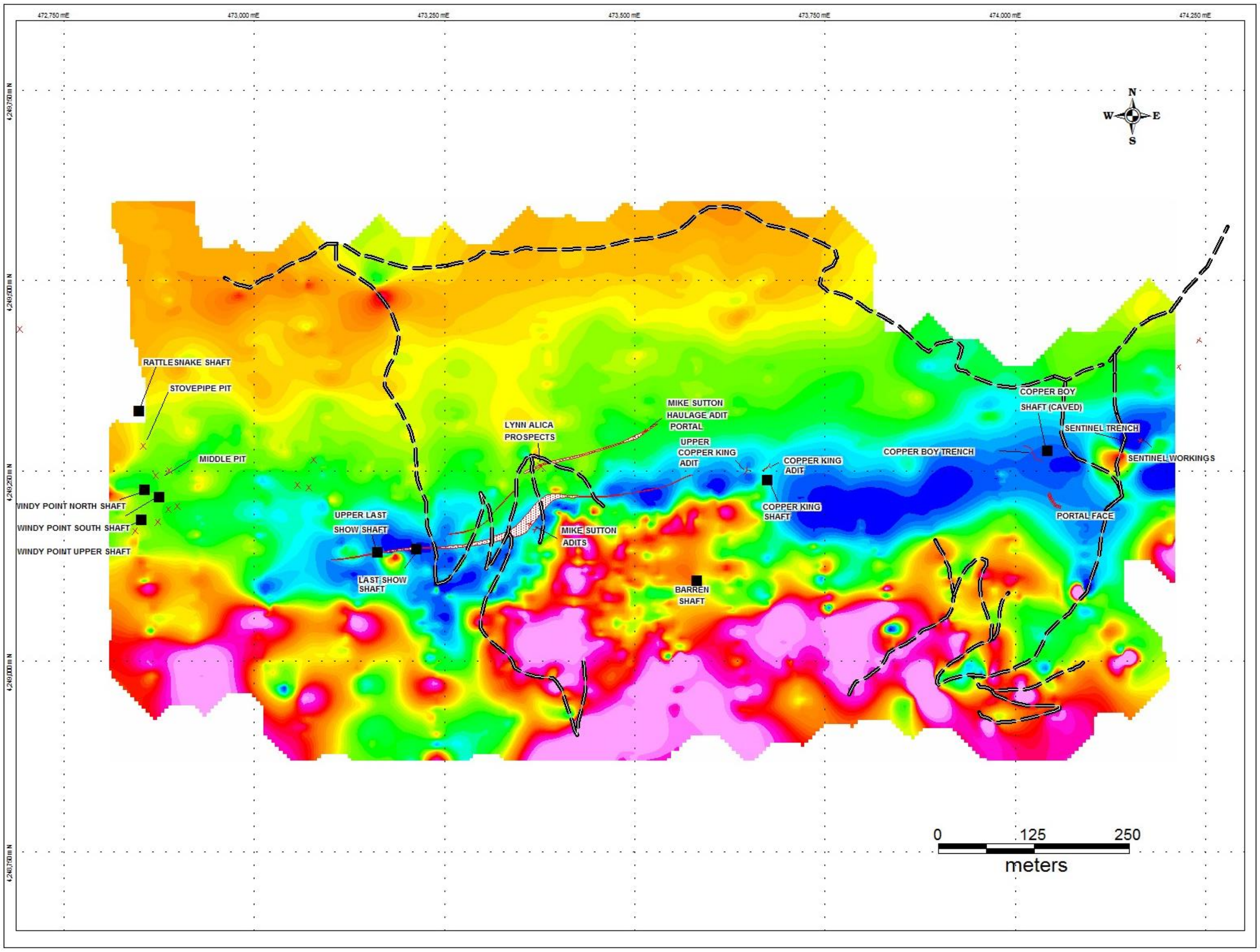


Figure 9.2: Total Magnetic Image for the Dawson Gold Project Area with Significant Gold Prospects

9.3.3.3 Results of the Induced Polarization Survey

The following are considered the key interpretive findings obtained from the induced polarization/resistivity survey data:

1) The chargeability response readily identified the weak sulphide content related to the main gold mineralization at Windy Gulch and Windy Point.

2) In general, known mineralization in the Dawson area shows a direct correlation between elevated chargeability and decreased resistivity.

3) Most of the zones appear to have an approximate northeast–southwest trend.

4) Chargeability anomalies are considered weak to moderate as sulphide content is generally low for the portion of the Dawson Project surveyed. Weak targets should not be overlooked especially when associated with resistivity and magnetic low features.

5) The main gold bearing structure corresponding with Zone 1 (Windy Gulch Zone) becomes very weak, and difficult to trace to the east past line L42W. Zone 1 is designated Z1 in Figure 9.3.

6) Several additional zones of interest have been identified for the survey area and require follow-up field investigation.

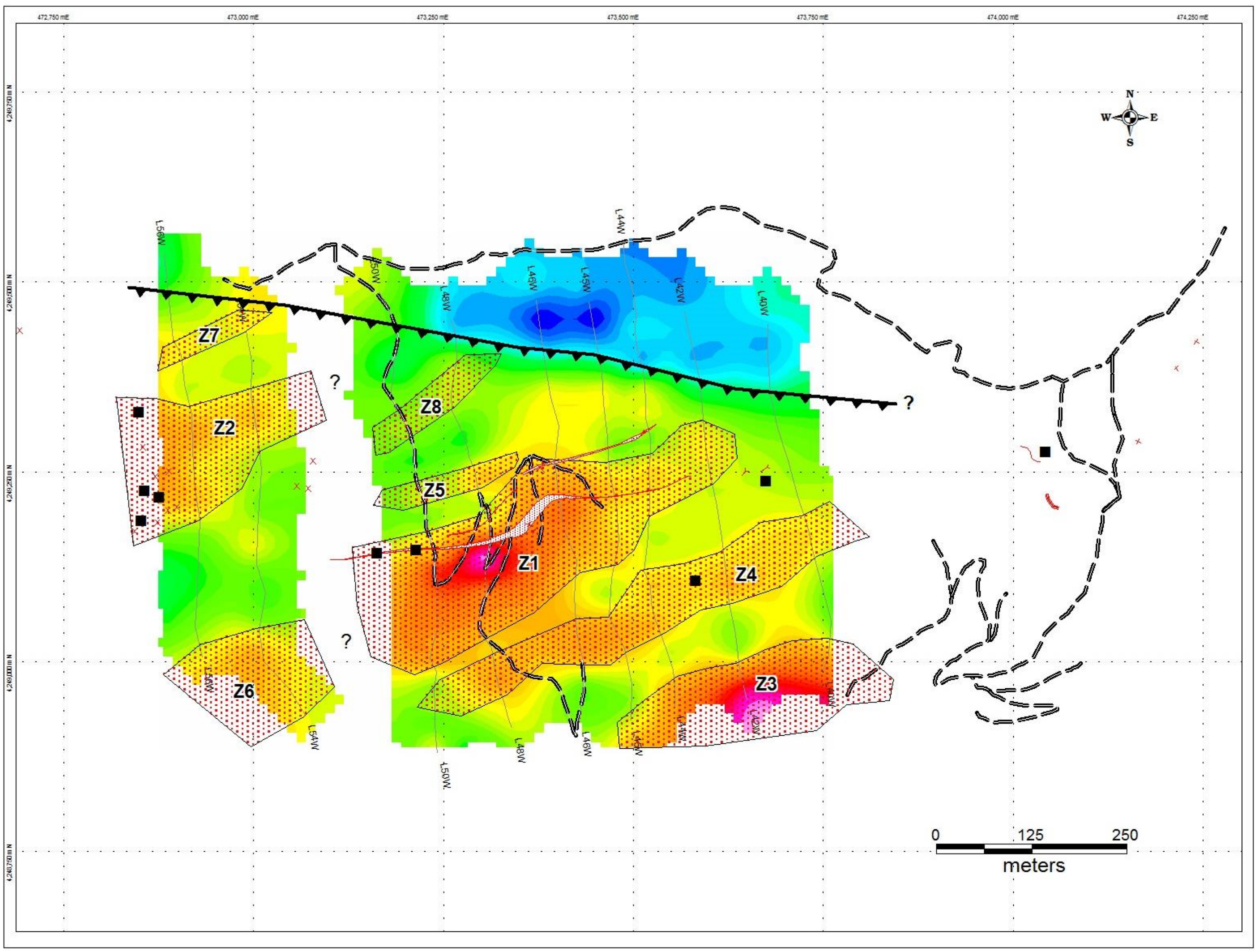


Figure 9.3: Interpretive M11 Chargeability Map Illustrating Zones of Elevated Chargeability

9.3.4 LiDAR and Orthophoto Data Report

9.3.4.1 Area of Interest

Aerial LiDAR and photography was collected on April 4, 2016, of the Dawson Project area located near Cañon City, Colorado. The Area of Interest (AOI) for this project covers a total of 0.66 m^2. Imagery resolution was 3 inches for the area depicted below in purple and 8 inches for the entire project area. LiDAR was flown to achieve a point density of 0.5 per square foot.

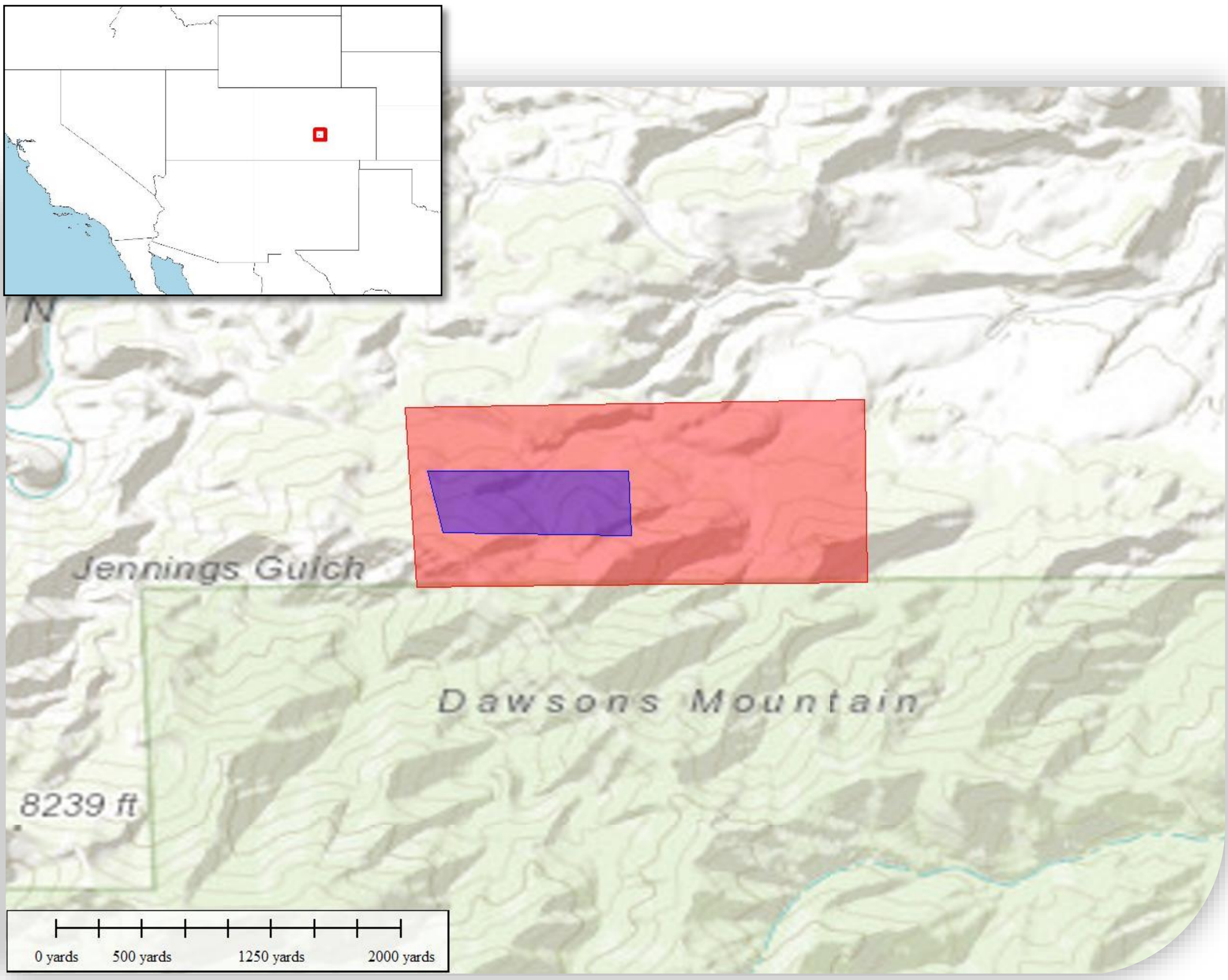


Figure 9.4: Location Map for Aerial LIDAR and Photography Survey at Zephyr's Dawson Project near Cañon City, CO (North is top of the page)

9.3.4.2 File Formats, Units, and Projection

Project deliverables include the following:

Digital Elevation Contours – 1 Meter and 5 Meter Interval in UTM and 2 Foot Interval in State Plane

- ESRI Shapefile format (.shp) – UTM
- ACAD DWG – State Plane

"Bare Earth" Digital Elevation Model – 1 m Grid

- ESRI Shapefile format (.shp)
- Delivered as one file clipped to the AOI

Digital Imagery – 8-inch and 3-inch nominal resolution

- Colour-balanced imagery in Tiff format
- Tiff reference files

Various other files

- 2 ft Contours in PDF
- 10 ft Contours in PDF
- High Resolution Imagery in PDF
- High Resolution Imagery with 2 ft Contours in PDF

Project Surround Files

- ESRI Shapefile format (.shp)

LiDAR Data Report

- PDF format

Table 9-2: Map Projection Information

Projection	State Plane Colorado Central	UTM Zone 13
Horizontal Datum	NAD83	NAD83
Vertical Datum	NAVD88	NAVD88
Geoid	GEOID12B	GEOID12B
Units	US Survey Feet	METERS
EPSG Code	2232	26913

9.3.5 Mineralogical Investigations

The text of this section is extracted from the Uranez 1990 annual report (Uranez, 1991).

Twenty-six core samples from the Dawson Segment have been mineralogically investigated by microscopy and X-ray diffractometry at the Uranerz laboratories in Bonn, Germany. Only one auriferous mineral can be determined by microscopy: native gold occurring in flakes between 0.001 and 0.24 mm in size. The gold locally shows varying reflectances indicating varying degrees of fineness. Two basic grain shape classes are presented at Dawson: cracks loosely occupied by flakes of 1 to 6 (or 8) µm thickness and 10 to 40 (maximally 80 µm) length and the other class yields irregular preferentially roundish grain shapes. The second class shows size variations from 4 to 48,000 µm. 22% of the gold grains of this two-dimensionally (larger sized) class have square sizes up to 100 µm; 91% of the 191 grains counted are up to 2,000 µm.

The gold is associated with sillimanite, sericite or biotite. It appears in cracks in quartz, along quartz grain boundaries and along the boundaries between quartz and the above-mentioned silicate minerals. It is furthermore associated with carbonate veinlets (siderite was X-ray confirmed in one sample). In another sample, gold was found in cracks of garnet; here associated with carbonatic and partly siliceous veinlets. In many instances the gold is associated with bismuthiferous minerals. In a very few cases. It occurs as inclusions in pyrite or chalcopyrite. The Bi-minerals determined so far are the sulfosalts hammerite, friedrichite, joseite (A+B), krupkaite and possibly bohdanowiczite. Bismuthinite is notably absent. Other ore minerals appearing are magnetite, hematite, limonite, locally pyrrhotite; further on pyrite, marcasite and rare chalcopyrite and sphalerite.

9.4 2016 Diamond Drilling

Zephyr conducted a core drilling program on the Property between June 1 and August 28, 2016. This program consisted of 17 drill holes at Windy Gulch that total 2,539.5 ft of drilling. These were designed to infill historical and 2013 drilling results and test the eastern strike extension of the Windy Gulch deposit. Details and results of the 2016 drilling by Zephyr are presented below in report Section 10.2.

10.0 DRILLING

10.1 General

Diamond drilling data from the Property included in the current Dawson Segment and Windy Gulch Segment resource estimates includes 55 holes completed by US Borax between 1981 and 1985, 43 holes completed by Jascan between 1987 and 1988, 20 holes (including 2 holes wedged off pilot hole GC40) completed by Uranerz between 1990 and 1991, 13 holes drilled by Zephyr in 2013 and 17 holes drilled by Zephyr in 2016. Details of each program are presented below under separate headings. All historical drilling information, including lithologic and sampling logs, assay results, collar survey data, and down hole survey information, was assembled from both digital compilations supplied by Zephyr and original hard copy company documents. All 2013 and 2016 Zephyr drilling information such as assay intervals, assay certificates, collar and downhole survey information, and lithologic quicklog summaries, were assembled by Mercator and Golder from digital sources provided by co-author Graves on behalf of Zephyr.

Table 10-1 below provides a summary of drilling information pertaining to the US Borax, Jascan, Uranerz, and Zephyr programs and includes all drill holes used in the current resource estimate. Collar coordinates, azimuth, and inclination information for each program appears in the corresponding discussion below and hole locations are presented in Figure 10.1. Only 56 holes fall within the confines of the current resource estimate but all contribute to geological models developed for the Dawson Segment and Windy Gulch Segment deposits.

Table 10-1: Company-Specific Listing of Diamond Drill Holes Used in Resource Estimate

Operator	Period	Total Drilled ft	Hole Series	No. of Holes Drilled in Dawson and Windy Gulch Segments
US Borax	1981–1985	46,250.3	GC03 to GC62*	55
Jascan	1987–1988	18,271.3	GC63 to GC98*, MS01 to MS07**	43
Uranerz	1990–1991	12,038	DA9001 to DA9008, DA9101 to DA9103, WG9101 to WG9107	20
Zephyr	2013	1,928	WG-13-08 to WG-13- 20	13
Zephyr	2016	2,539.5	WG-16-21 to WG-16-36	17

* Some holes in series not drilled in the Dawson Segment and Windy Gulch Segment areas; series also contains duplicate hole ID numbers from re-drills of holes lost in bad ground conditions
** Underground holes drilled from Mike Sutton workings

Figure 10.1: Windy Gulch Segment Drill Holes

Drill holes completed by US Borax, Jascan, Uranerz, and Zephyr on the Dawson and Windy Gulch Segment areas were drilled to test the east-northeast trending gold-bearing felsic gneisses and were drilled at azimuths that generally range from northwest to northeast with an average inclination of -58.1° (not including MS01 to MS07 underground holes). The average sample length for all drilling was 3.3 ft. Drill core from the US Borax, Jascan and Uranerz programs is stored in a hangar at the Fremont County Airport, CO, and core from the 2013 Zephyr program is stored at the Zephyr core facility in Cañon City, CO.

Prior to Zephyr's 2016 drill program, all drill holes were surveyed using a local non-earth grid. Drill hole collar casings for the Zephyr 2013 program were surveyed using this non-earth grid under the supervision of QP Mark Graves. Zephyr has modernized the grid system for the Dawson Project and all points are now registered utilizing NAD 83 Colorado State Plane, Central Zone (ft US).

10.2 Zephyr 2016 Windy Gulch Drilling Program

10.2.1 General

The 2016 Zephyr drilling program was conducted between June 1 and August 28 and involved the use of a single diamond drill. The drill was track mounted, operated by Godbe Drilling LLC of Montrose, CO. NQ sized (47.6 mm diameter) core was recovered from all drill holes. Mark Graves managed the drill program, providing on-site management, core logging, sampling, and geological control.

This program consisted of 17 drill holes at Windy Gulch that total 2,539.5 ft of drilling. The drill program was operated with only the dayshift. A nightshift wasn't employed. Down-hole surveys were generally collected every 50 ft. Three drill holes were lost due to difficult ground conditions. These were WG-16-30, WG-16-34, and WG-16-34a. Three drill holes went through pre-existing workings. These were WG-16-34, WG-16-34a, and WG-16-36.

Drill water was trucked to the site by a 5 ton truck owned by Fremont Paving and Redi-Mix of Cañon City, CO. Surface drill hole locations were surveyed by Edward-James Surveying of Pueblo, CO. The pre-existing non-earth mine grid is no longer used and has been replaced by the Colorado State Plane Central Zone (NAD 83) system. Diamond drill hole information is listed in Table 10-2 below.

Table 10-2: 2016 Zephyr Holes Completed on the Windy Gulch Segment: Colorado State Plane (NAD 83)

Hole ID	Grid Northing (ft)	Grid Easting (ft)	Grid Elevation (ft)	Hole Length (ft)	Dip (°)	Azimuth (°)
WG-16-21	1203058.45	3056158.58	7,190.30	205	-25	265
WG-16-22	1202976.71	3055965.22	7,214.08	178	-2	26
WG-16-23	1202974.69	3055964.95	7,211	146	-22	25
WG-16-24	1202941.56	3055955.68	7,206.59	229	-40	52
WG-16-25	1202983.19	3055959.55	7,213.46	155	-3	8
WG-16-26	1202978.15	3055940.51	7,207.69	124	-40	322
WG-16-27	1202938.15	3055944.65	7,205.26	178	-74	312
WG-16-28	1202931.30	3055949.20	7,207.41	181	-70	50
WG-16-29	1202976.54	3055960.77	7,209.49	134.5	-40	21
WG-16-30	1203077.73	3056156.68	7,184.61	196	-80	336

Hole ID	Grid Northing (ft)	Grid Easting (ft)	Grid Elevation (ft)	Hole Length (ft)	Dip (°)	Azimuth (°)
WG-16-31	1202833.55	3055797.55	7,120.11	184	-70	44
WG-16-32	1202832.07	3055789.30	7,120.21	145	-65	342
WG-16-33	1202842.91	3055799.92	7,119.49	125	-43	25
WG-16-34	1202961.38	3055947.20	7,206.15	49	-62	285
WG-16-34a	1202959.2	3055946.88	7,206.22	115	-62	285
WG-16-35	1202841.73	3055748.63	7,091.99	93	-28	13
WG-16-36	1202787.51	3055423.68	6,983.17	102	-45	322

10.2.2 Drilling Results

The 2016 Zephyr drilling program has validated previous drilling results, better established continuity of grade and thickness, and delimited the Windy Gulch Zone to both the east and west. No visible gold was seen in the core, even when examining high grade samples from the zone with a binocular microscope. Selected drilling highlights from the 2016 program are presented in Table 10-3.

Table 10-3: 2016 Zephyr Weighted Average Gold Highlights from Windy Gulch Segment

Hole ID	From (ft)	To (ft)	Interval (ft)	Approximate True Thickness (ft)	Au (g/t)
WG-16-21	138.0	156.5	18.5	14	1.6
Including	138.0	141.5	3.5		4.6
Including	170	176	6	4.5	2.0
WG-16-26	53.0	99.0	46.0	46.0	3.4
Including	73.5	99.0	25.5		5.0
Including	73.5	86.0	12.5		7.4
WG-13-27	66.5	138.0	71.5	60.5	4.0
Including	77.5	138.0	60.5		4.5
Including	90.0	104.5	14.5		15.2
Including	97.5	104.5	7.0		26.5
WG-16-28	133.0	147.0	14.0	8.0	1.8
WG-16-29	91.0	111.0	20.0	19.5	1.5
Including	108.0	111.0	3.0		7.2
WG-16-31	113.0	149.0	36.0	27.0	6.7
Including	131.0	146.0	15.0		13.8
WG-16-32	81.0	105.0	24.0	20.0	2.9
Including	81.0	84.0	3		17.1
WG-16-33	36.0	89.0	53.0	45.0	3.4
Including	64.0	69.0	5.0		8.1
And	77.0	86.0	9.0		9.7
WG-16-34A	57.0	109.5	52.5	39.5	2.0
Including	81.5	92.0	10.5		5.5
WG-16-35	42.0	73.5	31.5	31.5	4.0
Including	60.0	66.5	6.5		14.0

Windy Gulch drilling intersections have defined the dimensions of the zone. The true strike length (as measured in plan) ranges up to approximately 250 ft. Down-dip ranges to about 300 ft in the main part of the Windy Gulch mineralization. The maximum plunge length is about 725 ft and true thickness varies from 20 to 70 ft. The trend of the mineralization has an azimuth of 056° and dips to the southeast at 60°.

The host rocks for gold mineralization are better understood because of the 2016 drilling and mapping. Gold is now known to occur in four different lithotypes, three of which are hydrothermal or pneumatolytic in nature.

The Unit 1 hydrothermalite is a Property-wide hydrothermal map unit and was formerly considered a copper-rich massive sulphide facies in the volcanogenic gold model. The new characterization is that it is a copper-gold greisen related to the banded Stage 3 biotite aplite (formerly fine-grained biotite gneiss) with which it displays an intimate association. Megascopically, this unit is a mica-amphibole rock.

Unit 2 gold is hosted by anastomosing limonitic veins and veinlets that cross-cut the Stage 3 biotite aplite. These limonitic veins host bonanza grades of gold and at Windy Gulch are associated with a peraluminous aplo-pegmatite dyke containing textures suggesting it is a potential source for the gold-enriched hydrothermal fluids (Keith et al., 2016).

A third host for gold mineralization is a malachite-bearing massive biotite seen at 66.5 to 68.0 ft of WG-16-27. This intersection graded 9.65 g/t. Hydrothermalite of this sort is seen outcropping at Windy Gulch.

The fourth host for gold at Windy Gulch is the biotite aplite, which is the dominant rock type volumetrically. It is lower grade than the hydrothermalites. A mineral investigation of Dawson Segment rocks similar to the Windy Gulch aplite was conducted by Uranerz in 1990 and is seen in Section 9.3.3. The mode of gold occurrence in this Uranerz report likely describes that in the aplite at Windy Gulch.

10.2.3 Factors Effecting Reliability of Results

Core recovery varied from fair to excellent. For the biotite aplite, which was massive and competent, recovery was largely better than 80% to 90% and most often 100%. With respect to the higher grade hydrothermalitic rocks, which are quite friable, recoveries could vary from 50% to 75%. Given this drill program was conducted in the same area as the 2013 drilling then (Hilchey et al., 2013) would certainly apply which states

> *Zephyr's protocol for core loss grade assignment is to consider recovered core to be representative of the entire lost core section within a given sample interval. After review of the project data set, Mercator determined this approach to be acceptable for current resource estimation purposes due to the typically homogenous nature of the host felsic gneisses, in which adjacent drill holes with good recovery typically show similar gold grades over similar widths.*

A number of the higher grade historical holes at Windy Gulch should be selected for twinning.

10.3 Zephyr 2013 Windy Gulch Drilling Program

10.3.1 General

The 2013 Zephyr drilling program was conducted between April 2 and April 15 and involved use of one diamond drill. The drill was track mounted, operated by Godbe Drilling LLC of Montrose, CO, and recovered HQ (63.5 mm diameter) size drill core from all holes. Mark Graves coordinated the program and provided on-site supervision, core logging, sampling, surveying, and interpretive functions. Down-hole surveys were generally collected near the mid-point depth and bottom of each hole using a multi-shot instrument. Surface drill hole locations were surveyed by Cornerstone Land Surveying of Cañon City, CO. Drill hole details are listed in Table 10-4 and locations are presented in previous Figure 10.1.

Table 10-4: 2013 Zephyr Holes Completed on the Windy Gulch Segment

Hole ID	Grid Northing (ft)	Grid Easting (ft)	Grid Elevation (ft)	Hole Length (ft)	Dip (°)	Azimuth (°)
WG-13- 08	1202931.42	3055939.62	7,206.45	163	-45	0
WG-13- 09	1202931.42	3055939.62	7,206.45	230	-65	0
WG-13- 10	1202979.03	3055943.58	7,204.31	139.5	-45	0
WG-13- 11	1202957.55	3055944.96	7,205.04	143	-52	314
WG-13- 12	1202957.55	3055944.96	7,205.04	158	-62	314
WG-13- 13	1202957.55	3055944.96	7,205.04	184	-48	297
WG-13- 14	1202957.55	3055944.96	7,205.04	175	-48	277
WG-13- 15	1202822.94	3055788.77	7,120.21	148	-45	0
WG-13- 16	1202822.94	3055788.77	7,120.21	164	-70	0
WG-13- 17	1202878.91	3055801.17	7,112.48	82	-50	353
WG-13- 18	1202785.93	3055708.94	7,083.52	140	-45	320
WG-13- 19	1202785.93	3055708.94	7,083.52	134.5	-45	0
WG-13- 20	1202854.32	3055715.55	7,069.64	67	-50	0

Zephyr drilled a total of 1,928 ft in 13 drill holes at Windy Gulch and these were designed to confirm and infill historical drilling results and test the eastern strike extension of the deposit.

10.3.2 Drilling Results

The 2013 Zephyr program confirmed previous drilling results and demonstrated continuity of grade and thickness in the eastern and central deposit areas. Overall, mineralization was noted to occur in a fine-grained biotite gneiss that is locally mineralized with pyrite and variably gossanized. No visible gold was observed in drill core. At the time of the resource estimate, lithological quicklog summaries, and assay data were available for the

Zephyr drill holes and these were used in the resource models. Selected drilling highlights from the 2013 program are presented in Table 10-5.

Table 10-5: 2013 Zephyr Weighted Average Gold Highlights from Windy Gulch Segment

Hole ID	From (ft)	To (ft)	Interval (ft)	Approximate True Thickness (ft)	Au (g/t)
WG-13-15	89.0	110.0	21.0	21.0	9.16
Including	95.0	104.0	9.0	9.0	18.95
WG-13-16	87.0	108.0	21.0	17.0	1.81
WG-13-17	42.5	49.0	6.5	6.5	47.40*
WG-13-18	103.0	118.0	15.0	13.8	1.46
WG-13-19	106.0	118.0	12.0	12.0	2.84
WG-13-20	41.0	48.0	7.0	7.0	3.72

**40% core recovery over sample interval*

Windy Gulch Segment drilling intercepts of the mineralized “exhalite” horizons demonstrate gold mineralization has a known strike continuity of approximately 570 ft and a known dip continuity length of approximately 260 ft. The mineralized “exhalite” horizons generally trend at 056° azimuth with a southeast dip of about -60°. The Windy Gulch Segment deposit remains open along strike and down dip in the eastern deposit area, and down-dip in the western deposit area.

Drill hole WG-13-11 intersected Mike Sutton workings from 19 to 23 ft, and nearby hole WG-13-12 intersected workings from 49 to 54 ft. Core recovery within the deeper mineralized “exhalite” horizon generally ranged between 90% and 100% but recovery in more friable gossanized wall rock was more variable. This typically was above 75%, but was locally as low as 40%, as seen in hole WG-13-17 in the 42.5 to 49 ft interval, located in the central Windy Gulch Segment deposit area. Zephyr’s protocol for core loss grade assignment is to consider recovered core to be representative of the entire lost core section within a given sample interval. After review of the project data set, Mercator determined this approach to be acceptable for current resource estimation purposes due to the typically homogenous nature of the host felsic gneisses, in which adjacent drill holes with good recovery typically show similar gold grades over similar widths.

10.4 Uranerz Drilling (1990–1991)

10.4.1 General

Between 1990 and 1991, Uranerz completed a total 10,584 ft of drilling in 11 surface holes (NX sized core) and 2 wedge holes (BX sized core) on the Dawson Segment, and 1,454 ft of diamond drilling in 7 surface holes (BX sized core) on the Windy Gulch Segment. Uranerz drilling at the Dawson Segment tested 1) higher grade plunging grade shoots by infilling US Borax and Jascan “exhalite” horizon intercepts, and 2) the extents of gold mineralization in the western deposit area. For the Windy Gulch Segment, Uranerz also followed-up the down-dip extension of gold mineralization from the US Borax mineralized road-cut trench area and Mike Sutton workings. Drill hole details are listed in Table 10-6 and their locations are presented in previous Figure 10.1.

Table 10-6: 1990–1991 Uranerz Holes Completed on the Dawson and Windy Gulch Segments

Hole ID	Grid Northing (ft)	Grid Easting (ft)	Elevation (ft)	Depth (ft)	Dip (°)	Azimuth (°)	Segment
DA9001	1202446.56	3057775.22	6,726.70	1034	-76	337	Dawson
DA9002	1202455.66	3057790.32	6,726.72	1008	-70	336	Dawson
DA9003	1202278.97	3057610.3	6,717.71	1282	-76	355	Dawson
DA9004	1202581.55	3057669.77	6,840.53	898	-71	342	Dawson
DA9005	1202472.07	3057837.83	6,725.57	865	-62	341	Dawson
DA9006	1202568.31	3057899.28	6,703.63	878	-67	356	Dawson
DA9007	1202344.63	3057865.95	6,657.68	1173	-72	334	Dawson
DA9008	1202569.5	3057897.63	6,703.45	918	-74	345	Dawson
DA9101	1202449.99	3057452.75	6,850.60	837	-56	343	Dawson
DA9102	1202447.49	3057448.75	6,849.60	724	-46	328	Dawson
DA9103	1202454.99	3057458.75	6,848.60	752	-52	328	Dawson
GC40W1*	1202831.353	3057828.479	6,028.65	108	-49	340.3	Dawson
GC40W2*	1202831.353	3057828.479	6,028.65	107	-46	340.3	Dawson
WG9101	1202910.37	3055895.98	7,179.93	202	-82	336	Windy Gulch
WG9102	1202910.37	3055895.98	7,179.93	220.5	-47	336	Windy Gulch
WG9103	1202882.1	3055848.68	7,149.12	200	-85	336	Windy Gulch
WG9104	1202882.1	3055848.68	7,149.12	226.5	-49	336	Windy Gulch
WG9105	1202937.17	3055938.54	7,204.70	197.2	-87	336	Windy Gulch
WG9106	1202855.93	3055804.88	7,119.19	205.3	-80	336	Windy Gulch
WG9107	1202962.88	3055979.08	7,230.67	202.5	-78	336	Windy Gulch

**Hole wedged from GC40*

10.4.2 Drilling Results

Uranerz drilling followed up and confirmed results from previous US Borax and Jascan programs. Seven out of the eight drill holes (DA9001 to DA9008) drilled to test the grade shoot interpretation intercepted gold mineralization. The three holes drilled to the west of GC71 to expand shallow resources found no significant mineralization (DA9101 to DA9103). Seven holes were drilled in the Windy Gulch Segment (WG9101 to WG9107) to confirm continuity of the targeted "exhalite" horizon in the east and down-dip directions and gold was encountered in all cases. Drilling results also showed that the lateral continuity of mineralization was often disrupted by faulting or by post-mineral pegmatite intrusions. Selected drilling highlights from the program are listed below in Table 10-7.

Table 10-7: 1990–1991 Uranerz Weighted Average Gold Highlights from the Dawson and Windy Gulch Segments

Hole ID	From (ft)	To (ft)	Interval (ft)	Approximate True Thickness (ft)	Au (g/t)	Segment
DA9001	945.5	947.6	2.1	1.7	210.93	Dawson
DA9002	933.6	935.0	1.4	1.1	123.22	Dawson
DA9005	772.0	812.0	40.0	40.0	5.79	Dawson
Including	772.0	781.0	9.0	9.0	13.10	Dawson
GC9006	735.0	791.0	56.0	45.8	9.53	Dawson
Including	744.0	760.5	16.5	13.5	14.71	Dawson
Including	780.0	781.5	1.5	1.2	114.03	Dawson
GC9008	821.0	832.0	11.0	8.9	7.70	Dawson
WG9101	130.5	134.0	3.5	2.1	30.86	Windy Gulch
WG9105	83.5	113.5	30.0	19.1	4.35	Windy Gulch
Including	86.0	87.5	1.5	1.0	17.73	Windy Gulch
WG9107	139.5	144.5	5.0	3.7	22.15	Windy Gulch

10.5 Jascan Drilling (1987–1988)

10.5.1 General

Between 1987 and 1988, Jascan completed a total 10,673.3 ft of drilling in 17 holes on the Dawson Segment, and 7,230.5 ft of diamond drilling in 19 surface holes and 367.5 ft of diamond drilling in 7 underground holes on the Windy Gulch Segment. NX sized core was recovered for all surface holes, and BX and AX sized core was recovered for the underground holes. At the Dawson Segment, Jascan drilling tested 1) the gap between surface and US Borax intercepts at depth, 2) the western strike extent of known mineralization, and 3) the wider gaps between US Borax intersections. At the Windy Gulch Segment, Jascan drilling tested 1) the down-dip and immediate western extension of mineralization returned from 1984 US Borax road-cut trench sampling, 2) large gaps between US Borax intercepts along strike and dip in the western deposit area, and 3) beyond western deposit extents. A fan of underground holes was also completed from the Mike Sutton adit. Drill hole details are listed in Table 10-8 and their locations are presented in previous Figure 10.1.

Table 10-8: 1987–1988 Jascan Drilling on the Dawson and Windy Gulch Segments

Hole ID	Grid* Northing (ft)	Grid* Easting (ft)	Elevation (ft)	Depth (ft)	Dip (°)	Azimuth (°)	Segment**
GC63	1202859.29	3055798.55	7,119	122.5	-48	357	Windy Gulch
GC64	1202856.29	3055799.55	7,119	112	-59.5	357	Windy Gulch
GC65	1202854.29	3055799.55	7,119	121	-68	358	Windy Gulch
GC66	1202682.29	3055722.55	7,133	310	-63	350	Windy Gulch
GC67A	1202545.37	3058095.08	6,600.88	881	-56	349	Dawson
GC68	1202777.55	3058330.54	6,555.40	692	-51	330	Dawson
GC69	1202779.63	3058330.86	6,555.53	626.5	-45	343	Dawson

Hole ID	Grid* Northing (ft)	Grid* Easting (ft)	Elevation (ft)	Depth (ft)	Dip (°)	Azimuth (°)	Segment**
GC70	1202888.24	3057618.28	6,847.86	411	-67.5	316	Dawson
GC71A	1202610.06	3057685.36	6,833.58	754.5	-58.8	321.5	Dawson
GC72	1202447.72	3057451.76	6,850.60	766	-49	349	Dawson
GC73	1202445.71	3057452.13	6,850.07	1034.5	-71	343	Dawson
GC74	1202448.34	3057452.88	6,850.30	745.5	-47	358	Dawson
GC75	1202611.29	3057687.05	6,833.24	765	-62	348	Dawson
GC76	1202447.25	3057450.48	6,851.18	936.5	-66	347.5	Dawson
GC77	1202474.48	3055277.95	7,153.30	583	-63	360	Windy Gulch
GC78	1202474.48	3055277.95	7,153.30	443.5	-45	360	Windy Gulch
GC79	1202662.08	3055497.76	6,999.79	522	-29	56	Windy Gulch
GC80	1202666.16	3055493.59	6,998.92	272	-33	23	Windy Gulch
GC81	1202658.79	3055493.54	6,999.80	568	-56	53	Windy Gulch
GC82	1202658.55	3055487.28	7,000.23	265	-49	341	Windy Gulch
GC83	1202665.05	3055482.99	6,999.08	244	-29.5	333	Windy Gulch
GC84	1202776.44	3055583.73	7,006.10	233	-27	26	Windy Gulch
GC85	1202576.41	3055686.3	7,143.69	590	-59	47	Windy Gulch
GC86	1202579.79	3055689.23	7,143.29	496.5	-36	47	Windy Gulch
GC87	1202581.99	3055684.68	7,143.10	531.5	-43	22	Windy Gulch
GC88	1202579.23	3055683.08	7,143.29	524	-60	16	Windy Gulch
GC89	1202684.19	3055727.55	7,133.60	546.5	-25	32.5	Windy Gulch
GC90	1202677.09	3055716.55	7,133.90	472	-58	338	Windy Gulch
GC91	1202793.19	3055710.85	7,082.10	274	-29.5	9	Windy Gulch
GC92	1202817.34	3057855.28	6,762.71	498	-40	351	Dawson
GC93	1202815.47	3057853.46	6,762.52	499	-47.5	334.5	Dawson
GC94	1202816.32	3057857.69	6,762.83	616.8	-54.5	15	Dawson
GC95	1203111.62	3058220.51	6,519.89	304	-54	341	Dawson
GC96	1203116.72	3058222.13	6,519.57	240	-33.5	6	Dawson
GC97	1203126.31	3058169.62	6,521.59	230	-24	325	Dawson
GC98	1202679.72	3057917.56	6,681.72	673	-60	345	Dawson
MS01	1203004.29	3055922.55	7,150	52	-25	315	Windy Gulch
MS02	1203001.29	3055926.55	7,150	56.5	-70	315	Windy Gulch

Hole ID	Grid* Northing (ft)	Grid* Easting (ft)	Elevation (ft)	Depth (ft)	Dip (°)	Azimuth (°)	Segment**
MS03	1203001.29	3055926.55	7,150	27.5	0	315	Windy Gulch
MS04	1203001.29	3055926.55	7,150	57.5	-50	285	Windy Gulch
MS05	1203001.29	3055926.55	7,150	59	-30	360	Windy Gulch
MS06	1203001.29	3055926.55	7,150	44	-25	60	Windy Gulch
MS07	1203001.29	3055926.55	7,150	71	-50	345	Windy Gulch

* Local grid coordinates
**Sentinel Segment drilling is included in Dawson Segment drilling

10.5.2 Drilling Results

Jascan drilling intersected a new zone of "exhalative" gold mineralization on the western side of the Dawson Segment (belonging to Zone 2). Eight drill holes testing up-dip extents of mineralization showed the favourable " exhalite" QB-QBG gneiss horizon was either not present, or when it was, gold mineralization was less well developed; however, local up-plunge infill targets remain to be tested in the western deposit area. At Windy Gulch, drilling confirmed 1) the presence of high-grade gold mineralization down-dip from the road-cut trench, and 2) the presence of a variably mineralized prospective "exhalite" horizon in the area tested. The drill holes completed in the Mike Sutton workings intersected multiple gold mineralized horizons but are characterized by very poor core recoveries. Some of the drilling highlights from the program are listed below in Table 10-9.

Table 10-9: 1987–1988 Jascan Weighted Average Gold Highlights from the Dawson and Windy Gulch Segments

Hole ID	From (ft)	To (ft)	Interval (ft)	Estimated True Thickness (ft)	Au (g/t)	Segment
GC63	61.0	73.0	12.0	10.8	14.28*	Windy Gulch
Including	61.0	64.0	3.0	2.7	39.53*	Windy Gulch
GC71A	675.7	689.0	13.3	11.5	14.73**	Dawson
Including	675.7	678.7	3.0	2.6	24.23**	Dawson
Including	686.0	689.0	1.5	1.3	55.75**	Dawson
GC73	930	939	9.0	5.8	5.35	Dawson
Including	933	936	3.0	1.9	13.13	Dawson
GC76	826.8	828.2	1.40	1.1	26.57*	Dawson
GC79	309.3	313.3	4.0	2.3	85.38*	Windy Gulch
Including	311.3	313.3	2.0	1.2	159.75*	Windy Gulch
GC89	216.8	219.8	3.0	1.9	19.68*	Windy Gulch
And	281.9	284.9	3.0	1.9	78.27*	Windy Gulch

*Average of original sample and Jascan check assay results
**Average of original sample, Jascan check assay results, and Uranerz re-assay results

10.6 US Borax Drilling Program (1981–1985)

10.6.1 General

Between 1981 and 1985, US Borax completed a total 41,058.3 ft of drilling in 47 holes on the Dawson Segment and a total of 5,192 ft of diamond drilling in 8 holes on the Windy Gulch Segment. NX sized core was generally recovered, but there were lesser amounts of BX, BQ, BW44, and AX sized core retrieved. US Borax drilling was initially focused in areas with old workings to test known surface expression of mineralization, and was followed by additional drilling step-outs along strike and dip. Drill hole details are listed in Table 10-10 and their locations are presented in previous Figure 10.1.

Table 10-10: 1981-1985 US Borax Drilling on the Dawson and Windy Gulch Segments

Hole ID	Grid* Northing (ft)	Grid* Easting (ft)	Elev. (ft)	Depth (ft)	Dip°	Az°	Segment**
GC03	1203080.75	3058415.75	6518.1	450.1	-50	12	Dawson
GC04	1203056.23	3058203.99	6542.4	397	-55	320	Dawson
GC05	1203056.23	3058203.99	6542.4	495.3	-75	320	Dawson
GC10	1203056.23	3058203.99	6542.4	210.8	-90	0	Dawson
GC11	1202972.37	3058245.56	6543.01	665.5	-80	334	Dawson
GC12	1202904.86	3058207.62	6565.98	600.6	-60	313	Dawson
GC13	1202904.86	3058207.62	6565.98	756.1	-75	313	Dawson
GC14	1202978.43	3058413.59	6533.87	618	-70	355	Dawson
GC15	1202939.18	3058344.83	6531.56	499	-50	350	Dawson
GC16	1202939.18	3058344.83	6531.56	592.5	-70	350	Dawson
GC17	1202721.63	3058348.98	6562.36	903	-63.5	339.4	Dawson
GC18	1202747.27	3058005.47	6669.82	614	-55	337.5	Dawson
GC19	1202747.27	3058005.47	6669.78	704	-70	335.8	Dawson
GC20	1202721.63	3058348.98	6562.36	914	-59	325.6	Dawson
GC21	1202747.27	3058005.47	6669.82	563	-45.2	332	Dawson
GC22	1202785.73	3058405.33	6566.47	744	-62.5	5.8	Dawson
GC23	1202751.69	3058007.27	6669.82	760.5	-65	360	Dawson
GC24	1202679.26	3057908.41	6687.05	728	-65.7	328.8	Dawson
GC25	1202547.3	3058107.91	6598.6	1013	-62.3	358.5	Dawson
GC26	1202679.26	3057908.41	6687.05	602	-53	328	Dawson
GC27	1202796.44	3058404.18	6566.12	774	-64	353	Dawson
GC28	1203477.34	3058061.1	6479.72	1603	-50.2	197	Dawson
GC29B	1202559.68	3057802.23	6764.82	804	-67	333	Dawson
GC31	1202653.28	3055473.56	6999.79	602	-65	24	Windy Gulch

Hole ID	Grid* Northing (ft)	Grid* Easting (ft)	Elev. (ft)	Depth (ft)	Dip°	Az°	Segment**
GC32	1202338.02	3058159.28	6621.06	1159	-51	347	Dawson
GC34	1202656.29	3055478.55	6999.79	547	-65	320	Windy Gulch
GC35	1202341.5	3058155.45	6618.96	1135	-52.9	355.7	Dawson
GC37	1202790.99	3055724.22	7091.54	473	-65	50	Windy Gulch
GC39	1202795.28	3055713.56	7082.1	433	-65	0	Windy Gulch
GC40	1202422.53	3058013.04	6628.86	919	-55	332	Dawson
GC41	1202658.29	3055476.55	6999.79	332	-69	342	Windy Gulch
GC42	1202717.28	3058361.02	6564.72	851	-66	357	Dawson
GC43	1202425.57	3058009.36	6628.85	914	-60	324	Dawson
GC44	1202542.61	3058093.28	6599.23	919.5	-62	334.5	Dawson
GC45	1202797.56	3058407.65	6565.68	823	-70	357	Dawson
GC46	1202425.73	3058010.85	6628.71	992	-66	334	Dawson
GC47	1202405.02	3058442.5	6622.25	1338.1	-69	342	Dawson
GC48	1203097.66	3058310.89	6503.77	595	-58	305	Dawson
GC49	1202344.71	3057872.16	6656.92	1203.8	-70	318	Dawson
GC50	1202277.29	3056028.55	7249	975	-55	347	Windy Gulch
GC51	1202180.09	3058176.25	6679.8	1422	-66	341	Dawson
GC52	1202254.28	3055858.56	7182	903	-45	338	Windy Gulch
GC53	1202286.59	3057610.75	6715.5	1080.6	-62	336	Dawson
GC54	1202117.79	3057863.35	6721.8	1682.6	-75	345	Dawson
GC55	1202245.28	3055854.56	7182	927	-55	328	Windy Gulch
GC56	1202291.09	3057614.65	6713.1	315	-73	330	Dawson
GC56A	1202291.09	3057614.65	6713.1	1298	-75	331	Dawson
GC57	1202119.19	3057862.15	6721.4	307	-55	336	Dawson
GC57A	1202119.19	3057862.15	6721.4	1219.3	-56	335	Dawson
GC58	1202291.69	3057602.75	6715.9	993	-53.2	348	Dawson
GC59	1202259.29	3058000.75	6649.9	1237	-70	334	Dawson
GC60	1202112.99	3057862.65	6724	1579	-73.5	334	Dawson
GC61	1202336.69	3058153.75	6622.8	124	-60	343	Dawson
GC61A	1202336.69	3058153.75	6622.8	1131	-61.5	339	Dawson
GC62	1202051.79	3058290.65	6657.2	1809	-67.5	335	Dawson

*Sentinel Segment drilling included with Dawson Segment drilling

10.6.2 Drilling Results

US Borax drilling programs confirmed gold mineralization down-dip and along strike from known surface expressions of mineralization and old workings. Selected drilling highlights from the programs are listed below in Table 10-11.

Table 10-11: 1981–1985 US Borax Weighted Average Gold Highlights from the Dawson and Windy Gulch Segments

Hole ID	From (ft)	To (ft)	Interval (ft)	Estimated True Thickness (ft)	Au (g/t)	Segment***
GC05	345.0	380.0	35.0	21.5	8.64*	Dawson
Including	356.0	370.0	14.0	8.6	18.00*	Dawson
GC13	613.1	633.1	20.0	12.3	29.18*	Dawson
Including	613.1	618.1	5.0	3.1	27.95*	Dawson
Including	628.1	633.1	5.0	3.1	87.00*	Dawson
GC20	725.0	730.0	5.0	4.2	15.25*	Dawson
GC27	586.0	621.0	35.0	25.5	7.29*	Dawson
Including	611.0	621.0	10.0	7.3	14.38*	Dawson
GC37	68.0	74.0	6.0	4.3	8.54	Windy Gulch
And	197.0	200.0	3.0	2.4	38.60	Windy Gulch
GC40	784.0	814.0	30.0	30.0	22.53**	Dawson
Including	784.0	790.0	6.0	6.0	42.31**	Dawson
Including	793.0	799.0	6.0	6.0	36.19**	Dawson
Including	805.0	811.0	6.0	6.0	30.42**	Dawson
GC44	783.8	810.8	27.0	22.5	17.89*	Dawson
Including	804.8	807.8	3.0	2.5	114.51*	Dawson
GC51	1,315.0	1,321.0	6.0	4.2	21.35	Dawson

* Average of 2 US Borax Au (g/t) grade values (original and re-assay)

** Average of 3 Au grade values for pulp splits (US Borax original and re-assay, and Uranerz re-assay)

*** Sentinel Segment drilling is included with Dawson Segment drilling

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Preparation and Analysis

11.1.1 Zephyr 2016 Program Sample Preparation and Analysis

Zephyr staff cut NQ core in half using an electric rock saw. Half of the core was placed in a plastic sample bag and the remaining half-core was left in the core box with a corresponding sample tag number. The bagged core samples were placed in 5 gal plastic buckets for shipping by UPS Corporate in Cañon City, CO. The sample weight delivered to the laboratory ranged from 1.5 to 2.5 kg. Two analytical laboratories were used during this drill program: Bureau Veritas in Vancouver, BC, and ActLabs in Ancaster, ON. At both laboratories, the samples were subjected to standard rock preparation procedures that included jaw crushing, pulverizing; and splitting. Core samples from 13 drill holes were analyzed at Bureau Veritas, whereas samples from two drill holes were analyzed at ActLabs. All samples were initially analyzed by standard 1 assay/tonne FA method. All samples having greater that 1 g/t gold were then prepared using the screen metallic method and re-assayed. Methods used at the two laboratories are seen below. This information was extracted from the 2016 Bureau Veritas and ActLabs web-sites.

Bureau Veritas Commodities Canada Ltd.

Sample Preparation: PRP70-250, PRP70-500, PRP70-1000

Rock and drill core is crushed to 70% passing 10 mesh (2 mm), homogenized, riffle split (250 g, 500 g, or 1,000 g subsample), and pulverized to 85% passing 200 mesh (75 µm). Crusher and pulverizer are cleaned by brush and compressed air between routine samples. Granite/quartz wash scours equipment after high-grade samples, between changes in rock colour and at end of each file. Granite/quartz is crushed and pulverized as first sample in sequence and carried through to analysis.

Metallic Screen Fire Assay:

Prepared samples of 500 g samples are screened through 150 mesh (106 µm), screens producing two sample fractions for analysis. The plus fraction is analyzed in its entirety by FA with gravimetric finish and reported as +Au. The minus fraction is analyzed by FA with atomic absorption (AA) or inductively coupled plasma (ICP) finish either once or in duplicate at 30 or 50 g charge weight depending on client request and reported as -Au. If values exceed 10 ppm in the minus fraction, the minus fraction may also need to be analyzed with gravimetric finish. Gold values of both fractions are reported along with a total gold content of the sample. (Alternative screen sizes / weights are available upon request.)

Fire assay is performed by custom-blending samples with FA fluxes, PbO litharge and a silver inquart. Firing the charge at 1,050°C liberates Ag ± Au ± PGEs (platinum group elements) that report to the molten lead-metal phase. After cooling, the lead button is recovered, placed in a cupel and fired at 950°C to render a Ag ± Au ± PGEs doré bead. The bead is digested for ICP analysis or weighed and parted in ACS-grade nitric acid to dissolve silver leaving a gold sponge. Silver is weighed for gravimetric determination; ACS-grade hydrochloric acid is added dissolving the Au ± PGE sponge for instrument determination.

Calculations

For single minus fraction:

TotAu (ppm) = ((+Au x +Wt)+(-Au x –Wt)) / TotWt

- +Wt is the total weight of + fraction sample
- TotWt = total weight of sample sieved
- -Wt = TotWt - +Wt
- Above is calculated with both + and - Au in ppm if +Au is reported as mg then

+Au (ppm) = (-Au (mg) / +Wt)*1000

For duplicate minus fraction:

-Au Avg = (-Au(1) + -Au(2))/2

TotAu (ppm) = (+Au x +Wt)+(-Au Avg x –Wt) / TotWt

Fire Assay Method Description: FA 100, FA300, FA400, FA500

The prepared sample is custom-blended with FA fluxes, PbO litharge, and a silver inquart. Firing the charge at 1,050°C liberates silver, gold, and PGEs that report to the molten lead-metal phase. After cooling, the lead button is recovered, placed in a cupel and fired at 950°C to render a silver, gold, and PGEs doré bead. The bead is then either digested with nitric and hydrochloric acids for instrumentation determination or weighed and parted with nitric acid to dissolve silver leaving gold, which is weighed directly. Silver is determined by difference of the doré bead from the gold in gravimetric analysis.

Activation Laboratories Ltd.

Sample Preparation

As a routine practice with rock and core, the entire sample is crushed to a nominal -10 mesh (1.7 mm), mechanically split (riffle) to obtain a representative sample, and then pulverized to at least 95% -150 mesh (106 µm). All steel mills are now mild steel and do not induce Cr or Ni contamination. Quality of crushing and pulverization is routinely checked as part of the quality assurance program.

Fire Assay-ICP: (1A2-ICP 30 or 50)

A sample size of 5 to 50 g can be used, but the routine size is 30 g for rock pulps, soils, or sediments (exploration samples). The sample is mixed with FA fluxes (borax, soda ash, silica, litharge) and with silver added as a collector, and the mixture is placed in a fire clay crucible. The mixture is preheated at 850°C, intermediate 950°C, and finish 1,060°C, with the entire fusion process lasting 60 minutes. The crucibles are then removed from the assay furnace and the molten slag (lighter material) is carefully poured from the crucible into a mould, leaving a lead button at the base of the mould. The lead button is then placed in a preheated cupel which absorbs the lead when cupelled at 950°C to recover the Ag (doré bead) + Au.

Gold is separated from the silver in the doré bead by parting with nitric acid. The gold (roasting) flake remaining is weighed gravimetrically on a microbalance. Two splits on the -100 mesh fraction are weighted and analyzed by FA with a gravimetric finish. A final assay is calculated based on the weight of each separated fraction and obtained Au values.

Metallic Screen

A representative 500 g split (1,000 g for Code 1A4-1000) is sieved at 100 mesh (149 µm) with fire assays performed on the entire +100 mesh and two splits on the -100 mesh fraction. The total amount of sample and the +100 mesh and -100 mesh fraction is weighed for assay reconciliation. Measured amounts of cleaner sand are used between samples and saved to test for possible plating out of gold on the mill. Alternative sieving mesh sizes, are available but the user is warned that the finer the grind the more likelihood of gold loss by plating out on the mill.

11.1.2 Zephyr 2013 Program Sample Preparation and Analysis

Zephyr staff cut core samples in half and then quartered one side using an electric rock saw. One-quarter core sample was placed in a plastic sample bag and the remaining three-quarters of the core was left in the core box with a corresponding sample tag. Bagged core samples were shipped by Fedex Ltd. from Cañon City to ALS Minerals (ALS) in Elko, Nevada. The sample weight delivered to the laboratory ranged from 1.5 to 2.5 kg. Upon arrival, samples were subjected to standard rock preparation procedures that included jaw crushing, pulverizing and splitting. Samples were then shipped by ALS to its lab in Reno, Nevada, for analysis.

Each quarter core sample was pulverized to 85% <0.075 mm, passed through a +0.1 mm screen, and analyzed by screen metallic gold methods (ALS code Au-SCR21). The screen metallic gold analysis involved the following procedures: 1) all +0.1 mm screen material was analyzed in its entirety by fire assay gravimetric method (FA-GRA) method and reported as the +fraction assay result; 2) the -0.1 mm material that passed through the sieve was homogenized and two sub-samples were analyzed by fire assay atomic adsorption (FA-AA) method (ALS codes Au-AA25 and Au-AA25D), these assays were averaged and reported as the –fraction result; and 3) the + and - fractions were weight averaged to yield a final screen metallic result (head grade) for the sample. The screened metallics method was used to provide results that more accurately reflect the grade of the drill core by capturing all coarse gold in the sample. A comparison of screened metallic results shows the 2013 screened metallic head grade results falling within the confines of the Windy Gulch Segment resource solids are on average approximately 9% higher than corresponding -fraction results.

11.1.3 Historical

US Borax, Jascan, and Uranerz reports documenting the company drilling programs do not provide detailed descriptions for sample preparation methods, analytical procedures, or security considerations. Reports indicate that all US Borax laboratory work was conducted internally by US Borax laboratories which would not have been accredited or registered to ISO standards. The 2013 site visit identified that US Borax core was split in half using a manual core splitter. Conventional core sample preparation procedures are assumed to have been used and report documents show that gold analysis was by conventional FA pre-concentration followed by FA-GRA, or to a lesser extent, conventional FA pre-concentration followed by FA-AA, or both. US Borax also conducted an extensive re-assaying program in 1984 on key drilling intercepts within the Dawson Segment and Windy Gulch Segment deposits, in which a larger sample size and smaller mesh size were used to create analytical pulps (Hambrick and Theye, 1985).

Historical ACA Howe reports show that Jascan assay and check assay laboratory work was carried out by various independent commercial laboratories. These included Hazen Research Inc. in Golden, CO; Bondar-Clegg and Company Ltd. of Ottawa, ON (later acquired by ALS Chemex); Skyline Laboratories (likely Skyline Assayers and Laboratories based in Tucson, AZ, and acquired by Actlabs in 1997), and Acme Analytical Laboratories Ltd. based in Vancouver, BC. The 2013 site visit showed that Jascan core was split in half using a manual core splitter. And it is believed that conventional core sample preparation procedures were used followed by gold analysis by conventional FA pre-concentration methods followed by instrumental or GRA finish, but reports do not specify the finish technique.

Uranerz documentation indicates that all its laboratory work was conducted by Bondar-Clegg Laboratories. Half core samples were split using a manual core splitter and prepared by conventional procedures. Conventional core sample preparation procedures are assumed to have been used and gold analysis was by conventional FA pre-concentration followed by either GRA or AA finish. Some samples having visible gold were processed using a screen metallic protocol.

Based on the above, Mercator is of the opinion the sample preparation and analysis methods used by US Borax, Jascan, and Uranerz were consistent with industry standards of their respective periods and therefore acceptable for current use. It is also noted that Hazen Research, Bondar-Clegg, Skyline and Acme Laboratories were well known, fully independent commercial entities that have operated under full spectrums of internal QA/QC protocols. US Borax was and remains a large corporation that is assumed to have provided internal analytical services at a level of reliability comparable to that available through the contemporary commercial laboratories.

11.2 Quality Control and Quality Assurance Programs (QA/QC)

11.2.1 General

Drill core sampling carried out by Zephyr during the 2016 drilling program on the Windy Gulch Segment was subject to a QA/QC protocol under supervision of independent consultant Mark Graves. The program included systematic insertion of blank samples, certified reference materials (CRMs), and the placement of analytical replicates. Non-mineralized pink granite was obtained locally for use as blank sample material for the 2016 program. All samples for the 2016 program were analyzed by Bureaus Veritas and ActLabs for gold. Core samples were first analyzed by standard one-assay tonne FA. Any samples exceeding 1 g/t were then analyzed by the screen metallic method.

A total of 335 samples were analyzed for gold in the 2016 Zephyr program. Data for the 2016 Zephyr replicate sampling program appear in Figure 11.1, and results show that intra-sample analytical heterogeneity with respect to gold grade is not an issue at Windy Gulch.

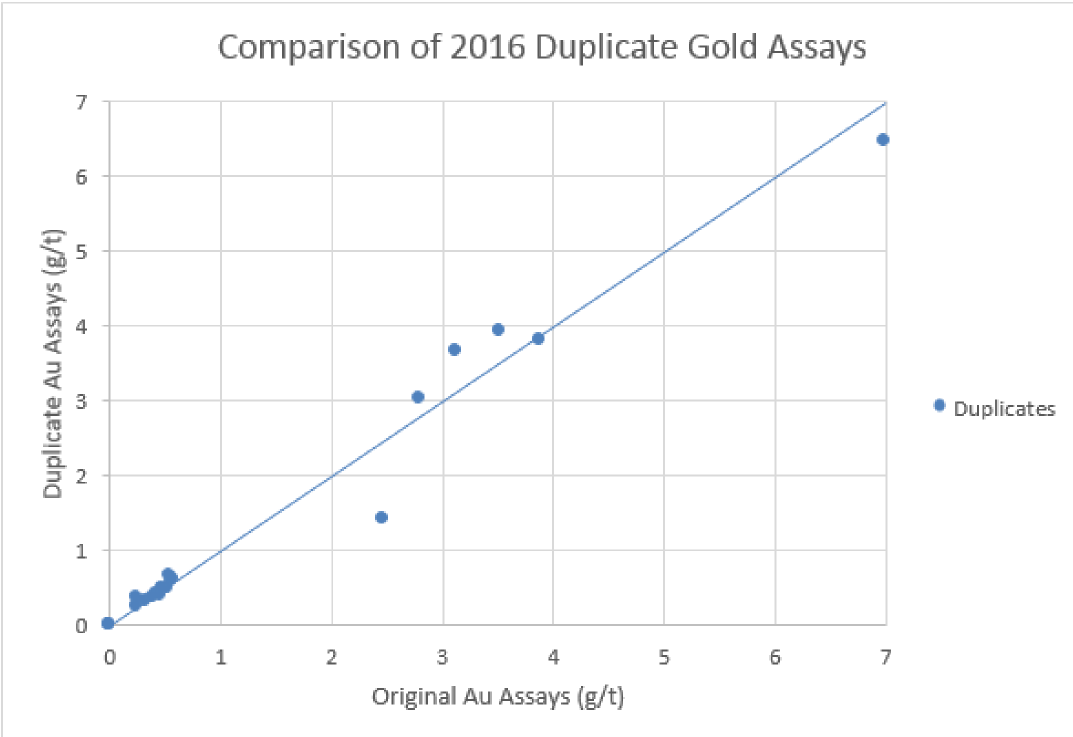


Figure 11.1: Zephyr 2016 Replicate Sampling Program (n=40)

Certified Reference Material Program

CRMs were obtained from CDN Resource Laboratories Ltd. of Delta, BC. Two CRMs were used during the 2016 Zephyr program. These were the same two standards as were used in the 2013 Zephyr drill program. Table 11-1 presents the certified mean values reported for these two materials.

Table 11-1: Certified Reference Materials Data for 2013

Certified Material	Certified Au Value (g/t)
CDN-GS-1L	1.16 ±0.10
CDN-GS-5L	4.74 ±0.22

CRM samples were received in 100 g paper packets which were split to 50 g packets and then individually placed in a plastic sample bag labelled per the normal core/rock sample numbering scheme. In total, results for 25 CRMs submitted for analysis in 2016 were reviewed for the purposes of this technical report. Reference samples were inserted into the laboratory sample shipment sequence at a rate of approximately one in every 13 samples.

Figures 11.2 and 11.3 present CRM results for the 2016 drilling program, and in both cases data consistently fall within reasonable analytical tolerances for the certified material.

Windy Gulch 2016 QAQC CDN-GS-5L

Standard CDN-GS-5L - Au

Mean	4.703 g/t Au	Expected Mean	4.740 g/t Au
Standard Devn	0.070 g/t Au sq	Expected Std Dev	0.110 g/t Au sq
Counts	11	% Bias (-ve when underestimated)	-0.79 %
Minimum	4.560 g/t Au	No of Outlier +/- 3 Std Dev	0
Maximum	4.790 g/t Au	% Outside Tolerance	0.00 %
Median	4.720 g/t Au	CV	0.01 %
Average HRD%	-0.40 %	Average HARD	0.67 %

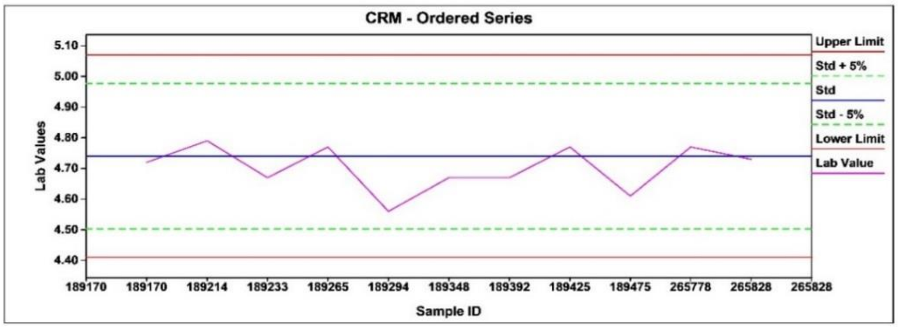


Figure 11.2: Certified Reference Standard CDN-GS-5L for 2016 Zephyr Program

Windy Gulch 2016 QAQC CDN-GS-1L

Standard CDN-GS-1L - Au

Mean	1.195 g/t Au	Expected Mean	1.160 g/t Au
Standard Devn	0.030 g/t Au sq	Expected Std Dev	0.050 g/t Au sq
Counts	14	% Bias (-ve when underestimated)	3.02 %
Minimum	1.150 g/t Au	No of Outlier +/- 3 Std Dev	0
Maximum	1.260 g/t Au	% Outside Tolerance	0.00 %
Median	1.195 g/t Au	CV	0.03 %
Average HRD%	1.47 %	Average HARD	1.53 %

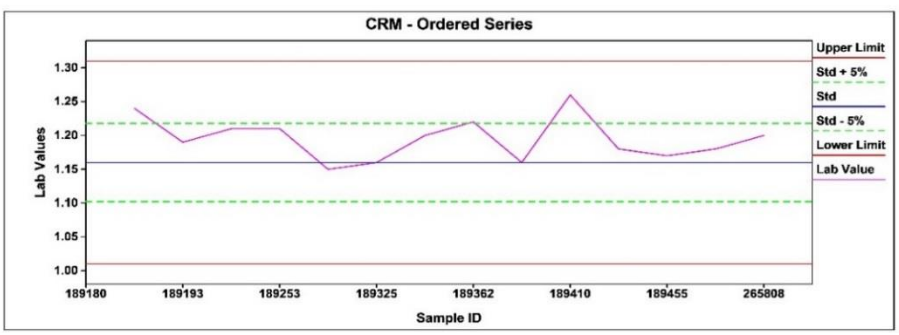


Figure 11.3: Certified Reference Standard CDN-GS-1L for 2016 Zephyr Program

11.2.3 Blank Sampling Program

Gold analytical results were reviewed for 15 blank samples submitted for analysis during the 2016 drilling program. Figure 11.4 presents these results. All blank samples returned gold values less than the 0.05 g/t detection limit for the analytical method except for a single sample which assayed 6 ppb, thereby indicating that significant sample cross-contamination did not occur.

Windy Gulch 2016 QAQC Blanks

Standard Blank - Au

Mean	0.003 g/t Au	Expected Mean	0.003 g/t Au
Standard Devn	0.001 g/t Au sq	Expected Std Dev	0.000 g/t Au sq
Counts	15	% Bias (-ve when underestimated)	9.33 %
Minimum	0.002 g/t Au	No of Outlier +/- 2 Std Dev	0
Maximum	0.006 g/t Au	% Outside Tolerance	0.00 %
Median	0.002 g/t Au	CV	0.32 %
Average HRD%	2.75 %	Average HARD	2.75 %

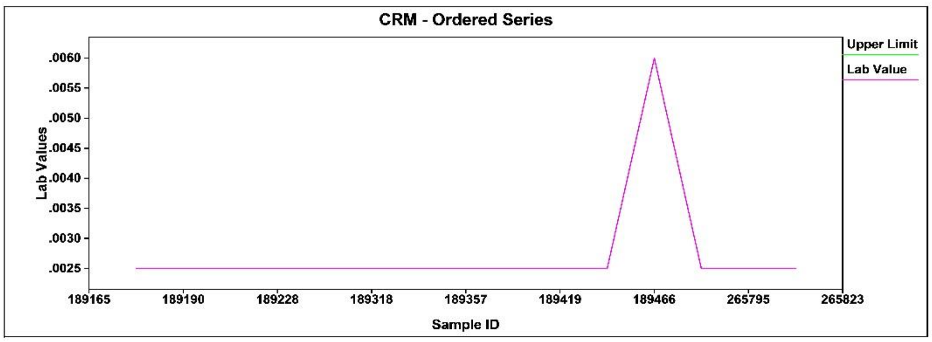


Figure 11.4: Blank Sampling Analyses for the 2016 Zephyr Program

11.3 Zephyr 2013 Windy Gulch Segment Program

11.3.1 General

Drill core sampling carried out by Zephyr during the 2013 drilling program on the Windy Gulch Segment was subject to a QA/QC protocol under supervision of independent consultant Mark Graves. The program included systematic insertion of blank samples and CRMs but did not include third party commercial laboratory check sampling. Non-mineralized pink granite was obtained locally for use as blank sample material for the 2013 program. All samples for the 2013 program were analyzed by ALS for gold by the screen metallic method.

A total of 448 samples (including 29 trench and pit samples) were analyzed for gold by the screen metallic method during the 2013 Zephyr program. A comparison between corresponding final screen metallic results and (-) fraction results appears in Figure 11.5 and results show that the impact of coarse gold is restricted to only a few samples a minimal on the results.

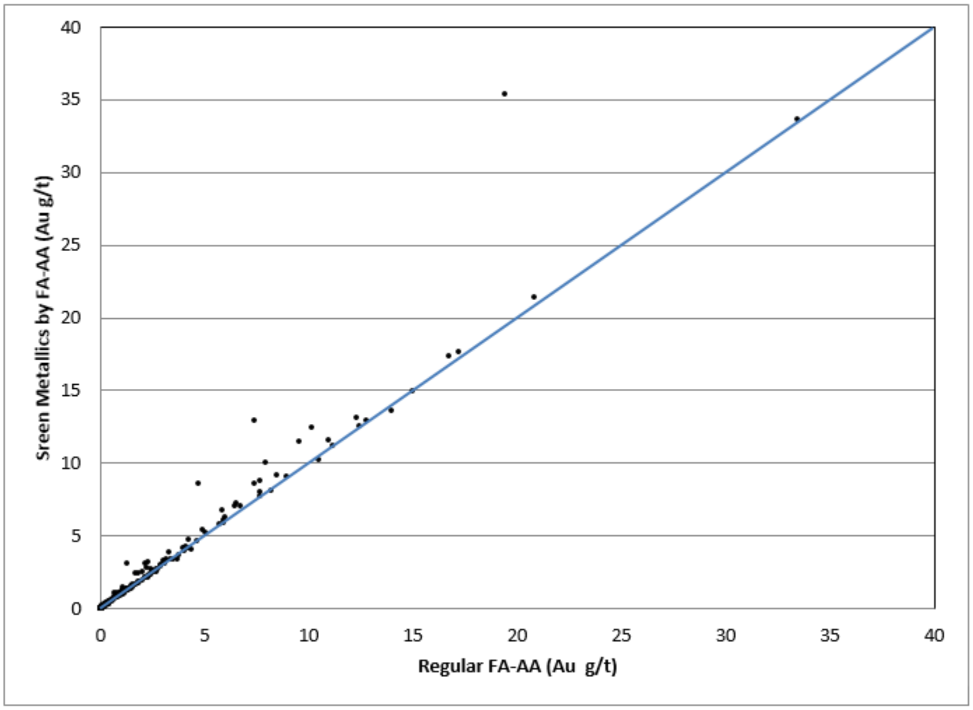


Figure 11.5: Final Screen Metallic Result by FA-AA vs. (-) Fraction Regular FA-AA Result (N=448)

The screen metallic (-) fraction result is the average of two pulp splits analyzed by FA-AA methods. The original and duplicate split results can be compared to assess precision, and a comparison of pulp split values from both methods appears in Figure 11.6 and results show an acceptable level of correlation along the 1:1 trend line.

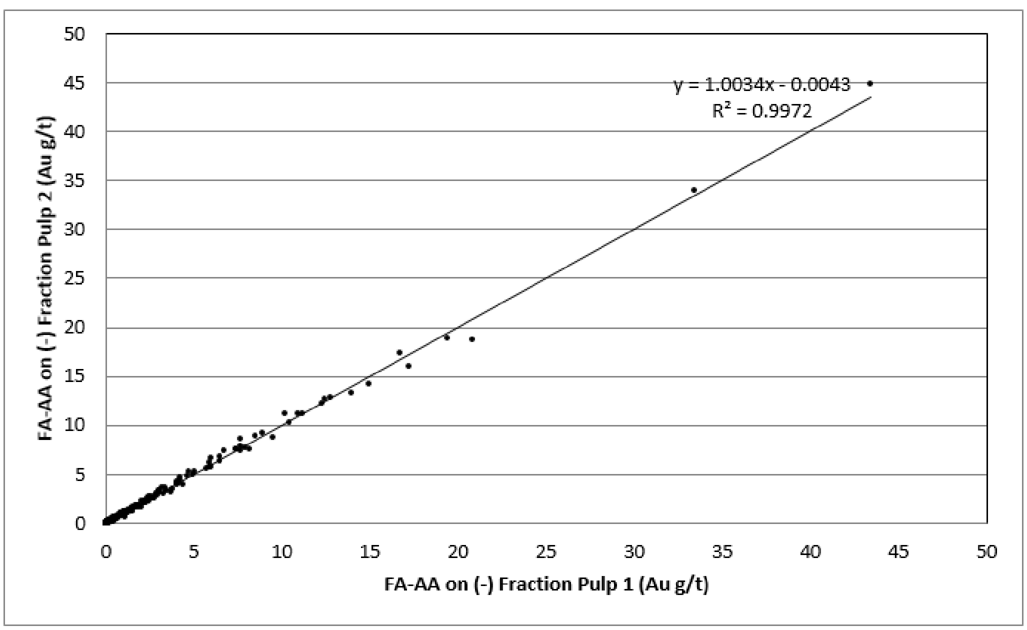


Figure 11.6: Comparison of Screen Metallic (-) Fraction Pulp Split Results (N=448)

11.3.2 Certified Reference Material (CRM) Program

Two CRMs were used during the 2013 Zephyr program. These were the same two standards as were used in the 2016 Zephyr drill program. Table 11-1 presents the certified mean values reported for the two analytical CDN-GS-1L and CDN-GS-5L.

CRM samples were received in 100 g paper packets, and were individually placed in a plastic sample bag labelled per the normal core/rock sample numbering scheme. In total, results for 22 certified reference samples submitted for analysis in 2013 were reviewed. Reference samples were inserted into the laboratory sample shipment sequence at a rate of approximately one in every 25 samples.

Figures 11.7 and 11.8 present CRM results for the 2013 drilling program and in both cases data consistently fall within the ±2 standard deviation control limits for the project.

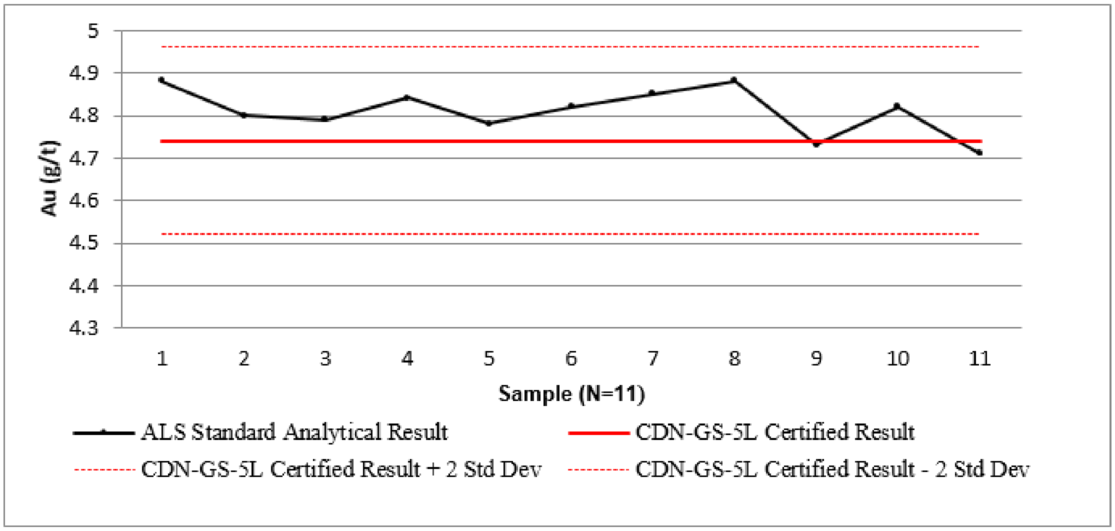


Figure 11.7: Certified Reference Standard CDN-GS-5L for 2013 Zephyr Program

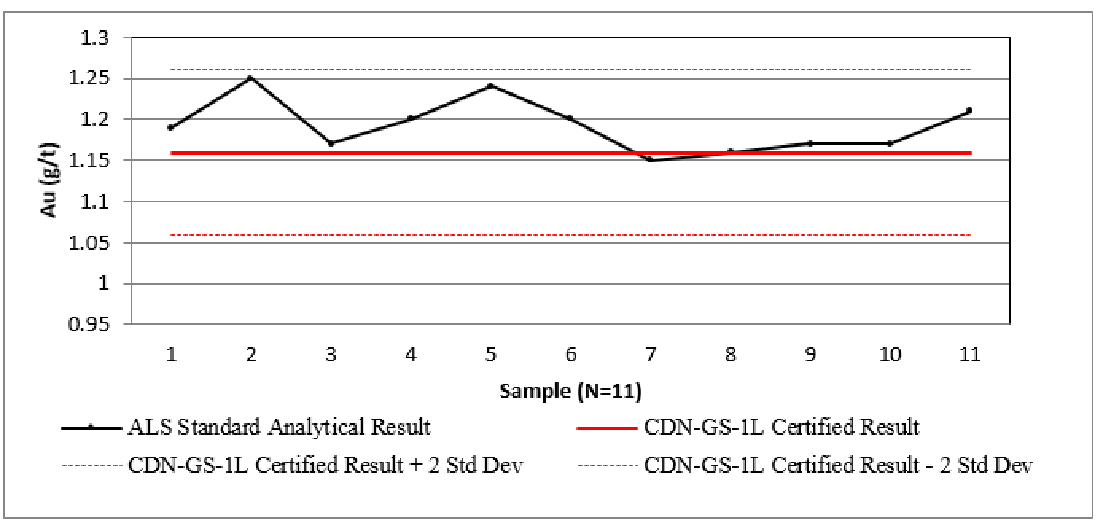


Figure 11.8: Certified Reference Standard CDN-GS-1L for 2013 Zephyr Program

11.3.3 Blank Sampling Program

Mercator reviewed gold analytical results for 25 blank samples submitted for analysis during the 2013 drilling program. Figure 11.9 presents these results. All blank samples returned gold values less than the 0.05 g/t detection limit for the analytical method and thereby indicate that sample cross-contamination did not occur.

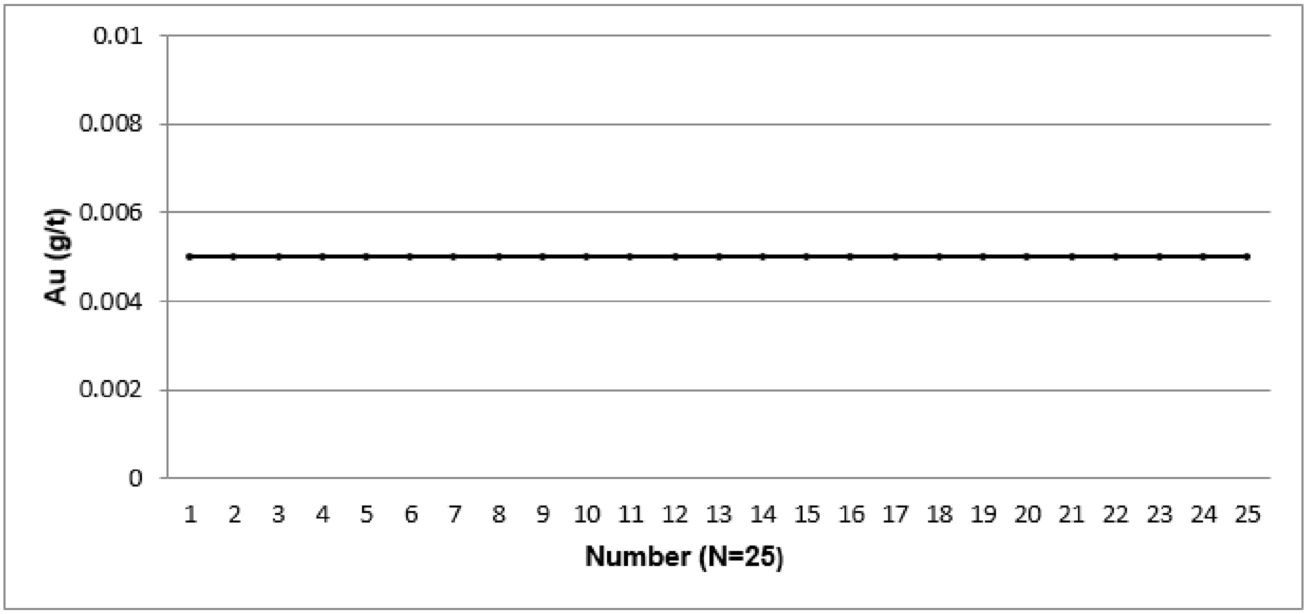


Figure 11.9: 2013 Zephyr Blank Sampling Program

11.4 US Borax (1981–1986), Jascan (1987–1988), and Uranerz (1991)

Hard copy reports describing US Borax, Jascan, and Uranerz drilling programs do not specifically address QA/QC issues. No evidence was noted of independent certified standards being inserted for any program, nor is there any evidence of systematic submission of blank samples or systematic provisions for duplicate sample splits to be prepared and analyzed. In Mercator's experience, this situation is not unusual for exploration programs of the period, which frequently relied upon internal laboratory QA/QC programs to ensure quality of data received.

11.4.1 1984 US Borax Re-assaying Program

Mercator compiled the results of 595 US Borax re-assay (check) samples from key drilling intercepts within 24 holes in the Dawson Segment and Windy Gulch Segment areas. When referring to re-assay sampling procedures, the original report only mentions that a larger sample of smaller mesh material was used (Hambrick and Theye, 1985). All re-assay samples were analyzed by FA-GRA. Original and re-assay data sets show generally good correlation, except for 47 re-assay results from hole GC37. All results are presented in Figure 11.10. Re-assay sample intervals for GC37 that collectively do not correspond with those in the original log are considered problematic. An outlier re-assay sample of 229 g/t Au in GC44 was returned for an original sample result of

<0.2 g/t Au, and review of the original drill log showed that visible gold had been identified in the original core. On that basis, the difference between the re-assay result and the original gold value has been attributed to "nugget effect."

Based on the above comparisons, Mercator concluded that the sample preparation and analysis methods used by US Borax, Jascan, and Uranerz were consistent with industry standards of their respective periods and, as reviewed, have been accepted for current use.

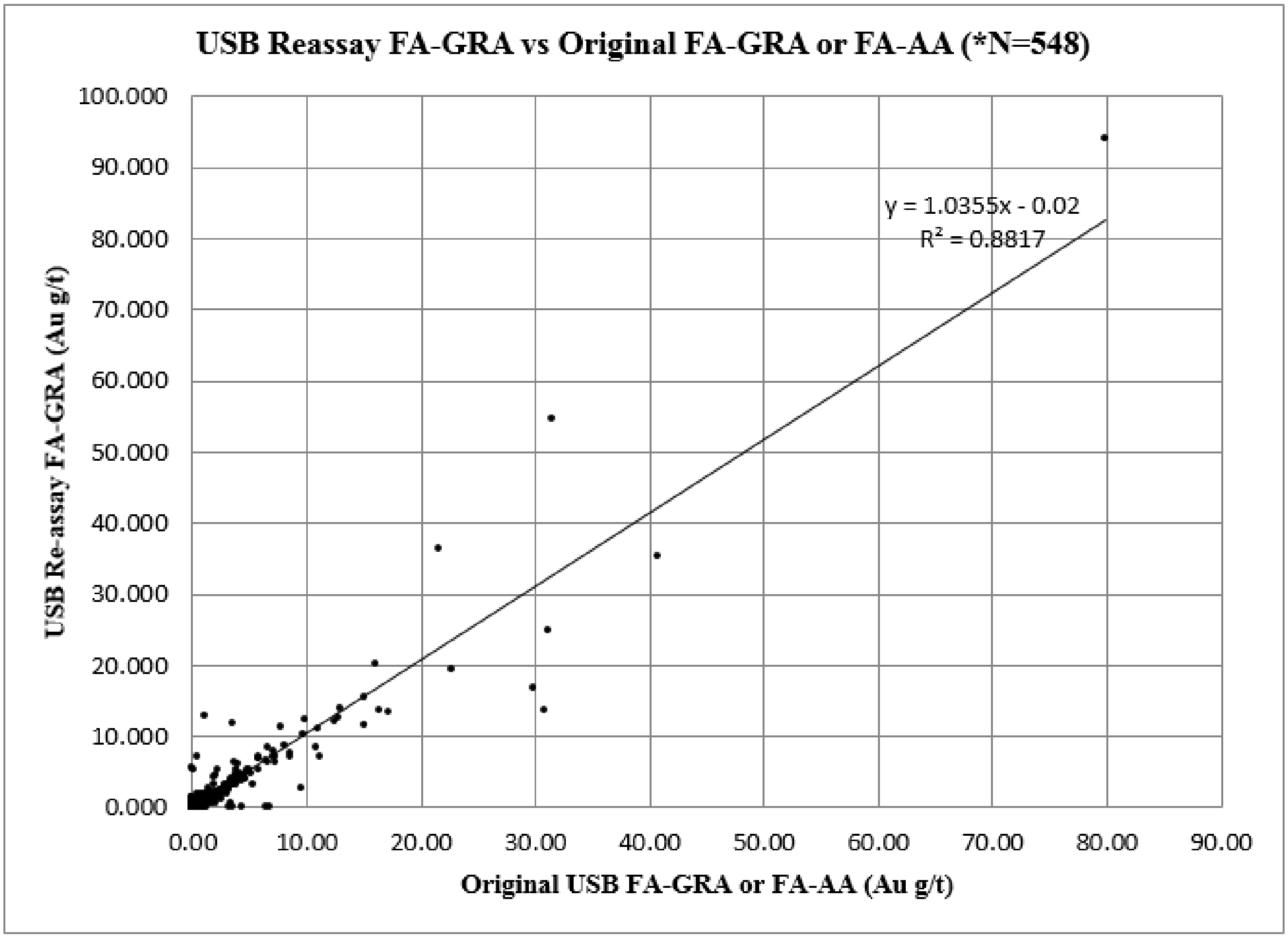


Figure 11.10: Comparison of Original US Borax Results to Re-assay Results

Note: Re-assay data from GC37 and 229 g/t and 95 g/t outliers from GC44 and GC40 not included.

11.4.2 1987–1988 Jascan Check Assay Pulp Programs

Mercator also compiled results of 387 Jascan check assays carried out on pulp splits collected from US Borax hole GC49 and from 29 Jascan holes that fall within drill hole series GC63 to GC98 and MS01 to MS07. Check assays were analyzed by one or multiple laboratories. All check assay data sets were graphed and show good correlation with grouping along a 1:1 trend line. Figure 11.11 compares a subset of gold check assay values belonging to the first and second analytical runs on the same sample pulp by the same lab (Bondar-Clegg Laboratories) on selected holes within the drill hole series GC79 to GC98, and MS02 to MS07. The strong grouping along the 1:1 trend line for these data illustrate the character of the check sample correlation. Jascan check assay results also provide additional but limited third party checks on the earlier US Borax assaying.

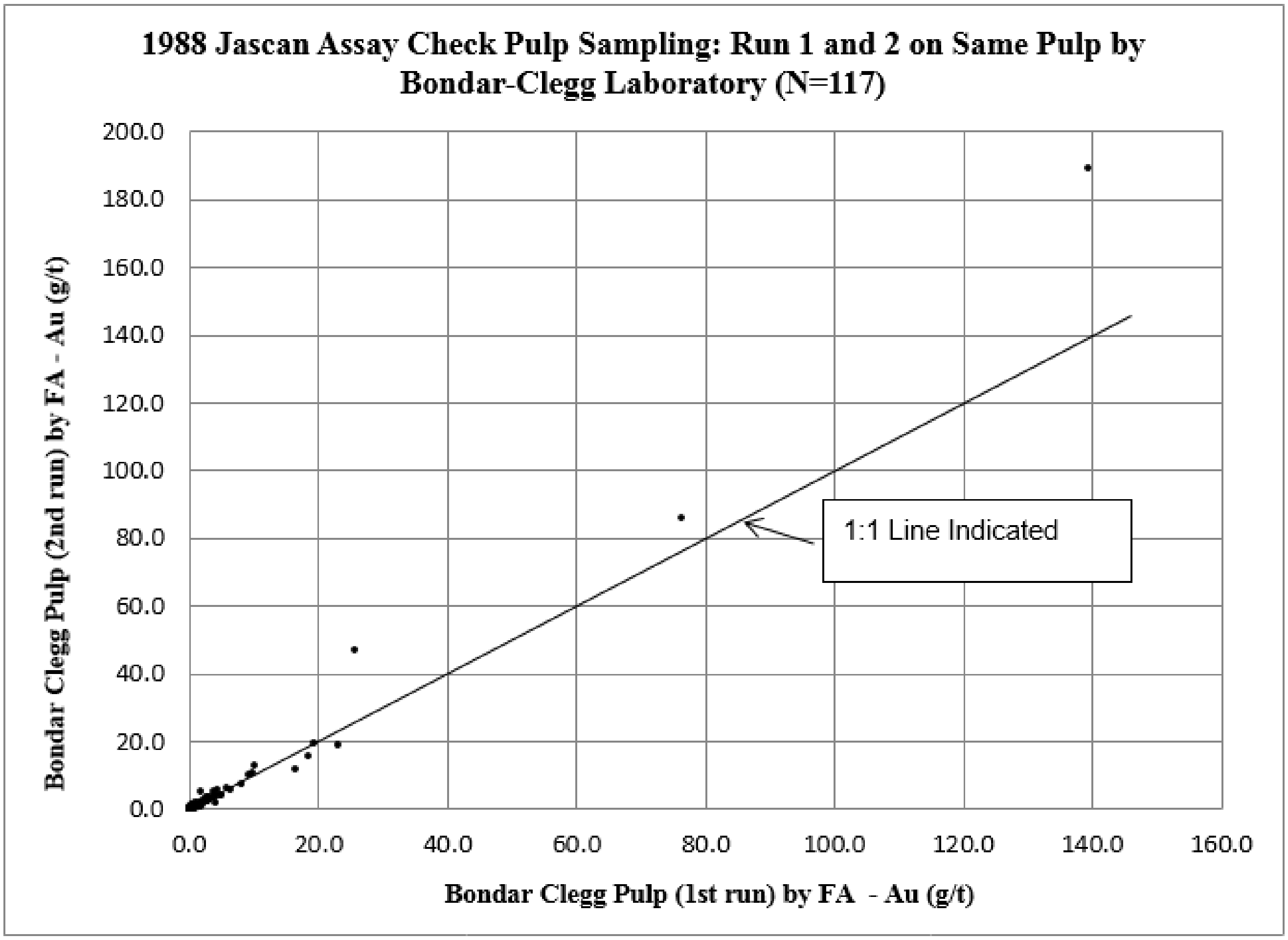


Figure 11.11: 1988 Jascan Check Assay Pulp Program – Subset of Data from Holes GC79 to GC98 and MS02-MS07)

11.5 Security

11.5.1 Zephyr 2016

Drill core was placed into NQ waxed cardboard boxes by Godbe Drilling LLC staff at the drill. The core was retrieved by Zephyr personnel at various times during the day, brought down the mountain by 4X4 truck, and transported to the core facility at 402 Valley Road, Cañon City, CO, for processing. Logging and sampling procedures were conducted by Mark Graves and included use of a pre-numbered sample tag system. The three-tag system included insertion of a sample tag in the sample box at the corresponding sample interval, insertion of a tag in the pre-numbered sample bag, and a third tag retained in the sample book for archive purposes. All core sample material was placed in a plastic bag and sealed for shipment. All core and samples were stored under supervision of Zephyr staff at the company's secure logging facility until shipment to the analytical laboratory.

Summary drill log descriptions for each hole were initially prepared and core sample intervals were marked on core for subsequent cutting. Sample records and drill logs were recorded in a notebook during logging and were then transferred to Microsoft Excel digital spreadsheets to facilitate data handling. Core was continuously sampled through the Windy Gulch Zone with two border samples collected at the hanging wall and footwall contacts. All core was split by electric diamond saw under the supervision Zephyr staff, and half-core samples were shipped by UPS Corporate, Cañon City, CO, for sample preparation. Sample handling, reduction, splitting and analysis for drill holes WG-16-21 to WG-16-25 and WG-16-28 to WG-16-36 were performed by Bureau Veritas Commodities Canada Ltd. These samples were shipped from Cañon City, CO, to a holding depot in Blaine, WA, for pick-up and transfer by Bureau Veritas to its laboratory in Vancouver, BC. Sample handling, reduction, splitting and analysis for drill holes WG-16-26 to WG-16-27 was performed by Actlabs in Ancaster, ON.

11.5.1.1 Comment on 2016 QA/QC Program Results

Sample preparation, analysis, and security practices used for the 2016 drilling program by Zephyr are compatible with current industry standards and are identical to the coreshed procedures performed during the 2013 drill program. The 2016 program also used the same blank material and analytical standards. Replicate sampling was conducted in 2016, which was not done for the 2013 core sampling.

11.5.2 Zephyr 2013

Drill core was placed into HQ waxed, cardboard boxes by Godbe Drilling LLC staff and brought down the mountain at the end of each shift. Zephyr personnel then transported the core by truck to the core facility at 402 Valley Road, Cañon City, CO, for processing. Logging and sampling procedures were carried out by co-author Graves and included use of a pre-numbered sample tag system. The three-tag system included insertion of a sample tag in the sample box at the corresponding sample interval, insertion of a tag in the pre-numbered sample bag, and a third tag retained in the sample book for archive purposes. All core sample material was placed in a plastic bag and sealed for shipment. All core and samples were stored under supervision of Zephyr staff at the company's secure logging facility until shipment to ALS.

Summary drill log descriptions for each hole were initially prepared and core sample intervals were marked on core for subsequent cutting. Sample records and drill logs were recorded in a notebook during logging and were then transferred to Microsoft Excel digital spreadsheets to facilitate data handling. Core was continuously sampled throughout drill holes until footwall sequence gneisses were intersected. All core was split by electric diamond saw under the supervision Zephyr staff and quarter core samples were shipped by Fedex Ltd. from Cañon City to ALS Minerals in Elko, NV, for sample preparation. Following transport by Fedex to the analytical laboratory, all sample handling, reduction, splitting and analysis were performed by ALS Minerals.

11.5.2.1 Comment on 2013 QA/QC Program Results

Mercator is of the opinion that sample preparation, analysis, and security methodologies employed during the 2013 drilling program by Zephyr are consistent with current industry standards and sufficient for this project; however, Mercator recommends that Zephyr incorporate an expanded QA/QC protocol in any future drilling programs. This should include systematic inclusion and analysis of duplicate pulp and reject splits as well as a drill core check sampling program with analysis at an independent accredited laboratory.

11.5.3 US Borax/Jascan/Uranerz

Original company reports do not specify details of security, sample handling, tagging, or shipping protocols. Bound hard copy drill logs reports were reviewed to identify core logging and sampling procedures and to determine whether evidence of security procedures was also present. Hard copy logs provided detailed information related to lithology, alteration, and mineralization, which was systematically recorded in the logs along with complete records of core sampling and analytical results, but no specific references to security were identified. For present purposes, it is assumed that security procedures consistent with industry standards of the respective periods would have been employed by US Borax, Jascan, and Uranerz. This assumption was independently confirmed with respect to work programs carried out for Jascan by ACA Howe Ltd. through discussion with Mr. W. Felderhof who in 2013 was a technical advisor to Zephyr. Mr. Felderhof was President and CEO of Jascan during the period in question and familiar with field protocols used at that time by A.C.A Howe.

12.0 DATA VERIFICATION

12.1 Golder 2016

Golder completed data verification as part of the process of completing this mineral resource estimate for the Windy Gulch segment. Data verification consisted of a site visit by the resource QP to the project site, verification of drill hole collar locations, review of logging and sampling procedures, review of sample chain of custody, collection of independent samples for metal verification, and a partial comparison of the drill hole assay database against original assay certificates.

12.1.1 Database Verification

Golder compared 921 Windy Point gold assays (of a possible total of 952) from the Zephyr database to the original Bondar Clegg assay certificates. Results of the comparison are summarized in Table 12-1. One minor error was identified and corrected resulting in a change from 0.008 oz/tn to 0.003 oz/tn. Thirty-one certificate assays were not available for hole DWP9002 but database values were low grade ranging from 0.002 oz/tn to 0.007 oz/tn with one value grading 0.033 oz/tn.

Golder verified the most relevant Windy Gulch samples that were contained within the main mineralized envelope. A total of 248 samples from the 2013 drill program (all samples) and 69 samples with grades greater than 0.0292 oz/tn (1 g/t) from the 2016 drill program (all screened metallic samples) were compared to the assay certificates from Bureau Veritas, Actlabs and ALS Minerals. Of the combined 317 samples, four issues were identified in the

2016 data where the original fire assay values were not replaced with the screen metallic values. This issue is not considered to be significant as the assay values were quite similar between assay methods. It is recommended that the database be updated with the screen metallic values prior to the next model iteration.

Table 12-1: Drill Hole Sample Data Validation

Segment	No. of Holes	No. of Assays	No. of Issues
Windy Point	16	921	1
Windy Gulch	35	317	4

12.1.2 Site Visit

A site visit to the Dawson Project was carried out by Brian Thomas, P.Geo., between August 2 and 3, 2016. The site visit was led by Mark Graves, P.Geo., the contract project geologist for the 2016 Windy Gulch drill program. The site visit consisted of the following:

- verification sampling and confirmation logging of Windy Point and Windy Gulch core samples
- confirmation of Windy Gulch drill hole collar locations
- observation of the current drill set-up
- review of the project geology and inspection of available rock outcrops
- review of drilling, sampling, analytical, and QA/QC procedures
- review of chain of custody for drill core and samples
- data hand-off
- review of core shack and project site security

Diamond drilling was ongoing at time of the visit by Godbe Drilling LLC. Figure 12.1 illustrates the drill set-up for hole WG-16-34.

Figure 12.1: Godbe Drill Set-Up of Hole WG-16-34

Figure 12.2 highlights an area of the Windy Gulch Zone at the upper road cut which has been mapped and sampled by Monte Swan and Mark Graves.

Figure 12.2: Windy Gulch Zone at the Upper Road Cut

Golder verified the location of three 2016 Windy Gulch drill hole collars while on site. All collar coordinates were found to closely match the Zephyr surveyed coordinates, generally within the accuracy of the GPS readings (±10 ft). A comparison of the collar coordinates is summarized in the Table 12-2. All coordinates are stated in Colorado State Plane, NAD 83 ft.

Table 12-2: Validation Check of Drill Collars

Hole Number	Source	Northing (ft)	Easting (ft)	Elevation (ft)
WG16-26	Zephyr	1202938	3055941	7,208
	Golder	1202948	3055937	7,211
WG16-27	Zephyr	1202978	3055945	7,205
	Golder	1202976	3055941	7,209
WG16-30	Zephyr	1203078	3056157	7,185
	Golder	1203080	3056150	7,191

Notes:
1) Golder coordinates measured using a handheld Garmin GPS with expected accuracy of approximately 10 ft.
2) All coordinates are reported in Colorado State Plane Central Zone, NAD_83

Golder did not visit the Windy Point Segment as the site has been rehabilitated and the historical workings are no longer accessible.

12.1.2.1 Confirmation Logging and Independent Sampling

Golder compared the drill logs to available core for selected Windy Point and Windy Gulch drill core intervals. The core matched the logged descriptions well and no notable differences were identified. Figures 12.3 and 12.4 provide examples of the Windy Point and Windy Gulch drill core available during the site visit. Figure 12.3 is a photograph of core from hole DWP9002 highlighting the interval between 101 ft to 106 ft (between intervals of scotch tape) that was included in the verification sampling.

Figure 12.3: Drill Core from Windy Point Hole DWP9002

Figure 12.4 illustrates the nature of the Windy Gulch Zone as seen in hole WG16-27.

Figure 12.4: Drill Core from Windy Gulch Hole WG-16-27

As part of the sample verification program, seven core samples and one standard reference sample were collected and transported back to Sudbury, ON, Canada where they were analyzed for gold by Actlabs using 50 g fire assay with gravity finish. One certified reference standard was submitted to Actlabs as a control on accuracy. The Actlabs laboratory in Sudbury is certified ISO 17025.

Five samples were taken for Windy Point and two were taken for Windy Gulch from quartered BX and NQ core. A summary of the selected samples and a comparison of assays is summarized in Table 12-3.

Table 12-3: Comparison of Verification Sample Results

Segment	Hole ID	Golder Sample ID	From (ft)	To (ft)	Length (ft)	Golder Au (oz/ton)	Zephyr Au (oz/ton)	Core Size
Windy Point	DWP9002	1310113	101	106	5	3.821	1.024	BX
Windy Point	DWP9003	1310114	66	71	5	0.009	1.674	BX
Windy Point	DWP9003	1310115	101	106	5	0.032	0.128	BX
Windy Point	DWP9004	1310116	11	17	6	0.187	0.344	BX
Windy Point	DWP9004	1310117	191	196	5	0.090	0.117	BX
Windy Gulch	WG-16-27	1310119	101.5	104.5	3	1.167	1.718	NQ
Windy Gulch	WG-16-26	1310120	73.5	77	3.5	0.057	0.493	NQ
Standard	CON-GS	1310118	na	na	na	0.130	0.138	na

Notes:
1) Golder check assay values that differ significantly from original Zephyr data are shown in red.

The Golder check assays are in general agreement with the original Zephyr data with a couple of notable exceptions as highlighted in red in Table 12-3 where Golder sample 1310113 is much higher than Zephyr results and samples 1310114 and 1310120 are much lower than expected. Golder re-assayed samples 1310114 and 1310120 from coarse reject with no significant difference in results. Differences may be attributed to the natural variability of the mineralization (nugget effect), the smaller sample size of the quartered core, or differing analytical techniques where, in the case of Windy Gulch samples, Zephyr has used a screened metallic process to better represent the coarse gold size fragment. Golder is satisfied that the results are within reasonable tolerances when considered in the context of the deposit geology and style of mineralization. Figure 12.5 provides a graphical comparison of the check assay results.

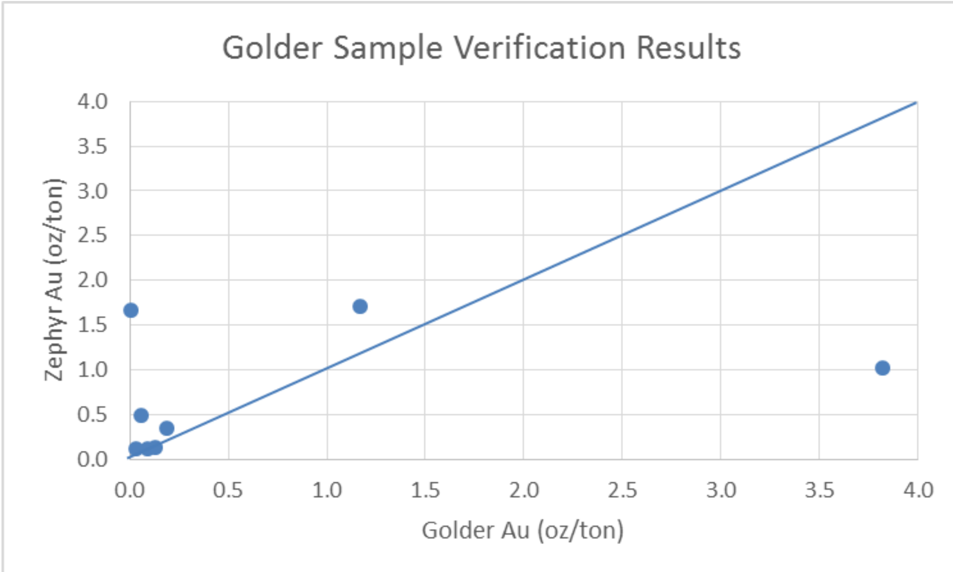


Figure 12.5: Scatter Plot of Windy Gulch and Windy Point Assay Comparisons

On completion of the data validation, site visit, and verification sampling, Golder has concluded that the drill hole and sample data are of suitable quality to support an Indicated or Inferred mineral resource estimate.

12.2 2013 Mercator, Graves

12.2.1 Review and Validation of Project Data Sets

Drill core sample records, lithological logs, available laboratory assay results, and associated drill hole information for all historical drill programs completed by US Borax, Jascan, and Uranerz were made available to Mercator in digital database and supporting hard copy format. Historical hard copy internal company reports, loose files, and correspondence were reviewed by Mercator. After initial spot checking of the database supplied by Zephyr against original source documents, it was determined that a comprehensive review of the digital data set for drill holes belonging to the Dawson Segment and Windy Gulch Segment should be completed. Mercator completed a detailed review, which consisted of checking all assay database entries against source documents consisting of 1) hard copy drill logs, 2) summary reports, or 3) laboratory documents. Collar coordinates and downhole surveys were checked against summary tables also found in hard copy reports. Database entries were corrected where mistakes were noted, and data were added where missing.

No original assay laboratory certificates exist for US Borax holes GC03 through to GC62 or Jascan holes GC63 to GC98 and MS01 to MS07, but assay results had been recorded on original drill logs and in summary assay tables in bound company reports and had been used in previous historical resource work. As part of the validation procedure, US Borax and Jascan holes were visually inspected on a sectional basis in Surpac, and assay grades and lithologies were found to correlate acceptably with drilling results for holes completed later by Uranerz with accompanying laboratory analytical records.

Core recovery results were also entered into the drilling database by Mercator and evaluated over mineralized intervals within the resource model.

Zephyr provided 2013 Zephyr drilling results in digital format, and Mercator compiled digital assay results from laboratory spreadsheets and checked 10% of entries to original values found in corresponding laboratory certificates supplied by ALS. Survey coordinates were compiled directly into the Surpac database from the digital files provided by Cornerstone Land Surveying of Cañon City, CO.

Manual review and checking of logging and sample records showed generally good agreement between original records and digital database values for all data sets. No historical or current drill holes were excluded from the resource drilling database due to lack of supporting records, but several items attributed to data entry errors were identified and appropriate changes were made in the database.

After completion of all manual record checking procedures, the drilling and sampling database records were further assessed through digital error identification methods available through the Surpac modelling software. This provided a check on items such as sample record duplications, end of hole errors, survey and collar file inconsistencies, and certain potential lithocode file errors. The digital review and import of the manually checked data sets through Surpac provided a validated Microsoft Access database that Mercator considers to be acceptable with respect to support of the resource estimation program described in this report.

12.2.2 2013 Independent Sampling

Mark Graves carried-out a site visit and an independent drill core check sampling program for historical drill core from the Dawson Segment deposit between March 25 and March 31, 2013. Discussions regarding the Dawson Segment deposit were held between Mr. Graves and Mercator at that time and drill core from representative holes covering the Dawson Segment deposit area were identified and transported from the storage hanger at the Fremont County Airport to the Zephyr core facility in Cañon City where they were inspected and then sampled. Many of the drill holes originally selected for check sampling could not be located and it is currently unclear if the hanger contains all drill holes belonging to the Property.

During inspection, it was noted that previous workers did not denote the location of their sample intervals in the core box and respective check assay intervals serve as best approximations measured from preserved downhole depth markers. Selected check assay intervals all showed good recovery except in one check assay (GC98) where the check assay interval was incomplete due to previous sampling. A total of 17 quarter core samples were collected from 4 Jascan holes (GC68, GC71A, GC75, and GC98), 3 Uranerz holes (DA9005, DA9006, and DA9007) and 1 US Borax hole (GC44). Visible gold was identified in hole GC98 at down hole depth of 554 ft. Representative core photographs in a pre-sampled state appear in Figures 12.6, 12.7, and 12.8 and the Fremont County Airport storage hanger is shown in Figure 12.9.

Figure 12.6: Felsic Biotite Gneiss in US Borax Hole GC44 (check sample 783.8–786.8 ft)

Figure 12.7: Felsic Biotite Gneiss in Uranerz Hole DA9007 (check sample 978.0–981.0 ft)

Figure 12.8: Felsic Gneiss in Jascan Hole GC98 (check sample 552.0–555.0 ft)

Figure 12.9: Fremont County Airport Storage Hanger – Core Boxes Wrapped in Plastic

Samples were analyzed by ALS in Reno, Nevada, an ISO 9001:2008 Certified, accredited, independent, commercial laboratory for screen metallic gold analysis (code Au-SCR21). One CRM sample (CDN-GS-5L) and one granite blank were included in the sample shipment. The blank granite sample returned a gold value of 0.1 g/t and was processed after a high grade gneiss sample that yielded 76.2 g/t Au. The elevated blank sample result may indicate the presence of low level contamination at the sample preparation stage of processing. The certified reference material returned a result within 2 standard deviations of the certified value. Figure 12.10 presents results of the 2013 independent check sampling program and these are interpreted as generally confirming historical gold grade levels in the respective sample sets.

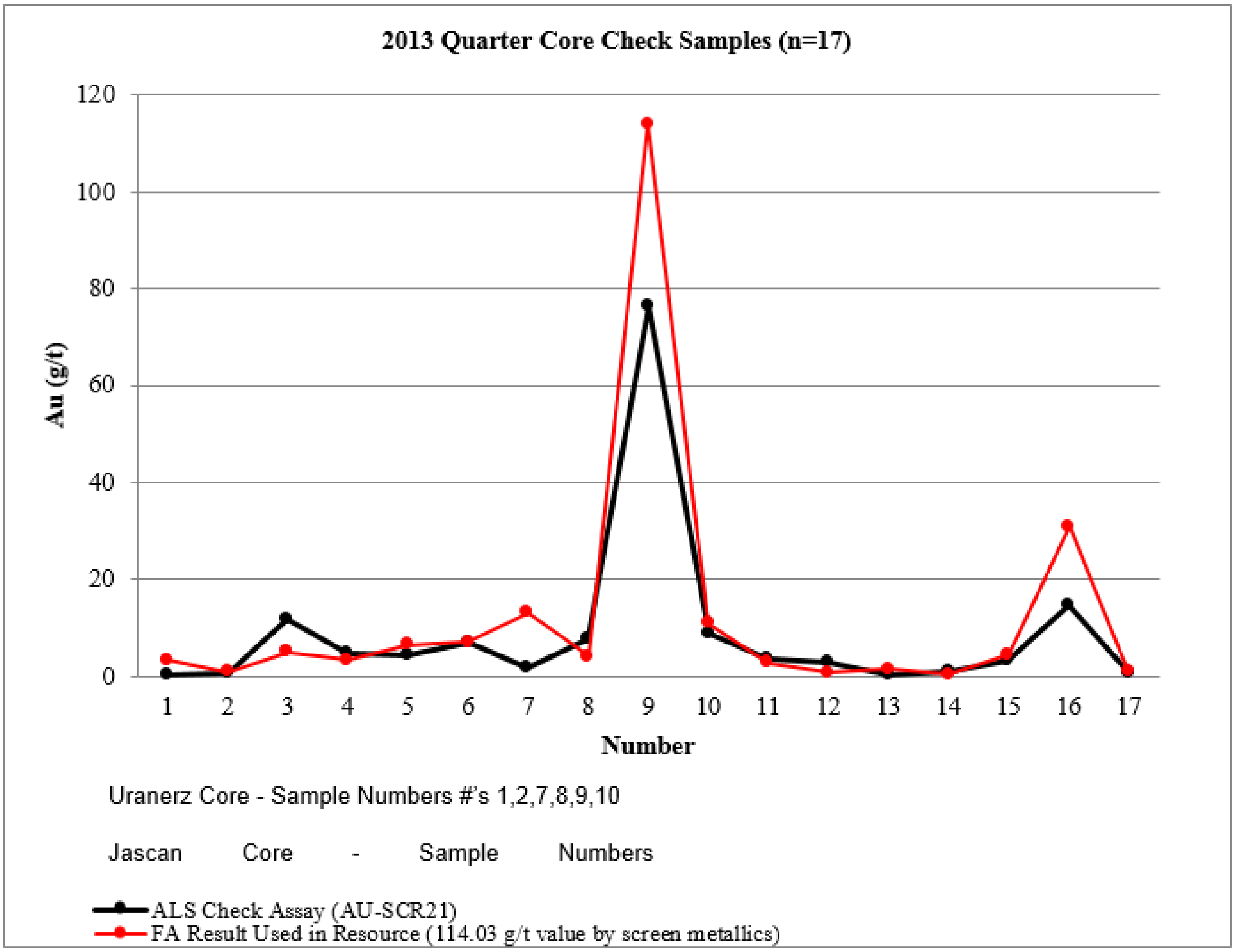


Figure 12.10: 2013 Check Samples for the Dawson Segment

Note: If multiple historical results were available for a given sample, they were averaged to generate the “original” assay result used in the resource database.

12.2.3 2013 Site Visit by M. Graves, P. Geo.

Mark Graves visited the Windy Gulch Segment area of the Property on a discontinuous basis between February 11 and May 12, 2013, while coordinating the 2013 Zephyr drilling program. General observations regarding the character of the landscape, vegetation, site elevations, surface drainage, road access, previous drill sites, and surface geology were made during the site visit. The historical Windy Gulch drilling access roads were rehabilitated and locally extended and three outcrop exposures of the prospective gold bearing zone were identified and one was chip sampled. Methods and results of the 2013 drilling and sampling are described in Section 9 and Section 10 and site visit and photographs appear in Figures 12.11, 12.12, 12.13, 12.14, 12.15, and 12.16.

The western-most historical Mike Sutton adit was opened and inspected. A historical underground casing was located (from Jascan MS series holes) during the site visit, and it was noted that road building to drill sites WG-13-08 and WG-13-09 exposed a BX sized drill casing near the understood location of Uranerz surface drill hole WG9105. However, the inclination of the casing is approximately -75° and directed to the north-northwest, which more closely matches the orientation of Uranerz drill hole WG9107, which is otherwise understood to lie approximately 50 ft to the east-northeast of the exposed casing. This discrepancy in drill collar location/survey information relative to surface cultural features such as roads and trails requires further investigation and exemplifies the need for acquisition of a well-constrained survey tie-in to UTM NAD 83 coordination.

Figure 12.11: April 2013 Drilling Underway on WG-13-18 Set-Up

Figure 12.12: Windy Gulch Zone Exhalite Horizon Exposed in Road-Cut (looking north)

Figure 12.13: Westernmost Adit of Mike Sutton Historical Workings

Figure 12.14: Uranerz BX Sized Drill Casing Exposed in Road-Cut

Figure 12.15: Location of 2013 Zephyr Core Logging Facility in Cañon City, CO

Figure 12.16: Zephyr Geo-Technician Sawing HQ-Sized Core

Based on observations made during the 2013 site visit, analysis of check sampling program results, and review of 2011 site visit and check sampling program results reported by earlier for Celtic, Mark Graves and Mercator have determined that, to the extent reviewed during these visits, ample evidence exists of the previous exploration programs carried out on the Dawson Segment and Windy Gulch Segment deposits and also of those carried out by Zephyr. Results of site visit check sampling, identification of historical workings and collars, core review, and checking of historical logging information against archived core and records for 2013 site visit purposes are considered acceptable, and no substantive inconsistencies have been recognized with respect to records of past work or associated results.

The authors of this technical report are not qualified to provide an opinion with respect to environmental conditions, potential hazards, or liabilities that may be present on the Dawson Property. However, during the course of site visit and drilling supervision programs, co-author Graves observed that various historical underground workings areas are present on the Property and that some of these are characterized by rusty-weathering waste rock piles. These piles may represent potential sources of acid rock drainage under certain surface drainage conditions. Additionally, hazards appear to exist locally with respect to open historical mine workings and pits.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

This section was prepared by H. Matt Bolu, P.Eng., of BOMENCO at the request of Zephyr management. The two most recent metallurgical test programs conducted on samples from the Dawson Gold Project in Colorado as reported by their respective laboratories, are titled as follows:

- "Characterization and Beneficiation Studies on Dawson Free Gold Bearing Ore", Hazen Research Inc. Project 008-464, February 15, 1991—("Hazen")
- "Metallurgical Testing of Samples from the Zephyr Minerals Ltd., Dawson Gold Project", Bureau Veritas Commodities Canada Ltd., Inspectorate Metallurgical Division Project No.: 1404310, February 2014 ("Inspectorate")

13.2 Test Program Objectives and Scope

The main objective of the 1991 Hazen test program was to investigate the response of each of the two composite samples from the Dawson Zone to selected gold beneficiation techniques. Additionally, a flotation gold concentrate, generated from gravity tailings, was subjected to a cyanidation study. The 2015 Inspectorate study in the meantime was undertaken by Zephyr and focused on gravity and flotation response of a composite from the Windy Gulch Zone. The overall objective of both test programs was to develop a precious metal recovery process from the samples submitted in sufficient detail and scope suitable for further project evaluation.

13.3 Identification of Test Samples

Zephyr has identified two high grade gold mineralization zones at the Dawson Project site: the Dawson Segment and the smaller Windy Gulch Segment. Zephyr reports that 10% to 20% of the resource may be contained within the Windy Gulch Zone, while the Dawson Zone contains the remaining. Therefore, the Dawson Zone is the more significant of the two gold zones identified, as it constitutes potentially greater than 80% of the total estimated gold content and resource tonnage. While Windy Gulch contains lower resource tonnage—although at similar gold grades—than the Dawson Zone, it could potentially be considered for initial open mining in order to generate early cash flow.

Two samples identified as "HCO UCC Altered Composite" ("altered") and "HCO UCC Unaltered Composite" ("unaltered") were received and tested at Hazen, while a sample identified as Composite 1 was received and tested at Inspectorate.

Origins of the drill hole intervals that formed the composites used in these two tests programs as provided by Zephyr are as follows:

Hazen test samples: Figure 13.1 shows a longitudinal section of the Dawson segment of the deposit on which Hazen metallurgical test work samples are based. The unaltered holes are circled in orange and the altered holes are circled in green. Hole 106 (part of the altered sample) is the thickest gold zone of the project (8 g/t over 21 m); all the other samples are around hole 106. These samples probably cover about 30% of the deposit and are representative of about 80% to 90% of the Dawson Segment. The other 10% to 20% is different (around holes 12 and 5) in that the samples have higher sulphide and copper content.

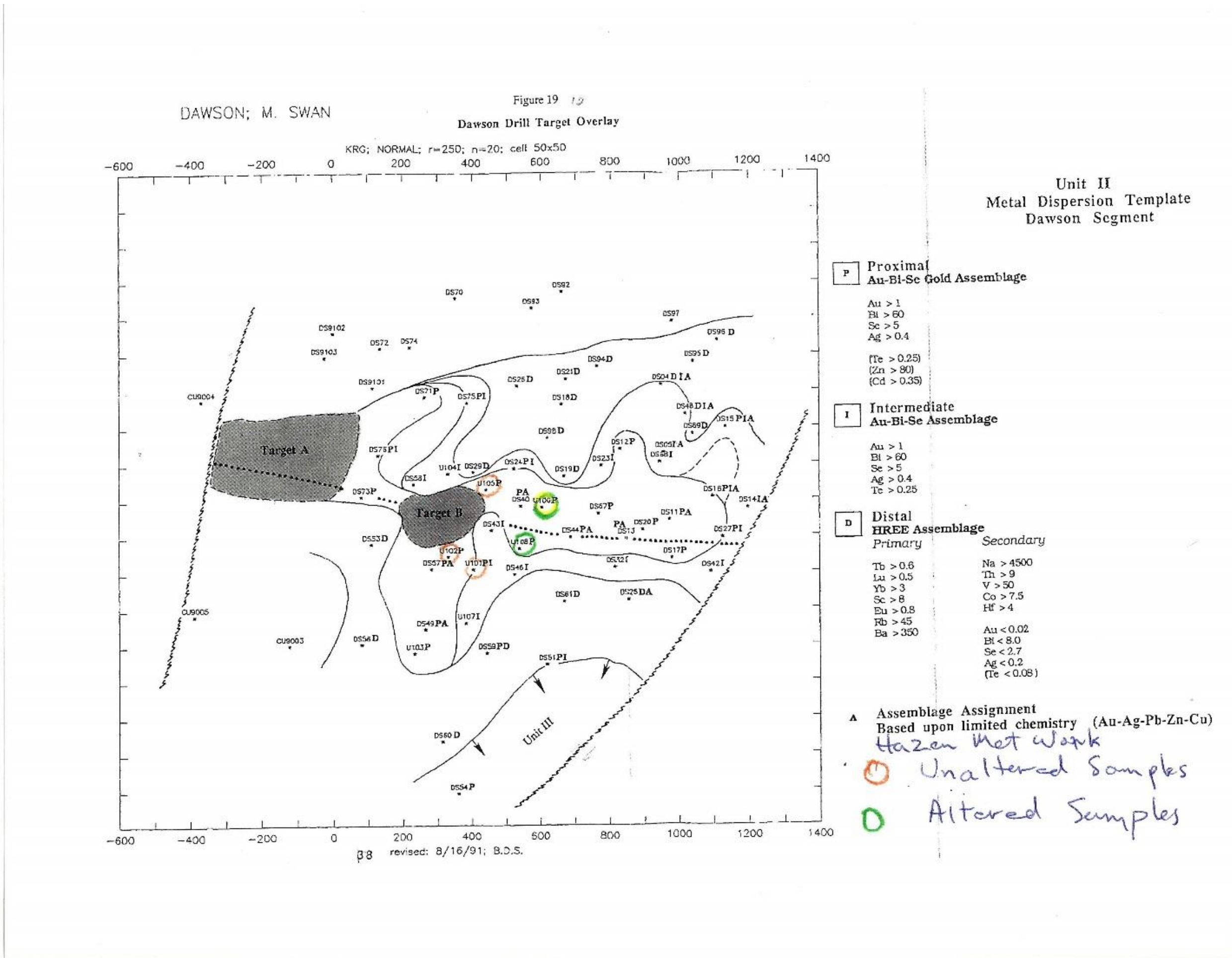


Figure 13.1: Metal Diversion Template – Dawson Segment

<u>Inspectorate test samples:</u> Figure 13.2 is a plan showing drill holes which provided the samples for the Inspectorate test program. All samples are from the Windy Gulch Segment of the deposit and are from within 98.4 ft of the surface, and almost all are altered (oxidized). These samples will probably have higher copper content than most of the Dawson Segment because a portion of the Windy Gulch samples contain an oxidized sulphide horizon. Approximately 75% of the Inspectorate test samples are from the northeast portion of Windy Gulch, while 25% are from the southwest portion. The met sample should be representative of the near surface resource at Windy Gulch, which is roughly 70% of the current resource at Windy Gulch.

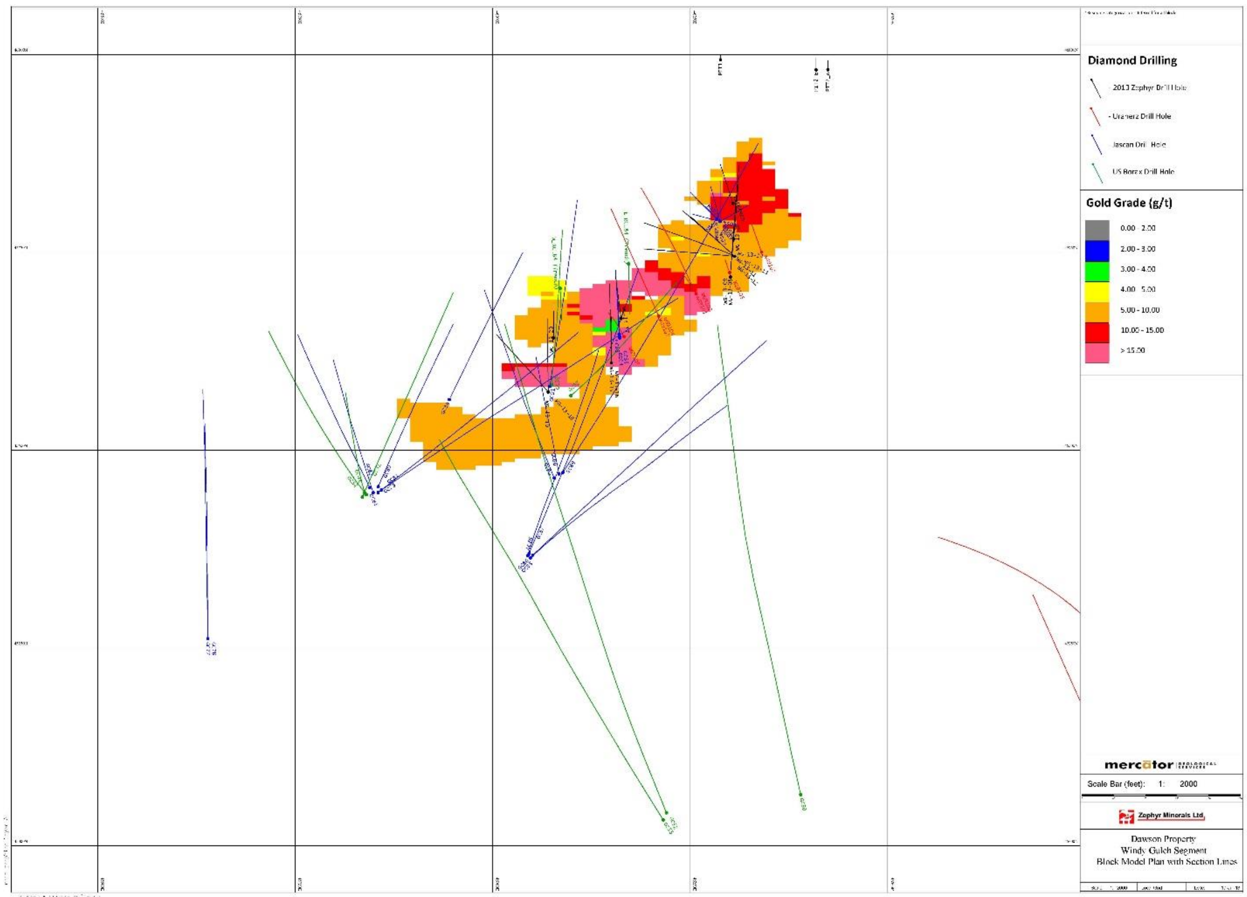


Figure 13.2: Section showing Block Model with Drill intercepts

Table 13-1 presents is a list of elements of prime interest and their direct analysis and calculated assays of the composite samples used in these two test programs by their respective laboratories.

Table 13-1: Calculated and Analysed Composite Samples

Element		Unit	Dawson				Windy Gulch	
			Comp: Altered		Comp: Unaltered		Comp: 1	
			Calculated	Analyzed	Calculated	Analyzed	Calculated	Analyzed
Gold	Au	g/t	10.1	10.7	11.5	16.9	6.80	8.26
Silver	Ag	g/t	2.2	1.0	1.3	1.4	3.7	4.2
Sulphur	S	%	n/a	n/a	n/a	n/a	0.47	0.51
Carbon (total)	C(t)	%	n/a	n/a	n/a	n/a	n/a	<0.02

Element		Unit	Dawson				Windy Gulch	
			Comp: Altered		Comp: Unaltered		Comp: 1	
			Calculated	Analyzed	Calculated	Analyzed	Calculated	Analyzed
Carbon (graphitic)	C(g)	%	n/a	n/a	n/a	n/a	n/a	<0.02
Mercury	Hg	ppm	n/a	n/a	n/a	n/a	n/a	0.03
Arsenic	As	ppm	n/a	n/a	n/a	n/a	n/a	<5

Dawson: The “altered” sample calculated head assays average from all tests was 10.1 g/t Au, and the analyzed direct head assay was 10.7 g/t Au. This compares with 10.1 g/t calculated from the drill hole assays. Silver assays were 2.2 g/t and 1.0 g/t for calculated and direct analysis, respectively. The “unaltered” sample calculated head assays from all tests was 11.5 g/t Au, and the analyzed direct head assay was 16.9 g/t Au. This compares with 18.2 g/t calculated from the drill hole assays. There were no impurity analyses conducted for the heads or concentrates for the Dawson samples in this test program.

Windy Gulch: As the analyses show, gold assayed 6.80 g/t and 8.26 g/t as average of calculated heads from all tests and direct analysis, respectively. This compares with 7.77 g/t gold calculated from the drill hole assays. Silver values were 3.7 g/t and 4.2 g/t as average of calculated heads from all tests and direct analysis, also respectively. Total and graphitic carbon were both less than 0.02%. Impurity analysis including mercury and arsenic in the composite showed no major cause for concern.

13.4 Comminution Tests

Samples from Windy Gulch were tested for grind hardness using the Bond Ball Mill Work Index test, and the results were as follows: Wi =17.2 kWh/tonne. This indicates a medium-high hardness.

13.5 Gravity Tests

Hazen test work: Preliminary gravity concentration tests were conducted to investigate the potential for gold recovery as a function of grind were conducted on both the "altered" and "unaltered" composites. Gravity concentration tests were conducted on both composites using a laboratory size shaking table for roughing and a Gemeni table for cleaning at nominal primary grind sizes of 80% passing 129 µm down to 47 µm. Gravity concentrate gold grade vs. recovery relationship for both composite samples are shown in Figure 13.3 below.

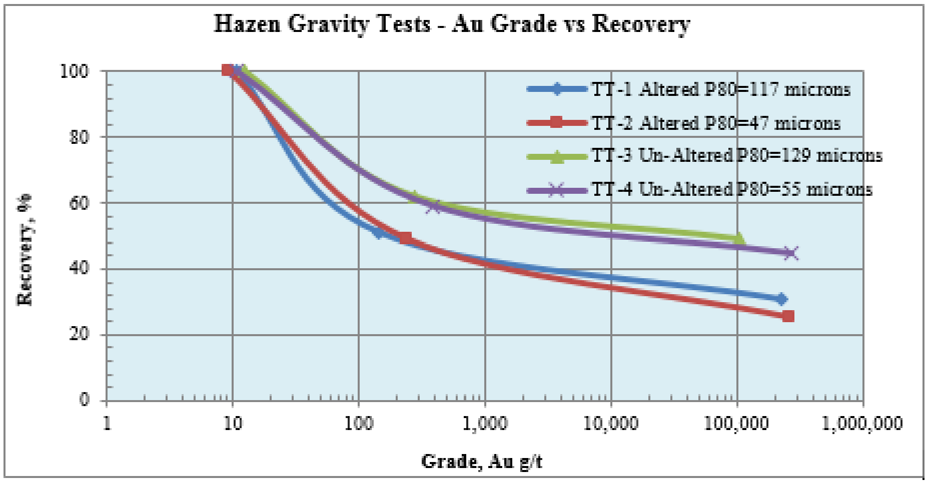


Figure 13.3: Hazen Gravity Tests

The results for the "altered" material (Tests 1 and 2) show that approximately 50% of the gold values were recovered to the shaking table concentrate, with grind size having no significant effect in the size ranges tested. Comparably, results for the "unaltered" sample (Tests 3 and 4) show gold recoveries to the concentrate to be about 60%. More importantly, the data show that 31% of the gold values for the "altered" and 49% for the "unaltered" materials were recovered to very high-grade free gold concentrates grading between ~6,500 oz/tn and 3,062 oz/tn, respectively, in Tests 1 and 3. Shaking table concentrate mass pull rates varied between 1.68% and 4.9% of feed mass ion the four tests.

As the results in the above chart demonstrates, there exists a high metallurgical and economic potential for gravity recovery of gold in these two Dawson samples, at relatively coarse grind sizes, and that gravity separation should be considered as part of the process development for Dawson.

<u>Inspectorate test work:</u> Preliminary gravity concentration tests were conducted using a centrifugal gravity concentrator to investigate the potential for gold recovery from the Windy Gulch composite samples at primary grind size ranges of P80=74 µm (G1) and P80=110 µm (GF2). A double pass through the centrifugal concentrator was followed by one upgrading stage on each primary gravity concentrate using panning. The results of the two tests, as graphically presented in Figure 13.4 below, showed that the first pan concentrates recovered 24% (GF2) and 31% (G1) of the gold into products assaying 2,113 g/t and 2,930 g/t respectively.

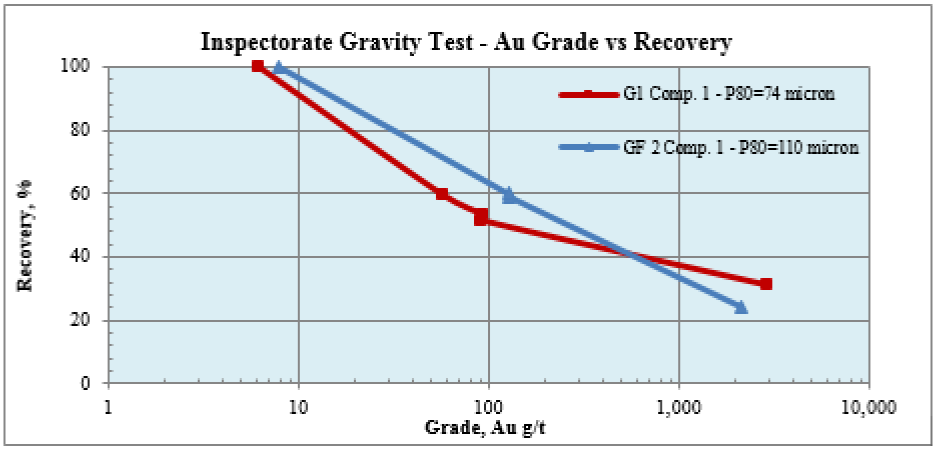


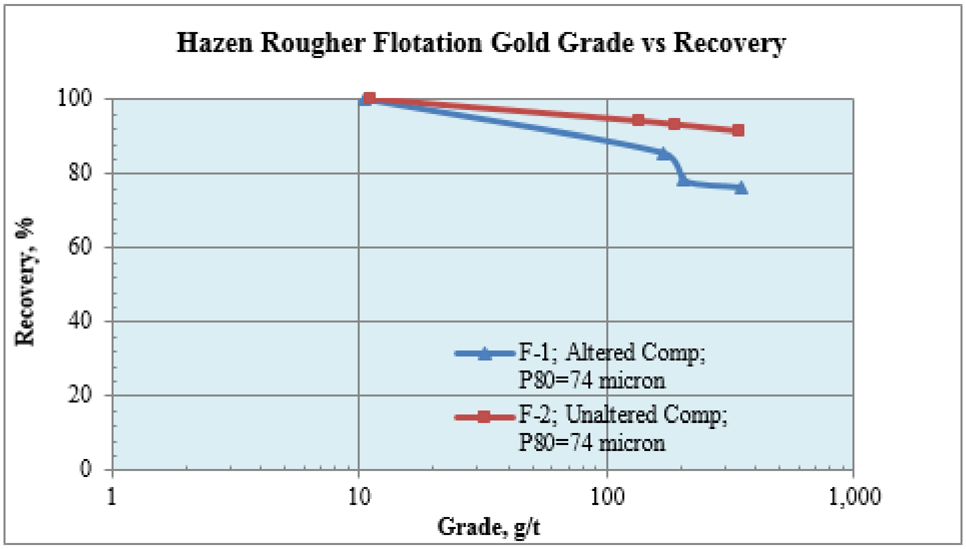
Figure 13.4: Inspectorate Gravity Test

The chart further indicates that higher gold recoveries, i.e., 40% and 50% at reduced concentrate grades of ~500 g/t and 100 g/t, respectively, may be projected.

It is apparent from these results, and similar to the Dawson samples discussed above, that there exists a respectable metallurgical and economic potential for gravity recovery of gold from the Windy Gulch samples at relatively coarse grind sizes. These results further demonstrate that gravity separation should be considered as part of the process development for the Dawson Project.

13.6 Flotation Tests

Hazen test work: Preliminary rougher flotation tests were conducted using conventional reagents and a nominal P80=74 µm primary grind size on altered and unaltered samples. Results of test work are graphically demonstrated in Figure 13.5 below.



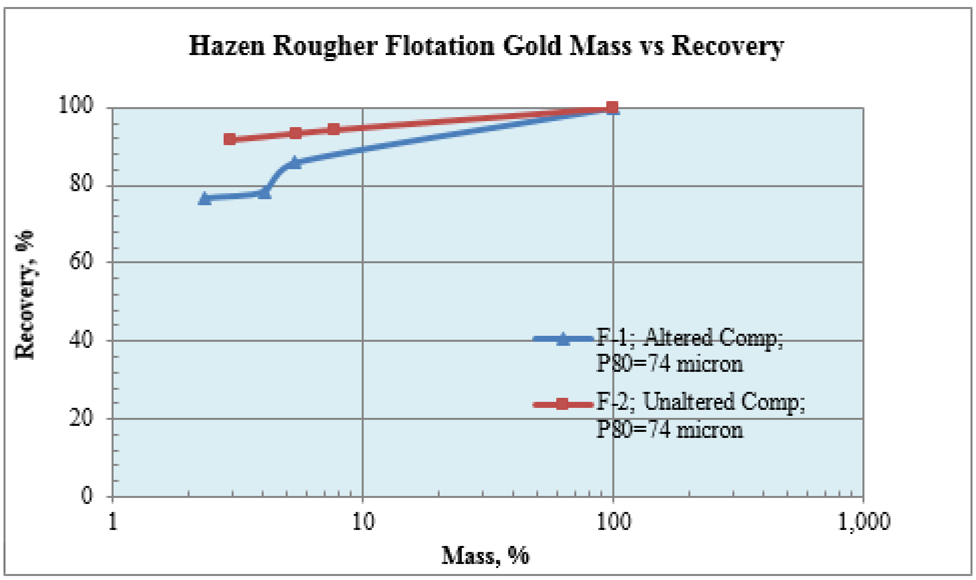


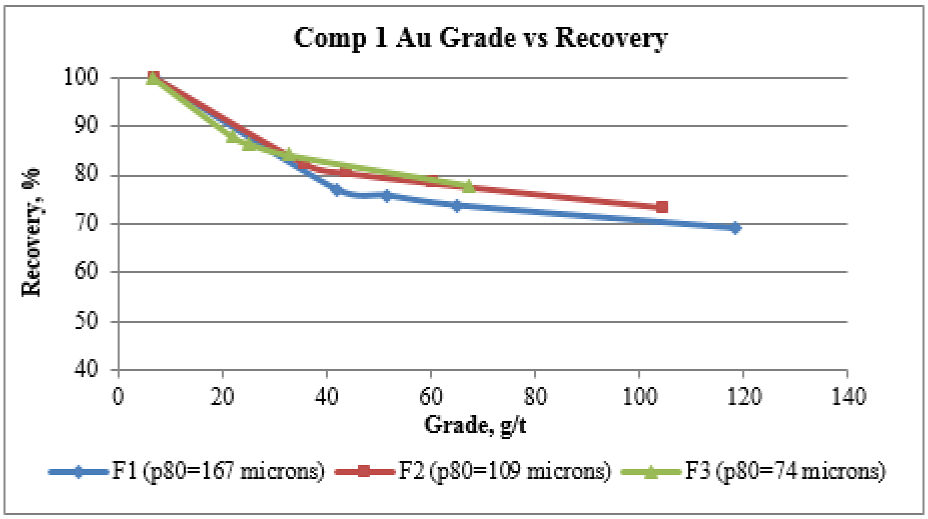
Figure 13.5: Hazen Rougher Flotation Golder Grade and Mass vs. Recovery

As shown in the above charts, nearly 86% of the total gold was recovered to a flotation concentrate assaying 169 g/t and at 5.4 weight percent of the feed for the "altered" sample (Flotation Test 1). Similarly and in Test 2, for the unaltered sample, over 94% of the gold was recovered to a flotation concentrate assaying 135 g/t and at 7.8 weight percent of the feed.

Inspectorate test work: Three rougher flotation kinetics tests were conducted on Composite 1 at P80 primary grind sizes of 167, 109, and 75 µm to investigate the effect of grind size and particle liberation on gold grade and recovery. All tests were run at a natural pH and using conventional reagents' flotation time over the four stages totalled 22 minutes. Metallurgical results are detailed in Table 13-2 and graphically shown in Figure 13.6.

Table 13-2: Inspectorate Test Work – Metallurgical Results

Test	Grind	Mass	Assays			% Rougher Recove		
No	**P80 (µm)**	**Pull, %**	**Au g/t**	**Ag g/t**	**% S**	**Au**	**Ag**	**S**
F 1	167	13.1	41.9	17.4	3.2	77.1	56.9	87.5
F 2	109	16.2	35.8	16.4	2.6	82.4	68.0	89.4
F 3	75	26.9	22.0	10.4	1.5	87.8	71.9	90.0



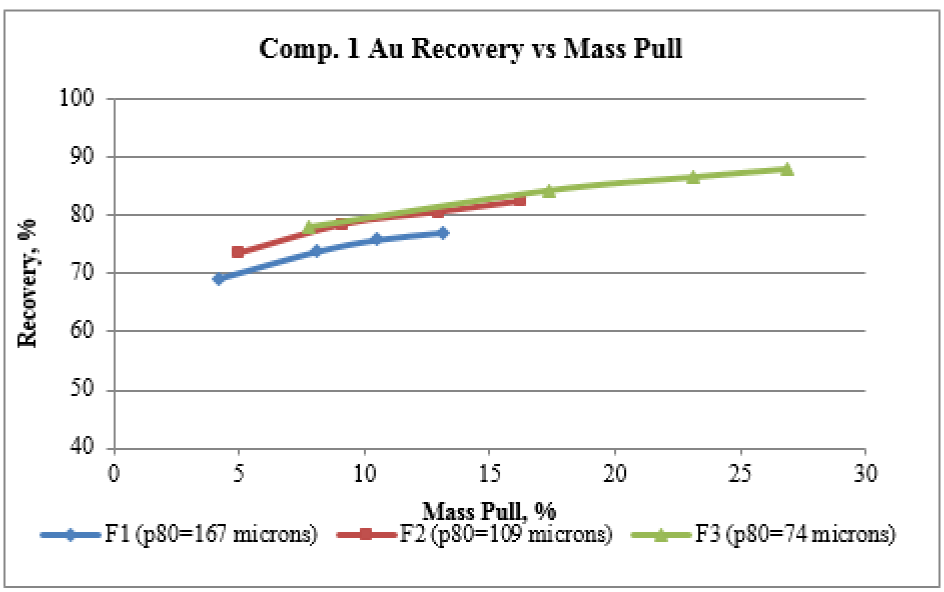


Figure 13.6: Inspectorate Test Work – Metallurgical Results

Variation of grind size between P80 of 109 and 74 µm did not appear to have a significant impact on gold recovery at a given mass pull while the coarse grind size of P80 167 µm resulted in lower recoveries. It appears that a coarser primary grind size than tested at Hazen (i.e., 109 µm) may be suitable for the Windy Gulch material.

A combined centrifugal gravity separation plus rougher-cleaner flotation test was conducted at 110 µm primary grind size. The test schematic and metallurgical balance of selected cumulative products are shown in Figure 13.7 and Table 13-3 below respectively.

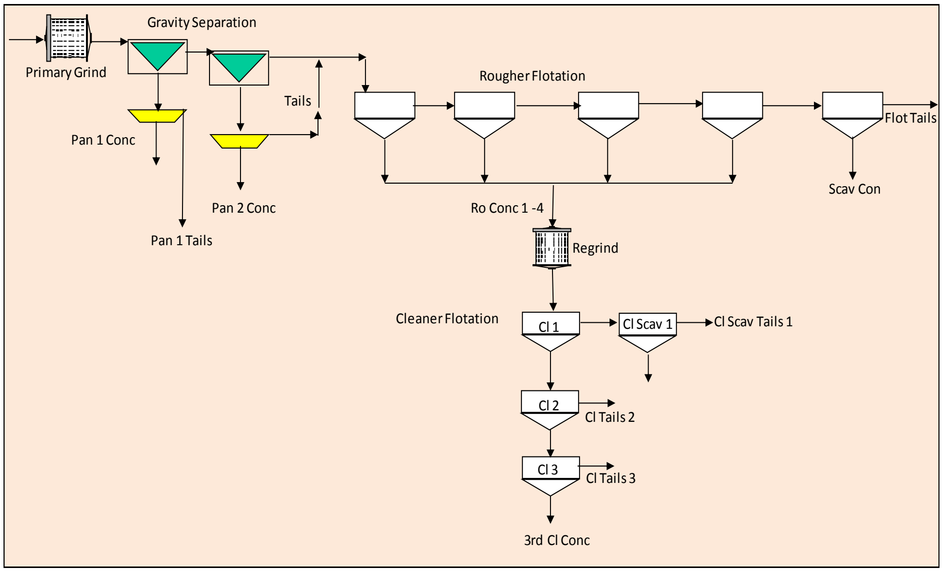


Figure 13.7: Combined Gravity+Flotation Test Schematic

Table 13-3: Combined Gravity+Flotation Test Metallurgical Balance

Product	Weight		Assay, g/t		Cum. Recovery, %	
	g	%	Au	Ag	Au	Ag
Pan Concentrate (1)	1.79	0.09	2,113.2	132.0	24.2	3.4
Pan Concentrate 1 + Pan Tail 1	72.32	3.65	127.7	18.9	59.0	19.9
Gravity Con: Comb' Pan 1+2 Cons + Pa	74.07	3.74	126.6	20.3	59.9	21.9
Gravity Conc + Flot 3rd Con	83.37	4.21	132.6	29.1	70.6	35.3
Gravity Conc + Flot 2nd Con	102.53	5.18	111.7	26.8	73.1	40.0
Gravity Conc + Flot 1st Con	219.54	11.08	55.9	16.2	78.3	51.6

The pan 1 concentrate recovered 24.2% of the gold at a grade of 2,113 g/t while total gravity concentrate recovery was 59.9% at a cumulative gold grade of 126.6 g/t. When combined with flotation first concentrate, recovery increases to 78.3%, but at a cumulative gold grade of 55.9 g/t.

It is apparent from these results that the Windy Gulch samples respond less favourably to flotation in particular and gravity separation in general than the Dawson samples tested at Hazen. This is highly likely due to the altered (oxidized) nature of the samples.

In conclusion, the results of gravity and flotation tests offer a technically viable process option for the recovery of precious metals from the Dawson and Windy Gulch resource.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Dawson Segment

14.1.1 Introduction

14.1.1.1 Standards and Definitions

The definition of mineral resource and associated mineral resource categories used in this report for the Dawson Segment mineral resource estimate are those recognized under NI 43-101 and set out in the Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves (the CIM Definition Standards). Assumptions, metal threshold parameters, and deposit modelling methods associated with this estimate are discussed below in report Sections 14.1.1.2 through 14.1.11.

14.1.1.2 Currency of Mineral Resource Estimate

In 2013, Mercator estimated mineral resources in accordance with NI 43-101 for the Dawson Segment and Windy Gulch Segment gold deposits and reported these in Hilchey et al. (2013). In 2015, Mercator confirmed currency of these two estimates to support NI 43-101 reporting of a preliminary mine plan assessment by MineTech for the Dawson Segment deposit. This work was reported in Hilchey et al. (2015). For current PEA purposes, Mercator has confirmed the continued currency of the 2013 Dawson Segment mineral resource estimate. An updated mineral resource estimate for the Windy Gulch Segment deposit has been prepared by Golder and is described in Section 14.2. The Golder estimate supersedes the 2013 estimate for the Windy Gulch Segment deposit completed by Mercator.

14.1.1.3 Overview of Mineral Resource Estimation Approach

The Dawson Segment deposit resource estimate is based on a 3D block model developed using GEOVIA Surpac 6.1.4 modelling software and a fully validated drill hole database. At the time of resource estimate preparation, Mercator developed and validated the project drill hole database inclusive of both the Dawson and Windy Gulch Segment deposits. This included 76,559.6 ft (23,335.37 m) of historical diamond drilling in 118 holes and 100.5 ft (30.63 m) of sampling in two trenches completed between 1981 and 1991 by US Borax, Jascan, and Uranerz, plus 1,928 ft (587.65 m) of diamond drilling in 13 holes and 87 ft (26.52 m) of chip sampling in one trench and two small pits (previously excavated) completed by Zephyr during the winter of 2013 on the Windy Gulch Segment. The Dawson Segment resource model was based on the composited results of 171 drill core samples from 27 separate drill holes and 2 wedge holes (GC40-W1 and GC40-W2) completed by US Borax, Jascan, and Uranerz that fall within the confines of the current resource limits. The remaining holes and assays in the Dawson Segment area contributed to the geological interpretations but did not meet the minimum grade threshold to be included in the model.

After the validated drill hole database was uploaded into Surpac, distribution statistics were calculated for the contributing gold data set after normalization of results to a common 5 ft (1.52 m) sample base. Continuous down hole grade composites measuring 5 ft (1.52 m) in length were used in the deposit block models. Frequency distribution and probability plots were prepared based on these results and high grade metal capping factors were determined. Composites for the Dawson Segment were capped at 1.17 oz/tn (40 g/t) Au.

Prior to digital deposit modelling, a complete set of digital 100 ft (30.48 m) spaced vertical cross-sections were generated for the Dawson Segment perpendicular to the regional strike. Lithocode and analytical gold assay data from the project database were displayed on drill hole and trench traces for all sections. The Dawson Segment was modelled after close inspection of the gold grade sections and geological surface models for the QBG-QB gneiss, sulphide-rich zone, and the footwall Pbu/Pmu gneisses. Mineralized intercepts showing gold grades of economic interest were identified, these being defined on the basis of a minimum gold grade threshold value of 0.12 oz/tn (4.12 g/t) Au over 5 ft (1.52 m) horizontal width. Two sharply defined high grade (>0.12 oz/tn [4.12 g/t] Au) stratiform gold horizons historically known as Zone 2 and Zone 2a were delineated within the QBG gneiss. Zone 2 locally extended into the overlying QB gneiss unit. A third higher grade gold domain historically referred to as Zone 1 was modelled within the QB gneiss but locally included the overlying sulphide-rich unit. Zone 1 is offset by a steeply dipping east-northeast striking fault in the eastern Dawson Segment area (Line E 48550).

Based on the minimum gold threshold value of 0.12 oz/tn (4.12 g/t) Au over a 5 ft (1.52 m) horizontal width, 3D solids were generated from the digital wireframe outlines created from the interpreted drill sections developed for the Dawson Segment. Upper limits of mineralization were defined during wireframing as being either 100 ft (30.48 m) extensions from sectional drill hole data or half the distance to the nearest drill hole. No gold domains were extended to surface. A total of eight gold domain models were assembled for the Dawson Segment. For resource modelling purposes, Zone 1 is composed of three separate gold domains, Zone 2 is composed of three separate gold domains, and Zone 2a is composed of two separate gold domains. Combined strike length of the higher grade Zone 2 gold domains is approximately 1,040 ft (316.99 m) and locally demonstrates an aggregated maximum mineralized dip component of up to approximately 1,025 ft (312.42 m). The main Zone 2a gold domain overlaps with the main Zone 2 gold domain in the central deposit area and has a combined strike length of approximately 875 ft (266.70 m) with a maximum dip component of up to approximately 585 ft (178.31 m). Moderately plunging southwest grade trends were visually identified in both zones. Zone 1 gold domains have a combined strike length of approximately 880 ft (268.22 m) and aggregate dip extent of up to 190 ft (57.91 m).

A model block size of 5 ft (1.52 m) × 16.5 ft (5.03 m) × 16.5 ft (5.03 m) - Y,X,Z, with partial percentage assignment to all blocks impinging on resource gold domain solids, was selected to provide definition of the relatively narrow stratiform gold zones that characterize the Property. A total of nine block model sub-domains were created within the eight gold domain solids to facilitate grade interpolation in areas of geometric irregularity. Grade interpolation was accomplished using ID^2 methodology and a search ellipsoid oriented along the 065°/-68° trend. The search ellipsoid was assigned a 200 ft (60.96 m) major axis range, 200 ft (60.96 m) semi-major axis range, 50 ft (15.24 m) minor axis range, and major axis plunge of -14° to the southwest. Grade interpolation was fully constrained within each gold domain solid and discretization for blocks in all domains was set at 1 × 1 × 1. In each gold domain, no more than nine composites were allowed to contribute to a block grade, with no limit placed on the number of included drill holes. A limit of four reporting composites per drill hole was also applied. Grade interpolation was fully constrained within the gold domain solids and a global density value of 164.19 lb/ft^3 (2.63 g/cm^3) was used for the model. This reflects an average of seven historical specific gravity (SG) determinations completed by Jascan in 1987 for mineralized samples representing a range of gold grades within the Dawson Segment.

Estimation procedures for the Dawson Segment mineral resource estimate are described below in more detail under separate headings.

14.1.2 Drill Hole Database

14.1.2.1 Units

All historical project information provided by Zephyr was delivered in local grid coordination and United States Customary unit measure prior to the evaluation of the Dawson Segment deposit. No accurate conversion factor to transform the drill hole database collar coordinates from original local grid coordination to UTM NAD 83 metric coordination had been developed at the time of the 2013 resource estimate preparation. As a result, resource block modelling was carried out using United States Customary unit measure. Mercator subsequently converted resource solid volumes to cubic metres, and the originally reported resource estimate was presented in metric tonnage and grade units. For current report purposes, Mercator was asked to convert previously reported metric tonnage and grade estimates and, where appropriate, associated support factor units, to United States Customary units. To provide continuity between this report and original estimation reporting presented in Hilchey et al. (2013), Mercator has provided herein calculated United States Customary unit values followed, in bracketed format, by the source metric equivalent value.

14.1.2.2 Database Validation

In total, 91,063.1 ft (27,756.03 m) of diamond drilling and 5,353 analytical results from 152 drill holes, and 21 analytical results from two road cut-trenches completed by US Borax, Jascan, and Uranerz between 1979 and 1991 on the Dawson Property were received in digital format from Zephyr. Eight banker's boxes containing historical original bound company reports, loose document files, and correspondence were also received from Zephyr. A preliminary review of the digital data set identified several typographic errors with respect to assay entries and collar coordinates, and it was determined that data subsets belonging to the Windy Gulch Segment and Dawson Segment consisting of 76,559.6 ft (23,335.37 m) of diamond drilling and 4,397 analytical results from 118 holes, and 21 analytical results from two road-cut trenches would need complete validation prior to use in the resource estimate. Validations included comparing all database entries for drill hole collar coordinates (local grid), downhole surveys, and gold assay values (and depths) to corresponding entries found in 1) summary tables from bound reports and loose notes, 2) original drill logs, or 3) original laboratory documentation (available for the majority of Uranerz gold assay results).

Supporting laboratory documentation was not available to confirm assay values from the 98 drill holes completed by US Borax (hole series GC-03 to GC-62) and Jascan (hole series GC-63 to GC-98 and MS01 to MS07) in the Dawson Segment and Windy Gulch Segment areas and validation was restricted to either hand written analytical results recorded in each drill log and typed analytical results in assay summary tables. US Borax and Jascan digital assay results available for the Windy Gulch Segment and Dawson Segment areas were also compared to original lithological log descriptions and locally checked on a sectional basis in Surpac against adjacent Uranerz drill holes having supporting laboratory documentation.

Mercator also compiled and validated 595 re-assay samples completed by US Borax in 24 holes that intersected the Dawson Segment and Windy Gulch Segment mineralized gold zones (hole series GC-5 to GC-44) and 387 assay checks on 300 samples completed by Jascan at multiple laboratories using sample pulps from 1 US Borax hole (GC49) and 29 Jascan holes. Hand-typed summary spreadsheets for the US Borax re-assay data were compared to the original sample intervals and gold values recorded in drill logs. A total of 47 re-assay results from GC37 were rejected due to sample interval inconsistencies with respective entries in the original logs, while the remainder were accepted. The remaining re-assay results showed acceptable degree of correlation with the original results and were incorporated into the resource estimate database. Comparison of Jascan pulp assay check values (check values for GC37 not included) also showed results demonstrating an acceptable degree of correlation.

After re-assay and assay pulp checks results were accepted, original gold assay values were averaged with re-assay and/or pulp check values to better approximate a mean gold value for the resource estimate. Sources of historical assay results used in the resource are summarized in Table 14-1. Unit conversion calculations from ounces per ton to grams per tonne were performed on Jascan and Uranerz gold results and finalized assay results were imported into GEOVIA Surpac 6.1.4 for resource estimation purposes.

Table 14-1: Summary of Analytical Results Available for Historical Drilling

Vintage	Hole Nos.	Historical Results Available	Source of Au Value Used in Resource Database
1981–1985 (USB)	GC03,GC04,GC10,GC14, GC22,GC25,GC26,GC31, GC34,GC35,GC41, GC45-GC62	USB** FA-AA to GC17, USB FA-GRA and USB FA-AA (locally missing) from GC17-GC62	USB FA-GRA; FA-AA when no FA-GRA or when FA-GRA was less than detection (<0.2 ppm Au)
1981–1984 (USB)	GC05,GC11,GC12,C13, GC15,GC16,GC17,C18, GC19,GC20,GC21,GC23, GC24,GC27,GC28,C29B, GC32,GC37,GC39,GC40, GC42,GC43,GC44	original USB FA-AA result to GC17; original USB FA-GRA for GC17-GC44;USB re-assay FA-GRA for mineralized intercepts; Uranerz re-assay samples for mineralized intercept in GC40	average of USB re-assay and FA-GRA original, or average of original USB FA-AA and re-assay if original FA-GRA not available; Uranerz re-assay values also averaged with USB re-assay and original values; re-assay results for GC37 not averaged
1987–1988 (Jascan)	GC63-GC98, MS01-MS07	Logs do not mention whether FA-GRA or FA-AA used, assumed to be FA-GRA; 387 check assay pulps on 300 samples in 29 holes; Uranerz resampled mineralized intercepts for GC71A	FA result, or FA result averaged with pulp check results when available; also averaged with regular FA Uranerz assay check values in GC71A
1990 (Uranerz) *	DA9001-9008	FA but sources do not mention whether AA or GRA finish was used; higher grade samples also analyzed by screen metallic analysis	FA except for 10 metallic screen results where FA results were not available

Vintage	Hole Nos.	Historical Results Available	Source of Au Value Used in Resource Database
1991 (Uranerz)	WG9101-9107, DA9101-9103	FA	FA

*Uranerz conducted limited screened metallics analyses for Au and these results were not used in the resource estimate except for 10 values where regular FA results were not available.
**US Borax further abbreviated to USB

A further 1,928 ft (587.65 m) of drilling and 419 screen metallic analytical gold results from 13 survey-controlled holes and 29 analytical results for chip sampling from one trench and two small previously excavated pits completed by Zephyr in 2013 were received in both digital laboratory spreadsheet and laboratory certificate format from QP Mark Graves and compiled by Mercator. One in 10 analytical entries was validated against the original laboratory certificates.

Based on a review of historical and current file and field information, Mercator determined the validated records for both historical and 2013 Zephyr drill hole data sets to be acceptable for resource estimation use.

14.1.2.3 Core Recovery

Core recovery information was compiled for both deposit areas from available logging sources and was imported into Surpac for assessment purposes. A total of 65 drill core recovery entries were available from the Dawson Segment deposit and these have a mean value of 96.78% core recovery with a minimum of 55% and maximum of 106%. On this basis, core loss was not considered a significant issue for the Dawson Segment area.

14.1.3 Geological Interpretation

14.1.3.1 General

The geological interpretation and discussion of mineralization presented previously in this report outline the main aspects of the geological models considered most appropriate for the current Dawson Segment mineral resource estimate. In summary, it has been modelled as being predominantly representative of a magmato-hydrothermal intrusion-related style of gold deposit. Gold mineralization is hosted by a sequence of Early Proterozoic siliceous felsic gneisses and lesser semi-massive sulphide/sulphide-rich zones and occurs in multiple horizons. Gold is generally associated with sillimanite, sericite, or biotite and is found in cracks in quartz and garnet and at quartz grain boundaries. Gold is also associated with carbonate/siliceous veinlets and rarely as inclusions in pyrite and chalcopyrite (Mettler, 1991). Gold mineralization contact zones are relatively discrete and have been documented to vary in true thickness from approximately 2 ft (0.61 m) to 12 ft (3.66 m) to as wide as 50 ft (15.24 m) (Alers, 2003). Gold is locally nuggety and has been identified as grains up to 3 mm in size (Mettler, 1991). It is commonly accompanied by 1% to 5% disseminated pyrite +/- chalcopyrite and is the only metal within the deposit areas being modelled that is considered to have economic significance at this time (Theye, 1989).

The Dawson Segment deposit is interpreted to lie on the south limb of a regional west plunging anticline (Wolfson 2011). Gold mineralization assessed in the current Dawson Segment resource estimate occurs as three east-northeast striking and moderate to steeply southeast dipping, tabular lenses historically known as Zone 1, Zone 2, and Zone 2a. Adjacent mineralization belonging to the historical Sentinel Segment deposit has been incorporated in the Dawson Segment resource estimate for current purposes.

Zone 1 gold mineralization is primarily hosted by siliceous QB gneisses but locally includes the lower portion of the overlying sulphide-rich unit containing 10% to 50% sulphides in a fragmental and chlorite matrix (Mettler, 1991). Sulphide minerals include pyrite with lesser amounts of chalcopyrite, pyrrhotite, sphalerite, and galena. Zone 2 gold mineralization occurs within the upper horizon of a siliceous QBG gneiss unit but locally extends into overlying QB gneisses. Zone 2a is also hosted by the QBG gneiss and underlies Zone 2. Zone 2 and Zone 2a locally converge and thicken in the central deposit area.

Previous work on the Dawson Segment indicates that gold mineralized sequence has undergone structural modification. Drag folds on the southern anticlinal limb have been interpreted to cause structural repetition of the sulphide-rich unit and associated thickening of the mineralized horizons (Wolfson 2011). Shearing along fold axes has also had the effect of producing local pinching and swelling of mineralized zones between drilling sections. The Dawson Segment is also cross-cut by east-northeast trending faults which locally offset mineralization. One of these cross faults was modelled in the eastern deposit area in association with the current resource estimate and likely offsets the Dawson Segment from the Sentinel Segment. At deposit scale, gold zones are considered to represent generally tabular bodies.

The portion of the Dawson Segment defined by historical drilling and incorporated in the current resource estimate occur within a series of digital solids that comprise a 0.12 oz/tn (4.0 g/t) Au peripheral constraint used in resource modelling. These solids occur along a combined strike length of approximately 1,215 ft (370.33 m) and define a combined dip extent of approximately 1,065 ft (324.61 m). Average thickness of the 0.12 oz/tn (4.0 g/t) Au peripheral constraint solids is approximately 12 ft (3.66 m) and thicknesses range from approximately 5 ft (1.52 m) to 35 ft (10.67 m).

14.1.3.2 Mineral Domains and Solid Modelling

14.1.3.2.1 Topographic and Bedrock Surfaces

A digital terrain model (DTM) of the topographic surface was created in Surpac for the Dawson Segment. A regional elevation contour map with 50 ft (15.24 m) contours in local grid coordination was compiled by JMS Geologic in Boulder, Colorado, and this was digitized by Mercator from a report by Alers (2003) and converted to a Surpac DTM topographic surface model. The DTM model served as a collar coordinate check for drill holes in the Dawson Segment area but was not used as a topographic constraint for the resource model.

14.1.3.2.2 Au Domain Models

A minimum 5 ft (1.52 m) horizontal support length was used to define gold domain thicknesses used in the Dawson Segment block model interpolation constraint. Mercator initially interpreted and developed 3D wireframe solid models for eight gold domains based on a subjectively applied 0.12 oz/tn (4.0 g/t) Au over a 5 horizontal ft (1.52 m) cut-off value and detailed sectional interpretations created for the entire zone. Sections included posted assays for gold and lithocoded rock units and were developed from drill hole traces projected to nominal 100 ft (30.48 m) section spacings. Lateral up and down dip extents of the gold domain solids were limited either to 100 ft (30.48 m) from the last intercept or half the distance to a constraining drill hole. The solid model was projected 100 ft (30.48 m) along strike from the last section that showed continuity and definition or half the distance to a constraining drill hole. The gold domain 3D wireframe solid models created for the Dawson Segment have a maximum combined strike extent of 1,215 ft (370.33 m) trending at approximately 065°/-68° (southeast) and a maximum combined dip extent of 1,065 ft (324.61 m). Average thickness of the 0.12 oz/tn (4.0 g/t) Au peripheral constraint solids is approximately 12 ft (3.66 m) and thicknesses range from approximately 5 ft (1.52 m) to 35 ft (10.67 m). The combined maximum vertical extent of the gold domains is approximately 1,008 ft (307.24 m). Five of the gold domains are stratigraphically hosted by the QBG gneiss and three are hosted by the overlying QB gneiss and sulphide-rich horizon.

Of the eight gold domains created to model the Dawson Segment, five were based on two or more drill hole intercepts and three were built around single drill hole intercepts. All gold domains for the Dawson Segment appear in Figures 14.1 to 14.3.

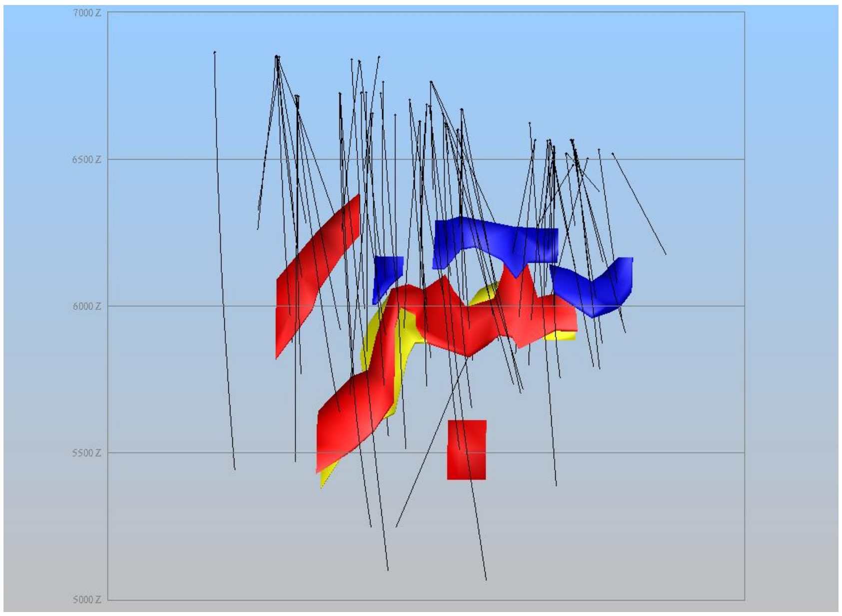


Figure 14.1: Longitudinal View of the Dawson Segment Surpac Au Domain Solid Models(View to northwest: blue=Zone 1 Au Domains, red=Zone 2 Au Domains, yellow=Zone 2a Au Domains)

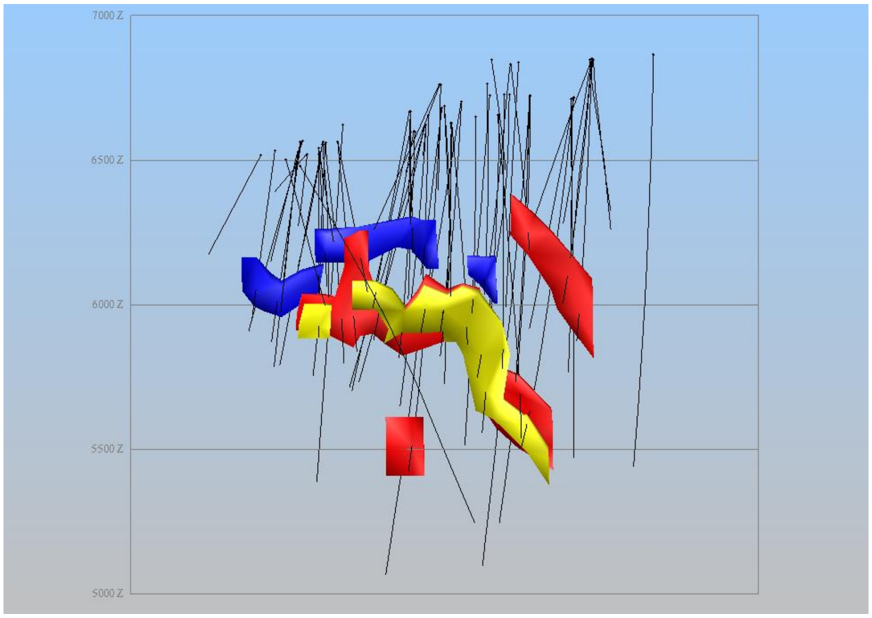


Figure 14.2: Longitudinal View of the Dawson Segment Surpac Au Domain Solid Models(View to southeast)

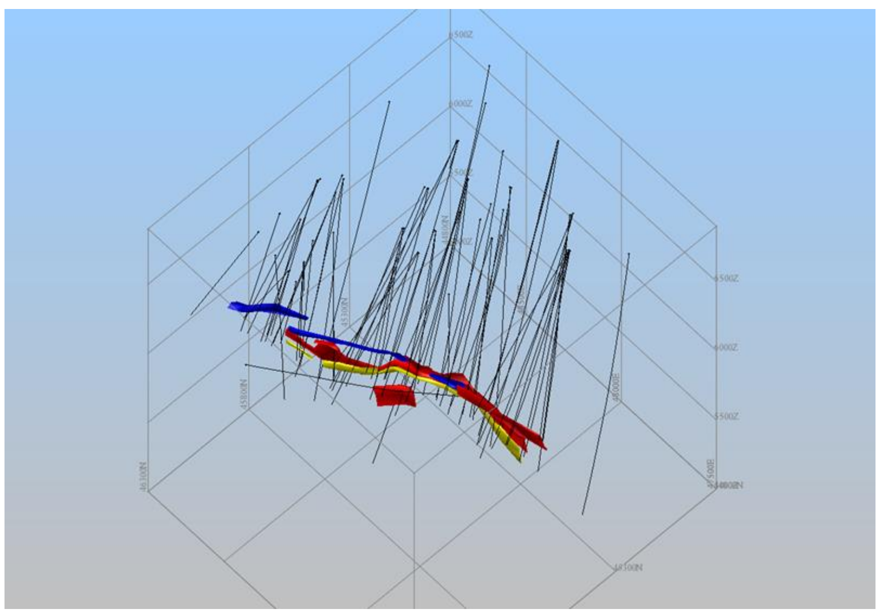


Figure 14.3: Isometric View of the Dawson Zone Surpac Au Domain Solid Models (View to southeast)

14.1.4 Drill Core Assay Composites and Descriptive Statistics

14.1.4.1 Drill Core Assay Composites

The project database, at the time of resource estimate preparation, contained 4,867 core sample records in 131 holes and 3 road-cut trenches, exclusive of quality control and quality assurance samples, including 171 core samples in 29 drill holes (including 2 wedge holes) occurring within the 8 gold domains of the Dawson Segment. Sample lengths range from 1 to 10 ft (0.31 to 3.05 m) within the deposit, with approximately 29.6% of samples measuring over 3 ft (0.91 m) in length. The frequency histogram and cumulative frequency distribution plot for sample lengths within the Dawson Segment resource estimate appear below in Figure 14.4.

Based on these results, a down hole assay composite data set at 5 ft (1.52 m) support length was developed for gold in drill holes intersecting the resource solid models. No lithological constraints were imposed on down hole assay compositing. Any composite with greater than 50% of its length falling within a gold domain was included in the composite data set to be used for resource estimation purposes.

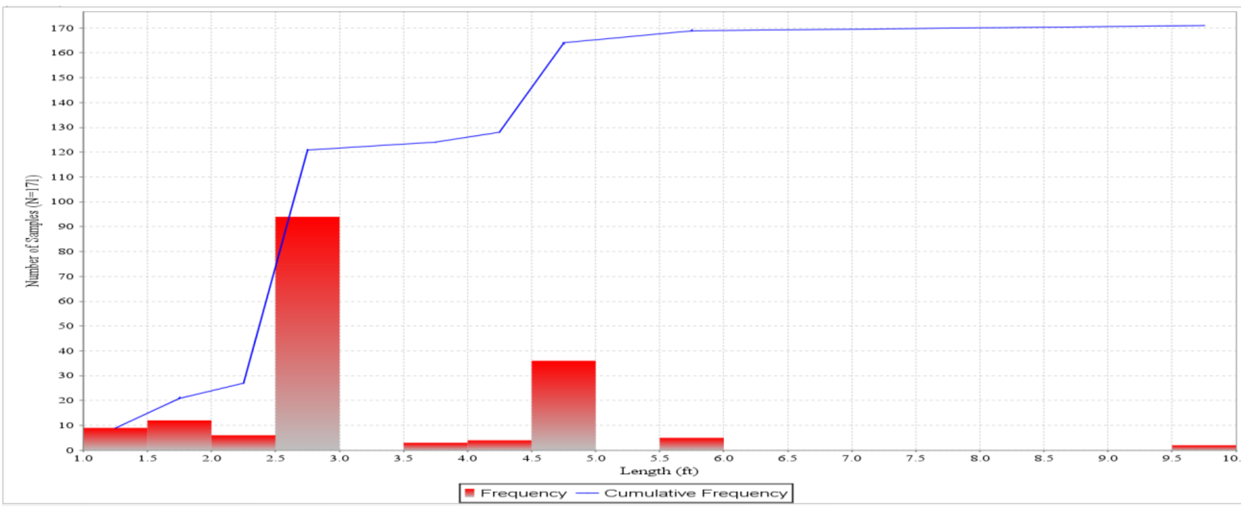


Figure 14.4: Cumulative Frequency Histogram of Sample Lengths in Dawson Segment

14.1.4.2 Descriptive Statistics

Descriptive statistics were calculated for the 5 ft (1.52 m) gold assay composite data set and the results are presented in Table 14-2. Distribution histograms, cumulative frequency plots, and probability plots for the uncapped composites for each zone were also prepared and were appended to the Hilchey et al. (2013) NI 43-101 technical report by Mercator. The maximum uncapped grade of composites for the Dawson Segment is 2.58 oz/tn (88.60 g/t) Au. Gold values approaching the maximum value appear to reflect intervals within the deposit that have coarse gold but limited lateral extent as defined by adjacent drill hole composite values.

Table 14-2 Dawson Segment Au Statistics for Uncapped 5 ft (1.52 m) Composites

Deposit	Dawson Zone
Parameter	Au
Mean Grade	0.32 oz/tn (10.98 g/t)

Maximum Grade	2.58 oz/tn (88.60 g/t)
Minimum Grade	0.0003 oz/tn (0.01 g/t)
Variance	214.323
Standard Deviation	14.640
Coefficient of Variation	1.333
Count	111

14.1.5 Outliers (High Grade Capping of Assay Composite Values)

A high grade cap for gold was applied to the 5 ft (1.52 m) assay composites to limit the influence of anomalous high grade results having restricted continuity. Composites within the Dawson Segment were capped at 1.17 oz/tn (~40 g/t) Au, corresponding to the 96.8 percentile level. A subjective check on the applicability of capping factors was carried out on the basis of logged geology and mineralization styles and it was concluded that the presence of 5 ft (1.52 m) intervals of gold mineralization at the selected capping grades was geologically reasonable, with potential for both strike and dip continuity at such levels. Descriptive statistics were calculated for the capped assay composites and are presented in Table 14-3. As expected, the mean capped gold composite grade decreases relative to the mean grade of the raw composite values presented above. The Coefficient of Variation for the capped data set is also lower, indicating that it are more statistically acceptable for resource estimation purposes. Distribution histograms, cumulative frequency plots, and probability plots for capped composites for gold were also prepared and were appended to the Hilchey et al. (2013) NI 43-101 technical report by Mercator.

Table 14-3: Dawson Segment Au Statistics for Capped 5 ft (1.52 m) Composites

Deposit	Dawson Segment
Parameter	Au
Mean Grade	0.29 oz/tn (9.82 g/t)
Maximum Grade	1.17 oz/tn (40.00 g/t)
Minimum Grade	0.0003 oz/tn (0.01 g/t)
Variance	94.208
Standard Deviation	9.706
Coefficient of Variation	0.989
Count	111

14.1.6 Resource Estimation

14.1.6.1 Grade Variography

Manually derived models of geology and grade distribution provided definition of the primary east-northeast striking and southeast dipping trend of the Dawson Segment that is characterized by presence of multiple thin stratiform zones of gold mineralization. To further assess the spatial aspects of grade distribution within the Dawson Segment, a series of experimental variograms were calculated for 5 ft (1.52 m) assay composites within two gold grade domains demonstrating best grade continuity along strike and dip extents (1 from Zone 2 and 1 from Zone 2a). Various lags were initially assessed at 15° increments within a plane corresponding to the average strike and dip of the two aggregated domains. Composites were assessed against a spherical model. Best variogram model results defined a major axis of continuity oriented within the plane of the deposit (locally 053°/-67°) with a -14°

plunge to the southwest and range of approximately 200 ft (60.96 m) at a lag of 46 ft (14.02 m) (Figure 14.5). Variograms for the semi-major and minor axis directions were inconclusive and the major axis range was applied to the semi-major axis. Ranges of 50 ft (15.24 m) were assigned to the minor axis.

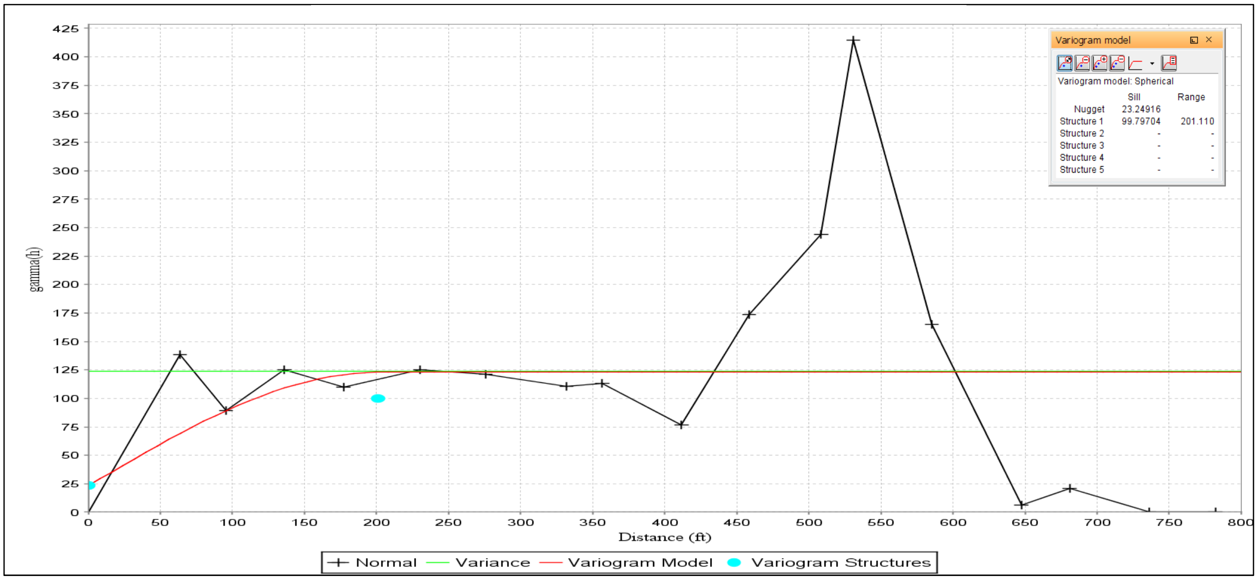


Figure 14.5: Dawson Segment – Variogram for Major Axis of Continuity (Lag 46, -13.78 plunge to 227.02)

Adequate assay data were not available to calculate individual variograms on the remaining six gold domains, and the axis range assignments for Zone 2 and 2a were therefore applied to the remaining gold domains within the Dawson Segment model. Directional parameters for each ellipsoid were manually adjusted for each interpolation domain or subdomain to conform to local geometric irregularities and these are summarized in Table 14-4.

Table 14-4: Ellipsoid Ranges and Orientations for Dawson Segment Interpolation Domains

Au Domain	Subdomain	Historical Name	Strike°	Plunge°	Dip°	Major and Semi-Major Axis Range (ft)	Minor Axis Range (ft)
zone_1a		Zone 1	249	-14	56	200	50
zone_1b		Zone 1	232	-14	62	200	50
zone_1c		Zone 1	225	-14	62	200	50
zone_2a		Zone 2	245	-14	69	200	50
zone_2b	zone_2b_sdom1	Zone 2	258	-14	75	200	50
	zone_2b_sdom2	Zone 2	235	-14	69	200	50
	zone_2b_sdom3	Zone 2	209	-14	69	200	50
	zone_2b_sdom4	Zone 2	236	-14	60	200	50
	zone_2b_sdom5	Zone 2	262	-14	75	200	50
zone_2c		Zone 2	222	-14	77	200	50

Au Domain	Subdomain	Historical Name	Strike°	Plunge°	Dip°	Major and Semi-Major Axis Range (ft)	Minor Axis Range (ft)
zone_3a	zone_3a_sdom1	Zone 2a	217	-14	65	200	50
	zone_3a_sdom2	Zone 2a	243	-14	49	200	50
	zone_3a_sdom3	Zone 2a	251	-14	62	200	50
	zone_3a_sdom4	Zone 2a	265	-14	79	200	50
zone_3b		Zone 2a	254	-14	65	200	50

14.1.6.2 Block Model Definition

Grid extents for the Dawson Segment block model were defined in local grid coordinates (ft) and appear in Table 14-5. At the time of resource estimate preparation there was no direct conversion from local grid into UTM coordination. Standard block size for the block model is 5 ft (1.52 m) × 16.5 ft (5.03 m) × 16.5 ft (5.03 m) (Y,X,Z) with no units of sub-blocking allowed. The block model is oriented on a grid azimuth of 0° with no dip rotation.

Table 14-5: Summary of Block Model Parameters for the Dawson Segment

Block Model	Minimum Coordinate (ft)	Maximum Coordinate (ft)
Dawson Segment	y = 44750, x = 47480, z = 4780	y = 46310 , x = 48965, z = 6925

14.1.6.3 Volume Assignment

Partial percentage estimation was used for volume assignment of solid models. Partial percentages were determined with the GEOVIA Surpac 6.1.4 module for all eight Dawson Segment solid models and adjusted in the block model where the main Zone 2a gold domain overlaps with the main Zone 2 gold domain.

14.1.6.4 Historical Workings

Historical workings are located along the Dawson Segment and date back to the early 1900s. Prospectors were following locally gold-bearing copper/iron gossans and/or copper-bearing iron sulphide zones (Wolfson, 2011). No level plans or sections showing historical workings were available to Mercator during preparation of the resource estimate, but as currently understood from mapping and drilling, they are small in scale and would have no material impact on the volume of the Dawson Segment resource. Future work should include locating and digitizing detailed workings plans to fully assess their impact on the resource model.

The Copper Boy Mine and Sentinel Mine workings occur along the Dawson Segment, and newspaper records indicate the deepest shaft at the Copper Boy Mine is approximately 173.9 ft (53.00 m) deep. The current Dawson Segment resource commences at approximately 270 ft (82.30 m) from surface and on this basis can be described as a blind deposit. Mercator notes that the Mike Sutton adits and Last Show shafts occur along the Windy Gulch Segment and are not included in the Dawson Segment mineralized zone. These workings are similarly characterized by limited documentation.

14.1.6.5 Estimation Methodology (Grade Interpolation)

Inverse distance squared (ID^2) grade interpolation was used to assign block grades within the Dawson Segment block model. As reviewed earlier, interpolation ellipsoid orientation and range values used in the estimation reflect a combination of trends determined from the variography and sectional interpretations of geology and grade distributions. Trends and ranges of the major, semi-major, and minor axes of grade interpolation ellipsoids used to estimate gold block grade were described previously in Section 14.1.6.1. Estimation parameters were derived from the gold composite data set and block model interpolation was fully constrained within the 3D gold domain solids. All blocks within or partially included by the 0.12 oz/tn (4 g/t) Au peripheral constraint solids were considered for grade interpolation. The minimum number of contributing composites used to estimate a block grade was set at 1 and the maximum number of contributing composites was set at 9, with no more than 4 contributing composites allowed from a single drill hole. Block discretization was set at 1Y × 1X × 1Z.

Resource interpolation involved eight gold domains and nine sub-domains for the Dawson Segment. Sub-domaining of the main gold solids was locally carried out to better accommodate their geometric irregularities. Slightly differing ellipsoid orientations were used to best-fit the ellipsoid to local deposit orientation. Sub-domain boundaries were soft with respect to grade interpolation and the entire assay composite population for the full domain was available for sub-domain grade interpolation purposes. Ellipsoids used for interpolations were omni-directional in the plane of the major and semi-major axes, and oriented to conform to the plane of each gold domain or subdomain.

14.1.6.6 Bulk Density

At the time of resource estimate preparation no substantive data set of density or specific gravity (SG) values existed for the Dawson Property. A total of seven historical SG determinations were available from the 1988 Jascan drilling on the Dawson Segment and include a range of mineralized intervals. Values range between 0.078 tn/ft^3 (2.50 g/cm^3) and 0.091 tn/ft^3 (2.9 g/cm^3) and an average SG value of 0.082 tn/ft^3 (2.63 g/cm^3) was used as a global density value for the Dawson Segment block model.

14.1.7 Mineral Resource Classification

14.1.7.1 Resource Category Definitions

Definitions (as of 2013 when this resource estimate was completed) of mineral resource and associated mineral resource categories used in this report are those recognized under NI 43-101 and set out in the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves Definitions and Guidelines (the CIM Standards). These are set out below:

Mineral Resource

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

Inferred Mineral Resource

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

Indicated Mineral Resource

An "indicated mineral resource" is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings, and drill holes, that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Measured Mineral Resource

A "measured mineral resource" is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings, and drill holes, that are spaced closely enough to confirm both geological and grade continuity.

14.1.7.2 Resource Category for the Dawson Segment Resource Estimate

Mineral resources presented in the Dawson Segment estimate have been assigned to the inferred resource category to reflect current levels of confidence given the spatial configuration of drill holes at this time. For the Dawson Segment, this reflects consideration of several factors, the most prominent of which is the lack of a fully documented assay data set with respect to the US Borax and Jascan drilling. Local intervals of poor core recovery within the mineralized zone and current lack of survey control in historical workings also contributed to assignment of inferred resource status.

14.1.8 Block Model Validation

Results of block modelling were reviewed in three dimensions and compared on a sectional basis with corresponding digital assay sections used for grade model solid development. The Dawson Segment deposit showed block model grade patterns that generally conform to an east-northeast striking deposit system, with a moderate to steep southeast dip. Overall, results of the visual inspection show an acceptable degree of consistency between the block model and the independently derived geological interpretations. Representative sections and isometric views are presented in Figures 14.6 through 14.11.

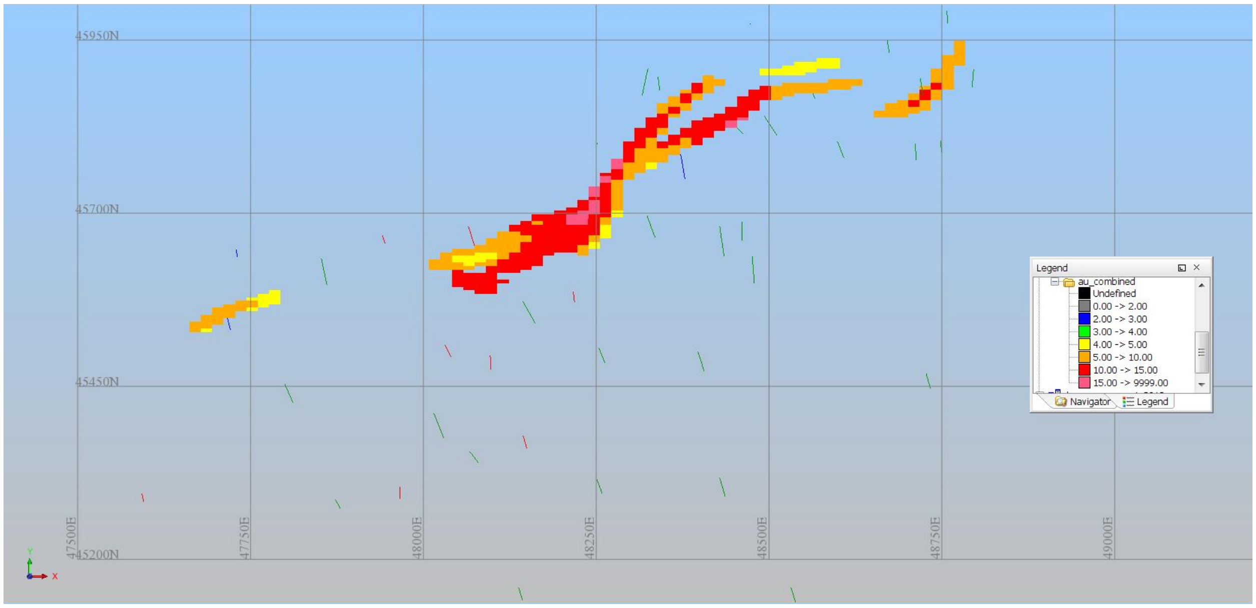


Figure 14.6: Dawson Segment Level Plan 6000 Elev. – Au (g/t) Block Grades

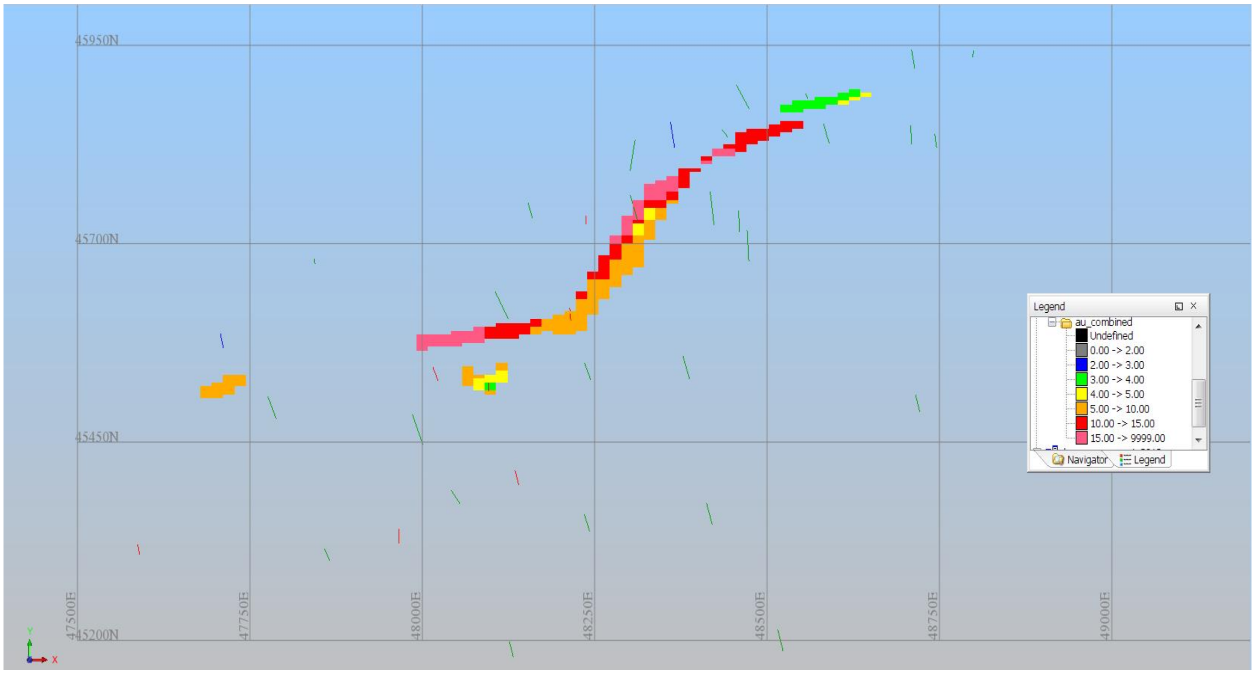


Figure 14.7: Dawson Segment Level Plan 5900 Elev. - Au (g/t) Block Grades

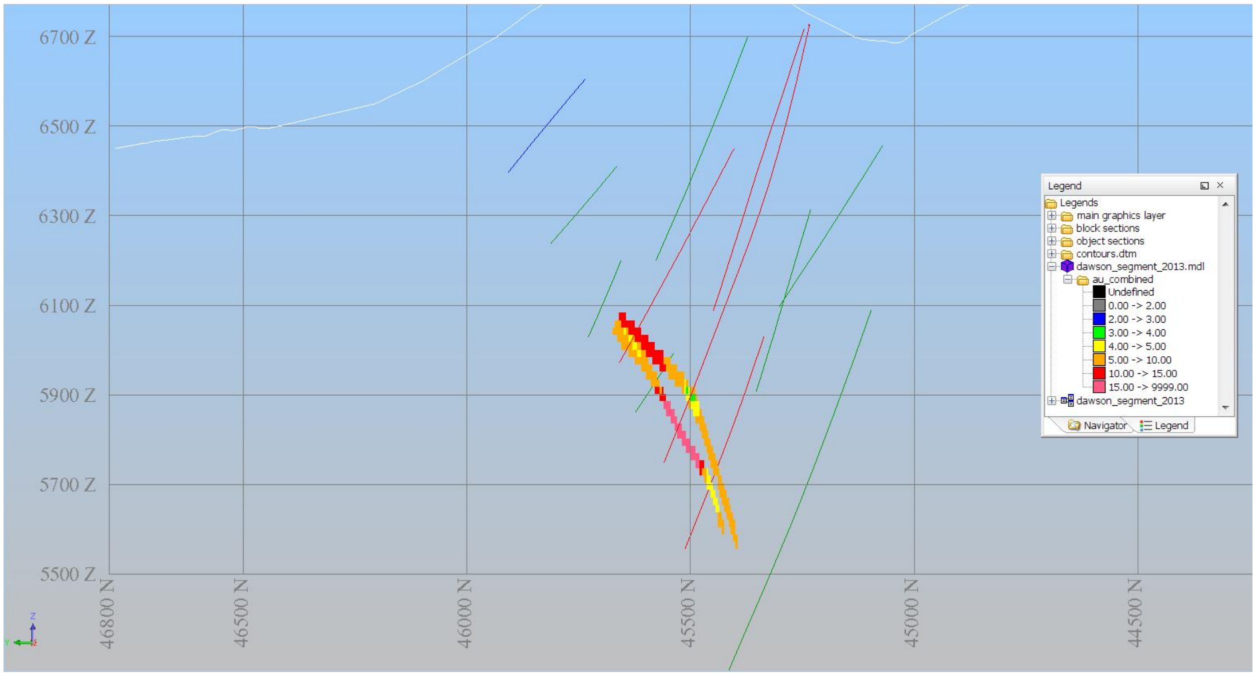


Figure 14.8: Dawson Segment Section 48100 East – Au (g/t) Block Grades (View to NE)

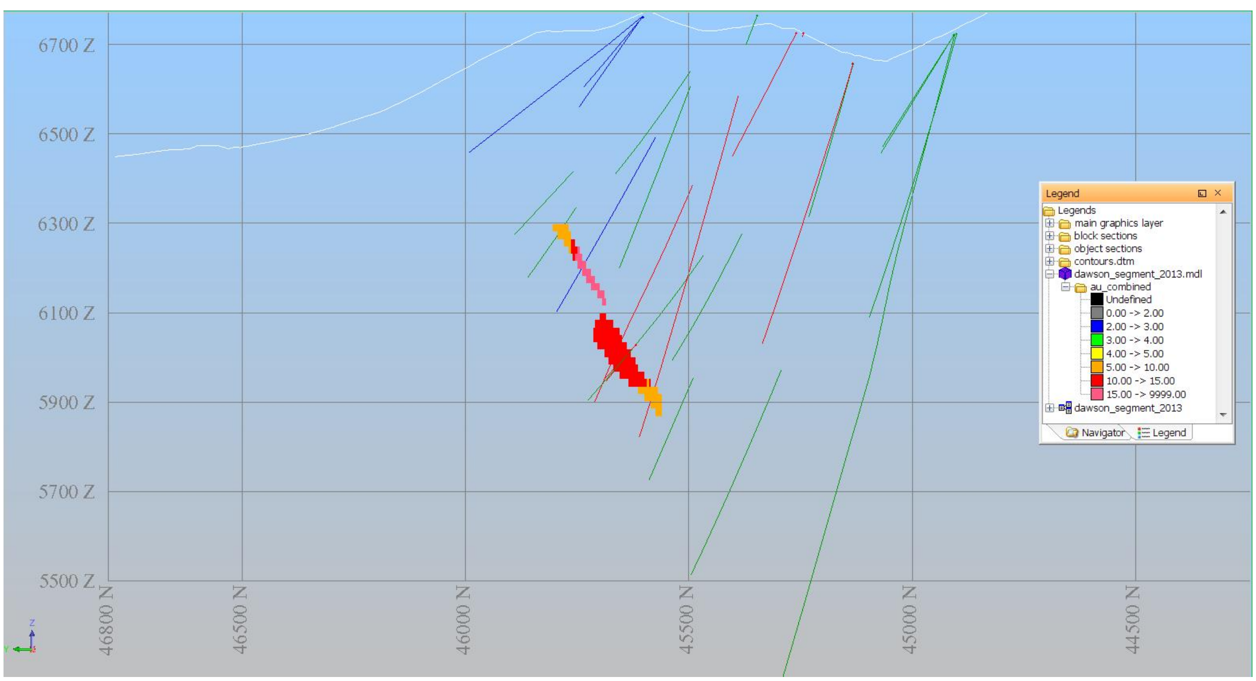


Figure 14.9: Dawson Segment Section 48200 East – Au (g/t) Block Grades (View to NE)

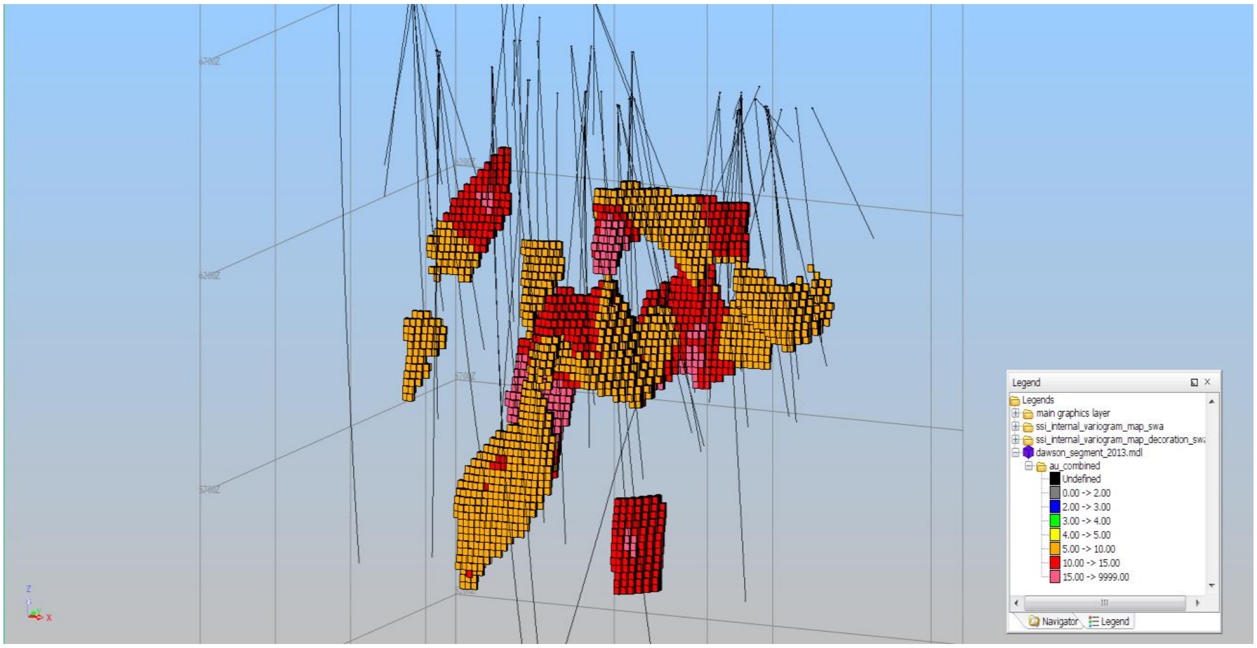


Figure 14.10: Isometric View of Dawson Segment Block Grades (>5 g/t Au - View to NW)

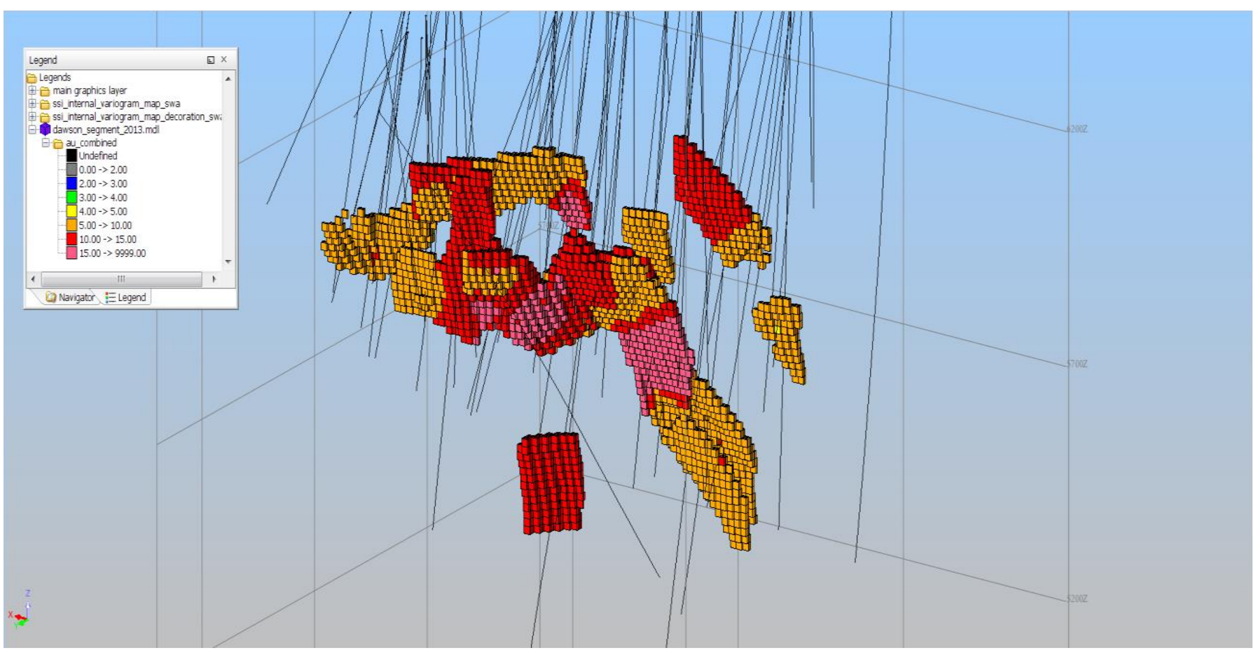


Figure 14.11: Isometric View of Dawson Segment Block Grades (>5 g/t Au – View to SE)

Descriptive statistics were calculated for the drill hole composite populations within the resource outlines and were compared to values calculated for the block model gold figures. Cumulative frequency plots and probability plots were also prepared and were appended to the Hilchey et al. (2013) NI 43-101 technical report by Mercator. The mean drill hole composite grades were found to compare acceptably with corresponding block grades, thereby providing a general check on the model with respect to the underlying assay composite populations. Results of the composite and block grade comparison are displayed below in Table 14-6.

Table 14-6: Comparison of Drill Hole Composite Grades and Block Model Grades for the Dawson Segment

Parameter	Au Composites	Au Blocks
Mean Grade	0.29 oz/tn (9.82 g/t)	0.29 oz/tn (9.90 g/t)
Minimum Grade	0.0003 oz/tn (0.01 g/t)	0.0003 oz/tn (0.01 g/t)
Maximum Grade	1.17 oz/tn (40 g/t)	1.16 oz/tn (39.88 g/t)
Variance	94.208	26.70
Standard Deviation	9.706	5.17
Coefficient of Variation	0.989	0.522
Count	111	9,414

The ID^2 block model interpolation methodology for the Dawson Segment was checked using Nearest Neighbour (NN) methodology within each gold domain. A NN block model was developed using omnidirectional search ellipsoids and weighted average composites created from the uncut assay data set over the full length of drill hole intersections within the resource solid models. An uncut ID^2 block model was generated for comparison using the same search ellipsoid parameters used in the resource ID^2 capped composite models. Global results of the uncapped ID^2 and NN models are presented below in Table 14-7 and in Figure 14.12. Both models at a 0 g/t Au cutoff value fill the constraint solids and thereby report similar tonnages. Grade results show acceptable global correlation at the lowest cut-off values and provide a basic cross-check on the preferred ID^2 interpolation method. Lower tonnages for the more declustered NN models at higher cut-offs primarily reflect influence of composite dilution over full drill hole intersection widths of grade domain models. This spatially limits the effect of length-limited high grade composites.

Table 14-7: Global Estimation Results for Dawson Segment at 0.00 g/t Au Reporting Threshold

Deposit / Method	Tonnes (Rounded)	Tons (Rounded)	Au (oz/tn)
Dawson Zone: ID2 Uncut	391,000	431,000	0.32 oz/tn (11.03 g/t)
Dawson Zone: NN Uncut	391,000	431,000	0.30 oz/tn (10.33 g/t)

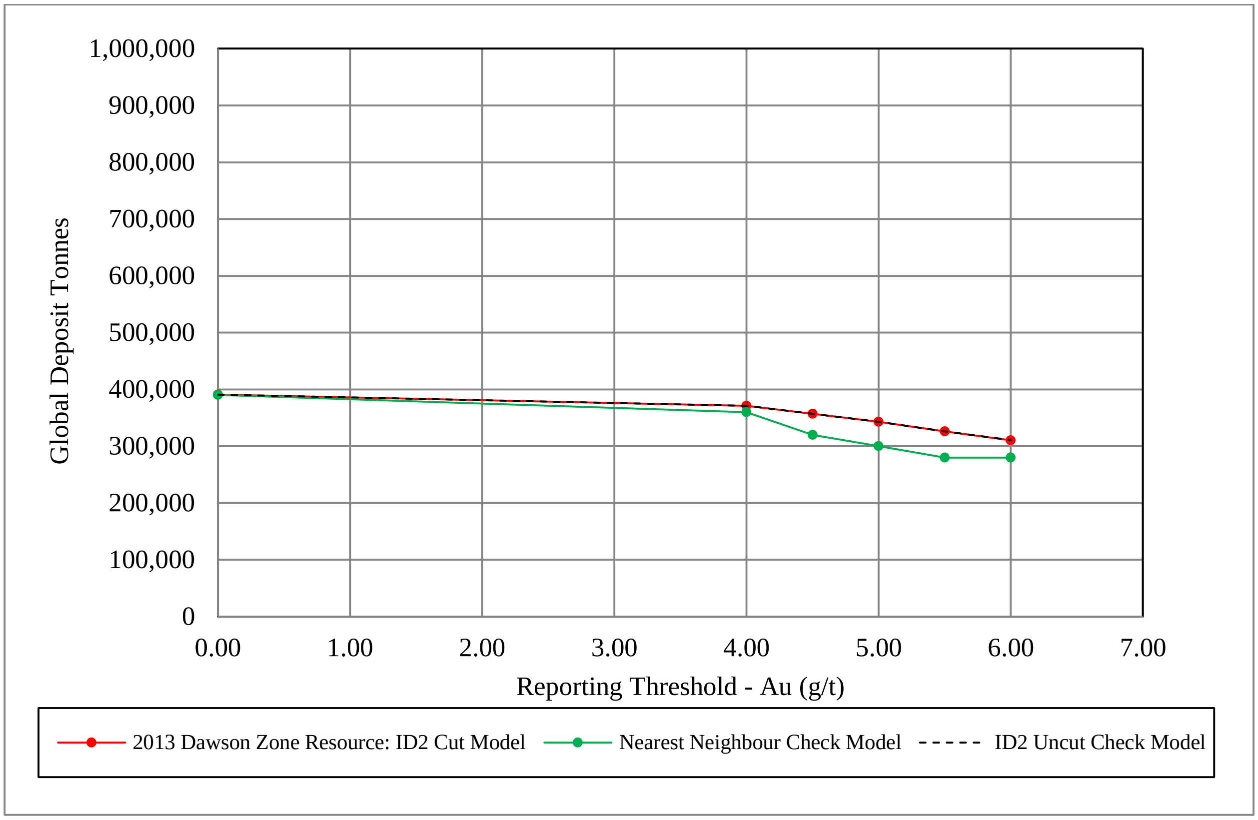


Figure 14.12: Comparison of Dawson Segment Block Model Interpolation Methods

14.1.9 Cut-Off Grade

A resource statement gold cut-off grade of 0.15 oz/tn (5.0 g/t) was used for the Dawson Segment resource estimate. Cut-off values at the preparation date of the Dawson Segment mineral resource estimate reflect technical input from Zephyr as well as Mercator's assessment of reasonable expectation of economic viability. More specifically, the steeply dipping, deeper, and narrow gold domains comprising a portion of the Dawson Segment are considered to have potential for conventional underground development and mining using both mechanized and narrow stoping methods. A gold price of US$1,200/ounce was used to support the cut-off grade assessment, along with assumed recovery factor of 90% for gold. Recoveries reflect historical assessments of these factors (e.g. AMSE, 1991). Mercator is of the opinion that future refinement of deposit modelling plus mine method and financial assessments could support reduction of the cutoff grade to a value to the 0.09 oz/tn (3 g/t) to 0.12 oz/tn (4 g/t) range. However, sharp telescoping of grade across narrow mineralized zone contact widths would limit the tonnage impact of such adjustment.

14.1.10 Dawson Segment Mineral Resource Statement

Block grade, block density, and block volume parameters for the Dawson Segment were estimated through the methods described in preceding sections of this report. The Inferred mineral resource estimate statement for the Dawson Segment is presented below in Table 14-8. Results are reported in accordance with Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines (CIM, 2010) as well as disclosure requirements of NI 43-101.

Table 14-8: Dawson Segment Mineral Resource Estimate Effective July 19, 2013

Resource Category	Au Cut-Off	Tonnes (Rounded)	Tons (Rounded)	Au Grade	Ounces**
Inferred	0.12 oz/tn (4 g/t)	371,000	409,000	0.29 oz/tn (10.09 g/t)	120,400
Inferred	**0.15 oz/tn* (5 g/t)**	**343,000**	**378,000**	**0.31 oz/tn (10.55 g/t)**	**116,300**
Inferred	0.18 oz/tn (6 g/t)	310,000	342,000	0.32 oz/tn (11.08 g/t)	110,400

*Resource statement cut-off value of 0.15 oz/tn (5 g/t) Au is highlighted by bolding
**Ounces may not sum due to rounding

Notes:
(1) Tonnes and tons have been rounded to the nearest 1,000.
(2) Ounces have been calculated from reported tonnes and g/t Au grade and are rounded to the nearest 100 ounces
(3) Contributing 5 ft (1.5 m) assay composites were capped at 1.17 oz/tn (40 g/t) Au
(4) The resource statement cut-off grade of 0.15 oz/tn (5.00 g/t) Au is highlighted in Table 14-8 above through bolding and reflects underground development potential based on a Au price of $US1,200/ounce.
(5) A density value of 0.082 tn/ft^3 (2.63 g/cm^3) was used for the Dawson Segment
(6) Mineral resources were estimated in conformance with the Canadian Institute of Mining, Metallurgy and Petroleum – Standards on Mineral Resources and Reserves – Definitions and Guidelines, as referenced in NI 43-101.
(7) The rounding of tonnes as required by NI 43-101 reporting guidelines may result in apparent differences between tonnes, grade and contained ounces.
(8) Mineral resources are not mineral reserves and do not have demonstrated economic viability. This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
(9) The quantities and grades of reported Inferred Mineral Resources are uncertain in nature and further exploration may not result in their upgrading to Indicated or Measured status

14.1.11 Comparison with Previous Mineral Resource Estimates

A series of historical mineral resource estimates appear in previous reporting associated with the Dawson Segment and these were itemized earlier in Section 6.3 of this report. These estimates were prepared sequentially by US Borax in 1985, ACA Howe in 1987 and 1988, and Uranerz in 1990 and 1991. Increased levels of drilling information and deposit knowledge were available for successive explorers. Estimates reflect various cutoff values and only ACA Howe applied grade capping (1.0 oz/tn). US Borax and ACA Howe used polygonal estimation methods and Uranerz used digital distance weighting methods.

On a tonnage comparative basis, the lowest historical tonnage estimate was reported by Mettler (1992) for Uranerz, as 263,000 tons grading 0.46 oz/tn Au (uncut) at a 0.15 oz/tn cutoff (238,593 tonnes grading 15.77 g/t Au at a 5.14 g/t cutoff). The highest tonnage estimate was reported at a much lower cutoff value by Theye (1989) for ACA

Howe and totalled 465,128 tons grading 0.24 oz/tn at a 0.08 oz/tn cutoff with grade capping at 1 oz/tn (421,964 tonnes grading 8.23 g/t Au at a 2.74 g/t cutoff with grade capping at 34.28 g/t).

As stated previously in this report, all of the estimates mentioned above are historical in nature and a qualified person has not done sufficient work to classify any of them as current mineral resources or mineral reserves. Zephyr is not considering these to be current mineral resources or mineral reserves and they should not be relied upon. All are superseded by the current mineral resource estimate for the deposit.

The 2013 Dawson resource estimate by Mercator falls within the tonnage and grade range for historical estimates noted above and reflects incorporation of new core drilling results not available to previous workers. Application of grade capping distinguishes the Mercator estimate from those of US Borax and Uranerz and estimates show the Dawson deposit to consist of moderately high grade gold mineralization occurring in a series of discrete, steeply dipping, tabular mineralized bodies.

14.2 Windy Gulch

14.2.1 Introduction

The 2016 mineral resource estimate for the Windy Gulch Segment was completed by Brian Thomas, P.Geo., with senior peer review provided by Greg Warren and Jerry DeWolfe, P.Geo. The resource estimate is based on assay data from drill and surface channel sample programs completed by Zephyr and its predecessors between 1983 and 2016. All modelling procedures and resource definitions conform to CIM best practice guidelines as defined in May 2014 and referred to in NI 43-101 regulations.

The software used to complete the resource model and estimate was Datamine Studio 3, release 3.24.25.0 (Datamine). The project was completed using extended precision and all processes were documented by recording HTM scripts for future reference.

14.2.2 Drill Hole Data

A total of 80 holes were provided by Zephyr, containing 2,297 assay intervals and having a total core length of 18,837 ft. All drill hole data were provided as of October 17, 2016.

Historical collar data, prior to 2016, were provided in a local grid coordinate system. Golder translated the collar locations into the Colorado State Plane coordinate system (NAD 83) based on the known location of Dawson hole GC59 in both coordinate systems. On completion of the translation, the collar locations were adjusted 10 ft to the north and 10 ft to the west in order to best fit the topography and road locations. This shift in data was validated based on surveyor notes found for the Dawson Project indicating that actual hole locations were found to be off approximately 10 to 12 ft (3.0 to 3.7 m) in the north and west directions. Historical surface hole collar locations are no longer available for re-surveying. The 2016 drill holes were surveyed in the Colorado State Plane coordinate system and did not require translation.

The Windy Gulch drill hole data were imported into Datamine from electronic CSV files (comma-separated values). Minor interval overlap errors were identified in the lithological data which were corrected prior to modelling.

The drill hole file was reviewed in 3D to validate the accuracy of the collar locations, hole orientations, and down hole trace, and the assay data were analyzed for out of range values. The drill hole database was determined by Golder to be of suitable quality to support an indicated or inferred resource estimate, but insufficient for a measured resource due to concerns over the accuracy of historical drill hole collar locations and insufficient historical QA/QC data.

14.2.3 Geological Interpretation

14.2.3.1 Mineral Domains and Sample Selection

A total of five mineral domain wireframe solids were created to represent mineralization in the Windy Gulch Segment as illustrated in Figure 14.13 and Figure 14.14. Zone 1 is the main mineralized zone and is shown in red. Zone 2 is a broad footwall zone below Zone 1 shown in pink. Zone 3 is a small zone in the footwall of Zone 1 shown in green. Zone 4 is a small zone in the footwall below Zone 2 shown in blue and Zone 5 is a small high grade zone in the hanging wall of Zone 1 shown in purple.

An approximate 0.015 oz/ton (0.5 g/t) Au cut-off was used to constrain the 3D mineral solids in areas of continuous mineralization; however, some lower grade material was included to maintain geological continuity locally.

These wireframe solids represent hard boundaries between each zone, meaning that samples were constrained to a single zone and not used to estimate multiple zones.

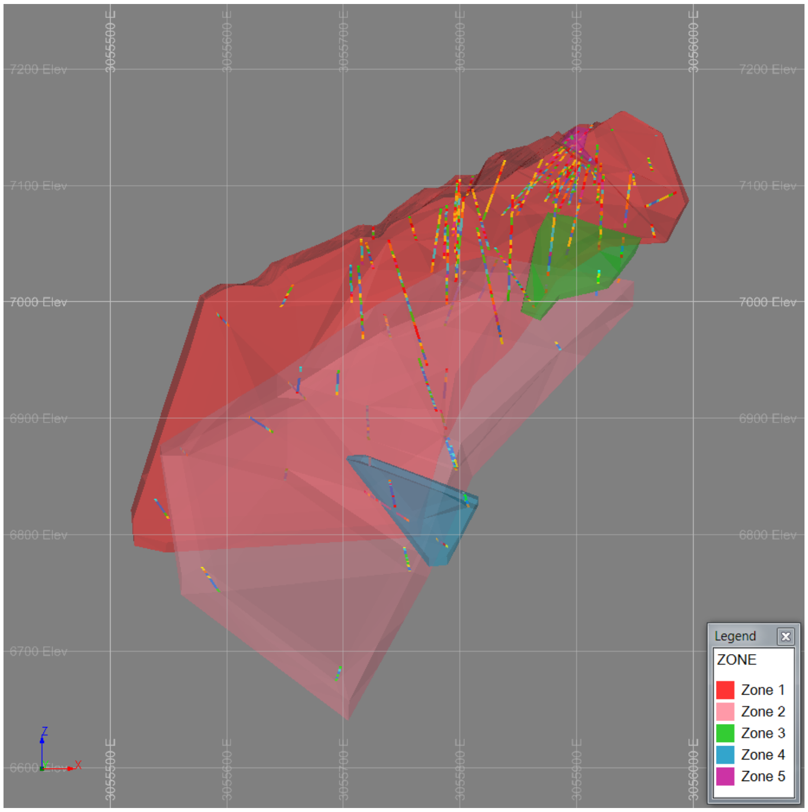


Figure 14.13: East–West View of Mineral Domains Windy Gulch Segment (facing north)

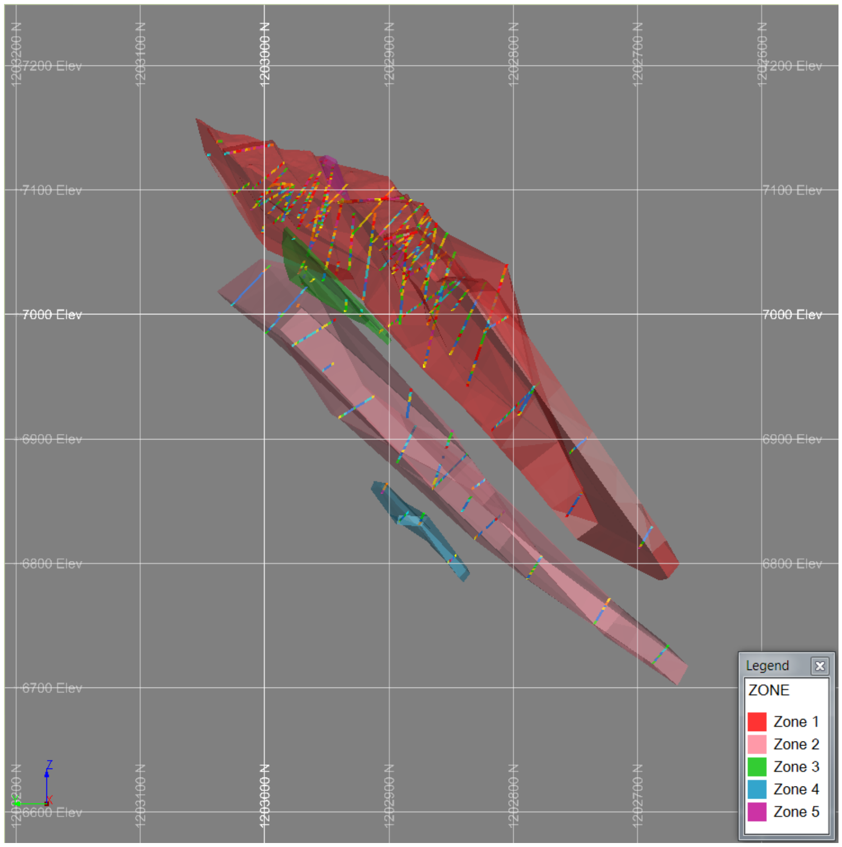


Figure 14.14: North–South View of Mineral Domains Windy Gulch Segment (facing east)

Raw sample intervals were captured inside each domain wireframe and verified visually to confirm the accuracy of the process. A total of 1,125 samples were captured for a total sample length of 3,394 ft (1,034 m) within all five domains. Table 14-9 provides the sample break down by domain.

Table 14-9: Summary of Captured Samples Tamarack North Project

Domain	No. of Holes	No. of Samples	Total Sample Length (ft)
Zone 1	57	890	2,747
Zone 2	17	179	492
Zone 3	7	26	74
Zone 4	4	17	49
Zone 5	5	13	32
Total	**90**	**1,125**	**3,394**

14.2.4 Exploratory Data Analysis

Descriptive statistics combined with a series of histograms, X-Y scatter plots and probability plots were used to analyze the grade distribution of each sample population and to assess for the presence of outliers in for each domain.

14.2.4.1 Descriptive Statistics

Table 14-10 provides a summary of the descriptive statistics for the sample populations captured from within each mineral domain.

Table 14-10: Descriptive Statistics of the Windy Gulch Segment Sample Population

Domain	Field	Samples	Min (oz/tn)	Max (oz/tn)	Mean (oz/tn)	STD Deviation	Skewness	Coefficient of Variation
Zone 1	Au	890	0.01	2.283	0.062	5.81	7.40	2.72
Zone 2	Au	179	0.01	4.659	0.036	10.60	13.89	8.66
Zone 3	Au	26	0.01	0.083	0.023	0.76	1.32	0.98
Zone 4	Au	17	0.01	0.231	0.047	2.43	1.96	1.52
Zone 5	Au	13	0.01	0.337	0.109	3.94	0.91	2.34

Note: Sample statistics weighted by length for all domains except Zone 1, which was weighted by a declustering weight.

Figure 14.15 and Figure 14.16 provide examples of the frequency distribution of the Au sample populations for Zone 1 and 2. The gold population was found to be positively skewed.

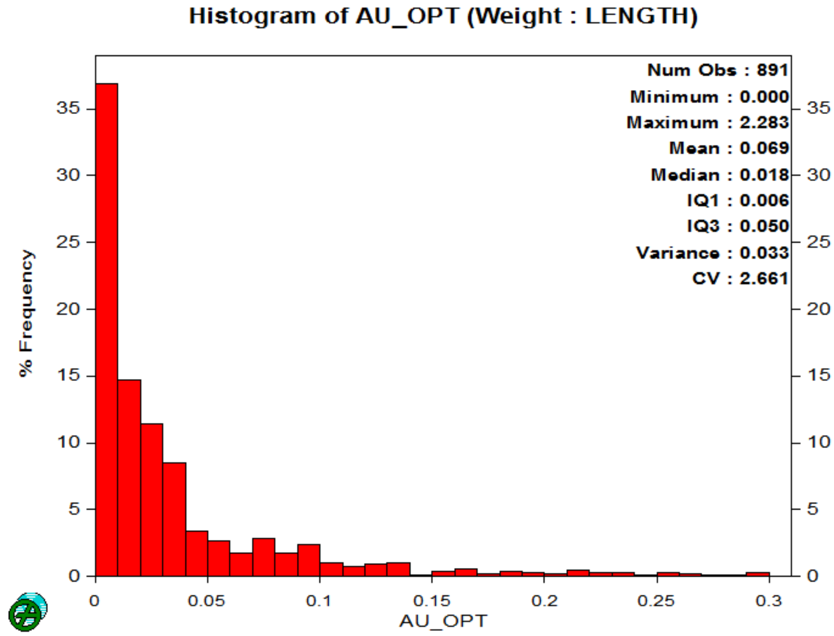


Figure 14.15: Histogram of Au for Zone 1

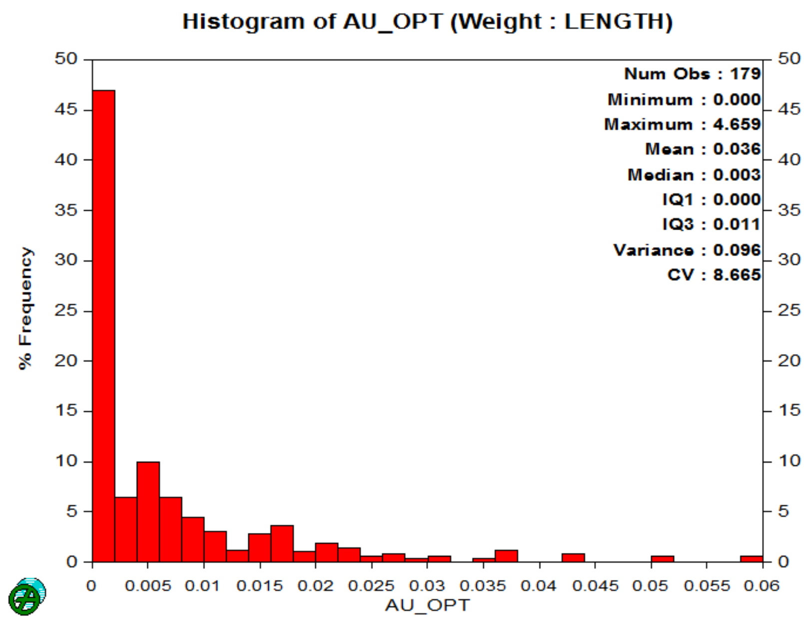


Figure 14.16: Histogram for Zone 2

Un-assayed intervals were assumed to be waste and assigned a metal value of 0.00007 oz/tn (0.0025 g/t) representing half the detection limit for Au analysis. There was only one interval with absent Au assays for the entire captured sample population.

14.2.4.2 Outliers

X-Y scatter plots were generated for all zones in order to assess the sample population for outlier values. High grade outlier data have the potential to bias the block model estimates if they are not handled by top cutting or otherwise restricting their influence through other estimation criteria. A minor number of high grade outliers were identified in the Au population of Zone 1 as shown in Figure 14.17. The identified Au outliers were top-cut to a value of 1.167 oz/tn based on review of the scatter plots and to maintain consistency with the Dawson Segment. Top cutting reduces the value of an outlier to a set maximum value.

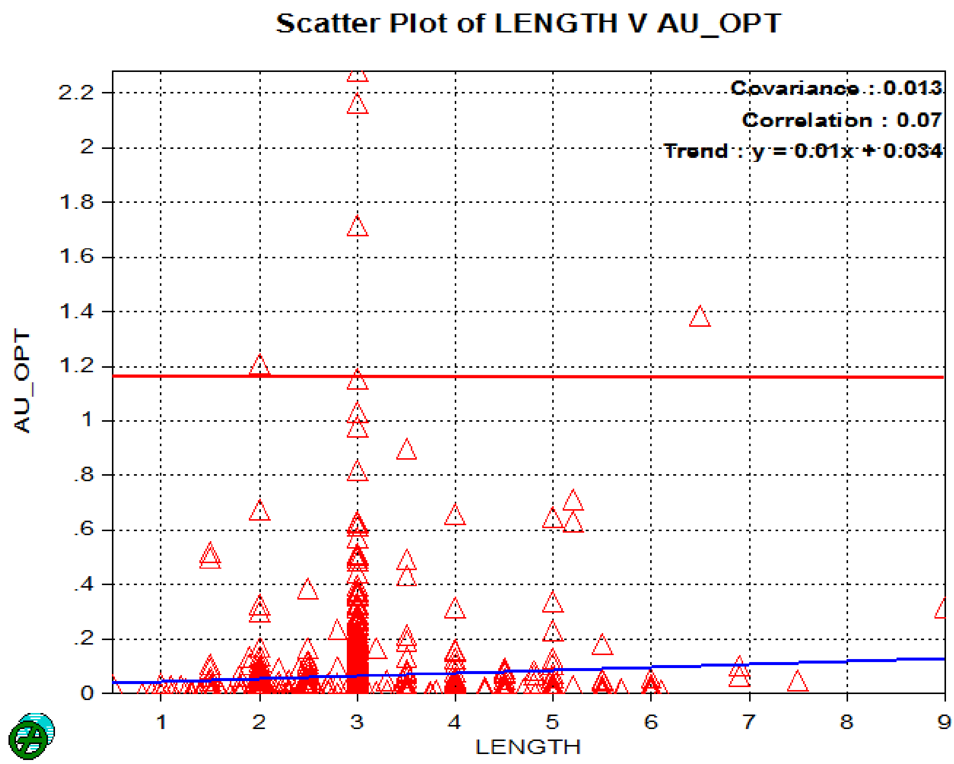


Figure 14.17: Scatter Plot of Au vs Sample Length (Zone 1)

Table 14-11 summarizes the number of samples that were top cut for each zone.

Table 14-11: Summary of Top Cuts

Zone	Metal	No. of Samples Cut
Zone 1	Au	5
Zone 2	Au	1
Zone 3	Au	0
Zone 4	Au	0
Zone 5	Au	0

14.2.5 Compositing

Compositing samples is a technique used to give each sample a relatively equal length weighting in order to reduce the potential for bias due to uneven sample lengths. A histogram of raw sample length was generated for Zone 1 in order to determine the most common sample length used at the Windy Gulch Segment as shown in Figure 14.18.

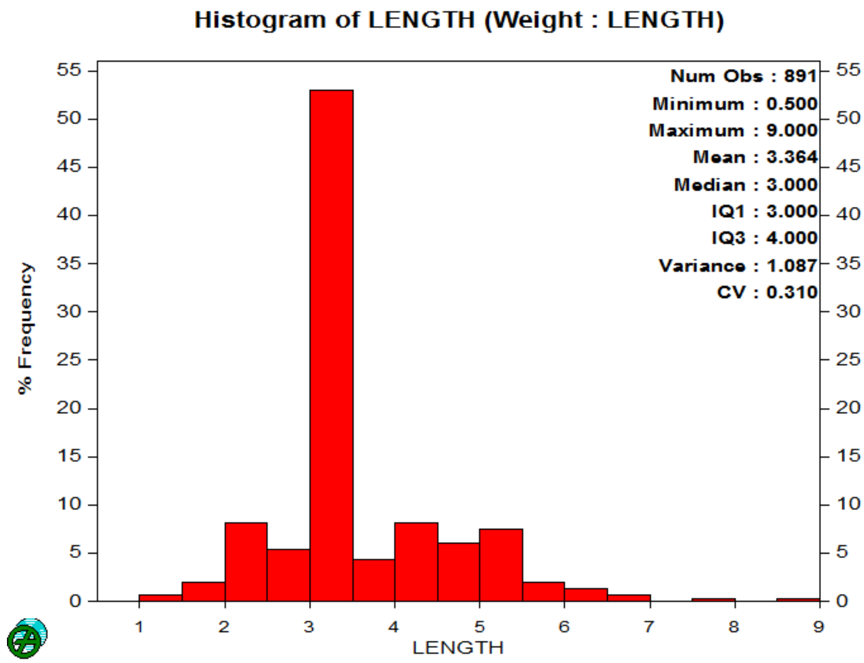


Figure 14.18: Histogram of Raw Sample Length (Zone 1)

Samples captured within each mineral domain were composited to a mean length of 3 ft (0.9 m). This interval was chosen because it is the most common sample length and provides a reasonable level of sample support. An option to use an average composite length was chosen in order to prevent the creation of short composites that are generally created when using a fixed length.

The composite samples were validated visually in plan and section, and a histogram of composite length was generated in order to confirm that the compositing was completed as expected. The histogram displayed a normal distribution around the chosen composite length of 3 ft. Statistical comparisons confirmed the total length and mean grade of the composites were found to match that of the raw captured samples.

14.2.5.1 Bulk Density

A total of 102 density measurements were obtained from core samples taken during the 2013 and 2016 drill programs. The density values were determined from pulp by means of a gas pycnometer.

These values were used to estimate bulk density into the block model for Zone 1 using ID^2 and NN interpolation. If blocks did not receive a density value based on ID^2 they were assigned the NN value, or a default value of 2.75 tonnes per cubic metre (t/m^3), based on the declustered mean of Zone 1. There was insufficient data available to estimate any of the other zones. A default density value of 2.73 t/m^3 was assigned to Zones 2 through 5 based on a declustered average of Zone 1 density values less than 3.0 t/m^3. Zone 1 contains sections of pyrite rich banding that have a density values greater than 3.0 t/m^3.

For imperial unit reporting, bulk density (BD) was converted from tonnes per cubic metre (t/m^3) to tons per cubic foot (tn/ft^3) based on the formula:

BD Imperial = BD Metric x 0.0312139803

14.2.6 Resource Estimation

14.2.6.1 Unfolding

The “unfold” process, within Studio 3, was used to transform the composite sample data from Cartesian coordinates into an Unfolded Coordinate System (UCS), as defined by the geometry of the footwall and hanging wall contacts of the wireframe solids for Zone 1 and Zone 2. This transformation essentially accounts for bends, pinches, and swells in the mineral model, allowing for more robust variogram calculations and grade estimation, lowering the potential for grade bias in the block model. This was considered an appropriate process to employ given the variable orientations of each 3D mineral wireframe. Unfolding was not used for Zones 3 to 5.

Strings representing the footwall and hanging wall contacts of the deposit were constructed and tagged in vertical section perpendicular to the plunge direction of mineralization as shown in Figure 14.19. These strings are then used to transform the composite samples into the unfold coordinate system. The same unfold strings are used in the grade estimation process to unfold the blocks into the same transformed system as the composite samples. The process unfolds discretization points from the prototype model and estimates the grades for each in the unfold co-ordinate system. The process then assigns the estimated grades back to the corresponding cell in the Cartesian model. In the unfold coordinate system, the X-axis is assigned to UCSA, which represents the across strike direction (thickness) of the zone, the Y-axis is assigned to UCSB, representing the down-dip direction, and the Z-axis becomes UCSC, representing the strike direction of the zone.

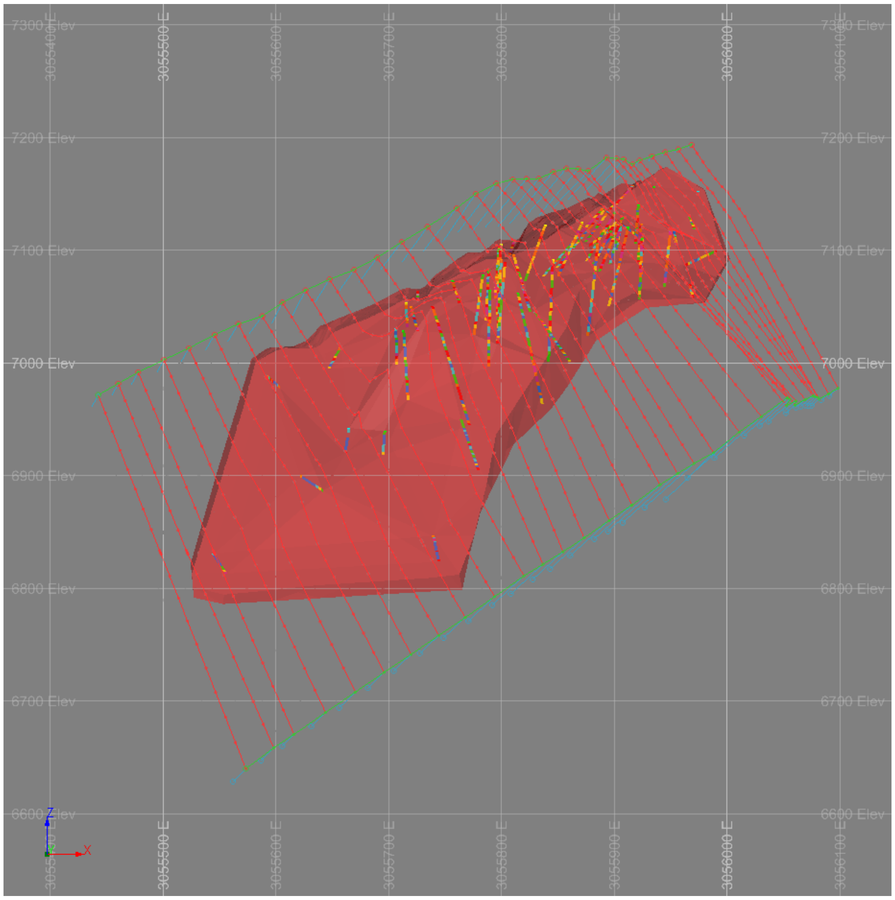


Figure 14.19: Zone 1 Unfold Strings, East–West View Facing North

The unfolded samples were validated visually in unfold space for each zone. Quadrilateral strings (Quads) created during the process were inspected to confirm that unfolding had performed as expected.

Visual inspection of the NN model confirmed that the unfolding process had worked as expected for all zones.

14.2.6.2 Grade Variography

Experimental gold variograms were generated from the unfolded composite data for Zone 1 in order to assess the spatial variability for the purpose of assigning Kriging weights to the composite samples and define the approximate dimensions of the search ellipse volume. Samples situated in the directions of preferred geological continuity receive higher Kriging weights, resulting in a greater influence on the block estimate.

Pairwise relative experimental grade variograms were generated based on the parameters outlined in Table 14-12. Variograms were not generated for Zones 2 through 5 due to insufficient data.

Table 14-12: Grade Variogram Parameters

Elements	Zone 1
Rotations	0
Lag Distance	20 ft
Number of Lags	15
Sub-lag Distance	5 ft
Number Lags to be Sub-lagged	5
Regularization angle	30°
Number of Azimuths	2
Cylindrical search radius	30 ft

A two-structure spherical variogram model was fitted to the data. The variogram model for Zone 1 is provided in Figure 14.20 to Figure 14.22. Numbers shown on the data points represent the number of sample pairs used to calculate the variogram point value. Note that the number of pairs available within the first 20 ft (6 m) are generally low in the down-dip and along strike directions as a function of the drill spacing. These points are deemed to be less reliable for variogram modelling.

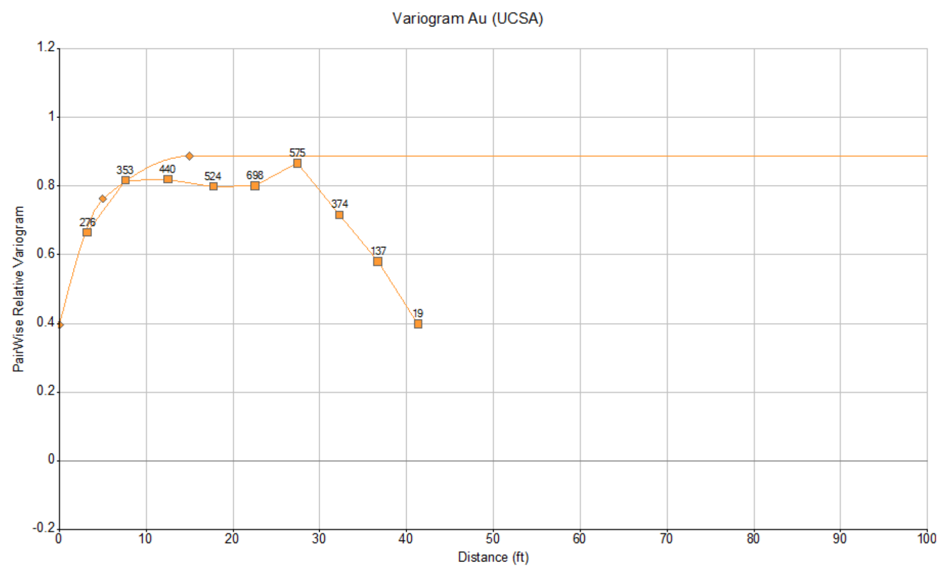


Figure 14.20: Zone 1 Across Strike Direction (UCSA) Au Variogram Model

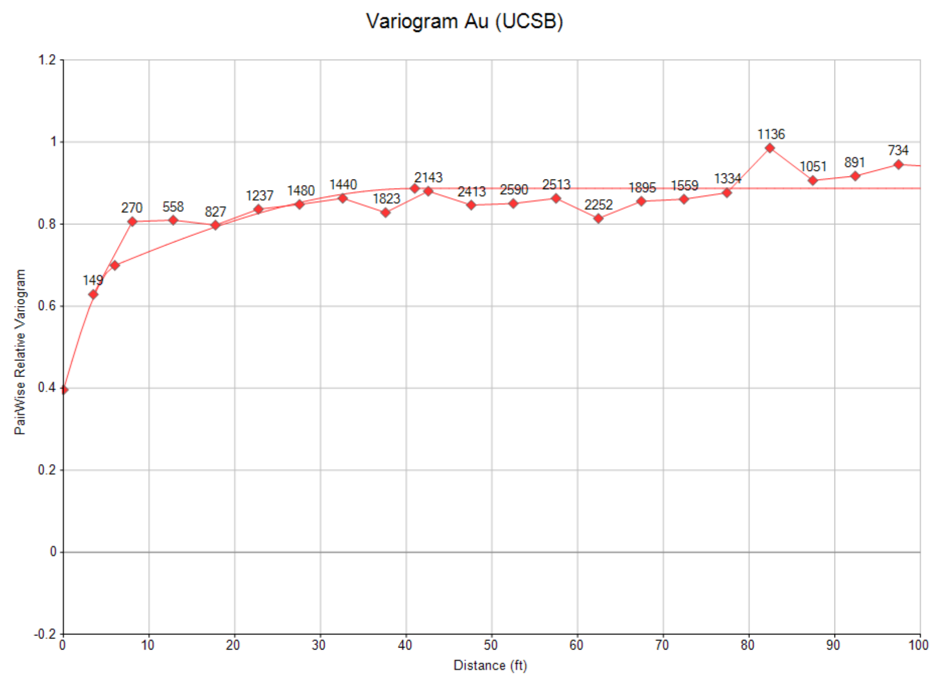


Figure 14.21: Zone 1 Down-Dip Direction (UCSB) Au Variogram Model

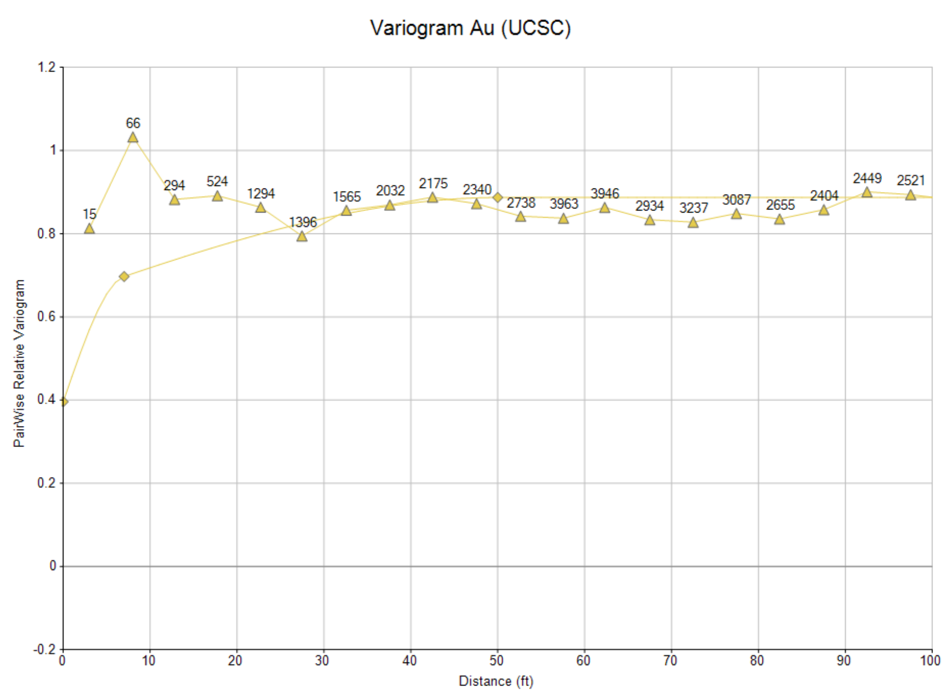


Figure 14.22: Zone 1 Along Strike Direction (UCSC) Au Variogram Model

A summary of the variogram model is provided in Table 14-13.

Table 14-13: Summary of Zone 1 Variogram Model (Unfold Coordinate System)

ELEMENT	NUGGET	1st Structure (ft)				2nd Structure (ft)			
		X-Range	Y-Range	Z-Range	Variance	X-Range	Y-Range	Z-Range	Variance
Zone 1 Au	0.4	5	6	7	0.251	15	41	50	0.18

Note: In the unfolded coordinate system, X axis (vertical) is across the strike of mineralization, Y axis is down-dip, and Z axis is along strike.

The down-dip (Y-Range) and along strike (Z-Range) directions of the mineralization were determined to be the directions of greatest grade continuity. The second structure range of each axis was used as the basis to define the search ellipse dimensions used for interpolating into the 3D mineral resource block model.

14.2.6.3 Block Model Definition

The Windy Gulch prototype model covers a 3D block of ground in Colorado State Plane NAD 83 grid coordinates from 3,055,000 east to 3,057,000 east, 1,204,000 north to 1,201,800 north, and 6,300 to 7,600 ft elevation. Block shape and size is a function of geometry of the deposit, density of sample data, and the expected potential smallest mining unit (SMU). Based on this rational, a parent block size of 10 ft (east-west) by 5 ft (north–south) by 5 ft (elevation) was chosen. The block model definition parameters are summarized in Table 14-14.

Table 14-14: Summary of Block Model Prototype Details

Origin (Lower Left Corner)			Block Size (ft)			Number of Blocks			Extent (ft)		
X	Y	Z	X	Y	Z	X	Y	Z	X	Y	Z
3,055,000	1,201,800	6,300	10	5	5	200	440	260	2,000	2,200	1,300

Each mineral solid was filled with blocks using the parameters described in Table 14-14. The wireframe volumes were then compared to the block volumes to confirm there were no errors during the process.

14.2.7 Estimation Methodology

Block grades for Zones 1 and 2 were estimated using various interpolation methods including: Ordinary Kriging (OK), ID^2, Inverse Distance Cubed (ID^3), and NN. Ordinary Kriging uses the variogram model to calculate the weights assigned to each sample based on distance and direction of geological continuity. Therefore samples closest to the block centre in the directions of greatest geological continuity will receive the highest weighting. The inverse distance methods assign weights to samples based on the samples distance from the block centroid, with closer samples having a higher weighting independent of directions of geological continuity. Higher powers used result in closer samples having a higher influence on the block grade. The NN method assigns the block the grade of the closest sample and is generally used as a declustering technique and for model validation purposes as core sample grades are not representative of block grades due to the large volume differences (volume variance). Zones 3 to 5 were estimated using ID^2, ID^3, and NN only since there was insufficient data for variography and these zones were not unfolded (Table 14-15). Each block was estimated using a set of discretization points consisting of four points in the strike direction and three across strike and in the vertical directions. After review of the models, the ID^3 estimation was chosen as the final model as it was believed to be the most representative of the grade distribution and contained the least grade smoothing, making it the most suitable for reporting tonnage and grade using a resource cut-off.

Table 14-15: Summary of Estimation Methodology

Geological Doı	Interpolation Me	Unfolding	Top Cutting
Zone 1	OK, ID^2, ID^3, NN	Yes	Yes
Zone 2	OK, ID^2, ID^3, NN	Yes	Yes
Zone 3	ID^2, ID^3, NN	No	Yes
Zone 4	ID^2, ID^3, NN	No	Yes
Zone 5	ID^2, ID^3, NN	No	Yes

An anisotropic search ellipse radius of 5 ft (X-axis), 40 ft (Y-axis), and 50 ft (Z-axis) was used for the initial search distances for all zones and is based on the modelled second structure variogram range of Zone 1. If block grades were not estimated in the first search, then expansion factors of 1.5x and 3x were applied to the search ellipse dimensions for the second and third searches, respectively. The search parameters used for each zone are summarized in Table 14-16 . Note that for Zones 1 and 2, these search parameters are used in unfolded space during the interpolation process, where X is across the deposit, Y is down-dip, and Z is in the strike direction. The search distance of the X-axis was restricted to 5 ft in order minimize grade smoothing in the across strike direction. Search strategies for each domain used an elliptical search with a minimum of 4 samples and a maximum of 12 samples, with a sample restriction of a maximum of 2 samples per hole. Any un-estimated blocks were then assigned the NN model grades.

Table 14-16: Summary of Search Parameters (unfolded)

	1st Search					2nd Searc			3rd Searc		
Element	X-Range	Y-Range	Z-Range	Min. Samples	Max. Samples	SVOL Factor 2	Min. Samples	Max. Samples	SVOL Factor 3	Min. Samples	Max. Samples
Zone 1*	5	40	50	4	12	1.5	4	12	3	2	12
Zone 2*	5	40	50	4	12	1.5	4	12	3	2	12
Zone 3	40	50	5	4	12	2	4	12	4	2	12
Zone 4	40	50	5	4	12	2	4	12	4	2	12
Zone 5	40	50	5	4	12	2	4	12	4	2	12

** Zones 1 and 2 estimated in the Unfold Coordinate System (UCS)*

14.2.8 Mineral Resource Classification

Resource classifications were assigned to broad regions of the block model based on a combination of drill hole density and the search volume used to estimate the grade of the blocks. Areas where the drill hole spacing was on average less than 40 ft and the block grade was estimated in the first search volume were classified as indicated mineral resource. Areas where the drill hole spacing was wider than 40 ft and the block grades were estimated in the second or third search volume were classified as inferred mineral resource. Measured mineral resource was not outlined from the block model due to uncertainties related to the accuracy of historical drill hole survey data and lack of QA/QC information; otherwise the drill spacing would have been suitable in some closely drilled areas. Figure 14.23 and Figure 14.24 outline the mineral resource classifications assigned to the Zone 1 model. The green area is indicated mineral resource and the blue is inferred mineral resource.

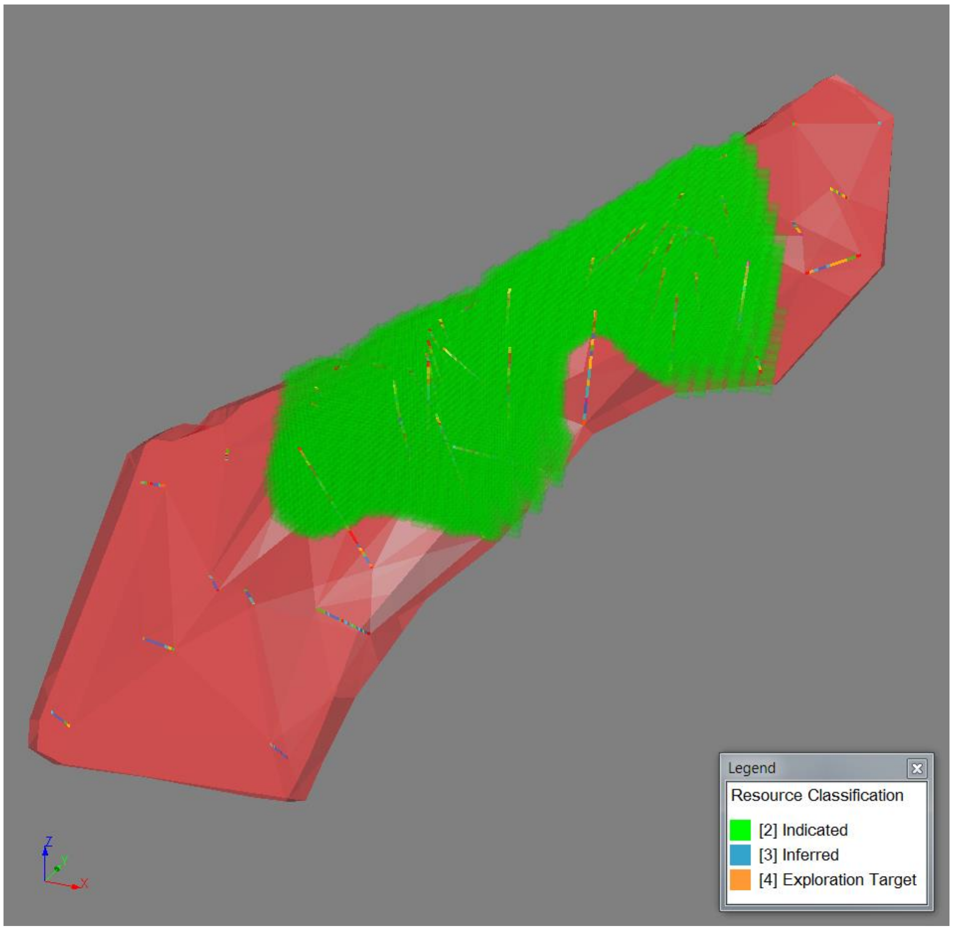


Figure 14.23: Distribution of Indicated Resource in Zone 1 (facing northwest)

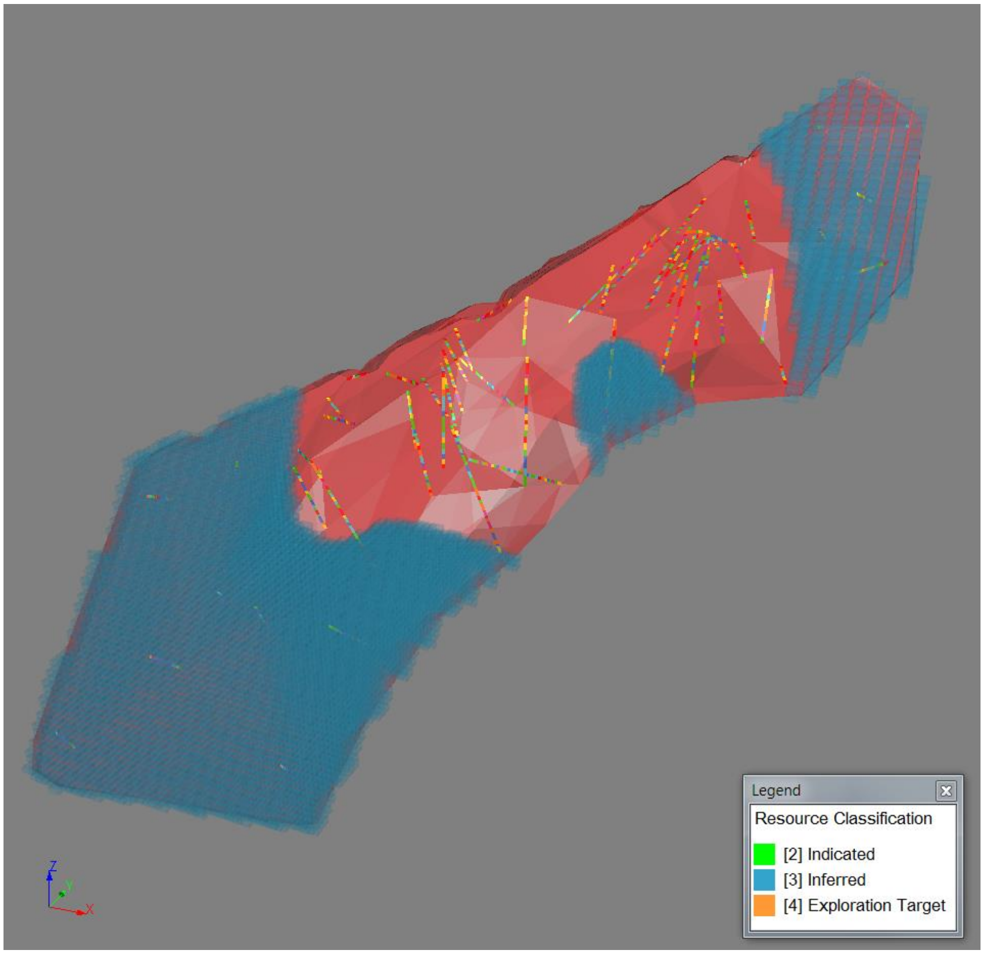


Figure 14.24: Distribution of Inferred Resource in Zone 1 (facing northwest)

Zones 2, 3, and 4 were classified entirely as inferred resource, and Zone 5, the small high grade zone in the hanging wall of Zone 1, was classified entirely as indicated resource.

Table 14-17 summarizes the global data density statistics by classification and domain.

Table 14-17: Global Data Density Statistics

Domain	Classification	No. of Holes	No. of Raw Samples	Tons Per Hole	Tons Per Raw Sample
Zone 1	Indicated Resource	44	733	4,158	250
Zone 5	Indicated Resource	5	13	65	25
Zone 1	Inferred Resource	15	158	8,908	846
Zone 2	Inferred Resource	17	179	14,961	1,421
Zone 3	Inferred Resource	7	26	1,068	288
Zone 4	Inferred Resource	4	17	1,307	308

The number of blocks estimated in each of the three search volumes was reviewed to ensure that the proportion of cells estimated for each was relatively consistent with the spacing of the drill hole data and the classification assigned to the model. Sixty-one and 92 percent of the blocks in Zones 1 and 5, respectively, were estimated within the first search volume while Zones 2 and 4 were considerably less. Zone 3 has similar statistics as Zone 1, but it was a QP decision, based on the distribution of holes to classify it as inferred resource. Table 14-18 summarizes the search volume statistics for each zone.

Table 14-18: Summary of Block Estimates by Search Volume

Domain	1st Search (%)	2nd Search (%)	3rd Search (%)
Zone 1	61	26	13
Zone 2	9	31	60
Zone 3	69	29	2
Zone 4	22	50	28
Zone 5	92	7	1

14.2.9 Block Model Validation

The model validation process included a visual comparison of block and composite grades in plan and section, along with a global comparison of mean grades and swath plots that compare average grades throughout regularly spaced sections of the model. Block grades were visually compared to the drill hole composite data in all domains to ensure agreement. No material grade bias issues were identified, and the block grades compared well to the composite data as demonstrated in Figure 14.25 and Figure 14.26 (regularized / partial percentage model shown).

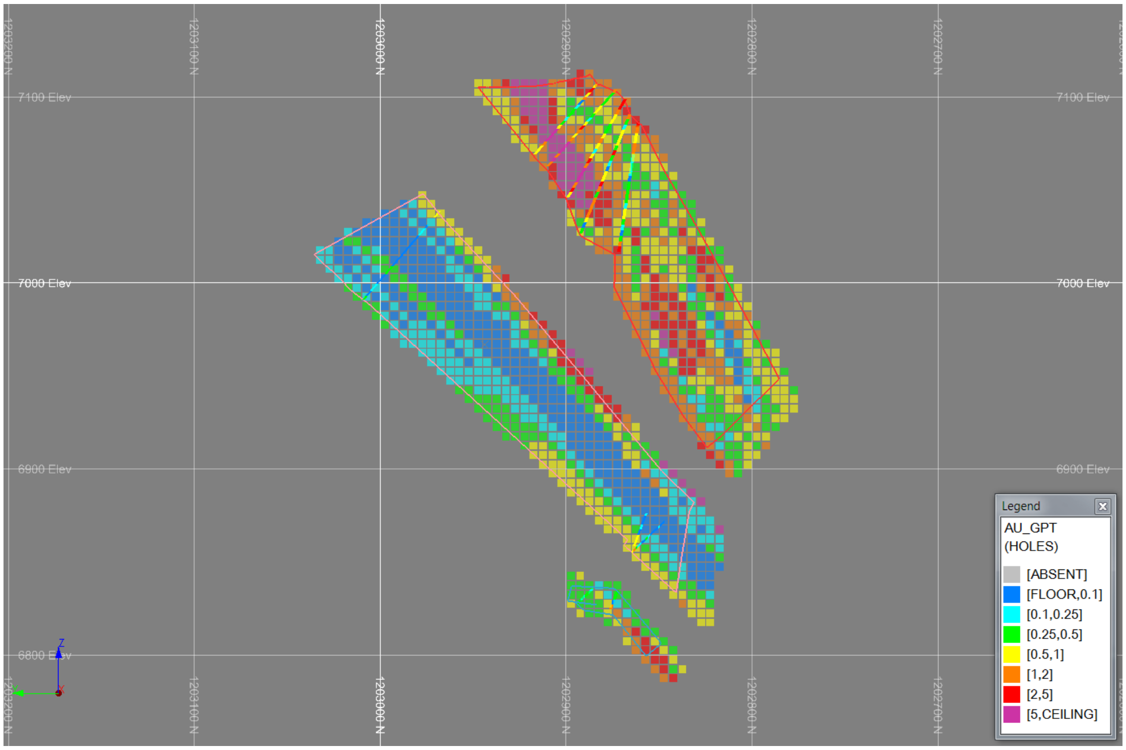


Figure 14.25: North–South Section 3055800 E (facing east)

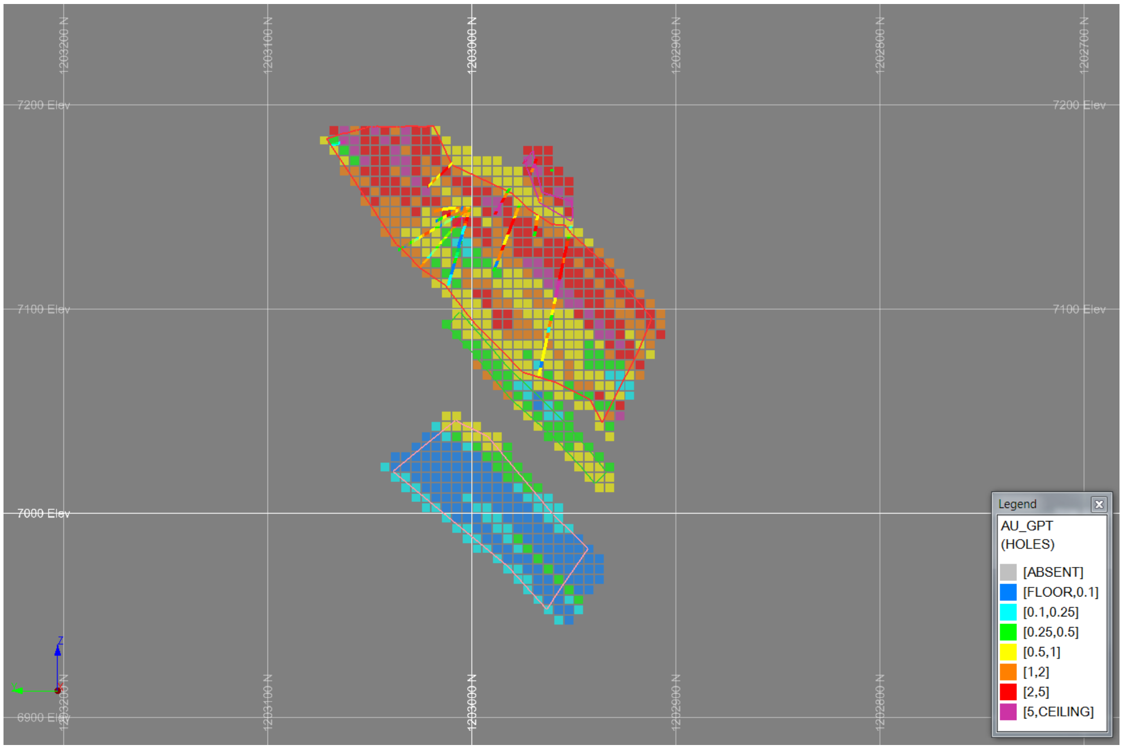


Figure 14.26: 138 Zone Domain North–South Section 3055920 E (facing east)

Global statistical comparisons between the composite samples, NN estimates and OK/ID estimates for each zone were compared to assess global bias. The NN model grades represent the declustered composite data. Clustering of drill hole data can result in differences between the global means of the composites and NN estimates. The global means of the NN, OK, and ID estimates should also be very similar to confirm that there is no global grade bias in the model. The results summarized in Table 14-19 indicate that no material grade bias was found in Zones 1 to 4. Zone 5 indicates a large difference between the NN model and the ID models, but this is a very small zone consisting of less than 300 t, so it is not entirely unexpected. One issue with using a NN model to represent declustered samples is that it is possible that not all samples will be assigned to a block in the model.

Table 14-19: Statistical Comparison of Global Mean Grades

Source	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5
	Mean Au (oz/tn)	Mean Au (oz/tn)	Mean Au (oz/tn)	Mean Au (oz/tn)	Mean Au (oz/tn)
Composites	0.064	0.018	0.022	0.047	0.109
NN Model	0.053	0.013	0.021	0.055	0.078
ID^2 Model	0.053	0.013	0.020	0.049	0.109
ID^3 Model	0.053	0.013	0.020	0.051	0.104
OK Model	0.052	0.012	n/a	n/a	n/a

Notes: For the purpose of calculating global statistics, composite samples were weighted by length and block grades weighted by tonnes.

A swath plot was generated from north–south slices throughout the Zone 1 block model and is presented in Figure 14.27. The swath plots compare the model grades to the declustered composite grades (NN model) in order to identify local grade bias in the model. Review of these swath plots did not identify any significant bias in the model that is material to the resource estimate as there was general agreement between the declustered composites (NN model) and the final ID^3 model grades. A small discrepancy is noted in the west end due to a low number of samples in this area.

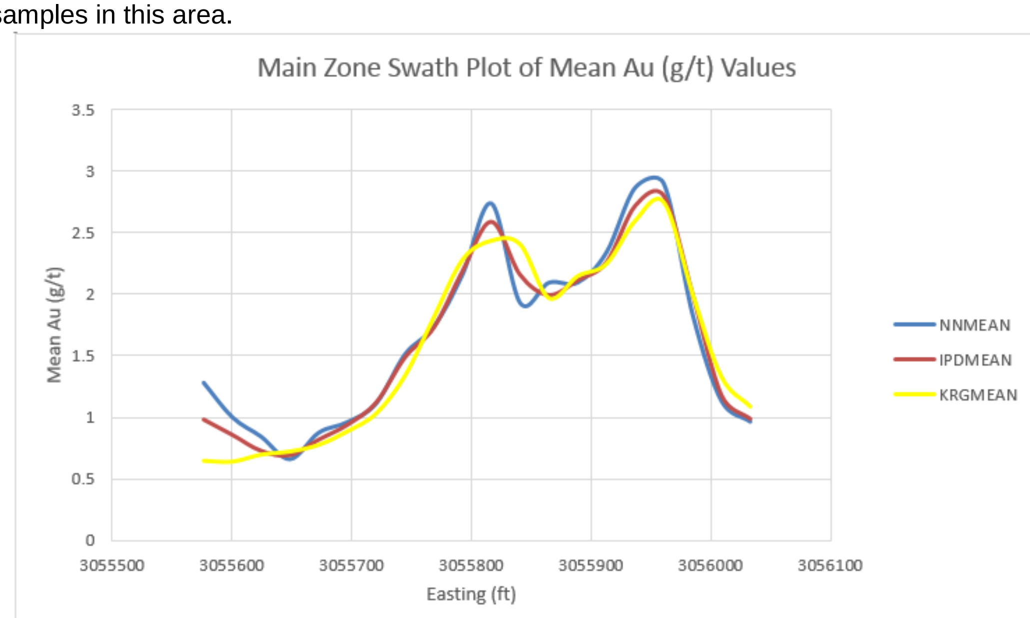


Figure 14.27: Zone 1 Swath Plot of Mean Au Values for NN, ID^3,and OK

14.2.10 Cut-Off Grade

The cut-off grades chosen to report the 2016 Windy Gulch open pit and underground mineral resource estimates were 0.035 oz/tn and 0.093 oz/tn Au respectively. These cut-offs represent a reasonable break-even mining and processing cost, based on an assumed gold price of $1,250 per ounce. The selected cut-off grades were derived based on the criteria in Table 14-20 and Table 14-21 and rounded to an appropriate level of accuracy.

Table 14-20: Open Pit Resource Cut-Off Grade Determination

Gold Price	$1,250	US$
Gold Recovery	80%	
Gold payable	100.0%	
Selling Costs	$170	
Royalty	1.5%	NSR
Dilution	0%	
Opex per tonne	$32.70	per ton
Revenue	$849.00	per ounce
	$30.86	per gram
Cut-off Grade	**1.2**	g/t
	0.035	oz/ton

Table 14-21: Underground Resource Cut-Off Grade Criteria

Gold Price	$1,250	US$
Gold Recovery	94%	
Gold payable	100.00%	
Refining Costs	$130	
Royalty	1.5%	NSR
Dilution	0%	
Opex per tonne	$105	per ton
Revenue	$1035	per ounce
	$33	per gram
Cut-off Grade	**3.2**	g/t
	0.093	oz/tn

14.2.11 Mineral Resource Statements

The mineral resource estimate for the Windy Gulch Segment is reported in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) and has been estimated in conformity with CIM Best Practise Guidelines for Estimation of Mineral Resource and Mineral Reserves (November 2003).

Mineral resources are not mineral reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of this mineral resource will be converted into mineral reserve.

Inferred mineral resources are too speculative geologically to have economic considerations applied to them to enable them to be categorized as mineral reserves.

The resource estimate was completed by Brian Thomas, P.Geo. (APGO #1366, APEGBC #38094), an independent QP as defined under NI 43-101. The effective date of this resource estimate is January 17, 2017.

Open pit mineral resources are reported at a gold cut-off of 0.035 oz/ton (1.2 g/t) constrained to a Whittle pit shell based on the parameters listed in Table 14-20 and a revenue factor of 1.25%. Underground resources were reported at a gold cut-off of 0.093 oz/ton (3.2 g/t) based on the parameters stated in Table 14-21. All reported resources were examined visually and found to have a reasonable level of continuity to support mining.

Table 14-22 and Table 14-23 state the mineral resources for the Windy Gulch Segment.

Table 14-22: Windy Gulch Open Pit and Underground Mineral Resource Estimate

Resource Classification	Tons	Au (oz/tn)	Au Ounces
Pit Constrained Resources (0.035 oz/ton cut-off)			
Indicated	67,000	0.11	7,300
Inferred	6,000	0.09	500
Underground Resources (0.093 oz/ton cut-off)			
Indicated	11,000	0.18	2,000
Inferred	14,000	0.19	2,700
Total Indicated	**78,000**	0.12	**9,300**
Total Inferred	**20,000**	0.16	**3,200**

Notes:
1) Pit constrained resources constrained to a pit shell and reported at a 0.035 oz/t Au cut-off.
2) All underground resources reported outside and below the pit shell at a 0.093 oz/t Au cut-off.
3) Resource tonnages have been rounded to the nearest 1,000 tons.
4) Grade estimates have been rounded to the nearest one hundredth of an ounce of gold.
5) Calculated Au ounces are rounded to the nearest 100 ounces.
6) Resource estimates do not include mining recovery or dilution factors.
7) Resource estimates have accounted for metallurgical recovery.
8) Calculated Au ounces may not add up correctly due to rounding.

Table 14-23 and Figure 14-16 illustrate the sensitivity of the Indicated open pit resource at various cut-offs (official resource highlighted in bold in Table 14-23).

Table 14-23: Sensitivities of Windy Gulch Open Pit Indicated Resource

Cut-Off (oz/tn)	Tons	Au (oz/tn)	Au oz’s
0.018	91,047	0.087	7,905
0.023	82,353	0.094	7,726
0.029	73,913	0.102	7,505
0.035	66,753	0.109	7,276

Cut-Off (oz/tn)	Tons	Au (oz/tn)	Au oz's
0.041	60,128	0.117	7,025
0.047	53,136	0.126	6,720
0.053	47,831	0.135	6,457
0.058	43,605	0.143	6,223

Notes:
1) Estimates listed for sensitivity purposes only and have not been rounded to reflect accuracy of the estimates.
2) Does not include Inferred Resources.
3) Mining recovery and dilution factors have not been applied to the estimates.
4) Bold represents the official resource scenario.

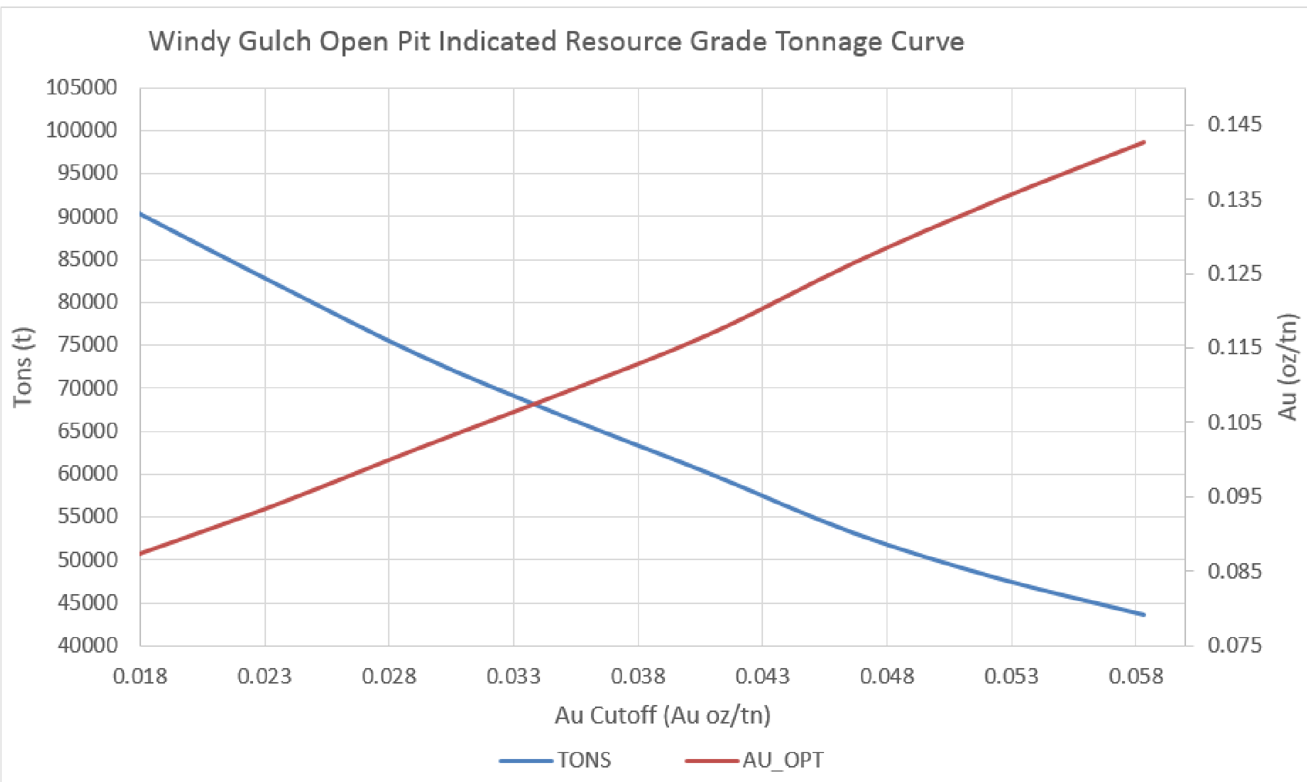


Figure 14.28: Windy Gulch Open Pit Indicated Resource Grade Tonnage Curve

Table 14-24 and Figure 14.29 illustrate the sensitivity of the indicated open pit resource at various cut-offs (official resource highlighted in bold in Table 14-24).

Table 14-24: Sensitivities of Windy Gulch Underground Indicated Resource

Cut-Off	Tons	Au (oz/tn)	Au (oz)
0.058	21,026	0.131	2,748
0.093	**10,881**	**0.184**	**1,999**
0.117	8,046	0.212	1,707
0.146	5,706	0.245	1,400

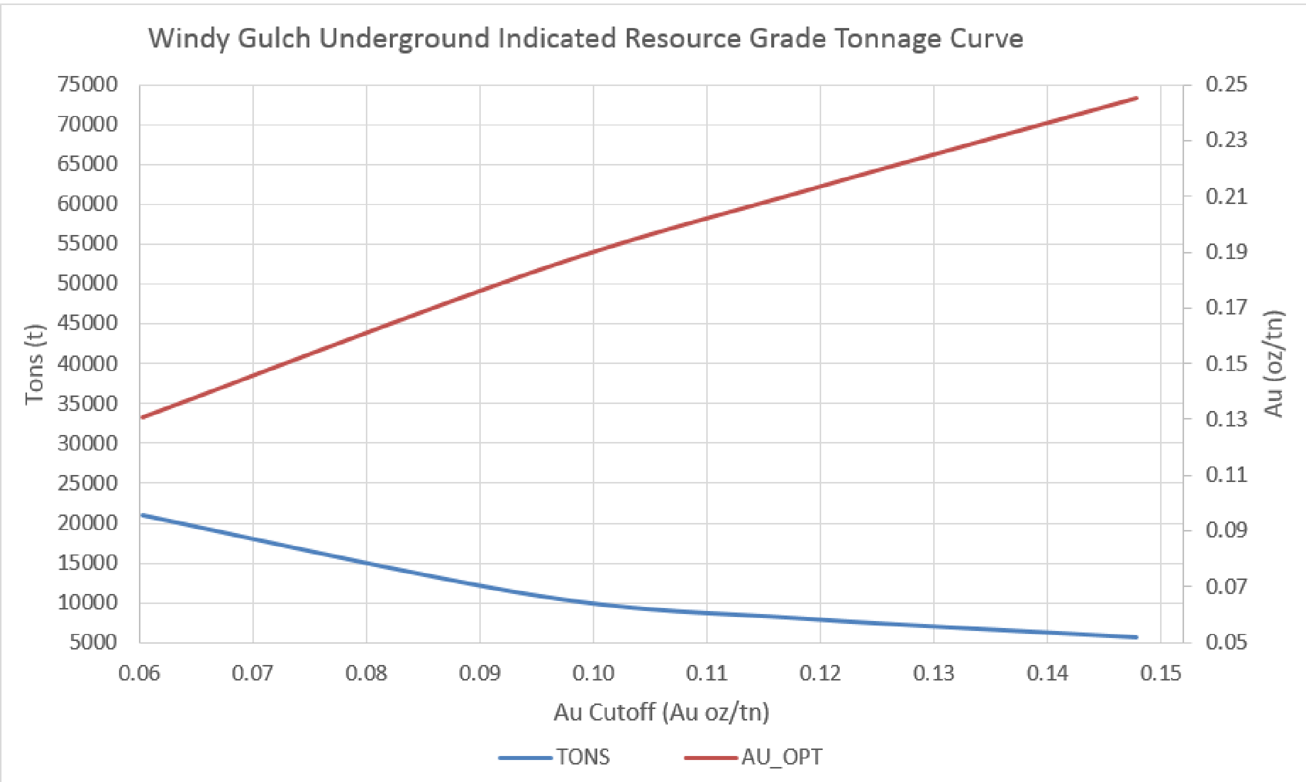


Figure 14.29: Windy Gulch Underground Indicated Resource Grade Tonnage Curve

14.2.11.1 Comparison to 2013 Resource

Table 14-25 compares the current resource to the 2013 resource. Modelling methods used for open pit mine scenarios are considerably different than those used for underground scenarios and can produce significantly different results.

Table 14-25: Resource Comparison

Golder 2016				Mercator 2013			
Resource Classification	**Tons**	**Au (oz/tn)**	**Au Ounces**	**Resource Classification**	**Tons**	**Au (oz/tn)**	**Au Ounces**
Open Pit Resources (0.035 oz/tn cut-off)				***Open Pit Resources***			
Indicated	67,000	0.11	7,300	Indicated	n/a	n/a	n/a
Inferred	6,000	0.09	500	Inferred	n/a	n/a	n/a
Underground Resources (0.093 oz/tn cut-off)				***Underground Resources (0.146 oz/tn cut-off)***			
Indicated	11,000	0.18	2,000	Indicated	n/a	n/a	n/a
Inferred	14,000	0.19	2,700	Inferred	54,000	0.3	16,000
Total Indicated	**78,000**	**0.12**	**9,300**	**Total Indicated**	NA	NA	NA
Total Inferred	**20,000**	**0.16**	**3,200**	**Total Inferred**	**54,000**	**0.3**	**16,000**

The sample population used to estimate resources for Windy Gulch contains a mixture of high and low grade samples due to the low cut-off grade used to outline a continuous zone of mineralization to evaluate an open pit mining scenario. The large difference between sample grades along with the relatively large block size, used to represent the SMU size for open pit mining, will have introduced a degree of smoothing (grade averaging) to the block model. Smoothing results from the weight averaging of sample grades used to estimate the block grade. Smoothing may not be an issue when reporting a global estimate (i.e. zero grade cut-off), but can become a problem when reporting resources above a grade cut-off, such as the case with Windy Gulch. Smoothing changes the distribution of a grade population resulting in a lower grade variance, which generally results in tonnages being reported too high and grades being reported too low. A certain degree of smoothing is expected due to the volume differences between the drill core samples and the block size (volume variance). The expected variance was calculated theoretically and compared to the final block model variance, indicating that the model had an appropriate level of smoothing and was not over-smoothed. Smoothing in the model is less likely to be an issue when reporting at lower grade cut-offs, such as in the 0.035 oz/tn Windy Gulch open pit scenario, but could be more of an issue when reporting at higher grade cut-offs, such as in the underground mining scenario. Underground mining may be able to achieve a higher degree of sorting, based on the smaller SMU size, and therefore may be able to achieve higher grades and lower tonnages and potentially more ounces than estimated.

The reader is cautioned that this block model and resource estimate was intended to evaluate an open pit mining scenario and may not accurately reflect conditions attainable from narrow vein mining (i.e. lower tonnage at a higher grade). Open pit mining consists of larger mining blocks and the same degree of sorting cannot be achieved as in underground mining, which generally results in higher tonnages at lower grades.

Golder is unaware of any known material project risks related to environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or any other potential factors that could materially impact the Windy Gulch Segment and resource estimate provided in this technical report.

15.0 MINERAL RESERVE ESTIMATES

Mineral reserves were not calculated for this report since a pre-feasibility study was not been completed and inferred resources were included in the analysis of both the underground and open pit analyses.

16.0 MINING METHODS

16.1 Underground Mining - Dawson

16.1.1 Block Model

The approximate geometries of the supplied wireframe segments that were used to model the Dawson underground resource are tabulated in Table 16-1. The majority of the Dawson deposit has a true thickness of less than 13 ft. Most mineralization is steeply dipping, between 50° and 70° to the south.

Table 16-1: Description of Mineralized Zones over Various Cross-Sections

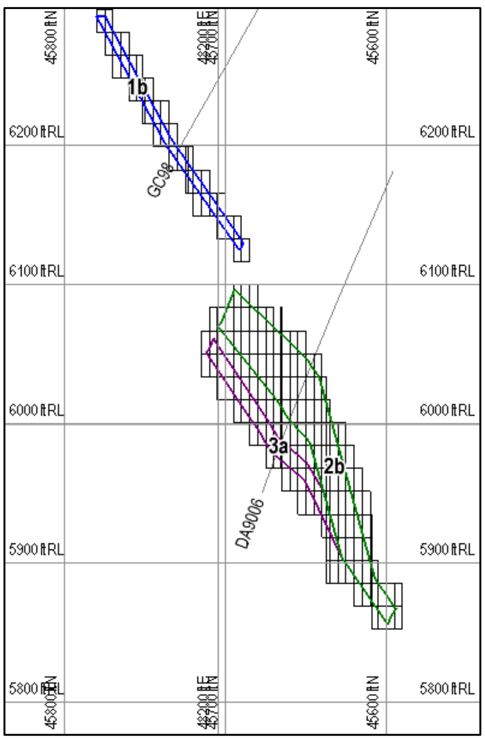
Zone	Azimuth (°)	Plunge (SW) (°)	Dip (S) (°)	Range in Thickness (ft)	Average Thickness (ft)
1a	249	-14	56	7–12	7
1b	232	-14	62	5–17	8
1c	225	-14	62	9–19	13.4
2a	245	-14	69	6–12	7
2b	258	-14	75	6–33	9.7
	235	-14	69		
	209	-14	69		
	236	-14	60		
	262	-14	75		
2c	222	-14	77	5–6	5.3
3a	217	-14	65	4–19	6.8
	243	-14	49		
	251	-14	62		
	265	-14	79		
3b	254	-14	65	4–5	4.4

To aid mine planning, the supplied “percent-type” or “block factored” model was sub-blocked into a model with irregular block sizes (refer to Table 16-2 and Figure 16.1). The resulting file is “Blocks, Dawson, Sub-blocked.dat.”

Table 16-2: Block Model Sub-blocking

Direction	Block Size (ft)	Number of Sub-blocks Added
East	16.5	2
North	5	5
Elevation	16.5	2

Original "Percent-Type" Model with Regular Sizes



Sub-blocked Model

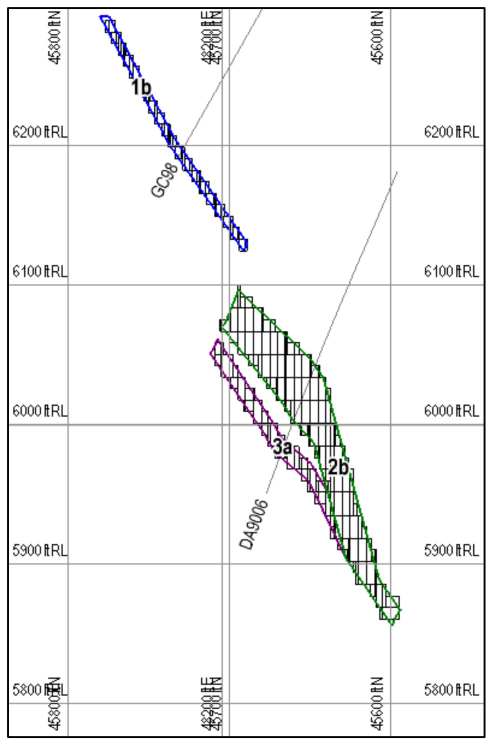


Figure 16.1: Block Model before and after Sub-blocking (Section G–G')

A resource specific gravity (SG) of 2.63 was used (Hilchey et al., 2013). This corresponds to a tonnage factor of 12.2 ft^3/tn.

16.1.2 Dilution

There are two main types of dilution: planned and unplanned (see Figure 16.2). Unplanned dilution is mainly caused by blasting overbreak. Proper blast design and blasting practice can reduce overbreak. High stress in the rock mass can also cause the walls of the stope to slough or fail, leading to unplanned dilution. Poor blasting practice will also contribute to unplanned dilution.

For this work, unplanned dilution was added to the non-diluted mineral resource as a function of the mining width. A skin of 6 inches of rock was added to both sides of the width, Table 16-3. When this skin is taken into account, dilution averages about 14.5% with a range from 7.5% to about 23%. Good blasting practice can help to minimize this unplanned dilution.

A diluting grade of 1 g/t (0.029 oz/tn) was used.

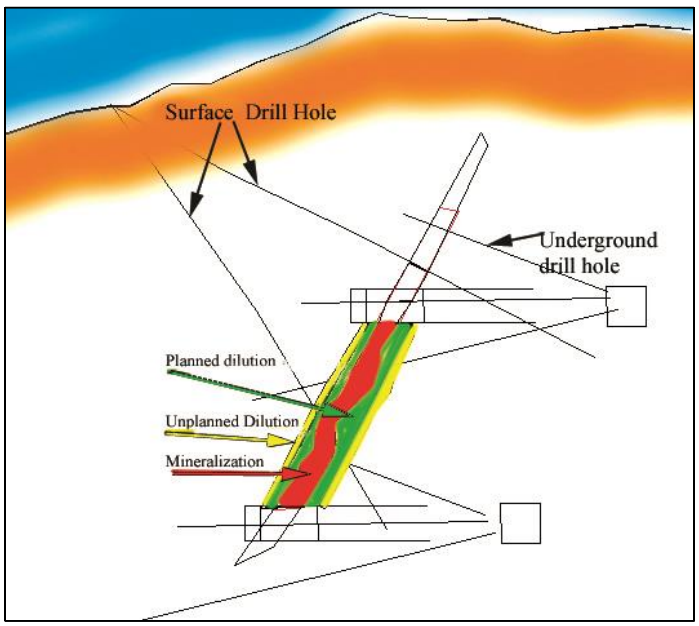


Figure 16.2: Dilution

Table 16-3: Dilution Possible from Overbreak and Blasting Damage in the Mine Stope Plan

Zone	Thickness (ft)	Thickness After Blasting (ft)	Blasting Dilution (%)
1a	7.0	8.0	14.3
1b	8.0	9.0	12.5
1c	13.4	14.4	7.5
2a	7.0	8.0	14.3
2b	9.7	10.7	10.3
2c	5.3	6.3	18.9
3a	6.8	7.8	14.7
3b	4.4	5.4	22.7

A minimum mining width of 5 ft was used to outline the mineral resources (Hilchey et al., 2013). This would be considered "planned dilution." However, longhole blasting using modern, in-the-hole production drills can be carried out as narrow as 3 ft. This presents an opportunity to reduce the planned dilution, thereby reducing the milling cost and increasing the mill feed grade.

16.1.3 Mining Recovery

A mining recovery of 95% is used in this report as a percentage of the resource that will be left as remnant pillars. Every effort will be made to recover profitable pillars. However, some losses are inevitable.

In the mining plan, approximately 85% of the resource is mined from the stopes and another 10% is recovered from the sill pillars. Mining the sill can be problematic if the stope above is filled with loose rock.

To recover the sill between stopes, development of the bottom sill of the stope above could include constructing a concrete beam at the base of the sill. The sill would be driven 12 ft high. This is because the stope will still be 10 ft high after the concrete floor has been constructed.

Recovery of the sill is done after the stope above has been completed and the stope below has been backfilled. The sill is recovered by drilling up-holes from the stope below. Recovery is in retreat fashion, starting at the end of the stope and working back to the raise (Figure 16.3).

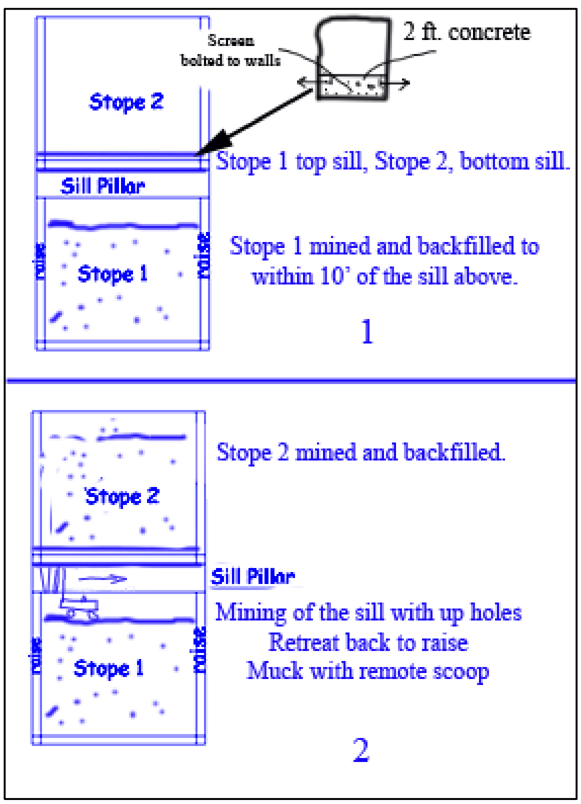


Figure 16.3: Schematic Illustrating the Steps Required to Recover the Sill Pillar

16.1.4 Mining Adjacent or Adjoining Zones

Many zones coalesce and bifurcate in places (refer to Figure 16.1 for an example). Coalescing zones, or zones with only a thin pillar of "waste rock" separating them, would be mined together as one. In many cases, the "waste" between zones is mineralized but below cut-off. This would be considered "planned dilution."

As the separation between zones increases, a decision would need to be made, on a case-by-case basis, whether the two zones would be most profitably mined as one or separately, leaving a waste pillar between them. This analysis would take into account that leaving a thin pillar may require additional ground support measures such as backfill or cable bolting.

16.1.5 Stope Mining Cut-Off Grade

A stope cut-off grade of 0.088 ounce per ton (oz/tn) was used to outline the resources that are eligible for mining since a 0.088 oz/tn outline gives reasonable mining widths and mineralization continuity. At the time of report writing, 0.088 oz/tn is *approximately* the break-even cut-off grade after mining, milling, and general and administration (G&A) costs are tallied. In practice, however, the profitability of each potential stope would be evaluated on a case-by-case basis. This evaluation would also take development into consideration.

If development is excavated through low grade material, a calculation will be made to determine if the material should go to a low grade stockpile rather than the waste pile. The material may be profitable as it only has to pay for processing, the cost of transport to surface, and overhead, including profit (breaking costs are sunk). The break-even cut-off grade for already-broken rock would likely be in the 0.029 oz/tn to 0.044 oz/tn range.

16.1.6 Potential Mill Feed – Dawson Segment

The potential mill feed for the Dawson Segment totals 449,000 tons grading 0.26 oz/tn, for 117,000 oz delivered to surface (shown in Table 16-4 and Table 16-5). At this point in time, the entire mineral resource is in the inferred category.

The potential mill feed for the Windy Gulch Segment is discussed in Section 16.2.

Table 16-4: Dawson Segment Diluted Potential Mill Feed with 5% Mining Losses

Main Level	Zone	tons in volume	Less 5% mining losses, tons	Available tons	oz/t	oz in mined material	Wall Dilution	Dilution Tons	dilution oz/t	oz in dilution	oz to surface	tons to surface	oz/ton to surface
6125	1a	2,778	139	2,639	0.25	652.8208	14.3%	377	0.035	13	666	3,016	0.22
6125	1b	41,954	2,098	39,856	0.30	11915.29	12.5%	4,982	0.035	177	12,092	44,838	0.27
6125	1c	2,706	135	2,571	0.12	305.1427	7.5%	193	0.035	7	312	2,764	0.11
6125	2a	23,237	1,162	22,075	0.35	7764.694	14.3%	3,157	0.035	112	7,877	25,232	0.31
6125	2b	11,403	570	10,833	0.31	3365.058	10.3%	1,116	0.035	40	3,405	11,949	0.28
5862	1a	6,076	304	5,772	0.25	1429.229	14.3%	825	0.035	29	1,458	6,598	0.22
5862	1b	569	28	540	0.24	130.4034	12.5%	68	0.035	2	133	608	0.22
5862	1c	39,036	1,952	37,084	0.19	7138.687	7.5%	2,781	0.035	99	7,237	39,866	0.18
5862	2a	16,146	807	15,338	0.14	2138.41	14.3%	2,193	0.035	78	2,216	17,532	0.13
5862	2b	125,677	6,284	119,393	0.31	37156.13	10.3%	12,297	0.035	436	37,592	131,690	0.29
5862	3a	59,399	2,970	56,429	0.32	17989.14	14.7%	8,295	0.035	294	18,283	64,724	0.28
5862	3b	5,278	264	5,014	0.12	605.4306	22.7%	1,138	0.035	40	646	6,152	0.10
5687	2a	368	18	350	0.15	53.46669	14.3%	50	0.035	2	55	400	0.14
5687	2b	20,028	1,001	19,026	0.20	3784.785	10.3%	1,960	0.035	69	3,854	20,986	0.18
5687	3a	12,424	621	11,803	0.71	8434.297	14.7%	1,735	0.035	61	8,496	13,538	0.63
5425	2b	26,723	1,336	25,387	0.21	5331.268	10.3%	2,615	0.035	93	5,424	28,002	0.19
5425	2c	12,268	613	11,654	0.38	4436.063	18.9%	2,203	0.035	78	4,514	13,857	0.33
5425	3a	13,736	687	13,049	0.18	2302.54	14.7%	1,918	0.035	68	2,371	14,967	0.16
Deeper	2c	1,127	56	1,070	0.38	405.3244	18.9%	202	0.035	7	412	1,273	0.32
Deeper	3a	268	13	254	0.17	43.91792	14.7%	37	0.035	1	45	292	0.16
Column Totals (rounded)		**421,000**	**21,000**	**400,000**	**0.29**	**115,400**		**48,000**		**1,700**	**117,100**	**448,000**	**0.26**

Table 16-5: Summary of Mineral Resources and Potential Diluted Mill Feed (0.088 oz/ton cut-off)

Non-diluted Mineral Resource Blocks			
	tons	**ounces**	**oz/ton**
Dawson Segment	421 k	121 k	0.288
Diluted and Mineable Potential Mill Feed			
	tons	**ounces**	**oz/ton**
Dawson Segment	449 k	117 k	0.26
Note: 0.088 oz/tn (3. g/t) block cut-off.			

16.1.7 Historical Mining Methods

There are historical workings, all of which are shallow (less than 100 ft) deep on the property, known as the “Mike Sutton Workings,” the “Last Show,” the “Haulage Adit Fault,” the “Copper Boy Workings” and the “Copper Boy” shaft. Old workings are found in the “Copper King” area where an adit was driven to intersect a southeast plunging (40° to 50°) structure. These old workings, including pits, shafts, and adits, which were primarily targeted on the copper-mineralized massive sulphide zone stratigraphically above the gold zones, are described in Hilchey et al. (2013). Shrinkage stoping was likely the method employed at most of these deposits.

No historical workings affect the current mine design in any significant way. The Dawson Segment being targeted for underground mining has no surface expression or outcrop, is entirely intact, and has not been subjected to any historical mining.

16.1.8 Geotechnical Considerations

The Dawson Project is hosted by hard rock that has been folded, faulted, and metamorphosed.

There is some debate regarding rock competence with respect to choosing a mining method. In the AMSE study (AMSE, 1991), AMSE chose cut-and-fill mining because “hanging wall rock conditions may prohibit the use of shrinkage stoping.” AMSE also felt that “the [mineralized rock] zones will be too narrow for mechanized equipment.”

Dynatec (1991), on the other hand, was “of the opinion that a backfill system of mining is not required. This opinion was arrived at based on the configuration of the [mineralized rock] and the geologic logs and calculated Rock Quality Designations (RQDs) from the drill holes…”

Because a site visit and personal examination of the core were beyond the scope of this work, the authors elected to rely on Dynatec’s opinion that backfill would not be required and that the rocks are generally strong enough to stand up while mining is being completed. Cut-and-fill stoping was ruled out due to its higher cost.

During operations, the hanging wall rocks may be cable-bolted from the top and bottom sills to reduce dilution. Diamond drilling will confirm stope geometry and the rock quality prior to mining.

During the initial mine development, geotechnical sampling for rock pressure should be done to determine the axis of the principal stress in the area. If weak hanging wall rocks are locally encountered that could significantly dilute the mineralized rock, then a cut-and-fill mining method could be substituted where required.

16.1.9 Groundwater

Groundwater inflow through the rock mass is not expected to be significant. Fracture porosity may carry some water into the underground workings near the surface. In the absence of detailed groundwater studies, groundwater infiltration is anticipated to be less than 100 gpm.

16.1.10 Proposed Mine Design

Several mining methods could be applied to this deposit, including “Alimak mining,” shrinkage stoping, and longhole sublevel stoping. The various advantages and disadvantages of each method were considered and longhole sublevel stoping was selected for preliminary mine design.

Mining will be near-completely mechanized. Jackleg and stoper drilling would be minimized. A mine plan was developed based on the criteria discussed in previous sections. These parameters are summarized in Table 16-6. The overall plan is shown in Figure 16.4 and Figure 16.5.

For the Dawson deposit, the portal will be at elevation 6,500 ft. The base of the known mineralization is at about 5,400 ft, a vertical distance of 1,100 ft.

The sills and the raises are driven in mineralization. The decline, haulage levels, and ventilation raise are all driven in waste rock.

Geological and geotechnical information is collected prior to mining a particular stope. Diamond drill holes, drilled from the footwall ramp, provide geological, grade, and geotechnical data prior to stoping and aid the stope design process.

Although there are some historical workings on the site, no existing underground workings affect, or are incorporated in, this mine plan.

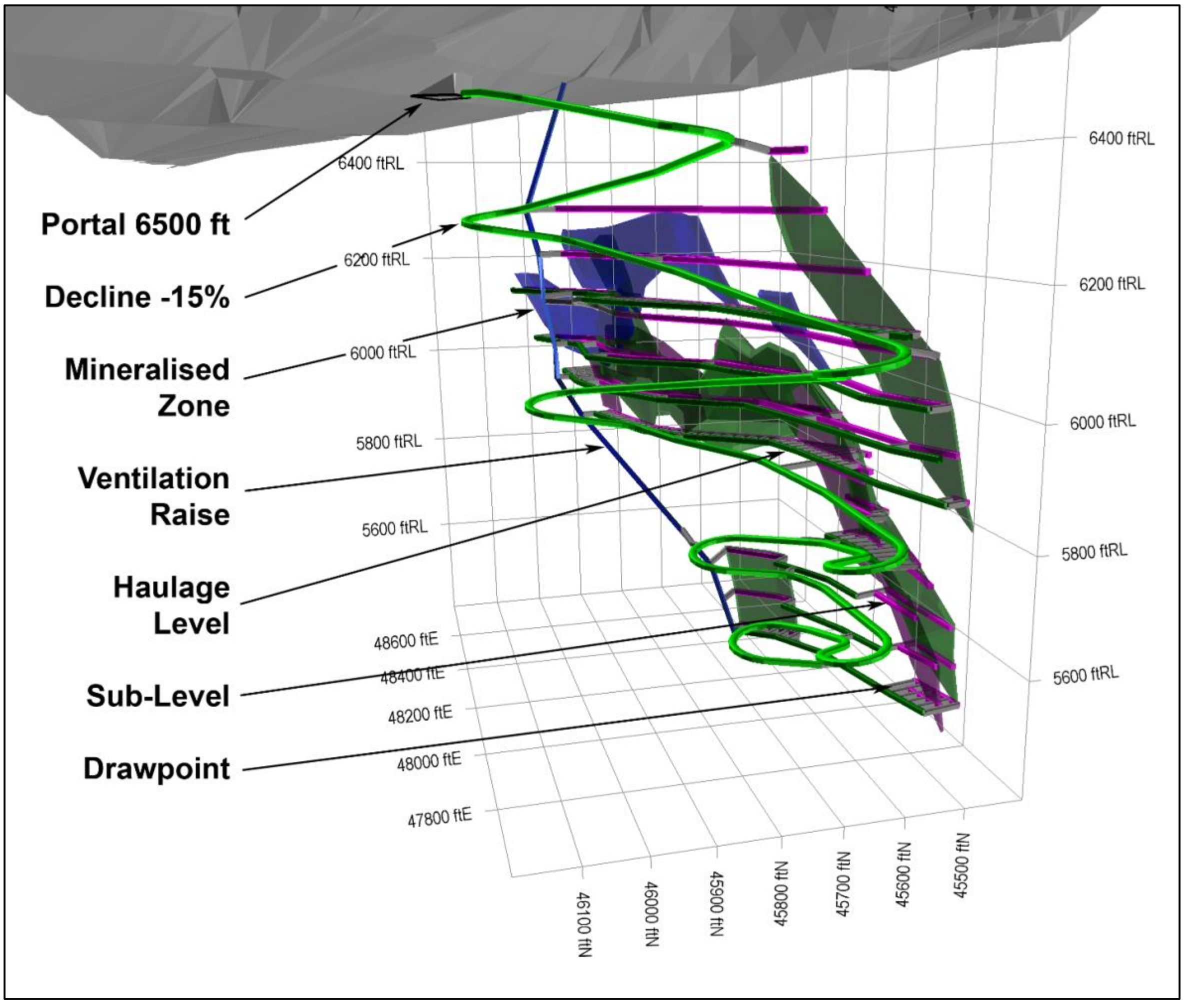
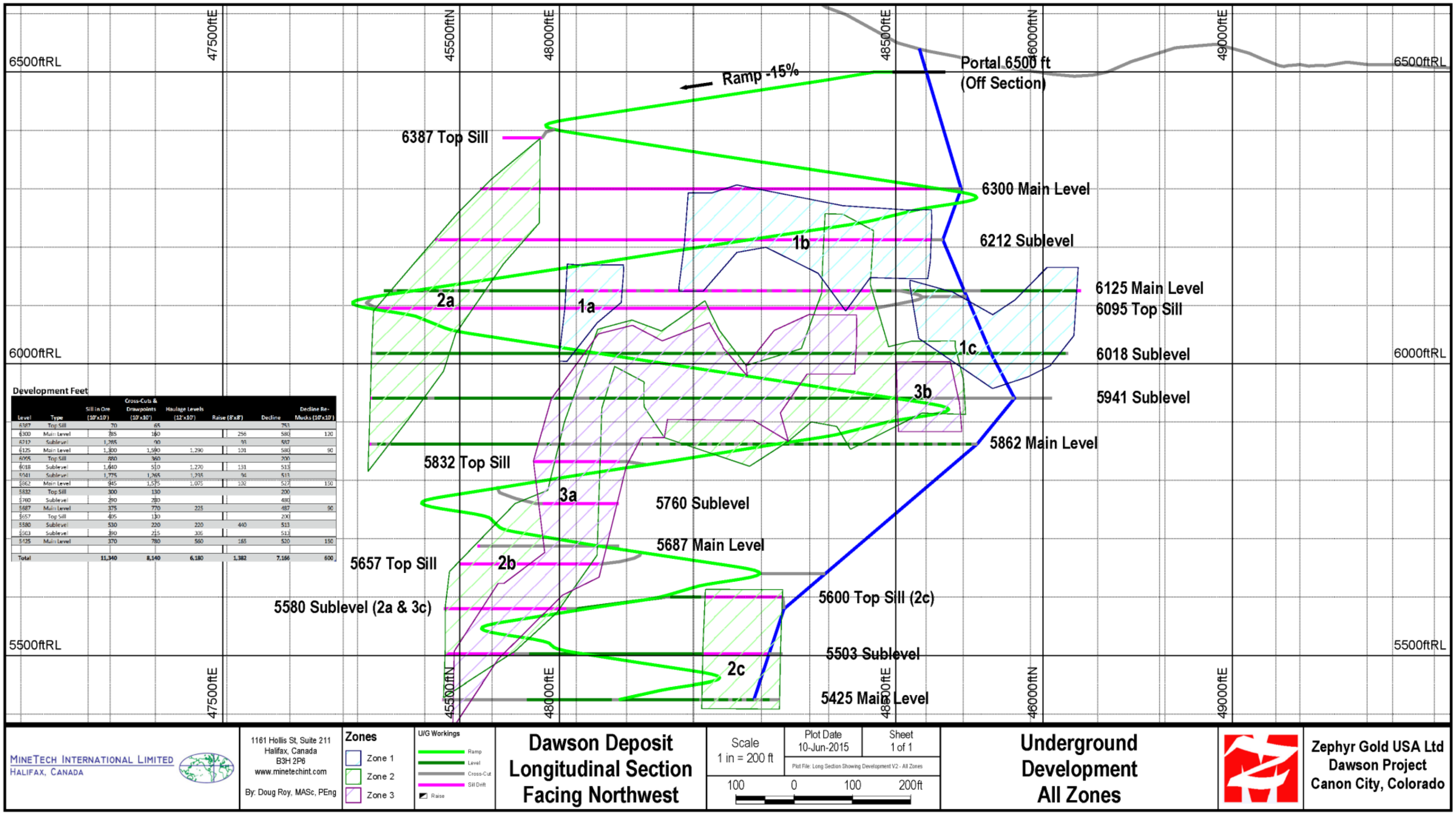


Figure 16.4: Three-Dimensional View of Dawson Underground Development (facing east)



Development Feet

Level	Type	Sill in Ore (10'x10')	Cross-Cuts & Drawpoints (10'x10')	Haulage Levels (12'x10')	Raise (8'x8')	Decline	Decline Re-Mucks (10'x10')
6387	Top Sill	70	65			753	
6300	Main Level	785	160		256	580	120
6212	Sublevel	1,285	90		93	587	
6125	Main Level	1,300	1,590	1,290	101	580	90
6095	Top Sill	880	360			200	
6018	Sublevel	1,640	510	1,270	131	513	
5941	Sublevel	1,775	1,265	1,235	94	513	
5862	Main Level	945	1,575	1,075	102	527	150
5832	Top Sill	300	130			200	
5760	Sublevel	290	280			480	
5687	Main Level	375	770	225		487	90
5657	Top Sill	405	130			200	
5580	Sublevel	530	220	220	440	513	
5503	Sublevel	390	215	305		513	
5425	Main Level	370	780	560	165	520	150
Total		11,340	8,140	6,180	1,382	7,166	600

Figure 16.5: Longitudinal Section Facing Northwest

Table 16-6: Mine Design Parameters

Item	Property
Mineralized Zone Width	5 ft to 33 ft
Mining Method	Longhole sublevel stoping
Available Mineral Resources and Potential Mill Feed	Refer to Table 16.5
Dilution	6 inches wall rock at 0.035 oz/tn, averaging 14.5%
Mining Recovery	95% Overall
Milling Rate	Average 300 tons per day, 365 days per year
Mill Downtime	1 shift per week preventive maintenance, 2 weeks per year breakdowns & major maintenance
Mill Nameplate Capacity	340 tons per day
Target Mining Rate	400 tons per day, 5 days per week
Stope Cut-off Grade	0.088 oz/ton
Haulage Level Spacing	175 feet
Sublevel Spacing	70 ft to 90 ft
Decline Gradient	-15%
Decline Size	15 ft x 11 ft (arched)
Haulage Drifts (Level Drifts)	12 ft x 10 ft (arched)
Raises	8 ft x 8 ft
Sill Drifts	10 ft high, width of zone [minimum 10 ft]
Cross-Cuts	12 ft x 10 ft
Draw Drifts	10 ft x 10 ft

16.1.11 Mine Development

The main development priorities are the decline, the main haulage drifts, and the ventilation/escapeway raise. During the first year's development work, waste rock will have to be hauled out of the mine, to be used for road construction and to flatten out areas on surface for the surface infrastructure. Once production mining commences, the waste rock will be used to backfill mined-out stopes.

Trackless equipment will be used. The equipment includes 20 ton trucks and load-haul-dump (LHD) units with a capacity of 1.5 yd^3 and 3.0 yd^3. A drill jumbo and a longhole drill will be used on the ramp and for stope mining, respectively.

The underground mine will be developed from a -15% gradient ramp. Once underground, infill diamond drilling will be carried out on and between current resource blocks to upgrade the resources to the indicated category. Drilling will also be undertaken on currently identified exploration targets that show potential to expand the resource base.

16.1.12 Portal

A portal will be positioned on the footwall side of the deposit, roughly 330 ft north-northwest of the deposit. An accurate elevation survey was not available for portal design work. According to the supplied digital terrain model, the elevation of the selected location is approximately 6,500 ft.

The terrain's slope in this location is approximately 20°. As an approximate guide, the solid rock overburden thickness over the slope's brow should be approximately 1 to 2 times the decline height, or 1.5 times 12 ft, equal to 18 ft of rock cover. It is recommended that approaching the brow, the ground have a +1° slope (1.7%) to the brow so that surface runoff will drain away from the portal.

16.1.13 Decline

Development of the deposit will be from a decline driven in the footwall gneisses. The decline is designed to stay between 60 ft and 100 ft from the mineralization by spiraling at the southwest and northeast ends of the deposit and shifting southeast as the deposit dips. Drilling the decline will be by electric-hydraulic jumbo. Mucking and haulage will be by 3 yd^3 LHD and 20 ton trucks. Ground support in the ramp will be rock bolts and screen.

For drilling a decline round, 12 ft drill steel could be used. These rounds would likely break to approximately 11 ft. Smaller openings such as cross-cuts and sill drifts would likely be drilled using shorter, 8 to 10 ft drill steel because a development round advance is typically in the neighbourhood of 80% of the opening's width.

The decline excavation will generate approximately 17,200 tons of waste over the 175 vertical feet between levels. The decline is planned to have a cross-sectional area of 160 ft^2, being 15 ft wide and 11 ft high. The decline is 10.5 ft high at the shoulder.

There will be a 30 ft bay or "remuck" cut into the walls every 300 ft along the decline. The LHD will load trucks from one of these bays to keep the tramming time manageable.

As the decline ramp is a semi-permanent structure, the ground support will include regular bolting and screening of the back and upper walls. Rock quality is expected to be good within the footwall gneisses. At this time there are no data on rock pressures.

16.1.14 Diamond Drilling

Once a drilling space is available, a number of holes will be drilled to further define the mineralized zones. At each drill set-up, at least three holes will be drilled, as illustrated below in Figure 16.6.

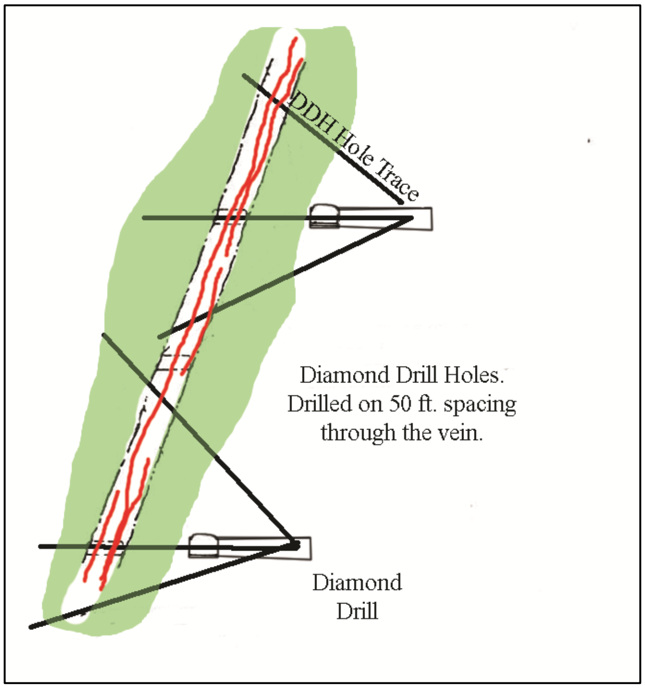


Figure 16.6: Diamond Drill Set-Up from a Former Remuck Station

16.1.15 Refuge Chambers

The remuck station near the 6300 level will be furnished as a refuge chamber and lunch room until the ventilation raise/escapeway has been completed. If an Alimak raise climber is used, the Alimak nest could then be outfitted as the 6125 level refuge chamber.

16.1.16 Ventilation Raise

The ventilation raise will be excavated 8 ft by 8 ft and will be equipped with ladders and landings. The ventilation fan air could be downcast to prevent the portal and decline from freezing in winter. In the unlikely event that freezing air in the vent raise causes ice buildup that could impede use of the escapeway, an air heater will be required. Mine ventilation is further discussed in Section 16.1.20.

16.1.17 Development Schedule

Development of the decline is on the critical path and should take priority. A second exit (the ventilation raise) and a refuge station (6125 level) are required prior to production mining. Development of the mine to the point at which production mining could start will take about a year. After the ventilation raise has been driven up from the 6125 level and after the refuge station has been established and equipped, production from the 6125 level could begin. Between the portal and the 6125 level, approximately 27,300 tons of mineralization will have been mined from the sills along with about 65,000 tons of waste. A summary of development and production milestones is included as Table 16-7.

Table 16-7: Dawson Mine Development and Production Schedule

	months	1	2	3	4	5	6	7	8	9	10	11	12										
BOD Decision to finance Dawson Underground Development																							
Prepare detailed site maps of topography options for plant site, tailings disposal, office site. Complete metallurgical flow sheet for plant, Preliminary feasibility study report.																							
Permitting and Public Meeting		To be completed prior to Month 1.																					
Retain senior mining, hourly and processing staff																							
Survey areas required for tailings area, portal and ventilation raise areas and the plant area, Complete Layout survey of plant, tailings area, access road etc. Prepare portal site, flatten an area for infrastructure.																							
Ancillary services and requirements																							
Complete design of infrastructure. Obtain all necessary municipal permits for construction. Construct sewage disposal field, roads, security fencing and construct office, dry, shop and warehouse around plant.																							
Mining exploration and Development - Underground																							
Detail labour and equipment requirements, cost and replacement schedule. Order underground equipment & inventory. Excavate portal area and ventilation/egress raise area to solid bedrock. Complete concrete work in portal and vent raise area . Portal El 6545 feet.																							
Begin underground work																							
Advance decline to the top of mineralization elevation, about 6300 elevation.																							
Cross cut to 6300 sill and excavate sill																							
Excavate Alimak nest for ventilation raise from x-cut off of 6300 L.																							
Excavate and equip vent raise- escapeway from 6300L to surface, 2 rounds/day																							
Continue Mine Development, Excavate Decline to 6237 elevation, 450 ft., Establish 6237 drll sill. Set up diamond drill in re-muck off ramp, section E, M and R areas.																							
Underground Diamond Drilling set up 1,2,3														**Continue Diamond Drilling exploration and development**									
Complete Resource Estimate, Prepare more detailed mine plan.															**BOD - Complete Development & start production**								
Decide to Proceed with mine development																							
Excavate 6125, 5862, 5687, and 5425 main levels, sublevels, and topsills, approximatly 10,500 feet development. Vent raises to connect main levels.														**Production Mining starts and Development continues if sufficient resources are demonstrated.**									

16.1.18 Stoping

Stoping will be carried out using a mobile, in-the-hole longhole drill. The mining cycle includes backfilling the stope as soon as possible before the stope walls deteriorate.

The sequence of stoping will depend upon the grade of the material, rock pressures, continuity of mineralization, and continuity of thickness of the mineralization. After detailed drilling, low grade areas can be identified and left as pillars. Maximum recovery of the deposit will be possible if backfilling is completed as the deposit is mined.

16.1.18.1 Stope Development

The capital development of a stope includes the decline, the footwall haulageway, cross-cuts to the sills, the ventilation raise, refuge chambers, water sumps, electrical substations, powder and cap magazines, and various storage places underground.

From the decline, two sill levels are driven approximately 175 ft apart. Raises are excavated from the bottom sill to the top sill every 200 ft along strike, beginning at the west end of the deposit. Sill drifts will be 10 ft high and the width of the zone, with a minimum width of 10 ft to accommodate mobile equipment. These sills are driven in the mineralized horizon at a slight uphill grade of 0.5% (1 ft in 200 ft) so that water will drain back to the decline and loaded vehicles will have a slight advantage going downhill.

The upper sill level will be driven 10 ft by 10 ft in mineralization, under geological guidance. It will be as wide as the mineralization but in any case, no less than 10 ft wide to accommodate the longhole drill. A bottom sill is then driven in mineralization to the west and east under geological control, in order to define the bottom of the stopes.

Once the bottom sill level has been excavated, mapped, and surveyed, a haulage level can be excavated in the footwall about 30 ft from the bottom sill. Draw points are excavated from the haulage level through the mineralized horizon and one round into the hanging wall.

Sublevel spacing for parallel blast holes drilled in the dip direction will be 50 to 100 ft apart. A 65 ft sublevel spacing has been incorporated.

16.1.18.2 Drop Raises

A raise is needed in each stope to provide a void for the initial production blasts. Drop raising is the proposed method for creating this void.

After the top and bottom sills have been excavated, a drop raise is excavated by drilling and blasting the raise from the top. A large hole, 6 to 8 inches in diameter, is drilled for relief. Drilling accuracy is necessary for the drop raise method. Once the raise is excavated, the walls are slashed to the full width of the zone and stope mining can begin.

There are several advantages to this method over conventional raise driving. It is safer as no persons are in the raise. It is cheaper since there are no ground support costs and mining can begin as soon as the raise has been excavated. Drill hole accuracy has improved greatly with rigid rods and down-the-hole hammer (DTH) drilling rigs. The dip of the structure will limit the practical length of the drop raise.

16.1.18.3 Production Mining

It will take at least six months to drive the decline to the 6125 sill elevation and another six months to develop the deposit for production. A summary of the development and production schedule is included in Table 16-8. Drilling and blasting for stope production will be done with 2.5-inch diameter holes with average lengths of 60 ft. One blasthole drill will be required for production. In areas with narrow stope widths, relatively fine fragmentation is required to promote the free flow of rock down the stope to the drawpoints. This can be achieved by employing proper blast design and conscientious blast implementation.

Production mining will be carried out over five days per week, with a targeted mining rate of 400 tons per day. Waste mining will be approximately 375 tons per day, 250 tons from the ramp and another 125 tons from other development. It is assumed that approximately 82% of the time the worker spends underground is productive time, with the remaining time used for travel to the workplace, breaks, and unplanned downtime.

At first, the only mineralized rock mined will be from the sills, raises, and the swell from the stope. As the mining proceeds upward, about 40% of the rock blasted is drawn out of the stope. The remaining rock will remain in the stope until stope mining has been completed.

Table 16-8: Production Schedule Summary

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Mining Equipment Capital	$6,170 k	$740 k	$650 k	$911 k	$911 k	$9,382 k
Initial Development	$4,231 k	$2,001 k	$ -	$ -	$ -	$6,232 k
Production Mining	$3,193 k	$8,131 k	$7,392 k	$10,377 k	$6,838 k	$35,920 k
Dawson Sill Tons	44 k	42 k	0 k	0 k	0 k	86 k
Dawson Stope Tons	0 k	40 k	102 k	92 k	130 k	364 k
Total Dawson Tons	44 k	81 k	102 k	92 k	131 k	450 k
grade (opt)	0.28	0.27	0.26	0.25	0.25	0.26
Total Ounces	14 k	24 k	28 k	25 k	30 k	121 k
Development Waste Tons	103 k	108 k	50 k	0 k	0 k	261 k

16.1.18.4 Backfilling

Once the stope has been emptied, development waste can be dumped from above to provide wall support and to dispose of waste rock (rather than trucking it to surface). When loading waste, the LHD operator must ensure that no oversize rock is included as it could cause the fill to hang up in the stope. The largest piece of backfill should be no bigger than 1/8th the width of the stope to backfill.

16.1.19 Equipment

Capital and operating costs for the main pieces of underground mining equipment are given in Section 21.

16.1.19.1 Power Source

All mobile equipment will be electric or diesel powered. Diesel equipment must be equipped with exhaust scrubbers to ensure that diesel particulate matter is kept below the regulated standard of 400 µg/m^3.

Battery-powered LHDs and trucks were selected for this mine design. Working heavily, the batteries last approximately four hours and can be swapped in 15 minutes. They take two hours to fully charge. This equipment is currently in use at Kirkland Lake Gold's Macassa Mine in Ontario, Canada.

Advantages of battery power over diesel include greatly improved working conditions (much quieter, no emissions), lower energy cost, lower maintenance cost, and reduced ventilation requirements. The main disadvantage is the higher capital cost. Future studies should include a detailed comparison between the capital and operating "life of equipment" costs of battery and diesel-powered equipment.

16.1.19.2 Cycle Times

The cycle time for a truck includes:

- the time it takes the LHD to load the truck
- the travel time from the loading point to the dumping point, using average uphill and downhill speeds
- the time to maneuver and dump the load
- the time to return to the loading point
- waiting time
- maneuver to load time.

The truck cycle time can be used to ensure the fleet is adequate for the job. A third truck is required after the 5862 level, and four trucks are required below the 5500 level.

16.1.19.3 Drilling Equipment

The main decline and haulage ways will be drilled using a one-boom electric-hydraulic jumbo. It is presumed that the sills would be drilled using a similar single boom drill jumbo. However, jackleg drills or "Long Tom" rigs would also be suitable.

16.1.19.4 Mucking and Loading Equipment

Mucking will be done with battery-powered 3 yd^3 LHDs that would load battery-powered 20 ton trucks. Smaller, 1.5 yd3 LHDs could also be used in narrower openings.

16.1.19.5 Hauling

Initially, two 20 ton trucks will haul waste and mineralized rock from the mine. Approximately 775 tons will be moved per day, approximately 390 tons per shift.

16.1.20 Ventilation

Ventilation requirements have been estimated based on providing 100 to 125 cubic feet per minute (CFM) of fresh air per rated horsepower of diesel powered equipment underground plus 100 CFM per person underground. An anticipated utilization rate was incorporated for each piece of mobile equipment.

During the initial development phase, while developing the decline and the ventilation/escape raise, fresh air will be delivered to the working face using a fan and flexible ducting. After the ventilation raise breaks through to surface, a fan arrangement will be installed over the raise in either an intake or exhaust set-up. Approximately 150 to 200 fan horsepower will be required during the production phase.

During the production phase, fresh air is channelled to the bottom of the mine and the main levels using air locks, regulators, and ventilation raises. Air is delivered to dead-end stopes and haulage drifts using flexible ducting. Small fans, between 10 and 20 HP, deliver air to the stopes.

Once the Dawson deposit has been developed, and prior to installing the main fan, a complete ventilation study should be completed using the data gleaned from the decline and level development.

16.1.21 Mine Services

16.1.21.1 Electrical

The main electrical trunk will run down the portal initially, then down the ventilation raise or a dedicated drill hole. A substation will be installed at each level. Electricity will be used to supply power to pumps, fans, electric jumbos, refuge stations, the jumbo drill, and possibly other major mobile equipment. Electric equipment has considerable advantages over diesel power, including longer machine life, less noise and most importantly, a cleaner underground atmosphere. It may be possible to also use electric-hydraulic jackleg drills.

Electrical substations will be located underground, with one substation serving each main level. After stopes are completed, it will be possible to relocate electrical equipment for reuse in new areas.

16.1.21.2 Compressed Air

Compressed air will be delivered to the underground using a main line of 4-inch steel pipe. The pipe will initially run down the ramp and then down the raise system once the raises are installed.

Branches from the main line will provide compressed air to the stopes via 4-inch steel pipe, and 2-inch steel and PVC pipes will deliver compressed air to the mining face. Equipment and activities that require compressed air include:

- jacklegs and stopers

- cleaning blastholes
- loading explosives
- refuge station pressurization

Small pneumatic pumps will be used to keep the decline face free of water.

16.1.21.3 Water Supply

Water will be used underground for drilling operations, dust suppression during mucking, and washing walls and backs for scaling and sampling purposes. Production water will be provided by water pumped from the polishing pond and gravity fed to the mine through the ventilation raise using a 2-inch steel pipe. Connections at each level will provide production water to each stope using 2-inch steel or PVC pipes delivering water to the active face.

16.1.21.4 Water Discharge

Water that accumulates at the active face will be pumped away using a small pneumatic diaphragm pump (Wilden type pump) or a small electric pump, using 2-inch steel or PVC pipe from the face to the sump.

Dirty water sumps should be connected by a system of overflow drain holes, with the cleaner water being pumped in stages, to the surface for clarification and reuse. The sump system should be designed so that the slimes can be cleaned out periodically.

A permanent pumping station will be constructed at the bottom of the mine that will pump water to the surface settling pond system. The pumping arrangement will be set up as a redundant parallel system, with either side capable of providing mine dewatering without the other.

16.1.21.5 Communications

A leaky feeder-type radio communication system is planned to be installed along with a wired phone system where necessary.

16.1.22 Maintenance

An experienced maintenance planner will run the department. Maintenance personnel will consist of mobile mechanics, industrial mechanics, electricians, and a drill doctor. The maintenance facility will consist of a maintenance shop and warehouse storage facilities.

16.1.22.1 Mobile Maintenance

A maintenance shop will be constructed on surface, near to the location of the underground portal. The shop will have two bays, with space for laydown in between bays. Mobile equipment will be brought from underground to surface for servicing, preventative maintenance, and repairs. The mobile maintenance team will have access to a mine utility vehicle, which will be used to access and service equipment underground as needed. Refuelling and lubrication of vehicles will be done on surface.

16.1.22.2 Ramp Roadbed

The maintenance of the main decline roadbed is important for mine tire life and haulage efficiency. A good ditch to channel water is a must. Occasional grading of the road is required.

16.1.23 Personnel

Hiring highly skilled hard rock miners with mining experience is highly recommended; however, some positions could be filled using less experienced miners. Lead hands, jumbo operators, scoop operators, production miners, and bolters would need to be skilled miners, while truck drivers and nippers could be less experienced.

16.1.23.1 Training

The applicable mining regulations (Coded of Federal Regulations, Title 30, Mineral Resources, Parts 1 to 199, revised as of July 1, 2014) require training for all new miners and refresher training for all miners every year. At least two mine rescue teams are required to be available when persons are underground at the mine.

16.1.23.2 Shift Schedule

Initially, it is planned to have four mining development crews to cover two shifts per day on a seven days on, seven days off schedule. For production mining, it may be possible to work on a two shifts per day, five days per week schedule and have the mill working seven days per week with one shift for maintenance.

There will be a gradual buildup of the mine employees.

16.1.23.3 Engineering, Geology and Surveying

Engineers will provide all plans for mining, using geological advice and guidance. A senior mine engineer will direct the technical services department. A junior engineer will also be hired, along with engineering students for work terms. Surveying will be done by a dedicated surveyor, with two employees splitting duties on a rotation. Geologists will maintain grade control and outline the valuable mineralization to keep dilution to a minimum.

16.1.23.4 Safety and Environment

It is anticipated that a staff person will be dedicated to training, safety, and environmental compliance. Two mine rescue teams are required, training and safety must be documented, and there are numerous records to be filled out and retained on site or submitted to regulators. It is a full-time position.

16.2 Open Pit Mining - Windy Gulch

16.2.1 Block Model

The Windy Gulch block model (mod_reg_final.dm) prepared by Golder has 10 ft by 5 ft by 5 ft blocks with a percentage field indicating how much of the block is considered indicated or inferred. The model has both metric and imperial grades and densities. Only the imperial density (short tons per cubic feet) and gold grades (ounces per ton) were used in the pit optimization. Both indicated and inferred blocks were included in this PEA.

The mod_reg_final.dm block model was imported into Whittle 4.6 (Whittle) pit optimization software. Whittle assigns a block value to each of the blocks within the model. The value is based on the modifying factors assigned. The block model was reblocked in Whittle to a 30 ft by 30 ft by 15 ft selective mining unit to better represent the size of blocks that could be mined with relatively small contractor mining equipment. Whittle retains the information of each of the 10 ft by 5 ft by 5 ft blocks combined into the 30 ft by 30 ft by 15 ft combined blocks so the contained gold in the reblocked model does not change, only the mining selectivity of the blocks is altered. The larger block size better represents the way the pit would have to be mined.

16.2.2 Geotechnical Sensitivity

Since no geotechnical information was available for Windy Gulch, Golder assumed a preliminary 45° overall slope angle and then performed a sensitivity analysis. A geotechnical assessment will be required to confirm whether the pit slope is required to be shallower or can be steepened. Variations to the pit slope will affect the pit size and economics. For example, shallower slopes will negatively impact the project economics.

Whittle was used to evaluate the effect of having an overall slope angle of 40° to 50°, both constrained (within the permit boundary) and unconstrained. The results of the analysis are summarized in Table 16-9.

Table 16-9: Sensitivity Analysis Comparing Geotechnical and Licence Constraints

Revenue Factor	Overall Slope (°)	Constraints	Waste (tons)	Mill Feed (tons)	Au (oz/tn)	Total Au (oz)	Strip Ratio
1.0	40	None	42,781	21,758	0.118	2,571	1.97
1.0	40	Lease	18,184	13,008	0.118	1,546	1.4
1.0	45	None	50,601	29,098	0.121	3,517	1.74
1.0	45	Lease	32,934	23,990	0.119	2,850	1.37
1.0	50	None	34,753	31,450	0.124	3,900	1.11
1.0	50	Lease	34,753	31,450	0.124	3,900	1.11

16.2.3 Equipment Assumptions

The terrain at Windy Gulch is steep and small, crawler-type drills are proposed. Excavation could be carried out using a relatively small excavator (25 to 45 ton). Articulated, six-wheel-drive haul trucks (20 to 35 ton) are appropriate for the steep, rough terrain. Any pit haul roads that are needed would be kept outside the cut as much as possible.

16.2.4 Whittle Analysis

A preliminary pit shell analysis was done in Whittle using an assumed gold selling price of $1,250/oz t. The following modifying factors were assumed for the base case pit shell analysis:

- Contract mining cost ($15/ton)
- Mining recovery (98%)
- Mining dilution (5%)
- Processing cost ($27.70/ton)
- General and administration cost ($xx/ton)
- Process recovery (80%)
- Selling price ($1,250/oz t).
- Selling costs (freight, treatment charges, totalling $171/oz t)

The analysis was also constrained by the permit boundary shown in Figure 16.7.

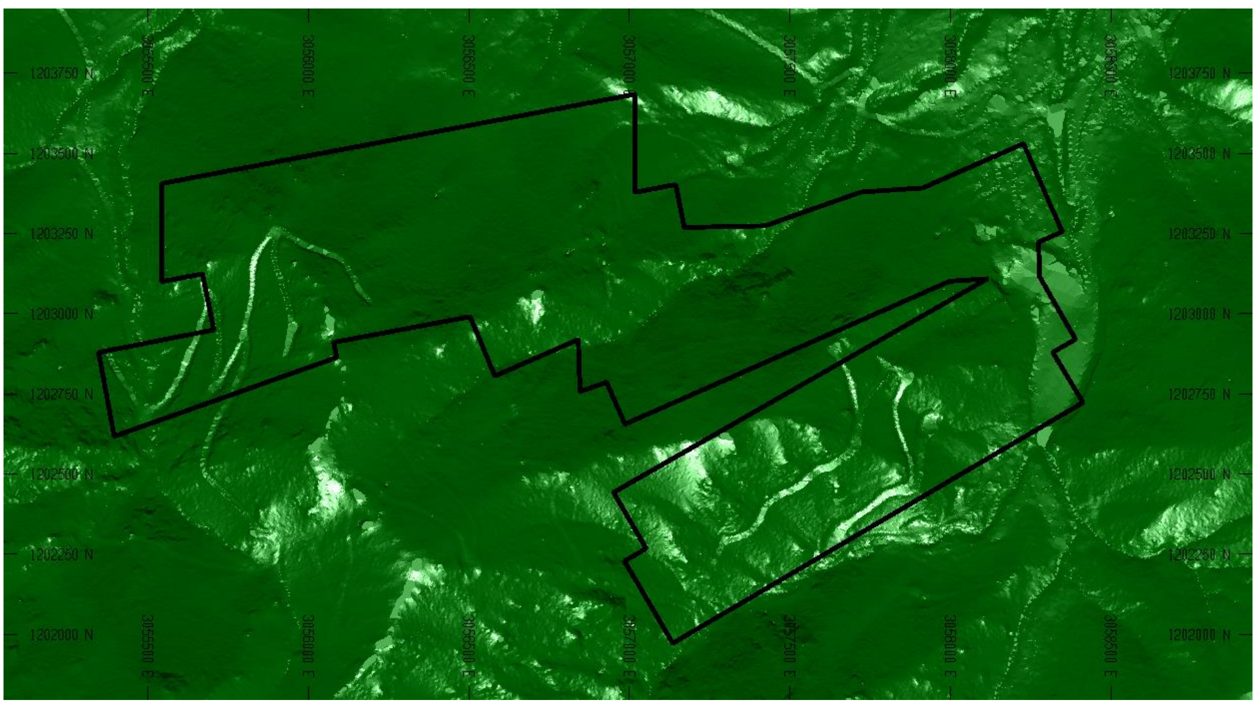


Figure 16.7: Whittle Constraint Boundary Based on Patented Leases

Whittle was used to evaluate the effect of the pit size when the selling price is varied between 0.5 and 1.4 times the base case selling price ($1,250/oz t). Each of the revenue factor pit shell's undiscounted value and mill feed, and waste tons is shown in Figure 16.8.

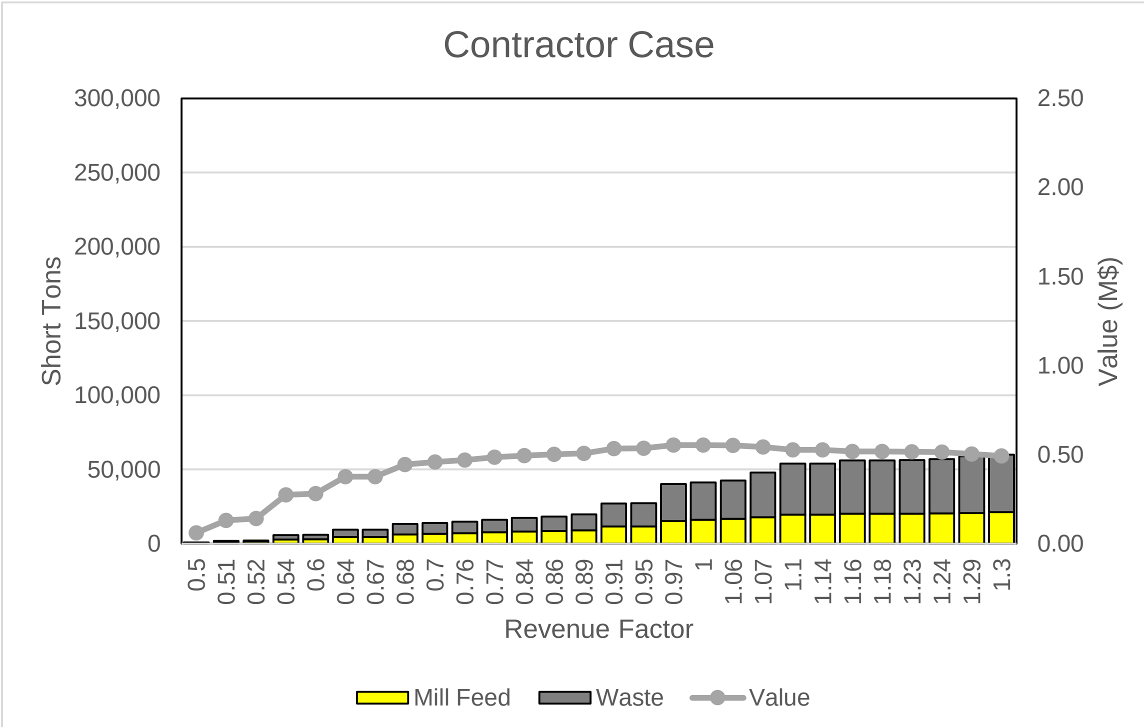


Figure 16.8: Whittle Revenue Factor Pit Shells Based on the Contractor Base Case

Figure 16.8 shows the mill feed and waste tons on one axis and the undiscounted value on the other axis for each of the revenue factors. The graph shows that by varying the selling price (revenue factor), the overall pit does not drastically change and the overall value of the pit also does not drastically change.

An owner operator case, Figure 16.9, was also analyzed as a potential alternative to mining the open pit with contractors. The mining cost of the owner operator case was reduced to $5/tn. The other modifying factors remained the same as the contractor case. Comparing Figure 16.8 and Figure 16.9 shows that the pit is sensitive to mining cost; however, the overall values of the pit shells created in this owner operator scenario do not vary significantly.

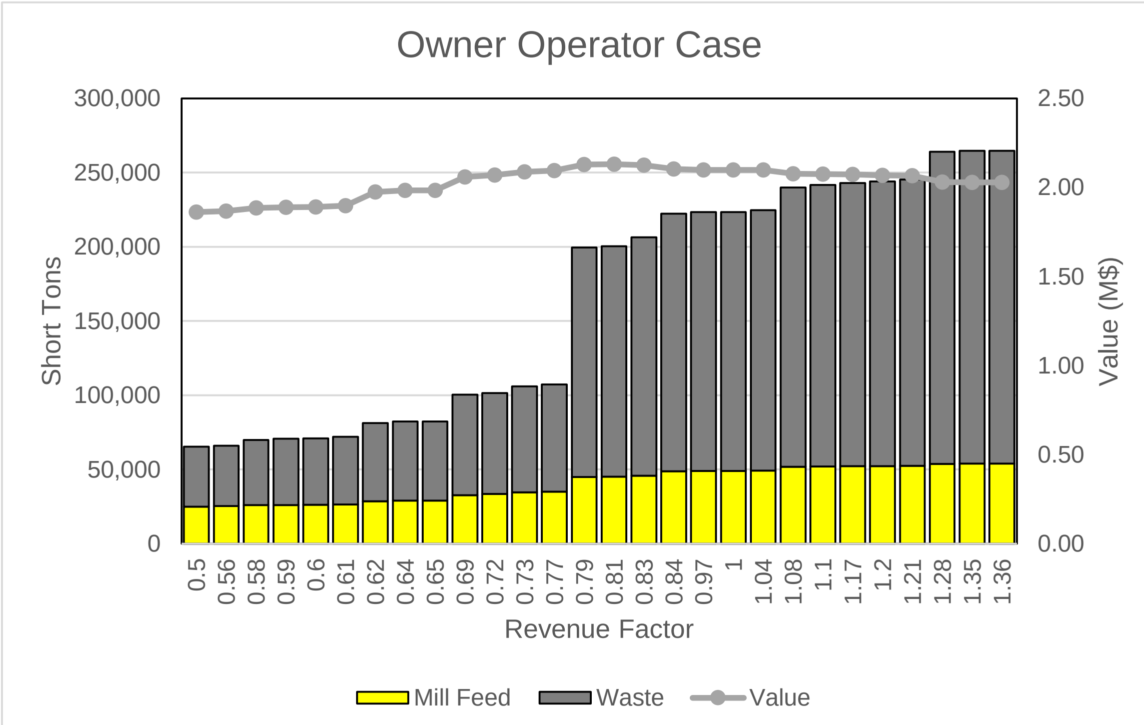


Figure 16.9: Whittle Revenue Factor Pit Shells Based on the Owner Operator Case

Golder presented the results of the sensitivity analysis to Zephyr and made a recommendation to consider a contractor base case since the size of the pit does not justify purchasing or leasing equipment, hiring and training a labour force, and supervising the construction A sensitivity analysis comparing the preliminary economic cashflow of the project including the small pit was done. Three scenarios were compared:

1) Assumed the pit would be mined before the underground in year one of the project.

2) Assumed the pit would be mined at the end of the project life (year five).

3) Assumed the pit would not be mined.

The result of the analysis showed the pit only provided approximately $100,000 increase in net present value (NPV) if mined at the end of the mine life, and a decrease in NPV if mined in year one. Golder presented the findings to Zephyr. Zephyr decided to omit the pit from the preliminary economic assessment until constraints, such as the mining licence boundary could be expanded, or until the potentially mineable resource could be increased. The small open pit could be profitable, however, with the assumptions made and the required infrastructure necessary to excavate the pit, the economic value does not justify including the Windy Gulch indicated and inferred resources in the economic assessment.

17.0 RECOVERY METHODS

17.1 Process and Plant Design – Introduction

The process design for Dawson plant is based on metallurgical test results provided in and the interpretations derived from the 1991 Hazen and the 2015 Inspectorate test reports. The findings of these test programs are summarized in previous sections of this report. The proposed process plant at Dawson project site near the town of Cañon City, in Fremont County, Colorado will be based on an annual plant throughput rate of 120,728 tn (109,500 t) of Run-of-Mine (ROM) ore from the mine based on a 365 days per year operation. Daily nominal throughput rate will be 330.8 tn (300 t).

Summary of major process design criteria is provided below.

Furthermore, the process is designed to enable the production of a marketable gold-silver concentrate containing several ounces of gold using conventional flotation techniques as well as a gravity concentrate containing 30-60 ounces of gold. The proposed ore processing facility will consist of a plant housing primary and secondary crushing circuits, fine ore screening and storage, grinding, gravity and flotation circuits, concentrate and tailings dewatering areas, a maintenance shop, and offices. These will be supported by ancillary circuits such as reagent preparation and utilities. Containers in a designated area will be used for on-site storage of reagents and other mill supplies. A portable trailer will house sampling preparation and assay laboratory equipment for mine, mill and environmental quality control needs of the operation.

A small gravity recovery circuit will be part of the grinding circuit to take advantage of the presence of any free gold in the plant feeds. Plant tailings containing both non-acid generating and potentially acid generating (PAG) streams from the flotation circuit will be combined and sent to a tailings thickener and downstream pressure filter for dewatering. Filtered tailings cake will be hauled to tailings management facility for storage while the thickener overflow and filtrate water will be recycled for re-use in the plant as process make-up water.

General arrangement plans and elevations for the process plant are provided in Appendix B.

Figure 17.1 below shows a simplified process flowsheet of the Dawson concentrator.

17.2 Process Plant Description and Major Equipment List

Preliminary metallurgical test programs to determine process parameters to aid in the design of the process plant are completed and the following plant description is prepared using the process flowsheet developed using the results of these programs as well as in-house experience with processing similar ores.

A summary equipment list showing select major process equipment and sizes is provided in Table 17-1 below. A complete list of major plant process and mechanical equipment providing preliminary sizing and power information on major mechanical and process equipment for the plant is also prepared for the development of preliminary capital cost for the mill.

Table 17-1: Mill Selected Major Process and Mechanical Equipment List

Equipment Description & Type		Nominal Capacity	Size or Model	No. Req'd	Est'd Inst'd Hp
Coarse Ore Dump	Hopper	50 t Live	322 ft^3 Live	1	
Vibrating Grizzly	Feeder	20 t/h	30 In X 10 ft	1	10
Primary Jaw	Crusher	60 t/h @ 1.6 In Css	20 In X 31 In	1	100
Secondary Cone	Crusher	72 tn/g @ 0.25 In Css	30 In Sh	1	100
Secondary Crusher Vibrating	Screen		6 ft X 12 ft	1	15
Fine Ore	Bin	300 tn Live	22 ft Dia. X 27 ft 6,100 ft^3 Live	1	
Crushed Ore	Conveyor		20 In X 236 ft	1	20
Secondary Crusher Screen Feed	Conveyor		20 In X 173 ft	1	20
Fine Ore Reclaim & Ball Mill Feed	Conveyor		30 In X 40 ft	1	15
Centrifugal Gravity	Concentrator		Semi Batch 400 Sq In	1	10
Cgc Concentrate	Shaking Table		Half Size: 4 ft X 9 ft	1	1
Primary Cyclone Feed	Pump		4 In	2	25
Primary Cyclone Feed	Pumpbox	44 ft^3 (Live)	43 In Dia. X 51 In	1	
Primary Ball	Mill		8 ft Dia X 12.33 ft Egl	1	450
Primary 1	Cyclone		10 In	2	
Rougher Flotation	Cell	100 ft^3	D-R 30	9	12.5
Regrind	Cyclone		4 In	2	
Regrind Ball	Mill		1.5 m Dia X 2.7 m Egl	1	135
Cleaner Flotation	Cell	100 ft^3	D-R 30	4	12.5
Crushing Grinding & Flotation O/H	Crane			1	10
Concentrate Thickener	Tank	15.4 tn/d Solids	8.3 ft Dia	1	
Concentrate Filter Feed	Tank	225 ft^3	6.6 ft Dia X 8 ft H	1	
Concentrate	Filter	15.4 tn/d Solids	63 ft^2	1	22
Tailings Thickener	Tank	360 tn/d Solids	60 ft Dia	1	
Tailings Filter Feed	Tank	2,200 ft^3	13.3 ft Dia X 16.6 ft H	1	
Tailings	Filter	360 tn/d Solids	1,066 ft^2	1	100
Tailings Filter Cake	Conveyor		36 In X 40 ft	1	10

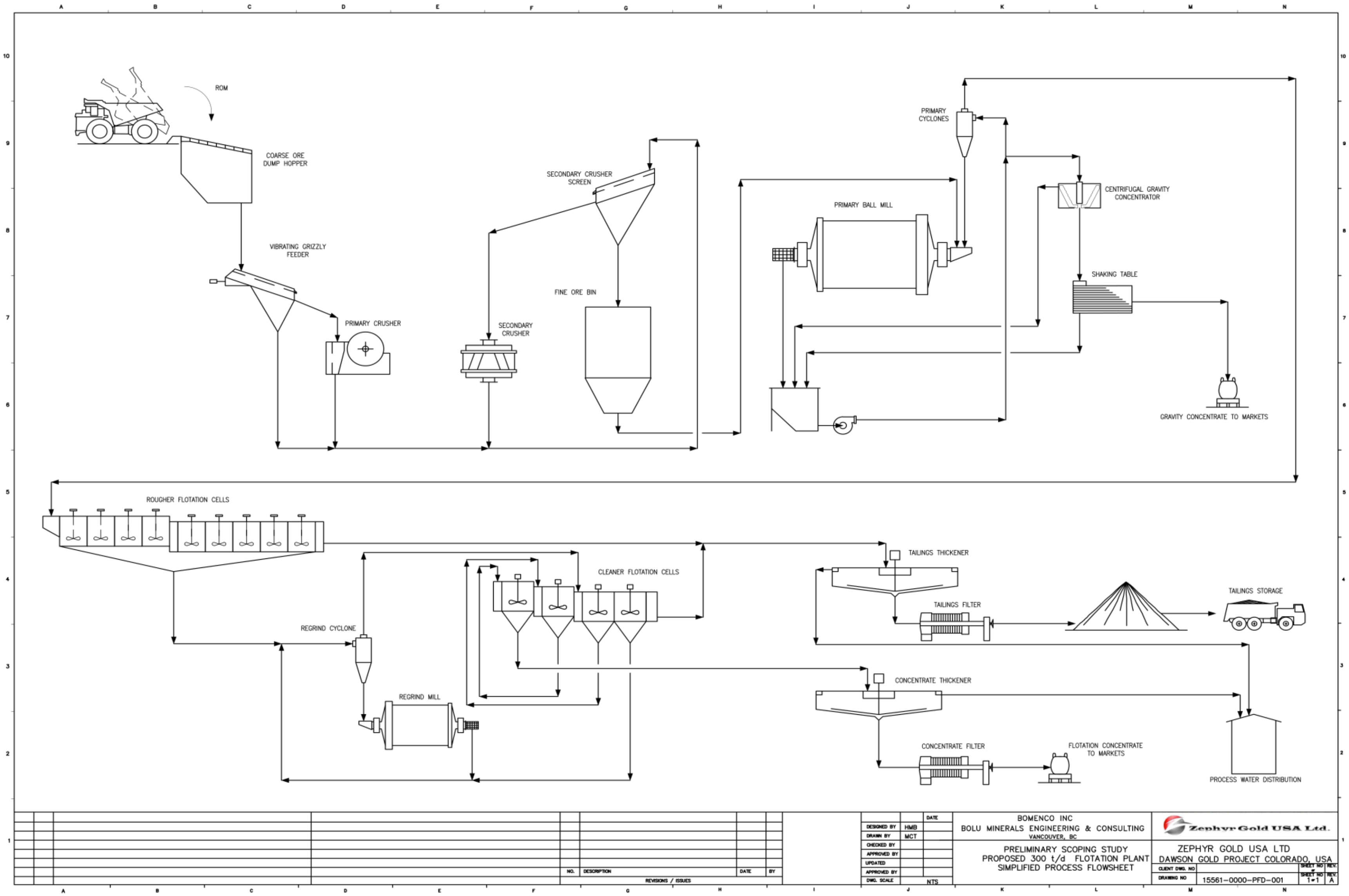


Figure 17.1: Simplified Process Flowsheet

17.2.1 Primary and Secondary Crushing

ROM ore from the open pit will be delivered by 20-ton ore trucks and dumped directly into a coarse ore dump hopper equipped with a grizzly on top and a vibrating grizzly feeder at bottom. The apron feeder will reclaim ore and feed it to a primary jaw crusher which will be set to produce a nominal 3-inch crushed product. The crushed ore will be removed by scissor conveyors to a vibratory screen which will be equipped with 2-inch top and 5/8-inch bottom opening screen panels. The nominal -5/8-inch screen undersize will fall by gravity into a 300-ton live capacity fine ore bin underneath the screen. The fine ore bin will provide approximately 20-hour surge capacity ahead of the grinding circuit. The +5/8-inch screen oversize on the other hand will feed a secondary cone crusher which will be set to deliver a -5/8-inch product. The cone crusher product will join the jaw crusher discharge on the crushed ore conveyor for transfer to the vibratory screen operating in closed circuit. A magnet placed over the crushed ore conveyor will help to remove tramp iron from the stream and a metal detector placed on the vibratory screen feed conveyor will help remove tramp iron ahead of the cone crusher. Dust is controlled by the use of a wet scrubber with the discharge slurry returning to the mill as recycle. The crusher will be scheduled to operate 16 hours per day or as required by the ore delivery and grinding circuit demands.

17.2.2 Grinding and Gravity Recovery

The fine ore, reclaimed by a belt conveyor/feeder from the fine ore bin, will feed the grinding ball mill. The grinding circuit will consist of a single ball mill operating in closed circuit with a hydrocyclone for classification. The grinding circuit is designed to treat fresh feed with an 80% passing size of 0.47 inches (12,000 µm) to produce a finished product with a target primary grind 80% passing size of 200 mesh (74 µm). The cyclone overflow will proceed to flotation while the cyclone underflow will gravitate to the ball mill for further size reduction.

A portion equivalent to approximately 70% of fresh mill feed tonnage will be diverted from the cyclone feed to the gravity circuit for the recovery of free gold. Major process equipment in this circuit will consist of a safety screen, a centrifugal gravity concentrator (CGC), and a shaking table. Gravity gold concentrate from the shaking table will be collected and may be processed on site to doré and separate marketing, or mixed with the flotation concentrate depending on its grade.

The safety screen oversize and tailings from both the centrifugal gravity concentrator and the table will join in a pump box for recycling back to the ball mill discharge / cyclone feed pump box for further classification and grinding. The safety screen oversize and tailings from the centrifugal gravity concentrator may go directly to the mill discharge pump box by gravity flow depending on plant layout in this area.

For the purposes of security, the shaking table may be operated during day shift only after storing the concentrate of the centrifugal gravity concentrator in a hopper.

17.2.3 Rougher/Scavenger/Cleaner Flotation and Regrind

Cyclone overflow from the grinding circuit gravitates to rougher/scavenger flotation bank of cells for bulk flotation concentrate recovery. A sulphide mineral collecting agent (PAX, potassium amyl xanthate) and a frother (MIBC, methyl isobutyl carbinol) are added ahead and during flotation for the promotion and collection of its precious metal and sulphide mineral content. Small addition of a secondary collector specific for the promotion of gold such as a dithiophosphate may be used. The rougher and scavenger flotation circuits produce their respective concentrates, which are then combined and reground ahead of cleaner flotation to produce a marketable grade final concentrate. Conventional trough style flotation cells providing 40 minutes of rougher/scavenger flotation residence time and 38 minutes of cleaner flotation residence time will be employed. The regrind mill will operate in closed circuit with a hydrocyclone to produce a target 80% passing regrind size of 400 mesh (37 µm).

Regrind cyclone overflow will gravitate to a bank of three-stage cleaner flotation cells operating in counter-current configuration. Third cleaner concentrate will head to dewatering, while the cleaner tailings will join the rougher/scavenger flotation tailings for dewatering for dry-stack tailings disposal.

The rougher/scavenger flotation circuit will produce approximately 75% to 80% of the plant feed mass, while the cleaner flotation circuit tailings stream will account for approximately 15% to 20% of the plant feed mass. The final concentrate product will account for the remaining 5% to 10% of the plant feed mass depending on mill feed and optimized concentrate grades targeted.

17.2.4 Concentrate Dewatering

The final cleaner flotation concentrate will first be thickened in a concentrate thickener to approximately 60% solids density. A flocculant will be used to aid in settling. Thickener underflow will be sent to a filter feed stock tank ahead of concentrate filtering. Concentrate filter cake at a target moisture content of 10% will be packaged into 2-ton bulk bags for shipment to markets. Filtrate water will be sent to the concentrate thickener, while the overflow from the thickener will be sent to process water tank as make up recycle water. It is estimated that approximately 5% to 10% of plant feed by weight or 1.6 to 3.3 tons per day will be recovered into the flotation concentrate.

17.2.5 Tailings Dewatering and Disposal

A dry-stack tailings system is proposed for Dawson. Combined tailings from rougher/scavenger flotation and cleaner scavenger flotation will be sent to tailings thickener for process water recovery and to aid in the subsequent filtration. The thickener underflow at a target solids density of +60%, will be pumped to a filter feed stock tank ahead of tailings filtering. A flocculant will be used to aid in settling. As in the concentrate dewatering, filtrate water will be sent to the tailings thickener while the overflow from the thickener will be sent to process water tank as make up recycle water.

Tailings filter cake at a target moisture content of 15% will be conveyed out to a temporary stockpile immediately outside of the mill building before being hauled intermittently by a truck to the designated tailings management facility for storage as described elsewhere in this report. **Note: the cost of this haulage is not covered in mill opex.**

17.2.6 Reagent Preparation

The following reagents will be required for the process. A sulphide mineral collecting agent (PAX, powder form) and a frother (MIBC, liquid form) are added ahead of and during flotation for the promotion and collection of precious metal and the associated sulphide mineral content. Small addition of a secondary collector, specific for the promotion of gold such as a dithiophosphate, may be used.

Reagents requiring mixing (PAX) will be mixed in a tank equipped with an agitator and then transferred to a head tank for distribution to their appropriate addition points in the mill. The head tanks will be situated so that all reagents will flow by gravity to a metering pump that will pump the required quantity to the desired location. Frother (MIBC) will not require mixing and it will be fed neat. A vendor-supplied flocculant mix system will provide 0.25% strength flocculant to the desired addition points. A secondary or a spare reagent may also be added from an Isotainer if/as required.

17.2.7 Services and Utilities

Services and utilities for the plant will include process water, filtered water, potable water, fresh water, high pressure compressed air, instrument air, and low pressure air blowers for flotation.

Fresh water for the mill and mine will be supplied, as required, from drilled wells. A tank and pumping system will be located at the source and it will be pumped to a fresh water holding tank at the plant. It is anticipated that drinking water will be hauled to site from the nearby town. It is estimated that 7.5 gpm (1.7 cubic metres per hour) of fresh water will be required as process and filtered water make-up.

Process water will be recycled to the extent possible through the concentrate and tailings dewatering systems. Required amount of process water will be filtered to reduce particulate matter to levels necessary for such uses as gland seal, reagent mixing and cooling water. Filtered water will be stored in a dedicated tank for distribution. Bottom portion of the filtered water tank will be dedicated to storing sufficient volume of water for firefighting.

Air for the mechanical flotation cells will be provided by a blower. High pressure air for general plant use and instrument air for plant will be supplied by two compressors one operating and one standby.

The plant will be serviced by a maintenance shop providing mechanical, electrical and instrumentation facilities. An assay lab will provide daily and shift quality monitoring information on solids and water samples for the safe and efficient operation of the plant.

17.3 Major Process Design and Production Criteria

The following major process design criteria were used for the preliminary design of Dawson plant (Table 17-2).

Table 17-2: Major Process Design Criteria

CONCENTRATOR GENERAL DESIGN BASIS		
AREA & CRITERIA	**Unit**	**Data**
GENERAL		
Plant design capacity	tn/yr	120,703
Life of mine (LOM)	yr	3.9
ROM Mill Feed Grades		
Gold	oz/tn (g/t)	0.27 (8.5 to 9.2)
Silver	oz/tn (g/t)	-
Sulphur	$\%S_T$	1.9
ROM Mill Feed Characteristics		
Specific gravity	-	2.8

Moisture, % H_2O	w/w	4
Maximum ROM mill feed size	in (mm)	24 (610)
OPERATIONAL CRITERIA		
Crushing Plant Design Capacity	tn/yr (t/a)	120,703 (109,500)
Operating days per year	d/a	365
Crushing Plant Operating availability	%	75
Crushing plant capacity, design	tn/d (t/d)	441 (400)
Grinding & Flotation Plant availability	%	92
Grinding & Flotation Plant capacity, design	st/d (t/d)	359 (326)
Operating shifts per day	-	2
Hours per shift	h	12
CRUSHING & SCREENING		
Haul truck capacity	t	20
Type of primary crusher	-	Jaw
Crusher product size P_{80}, nominal	in (mm)	3 (76)
Secondary screen openings	in (mm), top/bottom	2/0.6 (50/15)
Crusher type		Short Head Cone
Crusher Product size, nominal	in (mm)	-0.8 (-20)
Secondary screen undersize, nominal	in (mm)	-0.6 (-15)
Fine Ore Bin capacity, live	tn (t)	300 (272)
GRINDING AND GRAVITY CIRCUITS		
Bond ball mill work index	kWh/tn (kWh/t)	15.6 (17.2)
Grinding circuit	type	Single Ball mill
Classification type	-	closed circuit cyclones
Primary ball mill feed size, 80% passing size	in (µm)	0.47 (12,000)
Grinding circuit - Cyclone o/f P_{80}	mesh (µm)	200 (74)
Gravity concentration	type	CGC and shaking table
CGC feed rate as a % of fresh mill feed		70
FLOTATION AND REGRIND CIRCUITS		
Pulp pH	-	Natural
Aeration hold-up factor in flotation cells	%	15
Rougher-Scav Flotation residence time, design	min	40
Regrind Mill Type	-	Ball Mill
Regrind fresh feed F_{80}	mesh (µm)	200 (74)
Regrind product P_{80}	mesh (µm)	400 (37)
Regrind Feed Ball Mill Work index	kWh/tn (kWh/t)	12.7 (14)
Cleaner 1,2,3,4 Flotation residence times, design	min	20,12,10,16
CONCENTRATE AND TAILINGS DEWATERING		
Concentrate Thickener Feed Rate, Design	% of nominal	150
Concentrate Thickener Unit area thickening rate	$ft^2/tn/d$ ($m^2/t/d$)	2.44 (0.25)
Concentrate surge (Filter Stock) tank residence time	h	12
Concentrate Filter Feed Rate, Design	% of nominal	150
Concentrate Filter cake moisture content, nominal	%	10
Concentrate Filtration rate	$Lb/ft^2/h$ ($kg/m^2/h$)	20.5 (100)
Concentrate Bagging system		manual, bulk

Concentrate Bag capacity	tn (t)	2.2 (2.0)
Tailings Thickener Feed Rate, Design	% of nominal	150
Tailings Thickener Unit area thickening rate	ft^2/tn/d (m2/t/d)	2.44 (0.25)
Tailing Filter Type	-	Pressure Filter
Tailing Filter Feed Rate, Design	% of nominal	150
Tailing Filter cake moisture content, nominal	%	15
Tailing Filtration rate	Lb/ft^2/h (kg/m^2/h)	30.8 (150)
REAGENTS & UTILITIES		
Potassium Amyl Xanthate		
PAX Mixture Strength	%w/w	10
PAX Addition rate, total	Lb/tn (g/t)	0.10 (50)
Secondary Collector Mixture Strength	%w/w	100
Secondary Collector Addition rate, total	g/t	20
MIBC Mixture Strength	%w/w	100
MIBC Addition rate, total	Lb/tn (g/t)	0.05 (25)
Flocculant Mixture Strength	%w/w	0.02
Flocculant Addition rate, total	Lb/tn (g/t)	0.04 (20)
Low Pressure Flotation Air Supply pressure, design	psi (kPag)	9.3 (64)
Plant Air Supply pressure, design	psi (kPag)	150(1,034)
Instrument Air Supply pressure, minimum for design	psi (kPag)	110 (760)
Fresh Water Source		wells
Fresh Water Supply , average rate	gpm (m^3/h)	TBD
Fresh water tank residence time at average usage	h	24
Gland seal water supply pressure	psi (kPag)	70 (480)
Filtered Water Tank Residence time at average use	h	0.5
Fire Water Tank & Filtered Water Tank Source	-	Filtered Process Water
Process water tank residence time at average use	h	2
Process water supply pressure	psi (kPag)	70 (480)
Process water tank residence time at average use	h	0.5

17.4 Plant Mass and Water Balance

Mass balance of major concentrator streams and products are summarized in Table 17-3 below and a preliminary water balance for the process plant is depicted in Figure 17.2 below.

Table 17-3: Mass Balance of Major Process Streams

Process Stream		MASS BALANCE								
No	ID	SOLIDS			WATER			SLURRY		
		st/h	SG	usgpm	st/h	%Sol	usgpm	st/h	SG	usgpm
PRIMARY, SECONDARY CRUSHING & SCREENING										
101	ROM Delivery	18.4	2.80	26.4	0.8	96	3.1	19.2	2.60	29.1
102	Cone Crusher Feed/Discharge	18.4	2.80	26.4	0.8	96	3.1	19.2	2.60	29.1

Process Stream		MASS BALANCE								
103	Secondary Crusher Vibrating Screen Feed	36.7	2.80	52.4	1.5	96	6.2	38.3	2.60	58.6
PRIMARY GRINDING & CLASSIFICATION										
201	FOB Reclaim; Ball Mill Fresh Feed	15.0	2.80	21.6	0.6	96	2.6	15.7	2.60	23.8
204	Pr.Cyclone Feed Pump Disch.	62.8	2.80	89.8	61.5	51	245.7	124.1	1.50	335.1
206	Pr. Cyclone O/F, Ro.Flotation Feed	14.9	2.80	21.1	27.7	35	110.5	42.5	1.30	131.7
210	Primary Cyclone Feed	52.4	2.80	74.9	51.1	51	204.3	103.4	1.50	278.7
211	Feed to CGC (from Pr.Cyc Feed Split)	10.5	2.80	15.0	10.3	51	41.0	20.7	1.50	55.9
226	Comb`d Gravity Cct Tails (to Cyc Feed Pump)	10.4	2.80	14.5	20.4	34	81.5	30.6	1.30	96.0
227	Final Gravity (Table) Con	0.1	4.00	0.0	0.1	50	0.4	0.2	1.60	0.4
FLOTATION & REGRIND										
301	Rougher Tails	11.9	2.70	17.6	22.2	35	88.5	34.1	1.30	106.1
302	Rougher Concentrate	3.0	3.40	3.5	5.5	35	22.0	8.5	1.30	25.5
303	Regrind Cyclone Feed	10.7	3.50	12.3	12.6	46	50.2	23.4	1.50	62.5
304	Regrind Cyclone O/F	3.5	3.50	4.0	8.7	29	34.8	12.3	1.30	39.2
310	1st Cleaner Scavenger Concentrate	0.6	3.90	0.4	1.8	25	7.0	2.4	1.20	7.9
311	1st Cleaner Scav Tails (PAG Tails)	2.5	3.20	3.1	6.7	27	26.9	9.3	1.20	29.9
CONCENTRATE THICKENING & FILTRATION										
435	Concentrate Thickener u/f	0.5	4.80	0.4	0.4	55	1.3	0.8	1.80	1.8
442	Concentrate Filter Cake	0.5	4.80	0.4	0.1	88	0.4	0.5	3.30	0.4
TAILINGS THICKENING & FILTRATION										
451	Tails Thickener Combined Feed	14.4	2.80	20.7	28.9	33	115.4	43.3	1.30	136.1
455	Tailings Thickener u/f	14.4	2.80	20.7	11.8	55	47.1	26.2	1.50	68.3
462	Tailings Filter Cake	14.4	2.80	20.7	2.5	85	10.1	17.0	2.20	31.3

As the preliminary balance indicates, the plant will require 135 gpm of process water to operate while over 90% recycle rate will minimize fresh water usage. It is estimated that the mine will supply 3.6 gpm of water in the form of ROM moisture and mine water, and 6.2 gpm of fresh water will be required for cooling, reagent mixing, process make-up, and potable water purposes throughout the plant. Drinking water will be hauled in from the nearby town.

The concentrator water balance is prepared by BOMENCO, and all water requirements outside of the plant are provided by other consultants.

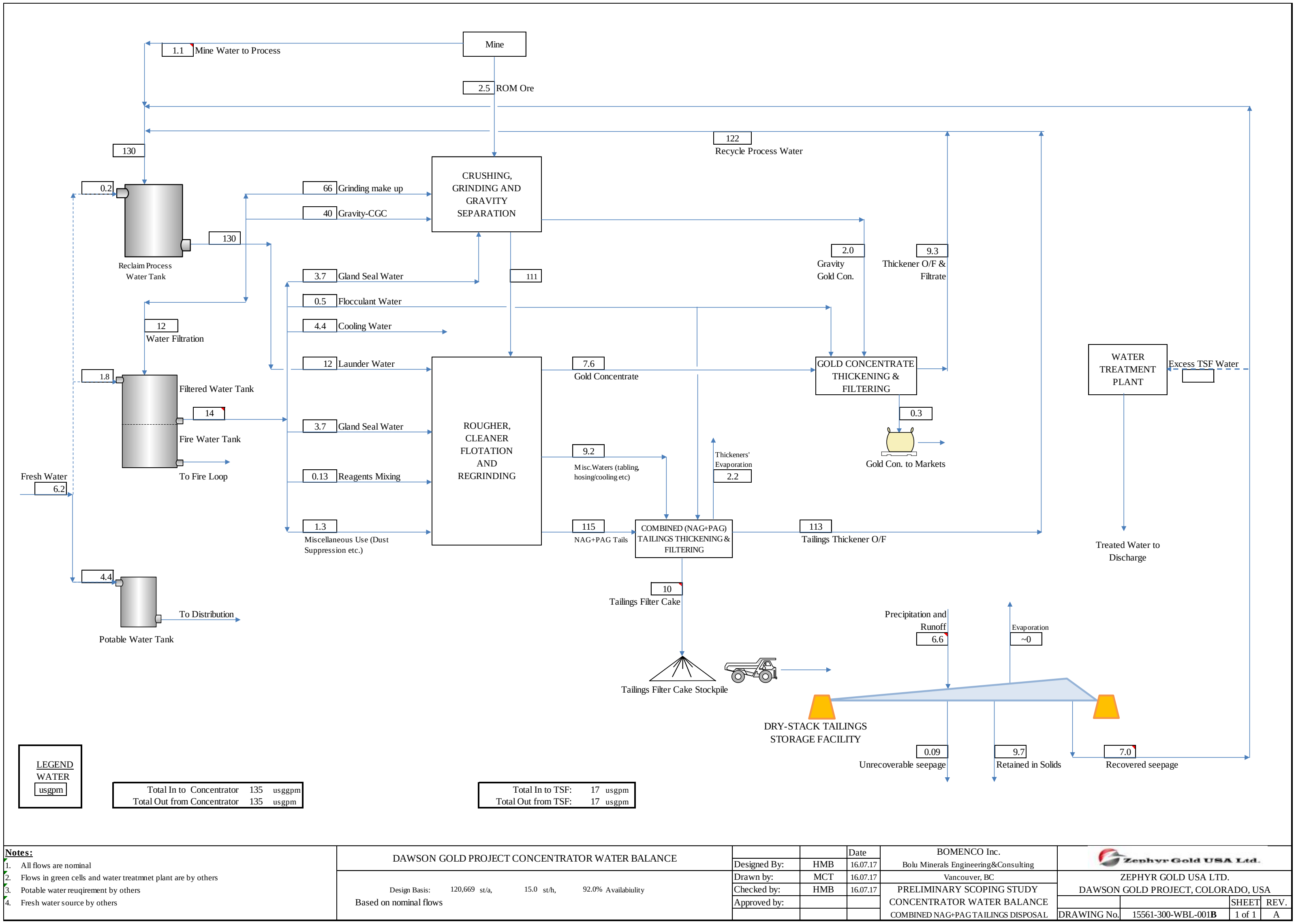


Figure 17.2: Preliminary Scoping Study Process Plant Water Balance

18.0 PROJECT INFRASTRUCTURE

18.1 Process Plant

Process plant building will house the following major processing facilities:

- ore receiving ramp and primary crusher feed hopper
- primary crushing
- secondary crushing
- transfer conveyors
- primary grinding and regrind mills
- rougher, cleaner flotation
- gravity recovery circuit
- reagent mixing and preparation
- concentrate and tailings dewatering
- process, fire and fresh water systems
- air (low/high pressure, instrument) systems
- mill maintenance facilities
- process plant offices

ROM ore from the mine will be transported by truck on a ramp to the hopper through a fixed grizzly. Grizzly undersize will be fed to the primary crusher via a vibrating grizzly feeder. The primary crushing structure will be re-reinforced concrete and steel and extended over the hopper to provide for shelter and reduce escaping dust. Primary crushed ore will be conveyed to a transfer tower on an inclined conveyor for furtherance to vibrating screen and the fine ore bin. Dust suppression and collection will be installed at conveyor transfer points to minimize and build-up of dust. Conveyors will be supported by steel bents and towers with concrete footings.

A small ROM ore stockpile may be required ahead of the primary crushing facility to allow for flexibility in mine ore deliveries and crushing schedule. A loader will be required to reclaim ore from the stockpile which will not be covered.

Process equipment will be housed in a steel structure plant building measuring 100 ft long and 50 ft wide. All major equipment will be placed on concrete foundations and on slab floor as appropriate. Floor slope will allow collection of spillage into sumps for pumping to their respective areas as required.

Electrical switchgear for the process plant will be located in a trailer placed next to the plant building.

18.2 Assay and Metallurgical Laboratory

Assay and metallurgical laboratories will be housed in a portable trailer assembled off-site with all necessary test and safety equipment. The trailer will be hauled to site and placed and secured on concrete slab near the process plant building.

18.3 Filtered Tailings Stockpile

Combined tailings, after thickening and filtering will be conveyed to a daily stockpile just outside the plant. Filtered tailings, will be transported to the storage facility by a haul truck intermittently as needed. Tailings thickener will be placed outside the plant building and will contain a berm around the base to contain any potential spillage.

18.4 Water Supply

Process water required for the plant will be stored in a tank which will be sized to supply sufficient water to last 30 min of operations. Supply of process water will be from concentrate and tailings thickener overflows, mine water, tailings storage recycle and fresh water makeup. A small portion of the process water will be filtered to meet requirements of select needs in the process plant such as reagent mixing, shaking table operation, cooling water etc.

Filtered water will also be used to supply the fire water tank for fire protection from the same filtered water tank. The outlets for the filtered water tank will be arranged in such a way that filtered water requirements can never draw down the level below the volume required for firefighting.

Fresh water will be supplied from water wells as required and will be stored in a fresh water tank ahead of treatment for distribution. Drinking water will be hauled in from the nearby town in bottles as required.

18.5 Fire Protection and Fire Water Pumping

All enclosed areas in the plant will be serviced by a sprinkling system. Portable extinguishers will be provided at key locations also. A fire truck will be stationed on site. Main fire water pumps will be placed in the in the plant adjacent to the filtered water tank. Two of the pumps will be electric and the third one will be diesel motor drive. A jockey pump will maintain supply pressure in the system at all times.

18.6 Warehouse/Process Equipment Parts Storage

A warehouse and parts storage facility will be allowed for in the combined structure that will also house the maintenance and admin offices. Additional storage, when needed will be provided in sea cans brought to site.

18.7 Utilities

Based on the power estimate provided by BOMENCO, the total connected load for the surface facilities is 1,265 kW. This total load is made up of the process plant and other site services. There are electric loads for the underground workings, including ventilation, dewatering, jumbo drills, and battery charging stations. An estimated 300 kW is added for the total mine load of approximately 1,565 kW. This converts to approximately 2,000 kVA, or 2 MVA with an assumed 0.8 power factor.

There are two potential utility connections that can support the proposed site, both supplied from the Cañon Plant substation. This substation, located beside the old Cañon power plant, distributes 13.2 kV, 69 kV, and 115 kV to the area. Black Hills Energy is the local distribution company in this area, so they own the substation and the transmission and distribution lines. One of the 13.2 kV circuits runs along Temple Canyon Road up to the old scrap yard. This is designated by Black Hills as South Cañon feeder 64021. Beyond this pole runs a single phase that extends to the cell phone tower. Black Hills proposes that this line from the intersection of Temple Canyon Rd and Mariposa Rd be re-conductored and extended to the site.

Regarding the capacity of this line, three-phase power supplied at 13.2 kV can offer 9 MVA, based on the current carrying capacity of conductors at that voltage. The mine load is estimated at 2 MVA, so there may be capacity in this line. Black Hills was consulted to evaluate the remaining capacity on these lines and the sensitivity to adding new loads onto it. They ran a flicker study that indicated that the starting or locked rotor current of the largest motor (estimated to be a 400 hp crusher motor) must be limited by a control mechanism such that at no time will the starting or locked rotor current exceed 1.5 per unit (P.U.) of the full load current of the motor. This may be limiting to the mine when the site is in full operation since the starting of a large motor, like a crusher, could potentially trip the entire site. There would be little capacity in the system to expand beyond the estimated 2.0 MVA. A more detailed study can be performed with an application and the results would be provided in several weeks with an electrical assessments and cost estimate.

The other potential utility tie-in is at a 69 kV transmission pole located 1,000 ft north of the intersection of Temple Canyon Rd and S 1st Street. This tie-in location is approximately 3.7 mi from the proposed mill, running along Temple Canyon Rd. Although the capital costs are higher than 13.2 kV, supplying the mine at 69 kV would provide a greater level of flexibility and reliability. The losses are also reduced when running at a higher voltage, so the operating costs would be reduced. Given the mountainous terrain, there are additional challenges and costs in installing transmission lines, but with the increased spacing of 69 kV poles, there is a smaller incremental cost compared to a 13.2 kV line. It is possible that the initial mine connect at 13.2 kV and then expand to a 69 kV service when the mill is ready to run at full load.

In either interconnection scenario, a substation would need to be installed at the mill site in order to step the voltage down to 4,160 V or 480 V. The crusher and the main ventilation fans are the larger loads, but it is estimated that they will operate at 480 V. The underground operations can be supplied at 13.2 kV or 4,160 V, depending on the total expected load. Most current mines are supplied at 13.2 kV since the cost to build a 15 kV network over one at 5 kV is unsubstantial but can gain three times the capacity. A standby diesel generator can be connected to the substation to provide emergency back-up power, and possibly start-up assistance for large loads.

18.8 Waste Rock Storage Area

A waste rock storage area (WRSA) will be located near the portal of the underground. The WRSA will be used to stockpile all waste rock that is hauled out of the underground. No geotechnical data are available for the foundation conditions in the area of the portal location. An assessment of the area will be required prior to selecting the WRSA location.

No geochemical analysis has been completed on the waste rock to date. Therefore, the assumption is that all material is potentially acid generating (PAG) and water runoff will be treated at the mill site, which is directly below the WRSA.

18.9 Filtered Tailings Storage Facility

Tailings will be stored in a filtered tailings storage facility (FTSF) in the valley immediately north of the proposed process plant. This location was selected due to its proximity to the portal and process plant, its relatively small upstream tributary watershed, and favourable storage capacity relative to other nearby sites. Tailings from gravity separation and flotation circuits will be filtered to their approximate optimum moisture content and will be spread and compacted in the FTSF. Advantages of the FTSF compared to conventional slurry tailings for this project include:

- reduced TSF footprint
- recycled water to the process circuit, reducing needs for make-up water
- increased recovery of process solutions at the plant
- reduced operational risks relating to stability and seepage
- operational flexibility for tailings placement and potential for expansion
- opportunity for progressive reclamation of the tailings facility during operations

The Dawson FTSF has been designed to store up to approximately 1.0 million tons of tailings over an approximately 10-year period, based on a mill throughput of 300 tons per day.

18.10 Tailings Characterization

For this study, one composite synthetic tailings sample was generated from 60 core samples obtained from exploration drilling of the Windy Gulch deposit. The Windy Gulch tailings were found to be non-plastic and have 60% by weight passing the #200 sieve (75 µm). The standard Proctor yielded a maximum dry density of approximately 111 pounds per cubic foot at an optimum moisture content of 15.9%.

Geochemical testing on the tailings sample included static testing to characterize the mineralogy and acid generation potential and accelerated weathering tests to characterize the metals leaching potential. Results from the acid base accounting testing of the tailings from the Windy Gulch Segment are mixed with respect to the potential for the material to be acid generating. The sulphur speciation data show that there is very little sulphur (0.1 weight percent) remaining in the tailings, and of that, only 0.01 weight percent is sulphide-sulphur. In addition to the low concentrations of total and pyritic sulphur in the tailings, the neutralization potential ratio and net acid generation pH results indicate the material is classified as non-potentially acid generating; however, the net neutralization potential results show the tailings to be "uncertain." The acid neutralization potential (NP) for all splits greatly exceeded the acid generation potential (AP); however, NP was not due to the presence of calcite as there was no "fizz rating." Therefore, the NP must be derived from iron oxides, reactive aluminosilicate, or silicate minerals.

The leachability of constituents from the tailings was tested using the Synthetic Precipitation Leaching Procedure (SPLP) and the Meteoric Water Mobility Procedure (MWMP). These procedures were performed on a subsample of the three splits of the composite tailings sample. The concentrations of most constituents in the leachates from both the SPLP and MWMP were well below standards for domestic drinking water and agriculture published by Colorado Department of Public Health & Environment Water Quality Control Commission (WQCC). However, the leachate concentrations of four constituents (cobalt, copper, manganese, and selenium) were very near to, or exceeded, the water quality standards. The concentrations of cobalt in the leachates for both the scaled SPLP and MWMP results were slightly greater than the agricultural standard of 0.05 mg/L. The concentrations of copper, manganese, and selenium in the leachates were approximately an order of magnitude greater than the domestic water supply and/or the agricultural water quality standards. These data indicate that there is potential that meteoric water infiltrating through the tailings could mobilize the constituents at levels that could negatively impact waters of the State. In addition to the four metals in the leachate that exceeded a water quality standard, the pH of the leachate from both the SPLP and MWMP were below pH 6 and in two MWMP samples were below pH 5. While the paste pH and net acid generation pH values were very close to or above pH 6, the lower pH in the SPLP and MWMP leachates suggest that there is some potential for the tailings to be slightly acidic, at least initially before the oxides, and/or aluminosilicates/silicates provide buffering due to their weathering.

Geochemical characterization of the Dawson and Windy Point tailings has not been undertaken. Additional testwork will be needed to characterize the geochemistry of the tailings.

18.11 Geotechnical Site Conditions

The Dawson project site is located in the northern Wet Mountains in the Southern Rocky Mountain Physiographic Province and is characterized by Proterozoic volcanic and sedimentary units. The Reconnaissance Geologic Map of the Royal Gorge Quadrangle (Taylor et al., 1975) maps nearly the entire footprint of the proposed Dawson FTSF as Cretaceous Dakota Sandstone and Purgatoire Formation. The Dakota Sandstone is described as a yellowish-brown fine-grained sandstone containing some shale of the Dry Creek Member in the upper middle portion of the section. The Purgatoire Formation consists of shale and sandstones of the Glencairn Shale Member and Lytle Sandstone Member. The formation is reported to have a total thickness of approximately 300 ft.

Dakota sandstone outcrops can be observed to dominate the south facing slopes of the valley at the northern portion of the proposed Dawson FTSF footprint, whereas alluvial/colluvial deposits overlie bedrock on the north facing slopes of the valley. Bedrock outcrops are present along parts of the drainage bottom. The sandstone outcrops were slightly weathered, medium strong to strong, and slightly fractured.

Two monitoring wells were installed at the project site in 2014. One well is located downstream of the proposed FTSF ("Dawson North") and the other well is located approximately 1,000 ft south ("Dawson South"). Depths to groundwater has been monitored quarterly by Zephyr since October 2014. The depth to groundwater in the

Dawson North monitoring well, which is more representative of the FTSF than the Dawson South monitoring well, was reported to be 190 ft at the time of installation (September 2014) and has varied from 168 to 176 ft in quarterly readings taken between October 2014 and October 2015.

Three test pits were excavated within and in the vicinity of the proposed FTSF footprint in August 2016. Subsurface conditions encountered in the test pits consisted of approximately 5 to 24 inches of topsoil overlying dry to moist, moderately dense clayey sand with gravel to sandy clay with some gravel and cobbles. All test pits were excavated to depths of 9 to 10 ft (maximum reach of the backhoe), and no groundwater was observed in any of the excavations. Bedrock was encountered at a depth of 9 ft in one test pit excavated near the bottom of the drainage.

18.12 Seismicity

As part of the Dawson FTSF pre-feasibility study, Amec Foster Wheeler conducted a site-specific seismic hazard study using both deterministic and probabilistic analyses. The probabilistic seismic hazard analysis estimated the following peak ground accelerations associated with various return periods:

- 0.04 g for an approximate return period of 500 years
- 0.11 g for an approximate return period of 2,500 years
- 0.17 g for an approximate return period of 5,000 years

The peak ground acceleration for the maximum credible earthquake is estimated to be 0.11 g based on the 84th percentile deterministic spectra for the larges earthquake on the closest Quaternary fault. The seismic hazard study is documented by Amec Foster Wheeler (2016a).

18.13 Key Design Components

Tailings will be hauled by truck from the filter plant to the FTSF site, where they will be spread in thin lifts and compacted. The key design components of the FTSF include the following:

- The foundation will be prepared by clearing and grubbing of significant vegetation within the FTSF footprint area and stripping and stockpiling of topsoil for use in reclamation of disturbed areas.
- No liner system is included in the FTSF design based on the low seepage rates observed from compacted filtered tailings stacks and the relatively benign geochemistry of the tailings. However, an underdrain system will be constructed within the drainage bottoms of the FTSF to capture any seepage from the filtered tailings stack as well as any potential shallow groundwater or seepage. The seepage will be directed to a lined contact water pond downstream of the FTSF. Contact water collected in the pond will be recycled back to the process plant, evaporated or treated, and released in accordance with State environmental requirements.
- A starter buttress will be constructed within the valley bottom at the toe of the FTSF to provide lateral confinement of tailings at the operations start-up. The starter buttress will be constructed with non-mineralized rockfill and will also provide stability, drainage, and erosion protection to the toe of the filtered tailings stack.

- The FTSF design includes two zones for tailings placement to achieve physical stability of the facility while also providing operational flexibility of tailings during times of heavy precipitation or upset filter plant conditions: 1) Shell Placement Area (Zone 1), and 2) General Placement Area (Zone 2). The Shell Placement Area forms a 110 ft wide zone (measured horizontally) at the exterior shell of the FTSF and shall be compacted to at least 95% of the maximum dry density as determined by the standard Proctor test (ASTM D698). Tailings in the General Placement Area shall be compacted to at least 90% of the standard Proctor dry density.
- A contingency tailings impoundment will be located near the process plant for temporary tailings storage during times when the tailings filter plant is off-line for maintenance or operational problems.
- The tailings will be stacked at an overall slope of 3H:1V with intermediate benches to control erosion and runoff.
- An erosion protection layer, consisting of inert, non-potentially acid generating fine waste rock, will be progressively placed on the downstream slope of the FTSF during operations for erosion protection. Since the erosion protection layer will be progressively placed on the tailings slope during operations it will become part of the reclamation cover.
- Surface water runoff shall not be allowed to run-on to the FTSF to the extent practicable. Stormwater runoff that has not come into contact with tailings ("non-contact" water) will be captured by perimeter diversion channels and routed around the south margin of the FTSF to discharge to the natural drainage east of the FTSF. The non-contact diversion channels have been designed to intercept runoff water from the contributing watersheds tributary to the FTSF and direct the clean non-contact water around the FTSF. The FTSF perimeter stormwater channels will be relocated as the tailings dry stack footprint increases (approximately every two years).
- Perimeter diversion channels will be constructed around the FTSF to capture clean (i.e., non-contact) water and route it around the FTSF, thereby preventing clean water run-on to the FTSF.
- Contact water, such as runoff of direct precipitation onto the tailings surface or water from the foundation underdrain, will be routed to a geomembrane-lined contact water pond located downstream of the FTSF. Contact water collected in the pond will either be evaporated, recycled to the process plant, or treated (as required to meet water quality standards) and released. A pumping and piping system will be constructed to provide the capability to recycle the contact water to the process plant.
- For closure, a vegetative cover system will be constructed over FTSF. A closure channel will be constructed around the perimeter of the ultimate tailings facility to capture surface water runoff and prevent surface water runoff flow onto the reclaimed tailings facility. The final surface of the FTSF will be graded to promote runoff from direct precipitation to the closure channel. Seepage from the FTSF is expected to be negligible; however, seepage will be monitored and treated if necessary to meet water quality standards of the State of Colorado. Once demonstrated to no longer be necessary for water quality monitoring, the contact water pond will be decommissioned, re-graded to original topography, and re-vegetated.

A typical plan view and cross-section of the ultimate (1.0 million ton) FTSF is presented in Figure 18.1.

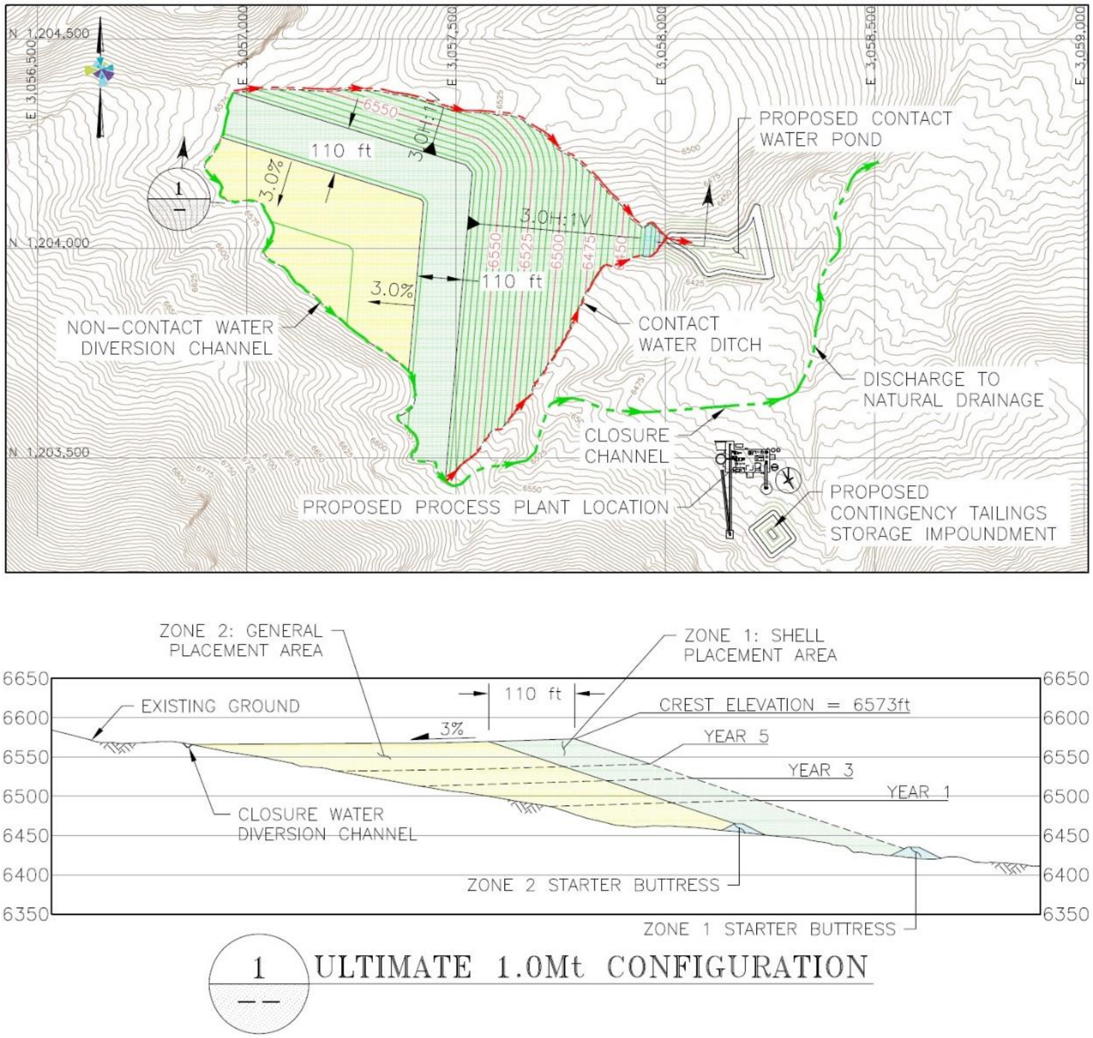


Figure 18.1: Typical Plan and Section of the Ultimate (1.0 million ton) FTSF

18.14 FTSF Stability and Seepage Analyses

The FTSF is designed for stability during operations as well as long-term stability after closure. As the tailings are compacted in an unsaturated condition, they are not considered to be susceptible to liquefaction during a seismic event. The stability analyses indicate that the factors of safety exceed 1.5 for static loading conditions. Furthermore, the FTSF is stable under seismic loading conditions as evidenced by pseudo-static factors of safety greater than 1.3, although some deformations of the facility may occur during a seismic event. Details for the stability analyses are provided by Amec Foster Wheeler (2016b).

One-dimensional seepage analyses were conducted to provide a first-order estimate of seepage rates from the FTSF in order to size the underdrains and contact water pond. Average seepage flows were estimated to be less than 1 gpm for the ultimate configuration of the FTSF.

18.15 FTSF Capital Cost Estimate

The FTSF will be initially constructed to provide tailings storage for approximately the first two years of tailings storage. The facility will be expanded over time as the tailings dry stack grows. Capital and sustaining capital costs include site preparation, clearing and grubbing, topsoil removal and stockpiling, and construction of foundation underdrains, toe buttress, surface water diversion channels and the contact water pond. Material quantities were estimated based on the design drawings and unit rates were based on vendor quotes, Colorado Department of Transportation (CDOT) construction cost databases, the Nevada Standardized Reclamation Cost Estimator (SRCE, 2016) and costs from other projects. Contingency of 20% was applied to the estimated costs. Capital costs for a 5 year FTSF are presented in Table 18-1. The cost estimate was developed to an accuracy of +/- 35%.

Table 18-1: Filtered Tailings Storage Facility Capital Cost Estimate

Item	Year 0 Cost (US$)	Year 2 Cost (US$)	Year 4 Cost (US$)
Site preparation, toe buttress and earthworks	$115,000	$44,000	$38,000
Underdrains	$57,000	$25,000	$11,000
Surface water channels	$195,000	$62,000	$89,000
Contact water pond	$121,000	-	-
Monitoring and instrumentation	$18,000	$5,000	-
Contingency tailings storage impoundment	$14,000	-	-
Total Construction Cost	**$520,000**	**$136,000**	**$138,000**
Contingency (20%)	$104,000	$27,000	$28,000
Total Construction Cost + Contingency	**$624,000**	**$163,000**	**$166,000**
Indiret Costs (10%)	$62,000	$16,000	$17,000
Total	**$640,000**	**$179,000**	**$183,000**

18.16 FTSF Operating Cost Estimate

PEA-level operating costs were estimated for the FTSF. Annual operating costs include road construction and maintenance, tailings haulage and placement at the FTSF, placement of the erosion protection layer, and surveillance and monitoring. The estimated annual operating costs of the FTSF are presented in Table 18-2.

Table 18-2: Filtered Tailings Storage Facility Operating Cost Estimate

Item	Annual Cost (US$)
Annual earthworks (roads, tailings and erosion protection layer placement)	$224,000
Annual Monitoring	$59,000
Total Annual Operating Costs	**$283,000**
Contingency (20%)	$57,000
Total Construction Cost + Contingency	**$340,000**

19.0 MARKET STUDIES AND CONTRACTS

No market studies have been undertaken for this preliminary economic assessment. Details of the mining licence contracts can be found in Section 4.0.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

Zephyr conducts exploration activities and intends to conduct mining activities in compliance with all applicable environmental, land use, and health and safety regulations and legislation. Zephyr is unaware of any existing environmental, land use, and health and safety issues or compliance challenges that have the potential to hinder future exploration, mine permitting, and mining at the Dawson Gold Mine. Zephyr's compliance team works directly with local, state, and federal oversight agencies to ensure that exploration, reclamation, and future mine permitting and development will be timely and a benefit to the surrounding community.

20.1 Exploration Permitting and Approvals

The Dawson Project site is in a historic hard rock mining area of Fremont County, Colorado. Exploration in the 1980's was approved by the United States Department of the Interior, Bureau of Land Management (BLM) on June 18, 1987 as a Notice of Intent to Conduct Prospecting. The prospecting plan was amended July 28 and October 15, 1987 to increase the impacted area and scope of exploration. On December 22, 1987 Fremont County approved a CUP (CUP 87-19) allowing precious metals surface and/or underground mine activity on property zoned Agricultural Forestry. The Mined Land Reclamation Division (currently the Division of Reclamation, Mining and Safety) received a Noitce to Intent (NOI) to Conduct Prospecting Operations on January 23, 1990. These approvals set the precedence that the project area was developing and suitable for hard rock mining.

Consistent with the 1980's exploration activities, Zephyr performs exploration with permits from the Colorado Division of Reclamation, Mining and Safety (DRMS) and the Fremont County Department of Planning and Zoning (FC). Exploration and permit modifications require notice to the BLM, the adjoining property owner.

20.1.1 DRMS Exploration Compliance

DRMS approved the NOI (P-2013-002) on February 8, 2013. Subsequent exploration notices were submitted and approved as Modification-1 (January 26, 2016) and Modification-2 (April 27, 2016). NOI notices include the proposed number of exploration holes and number and location of exploration drill pads and access roads. Modification-3 was received by DRMS November 21, 2016, proposing access roads that will allow further resource delineation and preliminary site investigation for mine development.

20.1.2 FC Exploration Compliance

FC Board of County Commissioners approved CUP 12-003 on February 26, 2013, allowing surface excavation and drilling exploration for precious metals within 593-acre area. FC granted an extension of the CUP on January 12, 2016, allowing continued exploration until February 26, 2019.

20.2 Reclamation Bond

DRMS evaluates the posted reclamation bond with each NOI modification. Zephyr posted a reclamation bond of $57,514 on January 27, 2016. This bond allows for up to 20 exploration holes and approximately 1,700 linear feet of access road. The proposed bond increase for Modification-3 is approximately $2,566 for an additional 9,700 linear feet of access road.

20.3 Environmental Compliance and Monitoring

Exploration does not require periodic or ongoing monitoring and reporting. Drill hole abandonment reports are submitted within 60 days of completion of exploration.

DRMS requires a 15-month baseline groundwater and surface water quality study as a component of the mine permit application. Two monitoring wells were constructed in October 2014. The wells and identified surface water convergent points were monitored monthly and sampled quarterly from October 2014 through December 2015.

20.4 Future Permitting

Zephyr will require approval from local, state, and federal agencies prior to commencement of mine development. The primary mining permits with a brief summary of agency oversight are listed in Table 20-1. State agency application submissions will begin in 2017.

Unlike state permit application reviews and decisions with state statute specified review durations, the federal government queues applications upon receipt. The BLM access road application was submitted understanding the review and decision date is undetermined.

Table 20-1: Agency Jurisdiction and Permit Summary

Agency	Permit	Permit Description
Fremont County Department of Planning and Zoning	Conditional Use Permit	The county government oversees that land use promotes the health, safety, and welfare of the community. FC considers CUPs for land uses that may be compatible with surrounding land uses and are not in conflict with the objectives of the zoning resolution.
Colorado Department of Natural Resources Division of Reclamation, Mining and Safety	Hard Rock/Metal Mining Regular (112) Operation Reclamation Permit	The reclamation permit establishes approved mining and reclamation activities that result in a stable post-mining land use that is harmonious with the surrounding land uses. DRMS holds a reclamation bond to ensure sufficient funds are available to complete reclamation in accordance with the approved plan.
Colorado Department of Natural Resources Colorado Division of Water Resources	Water Augmentation Plan (Substitute Water Supply Plan during augmentation plan review and approval period)	Tributary water rights are regulated by the State Engineer's Office in CO Division of Water Resources. The CO water law is protective of senior water rights including disruption of subsurface tributary waters. A water augmentation plan allows new water uses while protecting senior water rights.

Agency	Permit	Permit Description
Colorado Department of Public Health and Environment Air Pollution Control Division	Construction Permit and Fugitive Dust Plan	Air quality is regulated during site development, mining, and reclamation. Operators submit a fugitive dust plan that describes dust mitigation measures that will be implemented during all aspects of mine operations. Such as road construction, active mine areas, stockpiles, tailings piles and site development activities.
Colorado Department of Public Health and Environment Water Quality Control Division	Storm Water and Process Water General Permit including a Storm Water Management Plan	Water quality discharges both storm water and process water including runoff from tailings piles, waste rock piles, dewatering discharge, and process water must meet water quality standards prior to discharge that will not deteriorate the receiving waters.
US Bureau of Land Management	Access Permit for Transportation Systems on Federal Lands	Zephyr submitted BLM Form 299 on December 5, 2016, requesting access across 419 linear feet of public land in order to construct a site entrance from the county road that will be safe for mine employees, trucks and the public.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

This section presents items associated with the project capital costs. All figures are in US dollars unless otherwise stated. The estimate was prepared to a level of accuracy consistent with PEA standards.

A total of $33.2 million in capital expenditure (CAPEX) is estimated over the life of the project as summarized in Table 21-1.

Table 21-1 Capital Cost Estimate over Life of Project

Item	Total Cost ($M)
Surface Infrastructure	2.5
Underground Mining	9.4
Electrical	1.4
Mill and Tailings Management	10.8
Miscellaneous	9.0
Total	33.2

Note: Due to rounding, the sum of the items may differ from the total shown

Sustaining capital which includes underground mobile equipment replacements and ongoing underground development, was included in the operating costs.

21.2 Basis of the Capital Cost Estimate

Currency

The estimates are expressed in 2017 US dollars. No escalation or inflation factors were applied in forward estimates.

Contributors

The major contributors to the various sections are outlined in Table 21-2.

Table 21-2: Contributors to CAPEX/OPEX

Item	Contributor
Owner's Costs	Zephyr
Tailings Storage Facility	Amec Foster Wheeler/Zephyr
Access	Zephyr
Transmission Line	Golder
Process Plant	BOMENCO
Sustaining Capital	MineTech
Mine Closure and Reclamation	Zephyr/Environmental Alternatives
Environmental Studies/Permitting/Social Licence	Environmental Alternatives Inc.
Mine Development and Capital Installation	MineTech
Mine Equipment	MineTech
Mine Ventilation	MineTech
Contingency	Zephyr/Amec Foster Wheeler/Golder/BOMENCO/MineTech

Exclusions from the Capital Cost Estimate

The following items were excluded from the capital cost estimate:

- project finance arrangements and costs and amortization costs
- escalation due to currency inflation
- licence and permitting fees
- sunk costs
- unforeseen costs due to delays

21.2.1 Underground Mining Capital Cost Estimate

The total capital cost for the underground, excluding working capital, is **$15.6 million** (Tables 21-3 and 21-4). Working capital is excluded from Table 21-3. Working capital is usually three months' operating costs. In the case of Dawson, working capital would be approximately $2 million ($80/tn, 105,000 tn/yr).

A contractor would excavate the decline and stubs for levels for approximately $5 million ($1,800/ft).

Table 21-3: Summary of Capital Costs (Pre-production)

Item	Cost ($M)
Mining Equipment	6.17
Underground Development	4.23
Total* (Rounded)	**10.40**

*Excludes working capital.

Table 21-4: Details of Underground Capital Requirements

Dawson Project, Estimate of Underground Capital Requirements, May 2015, US$, Major Mining Equipment								
	Number of Item(s)	Motor/type	kW	Unit Delivery	Year 1	YEAR 2	YEAR 3	Total item cost (rounded)
Drilling								
Jumbo Drill, micro	2	d/e	40	$25,000	$585,000	$585,000		$1,170,000
Bar and Arm Drill	1			$5,000	$65,000			$65,000
Bazooka Drill	1			$1,000	$13,900			$13,900
Bit Sharpener	1			$2,000	$21,500			$21,500
Stopers	10			$750	$75,600			$75,600
Compressor	1	e	100	$10,000	$130,000			$130,000
Booster Compressor	1	e	60	$5,000	$50,000			$50,000
Blasting								
ANFO Loader	1	a		$1,000	$11,000			$11,000
Mucking								
3.5 yd LHD	1			$30,000	$324,000			$324,000
2 yd, remote control	1			$35,000	$275,000			$275,000
Scoop Tires	24			$200	$40,800			$40,800
Bobcat & spare parts	1			$2,000	$17,000			$17,000
Haulage - UG								
20 ton truck	2			$10,000	$1,018,000		$514,000	$1,532,000
Truck Tires	12			$200	$16,400			$16,400
Vehicles								
Service Vehicle	1			$10,000	$90,000			$90,000
Superintendent Vehicle	1			$10,000	$70,000			$70,000
Geology/Survey vehicle	1			$10,000	$70,000			$70,000
Personnel Transport	1			$30,000	$245,000			$245,000
Ventilation								
Main Fan and Motor	1			$10,000	$85,000			$85,000
Spare main fan plus motor	1			$10,000	$70,000			$70,000
Auxiliar fan	5			$2,000	$84,500			$84,500
Vent monitoring equipment	1			$1,000	$8,500			$8,500
Drainage								
High Head 150 kW pump	1			$8,000	$63,000			$63,000
Small pumps 15 kW	5			$2,000	$62,000			$62,000
Anciliary Equipment								
ger hoist, air powered (Stope use)	6			$1,500	$69,000			$69,000
cap lamps	50			$10	$2,450			$2,450
transformers, 500 kva	3			$5,000	$104,900			$104,900
First Aid Station, fully supplied	3			$200	$3,530			$3,530
Refuge Station	1			$4,000	$34,000			$34,000
Miscellaneous Equipment								
Scissor Lift	1			$15,000	$315,000			$315,000
Bolter for Scissor Lift	1			$10,000	$95,000			$95,000
Boom Truck	1			$10,000	$300,000			$300,000
Ground Support								
Shotcrete, 30 m^3/hr capacity, diesel, trailer mounted	1			$10,000	$79,750			$79,750
Grout Pump for cable bolting, air powered	1			$3,000	$21,750			$21,750
Initial inventory of cable bolts, rock bolts, screen, ground	1				$50,000			$50,000
Electrical Substation	1			$4,000	$204,000			$204,000
Battery Chargers	1			$500	$30,500			$30,500
		Yearly capital purchases			$4,801,000	$585,000	$514,000	
		Miscellaneous	15%		$720,000	$88,000	$77,000	
		Contingency	10%		$552,000	$67,000	$59,000	
		Yearly Totals				$740,000	$650,000	
					$6,073,000	initial Capital		

Sustaining Capital

It is estimated that the sustaining capital will be $740,000 in Year 2 and $650,000 in Year 3. Thereafter, the sustaining capital would be between 10% and 20% of the initial capital (15% was assumed).

Operating Costs

Operating costs were estimated at approximately $80/tn of mill feed. This cost includes overhead but not capital.

Refer to Table 21-5 for a summary of selected operating costs.

To determine the operating cost per ton of mill feed, costs were estimated on a monthly basis and summed over the life of the operation, then divided by the total mill feed.

The total underground personnel cost is approximately $4.5 million per year or approximately $20 for every ton broken. A two 10-hour shift option gives 7,300 operating hours per year for an average mine labour cost of $600 per operating hour. In terms of 100,000 tn/yr production, the labour component will be $45/tn (between $40/tn and $55/tn) delivered to the plant stockpile.

The costs per decline round were examined, using 12 ft drill steel.

Table 21-5: Summary of Selected Operating Costs

Item	Operating Cost ($)
Stoping (Incl. diamond drilling, labour, development)	80 per ton
Yearly Labour Cost at Full Production	4.5 million
Single Boom Drill Jumbo (Drillmaster 100)	17.70 per Broken Foot
Longhole Drill (Drillmaster 100 Longhole)	65–66 per drilling hour
LHD (Muckmaster 300EB)	57–58 per hour
Haul Truck (Haulmaster 800-20EB)	31–32 per hour

21.3 Mill Capital Costs

The total estimated direct capital cost for the process plant and its ancillary facilities is US$ 7.681 million. A breakdown of the summary of the capital cost estimate is shown in Table 21-6 below.

Table 21-6: Process Plant Direct capital Cost summary

Major Area	Cost, US$ M
Crushing	1.546
Process plant	5.131
Ttailings Dewatering(Dry Stack)	0.533
Air, Water, & Process Control Systems	0.286
Assay & Metallurgical Laboratory	0.186
Total:	7.681

Duties and Taxes are excluded from the estimate. Indirect costs, owner's costs and contingency are covered elsewhere in this section.

This estimate was prepared using preliminary budget quotations provided by vendors for major equipment. All other mechanical equipment and prices for construction and miscellaneous materials were based on in-house data on similar projects and reference databases. Blended labour rates used in this estimate are based on Nevada State Occupational Wage Rate Estimates. The cost of process piping, major electrical equipment, instrumentation and control systems and all required electrical installation costs were estimated as a percentage of the cost of the process equipment, based on similar facilities.

21.4 Mill Operating Costs

A summary of plant operating cost is provided in Table 21-7 below.

Table 21-7: Process Plant Operating Cost Summary

Description	Unit	Annual Cost ($)	Unit Cost, $/tn milled
Labour	**Personnel**		
Mill Staff	6	506,000	4.19
Mill Operations Labour	14	867,360	7.19
Maintenance Supervisory Staff	2	154,000	1.28
Maintenance Labour	6	346,000	2.87
Sub-total	**28**	**1,873,360**	**15.52**
Utility	**MWh/a**		
Power	8,820	881,967	7.31
Sub-total		**881,967**	**7.31**
Consumables			
Operating supplies		562,527	4.66
Maintenance Supplies		104,495	0.87
Sub-total		**667,022**	**5.53**
Total Process Operating Cost		**3,422,349**	**28.35**

Mill operating costs are composed of main categories of labour, utility power, and consumables including, operating and maintenance supplies.

Mill operating and maintenance labour and staffing levels, which include mill operations, maintenance, and metallurgical and assay labs, were developed using in-house experience and operations of similar size and type of mills. The mill will operate 2-to 12-hour shifts per day 7 days per week.

Annual bare salaries for the hourly personnel are based on 40 hours per week, 50 weeks per year plus 10% overtime. Loaded salaries are based on 28% overhead of annual salaries.

Mill power costs were developed from first principles using equipment motor power ratings, usage levels, and loads from vendor and in-house equipment data. Power unit cost, at $0.10/kWh, was based on similar operations in the region and as provided by the client.

Reagent consumption rates, as part of the operating supplies, were developed from metallurgical tests while prices were based on recent projects. Other mill consumables, such as grinding media and liner wear rates and prices were developed using in-house data and experience. Wear parts such as crusher plates and conveyor and screening panels were estimated using equipment capital costs and in-house data. Maintenance supply costs were estimated as a percentage of equipment capital costs as is the industry standard for this level of studies.

Mill operating supply costs include costs of supplies for the assay and metallurgical labs and miscellaneous operations expenses such as PPE, travel, light vehicles, and freight.

21.5 General and Administrative Costs

Dawson Project general and administrative costs are summarized in Table 21-8 below.

Table 21-8: General and Administrative Cost Summary

GENERAL & ADMINISTRATIVE COSTS	Personnel	Annual Cost, $	Unit Cost, $/tn milled
LABOUR			
General/Mine Manager	1	141,000	1.17
Warehouse Purchaser,Shipper, Receiver	1	51,000	0.42
HR, Payroll	1	64,000	0.53
H&S Supervisor, Nurse/EMT	1	51,000	0.42
Environmental , Community Liasion	1	51,000	0.42
SUB-TOTAL G&A LABOUR:	**5**	**358,000**	**2.97**
EXPENSES			
Safety Training & Mine Rescue		10,000	0.08
Safety Supplies		3,000	0.02
Personal Protective Equipment		30,000	0.25
Safety Awards		1,200	0.01
Mine Rescue, Emergency Response Training & Firefighting Materials		6,000	0.05
Medical Service/First Aid		5,800	0.05
Communications; Tel, Fax, Internet		24,000	0.20
Office Supplies (Computers, Furnishings, Couriers & Postage included)		4,000	0.20
Software Licensing		15,000	0.12
HR Recruitment and Expenses		6,000	0.05
Medical and Drug Testing		3,600	0.03
Employee Courses & Training		6,000	0.05

GENERAL & ADMINISTRATIVE COSTS	Personnel	Annual Cost, $	Unit Cost, $/tn milled
Insurances		72,000	0.60
Legal Services		24,000	0.20
Permitting Regulatory Compliance		18,000	0.15
Mineral Claims & Leases		7,500	0.06
Brokerage & Duty		1,200	0.01
Property Tax		48,000	0.40
Head Office Allowance/Accounting		100,000	0.83
Public Relations & Donations		20,000	0.17
Crew Transportation (From Parking to Site – 4 miles)		36,000	0.30
Process & Mining Consultations		24,000	0.20
Environmental Monitoring and Reporting		12,000	0.10
Building Maintenance		6,000	0.05
SUB-TOTAL G&A EXPENSES:		**503,300**	**4.17**
TOTAL G&A COSTS		**861,300**	**7.14**

Administrative staff levels are based on one 8-hour shift per day, five days per week. Similar to mill labour costs, annual bare salaries for the hourly personnel are based on 40 hour per week, 50 weeks per year plus 10% overtime. Loaded salaries are based on 28% overhead of annual salaries.G&A expense estimates are based on in house experience on similar size projects.

21.5.1 Infrastructure Capital Costs

Required utilities cost, including an electrical transmission line and substation, was estimated at $1.15 million.

All site storage and miscellaneous buildings are assumed included in the mill capital cost estimate. No cost estimate for the tailings disposal facility was used in the cashflow analysis.

21.5.2 Miscellaneous Capital Costs

Included in cashflow analysis were expected miscellaneous capital costs, which include:

- pre-feasibility and feasibility studies
- hydrogeological investigation
- geotechnical investigation
- environmental assessment
- permitting

- community licensing
- contingency on miscellaneous items

The total estimate for miscellaneous costs was $9.0million.

22.0 ECONOMIC ANALYSIS

This section presents items associated with the economic evaluation of the project. All figures are in US dollar terms unless otherwise stated.

The economic analysis was completed using best available information at the time. Due to risks and uncertainty related to global economic factors, government regulations, environmental considerations, and other inherent risks associated to the project, actual results may differe materially from those reflected in the analysis.

22.1 Summary

The economic analysis was compiled using Microsoft Excel. The cash flow was modelled by yearly periods. The nominal discount rate of 8% was used for this analysis.

A long term gold price of $1,250/oz was used. Table 22-1 summarizes the base case assumptions used in the cash flow model. Table 22-2 presents the payback periods.

Table 22-1: Base Case Assumptions

Item	Unit	Value
Gold Price	$/oz	1,250
Gold Gravity Recovery from Run of Mine	%	50
Gold Float Recovery from Run of Mine	%	42
Total Gold Recovery	%	92

Table 22-2: Payback Periods

Payback Period	Years
Discounted Payback Period	2.7
Nominal Payback Period	2.4

22.2 Revenues

22.2.1 Production Schedule

The revenues were derived from the production schedule developed by MineTech. The schedule provided tons of mill feed and grades.

22.2.2 Mill Recovery

The mill recovery from both the gravity and float circuits were provided by BOMENCO (Table 22-1).

22.2.3 Treatment and Refining Costs, Payable Gold Assumptions, and Freight

The treatment and refining costs, payable gold and freight costs are summarized in Table 22-3. These figures were provided byBOMENCO. For the base case, it is assumed the float concentrate would be trucked to a port in Los Angeles and shipped overseas. The shipping costs past the port are included in the treatment charge.

Table 22-3: Treatment and Refining Costs

Treatment Costs		
Gravity		
Shipping	$/oz	1
Refining	$/oz	5
Payable	%	99.5
Flotation		
Shipping	$/tn concentrate	150
Gold deduction	Oz/tn concentrate	0.029
Payable	%	95
Treatment Charge	$/tn concentrate	200
Refining Charge	$/oz	35

22.3 Operating Costs

Operating costs were supplied by Minetech and BOMENCO.

Table 22-4 summarizes the average life of mine operating costs for the project.

Table 22-4: Average Life of Mine Operating Costs

Item	Cost ($/tn) processed	Cost ($/oz)
Underground mining*	91	339
Processing and Tailings	31	116
G&A	7	27
Smelting and Refining	22	82
LOM Total Cash Cost**	151	563

* Underground mining cost includes development costs

** Cash cost includes mining costs, mine-level G&A, mill and refining cost, numbers may not add due to rounding

22.4 Capital Costs

The capital cost estimates imported into the model were provided by BOMENCO and MineTech. Details of the capital costs can be found in Section 21. The modelled capital costs are shown in Table 22-5.

Table 22-5: Total Capital Costs

Item	Cost ($M)
Surface Infrastructure	2.5
Underground mining	9.4
Electrical	1.5
Mill and Tailings	10.8
Miscellaneous	9.0
Total Capital Cost	33.2

Numbers may not add due to rounding. Total includes contingency.

22.5 Tax and Depreciation

Taxes were modelled based on information received by Environmental Alternatives, who discussed with the local county officials. The major taxes included in the model are:

- Federal tax, which is 35% on the net smelter return
- Colorado state income tax, which is 4.6% on the net smelter return
- Severance tax, which is 2.3% of the gross income over $19 million
- Personal property tax, which is 29% on all equipment over $7,500 multiplied by the mill levy (7%)
- County tax (minimum of the two methods)
  - Method A tax, which is 25% of the gross proceeds multiplied by 29% multiplied by the mill levy
  - Method B tax, which is 100% of the net proceeds multiplied by 20% multiplied by the mill levy

Depreciation was calculated using the modified accelerated cost recovery system (MACRS) seven-year basis. The MACRS allows for the capital costs to be deducted annually as depreciable assets. The MACRS seven-year basis is:

- 14.29%
- 24.49%
- 17.49%
- 12.49%
- 8.93%
- 8.92%

- 8.93%
- 4.46%

22.6 Royalties

A royalty of 3% on 50% of total revenues was applied to the cash flow to account for the agreement Zephyr has with The Allen Group. Because Zephyr has been paying royalties on the property to the amount of $429,000 to date, this amount has been deducted from the first year of royalties owed.

22.7 Project Valuation

On the basis of the revenue and costs outlined above, the project is estimated to have a pre-tax NPV (at 5% discount rate) of $35.5 million and a pre-tax internal rate of return of 66%**.** The project has a 2.4 year payback of initial capital before tax.

On the basis of the revenue and costs outlined above, the project is estimated to have a post-tax NPV (at 5% discount rate) of $22.1 million and a post-tax internal rate of return of 46%**.** The project has a 2.7 year payback of initial capital after tax.

The model and project valuation was compiled using the best available information at the time. Due to risks and uncertainty related to global factors, government regulations, environmental considerations and other inherent risks associated with mining projects, actual results may differ from those reflected in the model.

22.7.1 Sensitivity Analysis

A sensitivity analysis was done on the project to determine which of the key inputs into the cash flow have the greatest effect on the project. Figure 22.1 shows that the OPEX, the CAPEX, the gold price, the discount rate, and the treatment costs were varied by ±40% for the sensitivity analysis. The figure shows the gold price has the largest effect on the project cash flow and that the other inputs have less of an effect. For instance, varying the operating costs by -40% increases the cash flow by $21 million, while increasing the operating cost +40% decreases the cash flow by $21 million, from the base case of $22.1 million.

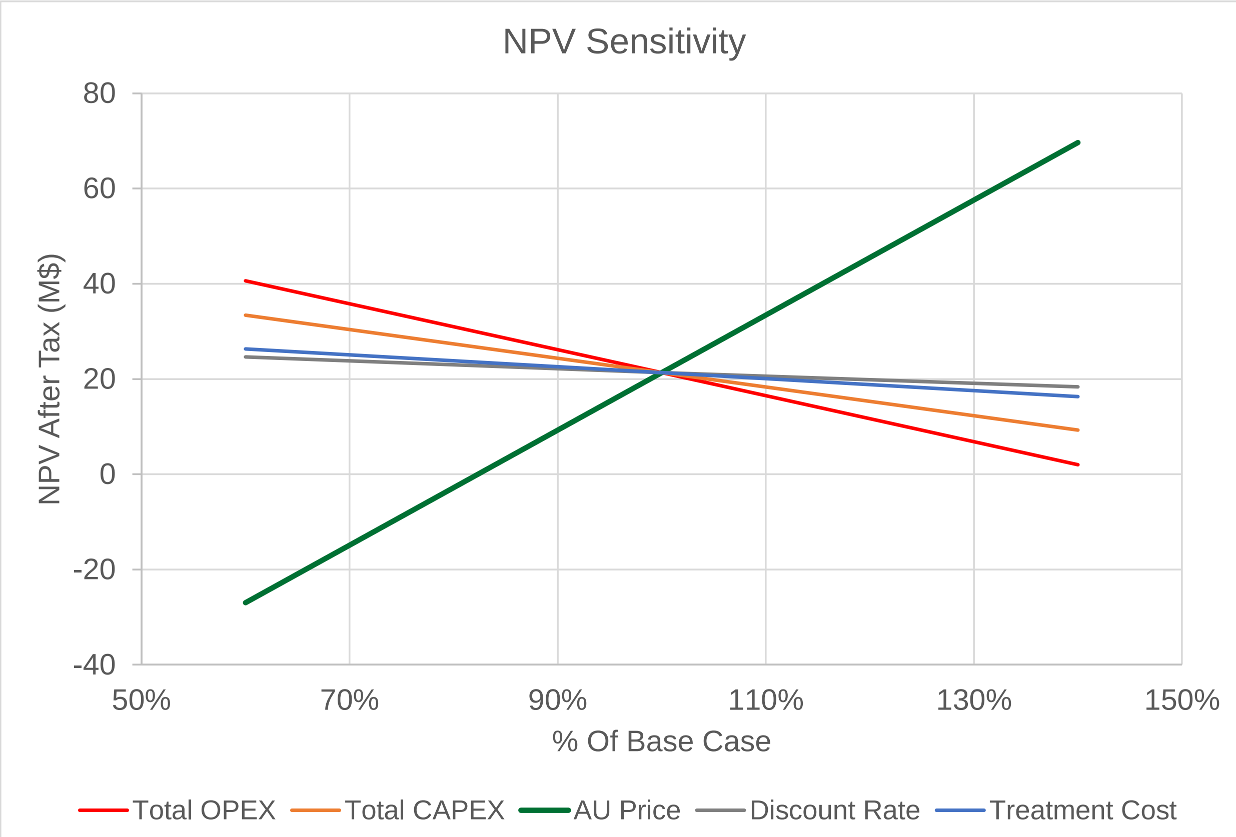


Figure 22.1: Sensitivity Analysis on Net Present Value (after tax)

The before tax NPV of the project is $35.5 million. Table 22.7 and 22.8 show the before and after tax at various discount rates. For this report the 5% discount rate was assumed.

Table 22-6: Before Tax NPV at Various Discount Rates

DISCOUNT RATE	PRE-TAX NPV US$ 000s
0%	43,023
5%	**35,519**
8%	31,679
10%	29,351

Table 22-7: After Tax NPV at Various Discount Rates

DISCOUNT RATE	POST-TAX NPV US$ 000s
0%	28,462
5%	**22,096**
8%	19,181
10%	17,411

23.0 ADJACENT PROPERTIES

There is no competitor activity near Zephyr's Dawson Project.

Almost 20 mi to the north of Cañon City, gold mining began in the Cripple Creek area in the 1890s. Underground operations at the time were following high-grade veins. The geology comprises Oligocene-aged intermediate volcanic breccia and volcanoclastic rocks. The gold occurs as micrometre size free gold and as gold-silver tellurides.

Newmont Mining Corporation currently operates two open pits extracting 350,000 to 400,000 ounces of gold per year. The gold is recovered from the ore by heap leaching. The heap leach pad is one of the largest in the world. Newmont employs 870 employees and contractors at the Cripple Creek and Victor Mines.

Over 23 million ounces of gold have been recovered from the Cripple Creek district since the 1890s.

24.0 OTHER RELEVANT DATA AND INFORMATION

The public has the conditional right to cross unpatented mining claims for recreational or other purposes and to access other Federal lands beyond the claim boundaries. This has not impacted exploration activities on the Dawson Project to date and there is no information now to indicate this would affect future exploration activities. Zephyr is of the opinion, confirmed by advice from its US legal counsel, that it has the legal right to access the Property via the historical road that has been in use since the early 1900s. This access road is gated near its junction with the Temple Canyon Rd. located to the north of the Property, and is currently maintained by Zephyr. A portion of the road passes through a parcel of land that has recently changed hands, and Zephyr is currently in the process of confirming its right of way on this section of road with the new owner.

In 2016, the non-earth grid used at the Dawson Project was tied into a global coordinate system. All holes from the 2016 drill program were surveyed using Colorado State Plane NAD 83. Two historical drill holes (DCK-9003 and GC-59) were also surveyed to assist in a property wide conversion of historical drill hole coordinates. Surveyors from the BLM have recently surveyed the corner-post locations of the patented claims as well as the northern border of the San Isabel National Forest that is located just south of the Dawson Project. BLM was contacted to obtain the coordinates of the patented claim locations. Several these corner-posts were relocated in 2016 and surveyed for confirmation. This will eventually assist the tie in of unpatented claim locations

Historically, the presence of magnetite in the rocks precluded the consistent use of downhole orientation survey instruments which depended on a compass system. Only a few downhole surveys were taken in the later programs: before 1986 an Eastman Whipstock downhole camera (which contains a magnetic compass) was used; in 1986 during a reverse-circulation drill program, Sperry-Sun equipment was used. Most of the drill hole traces were plotted as straight lines on historical plans and sections. Based on the drill logs, faulted and broken ground was common and could contribute to drill hole deviation. Zephyr used a multi-shot downhole surveying tool for both the 2013 and 2016 drill programs. Despite minor magnetite occurring in the granitic rocks of the Windy Gulch Segment most survey readings were as would be expected for the drill hole orientation. Any anomalous readings were surveyed a second time to obtain an acceptable azimuth.

To accommodate for rugged terrain, historical drill holes were commonly drilled as fans covering a range of azimuths. This results in variable drill hole density on regularly spaced drill sections and can make sectional interpretations more difficult.

Core recovery can be a problem in the hydrothermalitic rocks at Windy Gulch. To that end, Zephyr will be utilizing HQ drill equipment with expectation that larger core diameter will improve recoveries. Also, it recommended that recirculating return drill water and fluids down the drill holes not be practiced given the high gold grade and the resulting potential to contaminate the core.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Windy Gulch Resource

This mineral resource estimate represents an open pit mining scenario for the Windy Gulch segment of the Dawson Project. It has been prepared in accordance with CIM best practice guidelines and definitions as referred to in NI 43-101 regulations.

Due to a lower cut-off grade used to outline mineralization and the relatively large SMU size for open pit blocks relative to underground mining, a degree of grade smoothing will have been introduced into the model and resource estimates. When dealing with lower grades and larger block sizes, it is common that grades will be smoothed (averaged) due to the differences between high and low grade samples and greater differences between sample volumes and block volumes (volume variance). As a result, the resource tonnage may appear higher and the grade lower than reported from historical polygonal estimates, but the total ounces contained within the deposit are reasonably similar. Differences in contained metal may occur at higher cut-offs due to the smoothing effect, so the reader is cautioned that this resource estimate and block model is intended to evaluate an open pit mining scenario and may not accurately reflect conditions attainable from underground mining (i.e. lower tonnage at a higher grade).

Golder has outlined a mineral resource estimate consisting of 78,000 tons of Indicated Resource at a grade of 0.12 oz/tn Au reported at a 0.035 oz/tn cut-off, with an additional 20,000 tons of Inferred Resource with a grade of 0.16 oz/tn Au reported at a 0.093 oz/tn cut-off.

Mr. Brian Thomas, P.Geo., is the QP of the Windy Gulch resource, and has visited the site, collected samples for assay grade verification, and reviewed the geological data, including reports, maps, technical papers, digital data including lab results, sample analyses, and other miscellaneous information. The QP believes that the data presented are an accurate and reasonable representation of Windy Gulch and concludes that the database is of suitable quality to provide the basis of the conclusions and recommendations reached in this report.

The Dawson Project is considered to be a project of merit and has the potential for mining and increased resources through additional exploration.

25.1.1 Risks

Many drill holes within the outlined resource area are historical and may have data quality risks related to a lack of established QA/QC procedures along with some accuracy concerns regarding the collar locations. These issues may have the potential to affect the accuracy of the modelled volumes and the estimated grades in the block model.

The relatively low number of bulk density measurements and the use of average density values in some zones could affect the accuracy of the resource tonnage and contained metal. Density values are based on pycnometer volume measurements of pulp samples from drill core and may not accurately reflect the porosity or fracture spacing present in the rock mass, which could result in an over-estimation of resource tonnage.

Core recovery from diamond drilling, in some instances can be relatively poor, which could result in non-representative sample data.

Golder accounted for the above risks by being conservative with projected contacts and by assigning appropriate resource classifications to each domain. The resource classification provides a reasonable evaluation of the risks associated with the mineral resource estimates.

25.1.2 Opportunities

Based on the information collected to date, there is an opportunity to increase the size and confidence (resource classification) of the resource with future infill and exploration drilling. Inferred mineral resources in Zone 1 have a reasonable probability of being upgraded to an indicated mineral resource with the completion of adequate infill drilling. Indicated resources may be upgraded to measured resource with adequate infill drilling with 30 ft centres or less, and the completion of a twinning program to confirm the accuracy and quality of the historical drill hole data.

Zone 1 has potential to be extended down dip with additional step-out drilling. There is currently very little drill hole information in the down-dip direction beyond what has been captured in the model.

Windy Gulch also has reasonable Greenfield and Brownfield exploration potential due to the continuous nature of the host lithology, which has been traced over to the Windy Point Segment and is likely continuous with the Dawson mineralization as well.

25.2 Dawson Resource

This mineral resource estimate presented in this technical report for the Dawson Segment deposit was prepared by Mercator in accordance with NI 43-101 and the CIM standards on behalf of Zephyr in 2013. Since Zephyr management has confirmed that no new work material to the 2013 resource estimate has been completed on the Dawson Property since 2013, Mercator has deemed the 2013 estimate to be current for the purposes of NI 43-101 and this report.

The Dawson Segment deposit is currently considered by Zephyr to represent an intrusion related orogenic gold deposit hosted by Early Proterozoic faulted, siliceous, and peraluminous felsic gneisses with lesser semi-massive sulphide/sulphide-rich zones. This geological setting was interpreted by Zephyr at the time of the 2013 Mercator resource estimation program as a strongly deformed and metamorphosed volcanic exhalative system. Gold occurs in multiple mineralized horizons, is most commonly associated with sillimanite, sericite or biotite, and is sited in fractures in quartz and garnet grains and at quartz grain boundaries. The main mineralized horizon has been structurally modified to form pinch-and-swell features, thickened zones, and offsets, as well as relatively planar zones of mineralization that locally show southwest plunging grade shoots. Gold mineralization zones are relatively discrete and range in true thickness from approximately 2 ft (0.61 m) to as wide as 50 ft (15 m) (Alers, 2003). Gold is locally nuggety and is commonly accompanied by 1% to 5% disseminated pyrite +/- chalcopyrite (Theye, 1989). While gold is the only metal considered to have economic significance at this time, copper also occurs in association with the hosting sulphide-bearing unit and could be of future economic interest. Potential for development through underground mining using conventional narrow vein methods has been assumed for resource assessment of the deposit.

25.2.1 Risk

Continuity of gold mineralization as defined by current core drilling results is generally good, but the deposit has been locally disrupted by faulting and pegmatitic intrusions. In the Windy Gulch area, potential exists for interpretation of tight fold closures but this is not represented in the 2013 resource model. Rugged terrain has historically limited surface drilling access points in some areas and both downhole survey control and core recoveries for some historical diamond drill holes are incomplete.

25.2.2 Opportunities

Based upon results of the 2013 resource estimation program and associated geological interpretation, Mercator believes good potential exists for delineation of down-plunge (southwest) and up-plunge (northeast) extensions to the currently defined Dawson Segment deposit at the 0.15 oz/tn (5.0 g/t) Au cut-off grade level. Both areas warrant further evaluation through additional core drilling. Additionally, infill core drilling at closer spacing within the current resource limits will be required to upgrade Inferred resources to indicated or measured status.

25.3 Recovery Methods

The results of gravity and flotation tests conducted on samples from both Dawson and Windy Gulch segments and summarized in Section 13 above offer a technically viable process option for the recovery of precious metals at commercially respectable grades and recoveries of concentrates that may be marketed to smelters for further processing and refining. And based on these test results, a process flowsheet encompassing comminution followed by gravity and flotation processes was developed for the recovery of gold and silver from the Dawson ores. The plant flowsheet and design further include concentrate and tailings dewatering and all necessary ancillary facilities to enable the operation of a 330 ton per day plant.

25.4 Filtered Tailings Storage Facility

A PEA level design was prepared for the Dawson FTSF. Uncertainties and risks associated with the FTSF include the following:

- Although a liner system was not included in the FTSF design, it is possible that regulatory agencies will request or require a liner system. For the next study phase, it is anticipated that further evaluation of the geotechnical and geochemical characteristics of the tailings, as well as the subsurface geology and hydrogeology will be needed to justify the need for, or elimination of, a liner system for the FTSF.
- The water quality of seepage from the FTSF and other contact water collected in the contact water pond is uncertain based on a lack of geochemical characterization of tailings from the Dawson segment. During operations, contact water is anticipated to be recycled to the process plant. However, if contact water needs to be released during or following operations, some type of water treatment may be needed to meet water quality standards. Costs for a water treatment plant were not included in the estimated cost due to uncertainty of the water quality.

25.5 Preliminary Economic Assessment

A PEA was done on the Dawson Project encompassing a restricted portion of the Property, specifically two discrete areas known as the Dawson Segment and the Windy Gulch Segment. The PEA showed the following key highlights at a gold selling price of US$ 1,250/oz:

- pre-tax IRR and $NPV_{5\%}$ of 66% and $35.5 million and a 2.4 year payback of initial capital
- after-tax IRR and $NPV_{5\%}$ of 46% and $22.1 million and a payback of 2.7 years

The PEA included a five-year underground mine project encompassing the Dawson Segment of the Dawson Project. The Dawson Segment provides 450,000 tons of mill feed (which includes inferred material) at an average diluted grade of 0.27 oz/tn gold. This accounts to approximately 121,000 oz of gold in the mill feed prior to the estimated mill recovery of 92% by means of gravity and float recovery. The Windy Gulch Segment resource, including inferred resource, was evaluated assuming an open pit mining operation; however, due to the limited amount of material, the relatively high strip ratio, and the limited economic benefit the open pit provided to the overall PEA, Zephyr decided to omit the Windy Gulch mill feed from the PEA at this stage. Additional drilling and expansion of the mining permit application to include the relevant unpatented claims as well as the patented claims could increase the mining potential of the Windy Gulch resource.

The PEA was compiled using the best available information at the time. Due to risks and uncertainty related to global factors, government regulations, environmental considerations and other inherent risks associated with mining projects, actual results may differ from those reflected in the model.

The favourable results of the PEA of the base case scenario indicates the potential of the proposed project and supports the decision to continue to advance with further feasibility work.

26.0 RECOMMENDATIONS

26.1 Windy Gulch Resource

The Windy Gulch deposit requires further geological investigation to 1) expand the deposit foot print and 2) increase confidence in the current resource estimates. The following work program is recommended towards achieving these goals:

- Develop a new road cut between the existing upper and middle road cuts to expose mineralization for mapping and sampling as well as providing a better location for diamond drilling.
- Complete an HQ diamond drill program from the new road cut to infill gaps in existing drilling and test mineralization down dip as well as to determine if the larger core size reduces nugget variance.
- Complete a twinning drill hole program of a minimum of two to five historical holes to verify the quality and accuracy of the historical drill hole data.
- Re-evaluate the mineral resource on successful completion of the work program.

Estimated costs for the recommended work program are summarized in Table 26-1.

Table 26-1 Summary of Windy Gulch Recommendations and Cost Estimates

Recommendations	Estimated Cost ($)
New road cut	50,000
Diamond drilling	120,000
Twinning of historical holes	60,000
Updated resource estimate	35,000
Total Cost of Program	265,000

26.2 Dawson Segment Resource

No material work has been completed on the Dawson segment deposit since the 2013 resource estimate technical report prepared by Mercator. On this basis, and after review, Mercator and co-author Graves believe the following recommendations are applicable at the effective date of the current report. Items 1 through 4 directly reflect 2013 and more recent recommendations by Mercator and items 7 through 9 are provided by co-author Graves.

1) Additional core drilling should be carried out to evaluate the direct up-plunge and down-plunge extensions of the main gold grade trends that comprise the Dawson Segment deposit.

4) Infill core drilling should be carried out within the deposit limits to increase confidence in resource continuity and to thereby support future conversion of Inferred mineral resources to indicated or measured status. A new mineral resource estimate should be carried out after completion of this drilling and the drilling noted in item 1 above.

5) Zephyr should continue to review and update internal QA/QC protocols to include analysis of duplicate pulp and reject splits, core duplicate splits and drill core check sampling with analysis at a third party, independent, accredited laboratory.

6) The Dawson Segment resource estimate and associated resource files should be transformed to Colorado State Plane coordinate system (NAD83) to conform to the coordination system of the nearby Windy Gulch Segment deposit model.

7) Several holes should be drilled within the Dawson Segment deposit to compare screen metallic assaying to standard one-assay tonne gold analyses performed during the 1980s drill programs. The core could then also be utilized for geotechnical and metallurgical testing.

8) Archived drill core from the Dawson Segment and Copper King Segment should be re-logged to establish compatibility and context with the new intrusion related and shear zone hosted geological model.

9) Several holes should be drilled beneath the near-surface copper prospect at Copper King to explore for buried gold mineralization analogous to that occurring beneath the copper zone at the Dawson Segment deposit.

In the 2013 Dawson Segment technical report by Mercator, a two phase program totalling 1,500 m of diamond core drilling was recommended to address highest priority mineral resource extension and infill target opportunities, with this followed by preparation of a revised mineral resource estimate for the deposit prior to completion of a PEA. The core drilling and resource estimation aspects of the earlier recommendations remain valid at the current report date and have been augmented by three additional recommendations. A modified budget estimate of $700,000 is proposed to cover the updated recommendations presented above and in Table 26-2 below.

26.3 Recovery Methods

The following is recommended before the next phase of project development.

Representative samples of the mineable areas, including earlier months/years of mining as well as varying rock and mineral types, should be taken for a comprehensive metallurgical test program including gravity and cleaner flotation in locked cycle tests. The cost estimate for a comprehensive metallurgical test program described above is $50,000.

Grade-recovery relationship should be established to aid in determining economically and metallurgically optimum parameters. The cost to prepare the relationship is included in the above metallurgical test program.

Rod mill, ball mill, crushing, and abrasion work index tests should be conducted to aid in design, sizing and selection of major comminution equipment. The cost estimate to conduct the tests is $10,000.

Dewatering tests for thickening and filtration must be carried to optimize sizing of these unit operations. The cost estimate to conduct the dewatering tests is $15,000.

Marketing studies should be conducted to establish economically optimum smelting options for the concentrates. The cost estimate to conduct marketing studies for this project is $10,000.

Table 26-2: Summary of Dawson Segment Recommendations and Cost Estimates

Recommendations	Estimated Cost ($)
Diamond drilling (deposit infill and extension – 1500m)	$375,000
Core re-logging, sampling and mapping for Dawson and Copper King Segments	$50,000
Dawson Segment comparative drilling for screen metallics assessment (300m)	$75,000
Updated resource estimate and deposit modelling review	$60,000
Exploration drilling at Copper King Segment (300 m)	$75,000
Administration and reporting	$65,000
Total Cost of Program	$700,000

26.4 Filtered Tailings Storage Facility

The following recommendations are provided regarding the filtered tailings storage facility:

- The feasibility of tailings filtration to the optimum moisture content (approximately) has not been proven for the Windy Gulch or Dawson tailings. Consequently, the tailings filter plant sizing, capital cost, cycle times, operating costs, etc. are uncertain. Bench-scale pressure filtration testwork is recommended in the next study phase. The estimated cost for this test work is $15,000.
- Geochemical characterization of the Dawson and Windy Point tailings has not been undertaken. A synthetic sample of Windy Gulch tailings were shown to be "non-PAG" or "uncertain" potential for acid generation, depending on the evaluation method. Additional testwork will be needed to characterize the geochemistry of the tailings. The geochemical testing should include static and kinetic testing on each ore type from the Windy Gulch and Dawson segments. The cost for such geochemical characterization will depend on the number of ore types, but is estimated to be $30,000 to $50,000.
- The geotechnical investigations conducted at the filtered tailings storage facility as part of this study were limited to test pit excavations. Additional geotechnical investigation, consisting of 3 to 5 boreholes, is recommended for the next study phase to better characterize the subsurface conditions at the FTSF. The estimated cost for these geotechnical investigations is $50,000.

26.5 Preliminary Economic Assessment

Based on the findings of the PEA, additional exploration, definition drilling, and continuation of the completion of a mine permitting application is recommended. Estimated costs have been included in the sections 26.1 and 26.2.

Increased detail for the mine planning should be completed for the design, scheduling, and optimization of the underground operation. Identification of potential risks and opportunities should be addressed in the next level of planning to further improve the economic viability of the project. The estimated cost for optimizing and updating the designs is $200,000.

Geotechnical and hydrogeological investigations of the portal, underground development, and ventilation raises, as well as the waste rock storage area is also recommended to advance the project. The estimated cost for the geotechnical investigation for the underground mine development and waste rock storage area is $150,000. The estimated cost for a hydrogeotechnical investigation for the mine is approximately $50,000.

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Report Signature Page

This technical report on the Dawson Property, Colorado, USA is submitted to Zephyr Minerals Ltd.and is effective as of March 21, 2017.

Qualified Person	Responsible for Parts
Danny Tolmer, P.Eng., Golder Associates Ltd. March 21, 2017	Sections 1.9.3, 15, 16.2, 18.7, 18.8, 19, 20, 22, 25.5 and 26.5
Brian Thomas, P.Geo., Golder Associates Ltd. March 21, 2017	Sections 1.1, 1.6.2, 1.8.2, 1.9.2, 1.10.2, 2, 3, 12.1, 14.2, 25.1, and 26.1
Mark Graves, P.Geo. Independent Geological Consultant Date Signed:	Sections 1.2, 1.3, 1.4, 1.5,1.6.1, 4 to 10, 11.0, 11.1, 11.2, 11.5.1, 23, 24.0, and 27
Michael P. Cullen, P.Geo., Mercator Geological Services Limited March 21, 2017	Sections 1.6.1, 1.8.1, 1.9.1, 1.10.1, 11.3, 11.4, 11.5.2, 11.5.3, 12.2, 14.1, 25.2 and 26.2
Matt Bolu, P.Eng., Bolu Minerals Engineering & Consulting Inc. March 21, 2017	Sections 1.7, 1.10.3, 13, 17, 18.1, 18.2, 18.4, 18.5, 18.6, 21.3, 21.4, 21.5, 25.3, and 26.3
Doug Roy, P.Eng., MineTech International Ltd. March 21, 2017	Section 16.1, 21.1, 21.2,

Brett Byler, PE, AMEC Foster Wheeler March 21, 2017	Sections 1.10.4, 18.3, 18.9to 18.16, 25.4, and 26.4

APPENDIX A

Certificates of Qualified Persons

CERTIFICATE OF QUALIFIED PERSON

Brett Byler, PE
2000 South Colorado Blvd., Ste. 2-1000
Denver, CO 80222

I, Brett Byler, PE, am employed as a Senior Geotechnical Engineer with Amec Foster Wheeler Environment and Infrastructure, Inc.

This certificate applies to the technical report titled "National Instrument 43-101 Technical Report for the Dawson Property, Colorado, USA", effective date of March 21, 2017 (Technical Report).

1. I am a Registered Professional Engineer in the State of Colorado. I graduated with a Bachelor of Science in Geological Engineering from the Colorado School of Mines in 1995 and with a Master of Science in Civil Engineering from the University of Colorado at Boulder in 2003.

2. I have practiced my profession for 18 years. I have been directly involved in studies, design and construction of tailings storage facilities, heap leach pads and other mine infrastructure.

3. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

4. I have personally visited the Dawson Project site on June 16, 2015.

5. I am responsible for Sections 1.10.4, 18.3, 18.9, 18.10, 18.11, 18.12, 18.13, 18.14, 18.15, 18.16, 25.4 and 26.4 of the Technical Report.

6. I am independent of Zephyr Minerals Ltd. as independence is described by Section 1.5 of NI 43-101.

7. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

8. I have read NI 43-101 and form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 22nd day of March, 2017.

(signed and sealed)

Brett Byler, PE
Senior Geotechnical Engineer

BRIAN THOMAS
CERTIFICATE OF QUALIFICATION

I, Brian Thomas, P.Geo., do hereby certify that:

1. I am employed as a Senior Resource Geologist at:
   Golder Associates Limited
   33 Mackenzie Street, Suite 100
   Sudbury, Ontario, Canada, P3C 4Y1
   Telephone: 705-524-6861; Email: brian_thomas@golder.com
2. I graduated with a Bachelor's degree in Geology from Laurentian University of Sudbury, Ontario in 1994.
3. I am a member in good standing of the Association of Professional Geoscientists of Ontario (#1366) and a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#38094).
4. I have practised my profession continuously since graduation. My relevant background with respect to this project is over twenty two years of experience in mine geology and mineral resource evaluation of mineral projects nationally and internationally in a variety of commodities including eight years of gold mine experience with Placer Dome Ltd.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43 101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.
6. I am responsible for Sections 1.1, 1.6.2, 1.8.2, 1.9.2, 1.10.2, 2, 3, 12.1, 14.2, 25.1, and 26.1 of the technical report titled "National Instrument 43-101 Technical Report for the Dawson Property, Colorado, USA (Technical Report). I have personally completed a site visit of the Dawson Project on August 2, 2016.
7. I have had no prior involvement with the Dawson Project.
8. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying, all the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated at Sudbury, Ontario, this 22nd day of March, 2017.

(signed and sealed) Brian Thomas

Brian Thomas, P.Geo.
Senior Resource Geologist

Danny Tolmer
CERTIFICATE OF QUALIFICATION

I, Danny Tolmer, P.Eng., do hereby certify that:

1. I am employed as a Senior Mining Engineer at:
   Golder Associates Ltd.
   2920 Virtual Way
   Vancouver, British Columbia
   Canada V5M 0C4
   Tel: +1 (604) 296-4200; Fax: +1 (604) 298-5253; Email: dtolmer@golder.com
2. I graduated with a B.A.Sc in Mining Engineering, from the University of British Columbia, in 2004.
3. I am a member in good standing of the Association of Professional of British Columbia (#33590)
4. I have practised my profession continuously since graduation. My relevant background with respect to this project is I have over twelve years of experience in mine planning and mine economic evaluations with projects in North America and internationally in a variety of commodities including gold.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43 101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.
6. I am responsible for Sections 1.9.3, 15, 16.2, 18.7,18.8, 19, 20, 22, 25.5 and 26.5 of the technical report titled "National Instrument 43-101 Technical Report for the Dawson Property, Colorado, USA", effective date of March 21, 2017 (Technical Report).
7. I have not personally completed a site visit of the Dawson Project.
8. I have had no prior involvement with the Dawson Project.
9. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying, all the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated at Vancouver, British Columbia, this 22, day of March, 2017.

(signed and sealed) Danny Tolmer

Danny Tolmer, P.Eng.

Senior Mining Engineer

MINETECH INTERNATIONAL LIMITED
HALIFAX, CANADA

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Consent of Author

I, William Douglas Roy, M.A.Sc., P.Eng., do hereby consent to the filing of the written disclosure of the technical report titled, “National Instrument 43-101 Technical Report for the Dawson Property, Colorado, USA”, effective date of March 21, 2017 (Technical Report) and any extracts from or a summary of the Technical Report in the news releases and other disclosures of Zephyr Minerals Ltd, and to the filing of the Technical Report with securities regulatory authorities.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the Technical Report or that the written disclosure in the news releases and other disclosures of Zephyr Minerals Ltd contains any misrepresentation of the information contained in the Technical Report.

Dated this 21st day of March, 2017.

”signed and sealed”

William Douglas Roy, M.A.Sc., P.Eng.
Mining Engineer

Mark Graves

CERTIFICATE OF QUALIFICATION

I, Mark Graves, P.Geo., do hereby certify that:

1. I am employed as a Consulting Geologist at:
   Sole Proprietor
   99 Skyway Drive, Wolfville, Nova Scotia, Canada, B4P 1S3
2. I graduated with a Bachelor's degree in Geology from Dalhousie University, Halifax, Nova Scotia in 1978.
3. I am a registered member in good standing of the Association of Professional Geoscientists of Nova Scotia (Member Number 172) and Association of Professional Engineers and Geoscientists of Newfoundland and Labrador (Member Number 2911).
4. I have practised my profession continuously since graduation. My relevant background with respect to this project is 1) mapping in granitic terrains 2) managing drill programs in the search for vein and disseminated gold deposits and 3) have extensive mineral exploration experience for various other commodities.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43 101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.
6. I am responsible for Sections 1.2, 1.3, 1.4, 1.5, 1.6.1, 4 to 10, 11.0, 11.1, 11.2, 11.5.1, 23.0, 24.0 and 27.0 of the technical report titled "National Instrument 43-101 Technical Report for the Dawson Property, Colorado, USA", effective date of March 21, 2017 (Technical Report). I worked on the Dawson Property from June 1, 2016 to August 27, 2016 managing the drill program and collecting other geologic data that forms the basis for part of this report.
7. I have prior involvement with the Dawson Project. I am one of the qualified persons responsible for preparation of the 2013 Technical Report titled: Resource Estimate Technical Report for the Dawson Property, Fremont County, Colorado, USA, effective date: July 19th, 2013. I was also responsible for exploration in the Windy Gulch Segment area of the Dawson Project during February 11, 2013 through May 12, 2013 and visited the Dawson Project frequently during this time. In addition, in March 2013, I viewed and sampled historical drill core samples from the 1984 US Borax, 1987-1988 Jascan and the 1990 Uranerz drilling programs completed on the Dawson Segment of the Project.
8. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying, all the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated at Wolfville, Nova Scotia, this 22th day of March, 2017.

(signed and sealed)

Mark Graves, P.Geo.
Consulting Geologist

H.M. MATT BOLU, P. ENG.
Principal Process Engineer
BOMENCO (Bolu Minerals Engineering & Consulting) Inc.

CERTIFICATE OF QUALIFICATION

I, H.M. Matt Bolu, P. Eng.**,** do hereby certify that:

1. I am employed as the Principal Process Engineer at BOMENCO (Bolu Minerals Engineering & Consulting) Inc. with a business address at #310 – 304 West Cordova St., Vancouver BC, V6B 1E8.
2. I graduated with a M.Sc. degree in Minerals Engineering from the University of Birmingham, England (1978).
3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#27144).
4. I have practised my profession continuously since graduation. My relevant background with respect to this project is in operations, testing, design and engineering of gold and base and precious metals projects throughout in North America, Europe, South America and Asia.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (NI 43 101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purpose of NI 43-101.
6. I am responsible for Sections 1.7, 1.10.3, 13, 17, 18.1, 18.2, 18.4, 18.5, 18.6, 21.3, 21.5, 25.3, and 26.3 of the technical report titled “National Instrument 43-101 Technical Report for the Dawson Property, Colorado, USA”, effective date of March 21, 2017 (Technical Report). I have not done a site visit of the Dawson Project.
7. I have had no prior involvement with the Dawson Project.
8. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying, all the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated at Vancouver, BC, this 22nd day of March, 2017.

(signed and sealed)

H.M.Matt Bolu, P.Eng.

MICHAEL P. CULLEN, P. GEO.
CERTIFICATE OF AUTHOR

I, Michael P. Cullen, P. Geo., do hereby certify that:

1. I am employed as Chief Geologist with:

   Mercator Geological Services Limited
   65 Queen St.
   Dartmouth, Nova Scotia
   Canada
   B2Y 1GA

2. I graduated with a Master's Degree in Science (Geology) from Dalhousie University in 1984 and a Bachelor of Science Degree (Honours, Geology) in 1980 from Mount Allison University.

3. I am a member in good standing of the Association of Professional Geoscientists of Nova Scotia (Registration Number 064), the Association of Professional Engineers and Geoscientists of Newfoundland and Labrador (Member Number 05058) and Association of Professional Engineers and Geoscientists of New Brunswick, (Registration Number L4333).

4. I have practised my profession continuously since graduation. I have worked as a geologist in Canada and internationally since graduation and have extensive relevant experience in relation to resource estimation and geological studies of mesothermal and epithermal precious metal deposits as well as various other styles of mineral deposits.

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

6. I am responsible for Sections 1.6.1, 1.8.1, 1.9.1, 1.10.1, 11.3, 11.4, 11.5.2, 11.5.3, 12.2, 14.1, 25.2 and 26.2 of the Technical Report titled "National Instrument 43-101 Technical Report for the Dawson Property, Colorado, USA, effective date March 21st, 2017. I have not visited the Dawson Property site.

7. I supervised a mineral resource estimate prepared in 2013 by Mercator Geological Services Limited for the Dawson Segment gold deposit.

8. I am independent of the issuer, applying all of the tests in section 1.5 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1 and the sections of this Technical Report for which I am responsible have been prepared in accordance with that instrument and form.

Dated at Dartmouth, Nova Scotia, Canada this 22nd day of March, 2017.

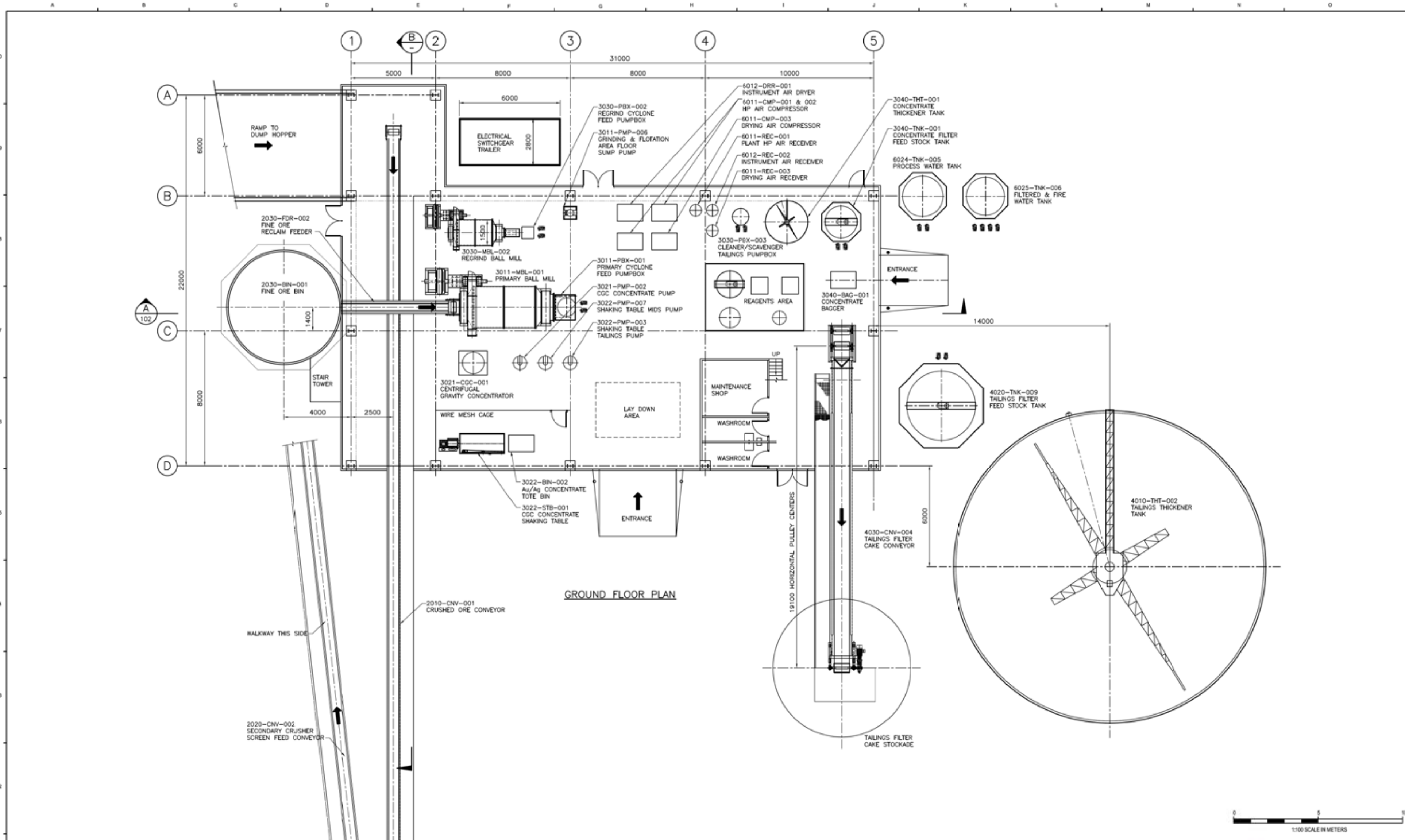
"Original signed and stamped by"

Michael P. Cullen

Michael P. Cullen, M. Sc., P.Geo.
Chief Geologist
Mercator Geological Services Limited

APPENDIX B

General Arrangement Plans



NO.	DESCRIPTION	DATE	BY
A	ISSUED FOR REVIEW	02MAY2016	JSM

REVISIONS / ISSUES — REFERENCE DRAWINGS

DESIGNED BY	HMB	11MAR16
DRAWN BY	JSM	11MAR16
CHECKED BY		
APPOVED BY		
DWG. SCALE	1:100	

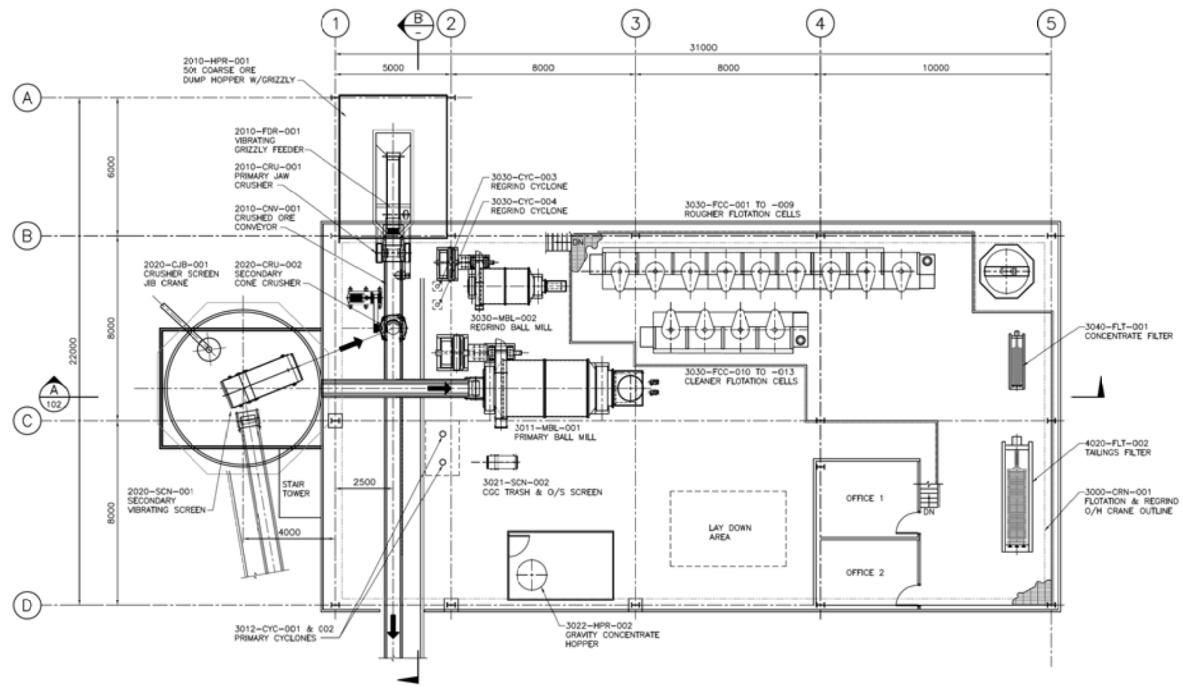
BOMENCO INC.
BOLU MINERAL ENGINEERING & CONSULTANT
VANCOUVER, BC

PRELIMINARY SCOPING STUDY
PROPOSED 300t/d FLOTATION PLANT
GENERAL ARRANGEMENT
PLAN - SHEET 1 OF 2

Zephyr Gold USA Ltd.

ZEPHYRGOLD USA LTD
DAWSON GOLD PROJECT, COLORADO, USA

CLIENT DWG. NO
DRAWING NO 15561-3000-GAD-100
REV. A



UPPER FLOOR PLAN

1:100 SCALE IN METERS

		DATE	
DESIGNED BY	HMB	11MAR16	
DRAWN BY	JSM	11MAR16	
CHECKED BY			
APPOVED BY			
DWG. SCALE	1:100		

NO.	DESCRIPTION	DATE	BY
A	ISSUED FOR REVIEW	02MAY2016	JSM

REVISIONS / ISSUES

REFERENCE DRAWINGS

BOMENCO INC.
BOLU MINERAL ENGINEERING & CONSULTANT
VANCOUVER, BC

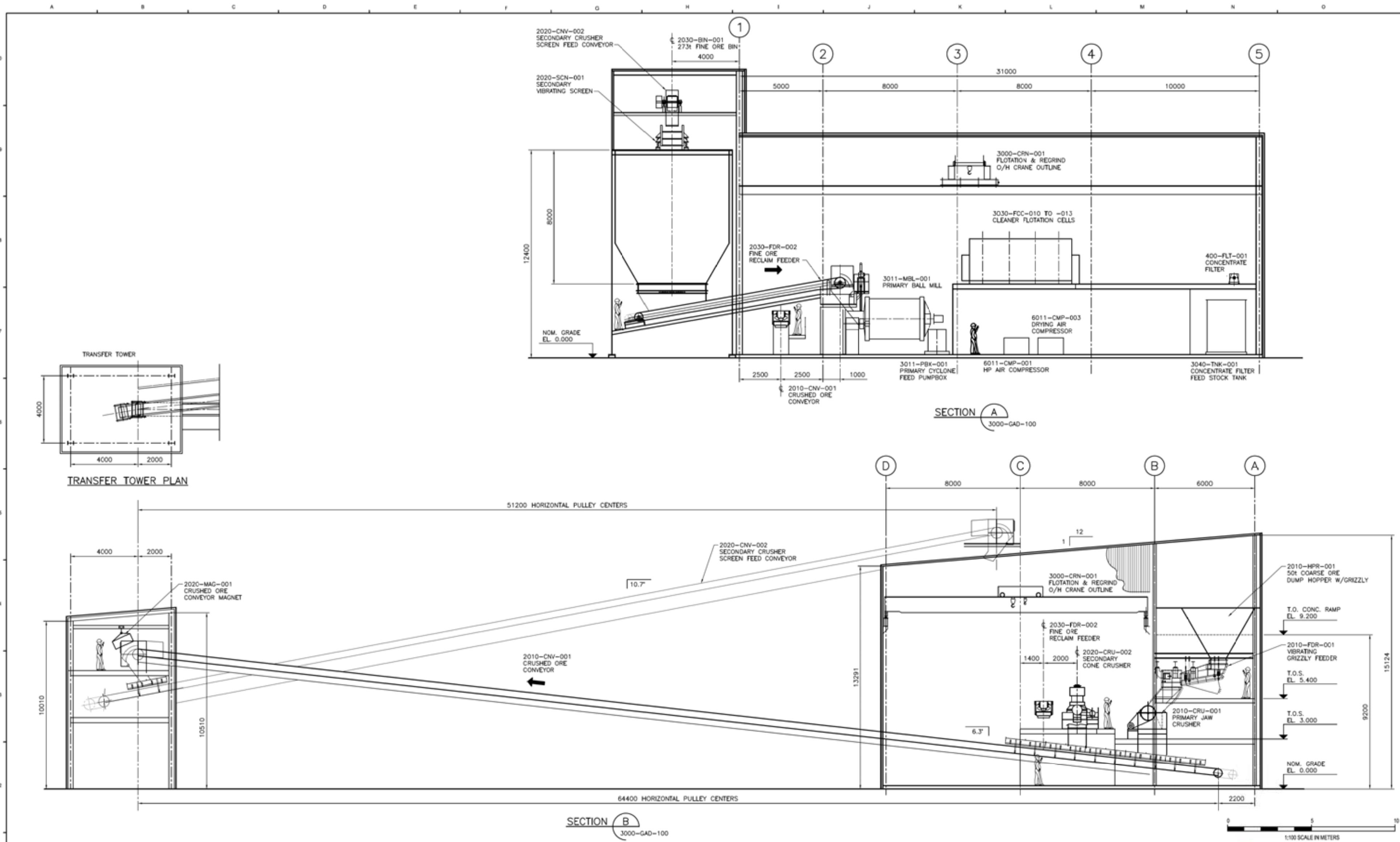
PRELIMINARY SCOPING STUDY
PROPOSED 300t/d FLOTATION PLANT
GENERAL ARRANGEMENT
PLAN - SHEET 2 OF 2

Zephyr Gold USA Ltd.

ZEPHYRGOLD USA LTD
DAWSON GOLD PROJECT, COLORADO, USA

CLIENT DWG. NO

DRAWING NO 15561-3000-GAD-101 REV. A



NO.	DESCRIPTION	DATE	BY
A	ISSUED FOR REVIEW	02MAY2016	JSM

REVISIONS / ISSUES

ELECTR. | INSTRU. | P.P. | MECH. | CIVIL | SERV. | PROCE. | P.E.M.

REFERENCE DRAWINGS

	BY	DATE
DESIGNED BY	HMB	26APR16
DRAWN BY	JSM	26APR16
CHECKED BY		
APPROVED BY		
DWG. SCALE	1:100	

BOMENCO INC.
BOLU MINERAL ENGINEERING & CONSULTANT
VANCOUVER, BC

PRELIMINARY SCOPING STUDY
PROPOSED 300t/d FLOTATION PLANT
GENERAL ARRANGEMENT
SECTIONS

Zephyr Gold USA Ltd.

ZEPHYRGOLD USA LTD
DAWSON GOLD PROJECT, COLORADO, USA

CLIENT DWG. NO.
DRAWING NO. 15561-3000-GAD-102
SHEET NO.
REV. A

As a global, employee-owned organisation with over 50 years of experience, Golder Associates is driven by our purpose to engineer earth's development while preserving earth's integrity. We deliver solutions that help our clients achieve their sustainable development goals by providing a wide range of independent consulting, design and construction services in our specialist areas of earth, environment and energy.

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