

NI 43-101 Technical Report

Frasergold Exploration Project

Cariboo Mining Division, B.C.

Latitude 52° 18' N, Longitude 120° 35' W

for

Eureka Resources Inc.

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by

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1. Summary

The purpose of this report is to update relevant geological and technical information on the Frasergold property, such as additional and sampling conducted by Teslin River Resources Corp. ("Teslin" or the "Company") during the 2011 exploration season and by Eureka Resources Inc. in the spring of 2015.

For purposes of clarification, much of the data in this current report comes from the March 20, 2007 technical report titled, "Summary Report and Exploration Proposal on the Frasergold Project, Cariboo Mining Division, B.C.", by authors Geoffrey Goodall, P. Geo. and K.V. Campbell, Ph. D., P.Geo.

Additional data has been incorporated from the January 31, 2008 NI 43-101 technical report "Summary Report and Exploration Proposal on the Frasergold Project, Cariboo Mining Division, B.C." by authors J. Sparling, P.Geo. and K.V. Campbell, Ph.D., P.Geo. and the November 15, 2009 NI 43-101 technical report "Report on the 2007 and 2008 drill programs on the Frasergold Project, Cariboo Mining Division, B.C." by authors K.V. Campbell, Ph.D., P.Geo. and G.H. Giroux, MA.Sc., P.Eng.

The current report was prepared by the Qualified Persons K.V. Campbell, Ph. D., P.Geo. and G.H. Giroux, MA.Sc, P.Eng. as a technical report meeting NI 43-101 requirements. This report will be posted on SEDAR.

The Frasergold Property claims are located approximately 50 kilometers east of the village of Horsefly, B.C. and 100 kilometers east northeast of city of Williams Lake, B.C. located on NTS map sheets 093A02 and 093A07 at approximately 52° 18' 17.62" North latitude and 120° 35' 16.14" West longitude. The property consists of 25 contiguous mineral claims covering approximately 3,616.89 hectares within the MacKay River valley, a tributary to the Horsefly River.

This report summarizes the geology, mineralization, resources and exploration potential of the Frasergold property located in the Cariboo Mining Division of central British Columbia and makes recommendations for additional exploration to define further zones of mineralization on the property.

The Frasergold property straddles the boundary between two major tectonic belts of the Canadian Cordillera; the Omineca Tectonic belt lies on the east side of the property while the Intermontane Belt occupies the west and central portions of the property. Three regional tectonostratigraphic terranes are present; Kootenay, Slide Mountain and Quesnellia terranes. The Slide Mountain and Quesnellia terranes are part of the Intermontane Belt which has been accreted eastward onto the Kootenay terrane of the Omineca Belt. The Eureka Thrust forms the tectonic boundary between these two Belts.

In the project area the Omineca Tectonic Belt is represented by Hadrynian to early Paleozoic quartz-mica schists and gneisses of the Snowshoe Group. These make up part of the Kootenay terrane; pericratonic, intensely deformed, variably metamorphosed rocks which appear to be stratigraphically related to ancestral North America. The Omineca Tectonic Belt is known for its prevalence of gold and tungsten mineral occurrences, such as those in the Barkerville gold mining camp to the north. The Quesnellia

Terrane is composed of metavolcanic and phyllitic rocks of Permian to Jurassic age. Numerous copper and gold deposits occur within this package of rocks, including the Mt. Polley mine 40 kilometers north of Frasergold. The Frasergold mineralization appears to fit the orogenic lode-gold deposit type; gold tends to occur in quartz veins with coarse particulate gold occurring in segregations of stringers, veins, boudins and mullions. Pervasive low grade gold mineralization is also found within the knotted phyllite strata where quartz is absent.

Exploration activities completed on the Frasergold property during the period 1981 through 1994 by the companies Amoco, Eureka and Asarco identified an area of gold mineralization that extends along a strike length of approximately 10 kilometers. Within this area are five mineralized zones that have been partially delineated by drilling and underground sampling. With further exploration, these zones may be determined to be continuous. Historic resource estimates of the Main Zone were completed in 1991 by Campbell et al (3,396,970 tons at 0.05 oz/T Au) and James Askew and Associates (6,612,675 tons at 0.055 oz/T Au). These estimations are not compliant with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") standards of definition, and therefore do not fulfill NI 43-101 reporting standards, and should not be relied upon. However, this information provides a relevant indication of the level of exploration conducted and the tenor of mineralization identified at the Frasergold property.

In the 2006 Hawthorne Gold Corp. ("Hawthorne") optioned the Frasergold property from Eureka and in 2007 completed minor trenching programs, oversaw rehabilitation of historical underground workings on the Main Zone, including rock bolting to allow detailed channel sampling throughout the workings, and conducted a diamond drill program consisting of sixteen (16) HQ holes totalling 3,615 meters. Aeroquest International Advanced Airborne Geophysics completed a helicopter-borne geophysical survey on the property during the 2007 field season. The approximate cost of the Hawthorne 2007 program was \$5.1 million (Sparling, 2008).

During the 2008 season Hawthorne completed 10,414 meters of NQ2 diamond drilling, regional mapping and several soil sampling programs. The approximate cost of the Hawthorne 2008 program was \$3.7 million (Whitehead, 2009).

The 2009 resource estimate is summarized below for a cut-off value of 0.50 g/t Au. All of the resources described are located on the Frasergold property owned by Eureka Resources Inc.

Zone	Class	Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		
				Au (g/t)	Au (grams)	Au (ounces)
Main	Measured	0.50	5,600,000	0.812	4,500,000	145,000
Main	Indicated	0.50	9,570,000	0.755	7,200,000	231,000
Main	M+I	0.50	15,170,000	0.776	11,800,000	376,000
Main	Inferred	0.50	8,270,000	0.670	5,500,000	177,000
NW	Inferred	0.50	19,180,000	0.740	14,200,000	457,000
SE	Inferred	0.50	43,000	0.632	27,000	900
Total	Inferred	0.50	27,493,000	0.718	19,727,000	634,900

Hawthorne's interest in the Frasergold property expired in 2010 and the claims were returned to Eureka with no encumbrances. The property was then optioned to Teslin River Resources Corp. ("Teslin") in 2011 who completed a limited geochemical soil survey on the property. In December, 2013 Teslin's option agreement expired and the Frasergold property was returned to Eureka.

In late April, 2015, Eureka conducted a geochemical soil survey in order to maintain the claims in good standing through to April and October, 2016.

A two stage program is recommended. Additional resource may be developed along the southeast and northwest projections of the main zone and within the 18ppm and Eureka Bowl grid areas. The first stage is continued soil sampling, road and drill site construction, and a 6,000m diamond drill program at an estimated cost of \$2,000,000. 1,500m of this drilling could be conducted in 2015, the remaining 4,500m completed in 2016. Contingent upon favorable results from the Stage I drilling, additional drilling would be recommended at a cost of about \$2,500,000.

2. Introduction

2.1. Background

This report summarizes the geology, mineralization, historic gold resources and exploration potential of the Frasergold project area located in the Cariboo Mining Division of central British Columbia and makes recommendations for exploration to define further zones of mineralization on the property. The report has been prepared at the request of Mr. Michael D. Sweatman, President of Eureka Resources Inc. ("Eureka").

This document is a technical report outlining the geological setting, results of exploration and the exploration potential of the Frasergold property. The report is compliant with Canadian Securities Administrators' ("CSA") National Instrument 43-101 ("NI 43-101") requirements and satisfies independent "Qualified Person" requirements under Part 4 of the instrument.

Sources of information for this report are referenced in Section 20 (References). No independent verification of the historical geological, geochemical or geophysical data was undertaken. However, there is no reason not to rely on the sources of this data and nothing has come to the attention of the authors which would cause them not to rely on such data. The authors have no reason to believe that any material facts have been withheld.

2.2. Qualified Persons

The Qualified Persons ("QP"), as defined by NI 43-101, responsible for preparation of this Technical Report are:

- K. Vincent Campbell, Ph.D., P..Geo. - Executive Consultant and President of ERSi Earth Resource Surveys Inc., ("ERSi") and
- Gary H. Giroux, MA.Sc, P.eng. - Executive Consultant and President of Giroux Consultants Ltd.

The primary author of this report, K. Vincent Campbell, is a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of British Columbia. He is a Qualified Person as defined by NI 43-101 and has extensive knowledge of the Frasergold property through his involvement of exploration programs during the periods 1984 to 1991 and 2007 to 2009. He co-authored the preparation of the March 20, 2007, January 31, 2008 and November 15, 2009 technical reports. His most recent visit to the property was on June 13, 2015 in order to assess access conditions and the current state of logging and other activities in the MacKay River valley.

Gary Giroux, the co-author of this report, is a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of British Columbia. He is a Qualified Person as defined by NI 43-101 and was responsible for the current resource estimation of the Frasergold project. He co-authored the preparation of the November 15, 2009 technical report. He last visited the property on September 10-11th, 2008.

All currency referred to in this report is in Canadian dollars. This report draws largely on information that was originally reported in Imperial measurements. Conversion to metric measurements is included where appropriate, in addition to reporting the Imperial measurements. Historic resource estimations were left in their original context.

All map coordinates in this report are based on Universal Transverse Mercator (UTM) Zone 10 projection in North American Datum 1983 (NAD83).

The Technical Report is intended to be read as a whole and sections or parts should not be read or relied upon out of context.

The effective date of this Technical Report is July 20th, 2015.

2.3. Definitions, Abbreviations and Conversions

asl	above sea level	mm	millimeter(s)
ARIS	Assessment Report Indexing System	MEM	Ministry Energy and Mines (B.C.)
BCFS	British Columbia Forest Service	MTO	Mineral Titles Online (B.C. MEM)
BCGS	British Columbia Geographic System	NAD	North American Datum
NI 43-101	Canadian National Instrument 43-101	n.a.	not available/applicable
DDH	diamond drill hole	NQ	diamond drill inside diameter of 47.6mm
cm	centimeter(s)	NTS	National Topographic Service
°	degree(s)	NOWR	Notice of Work and Reclamation
°C	degrees Celsius	nT	nano Tesla
DEM	digital elevation model	oz	troy ounce
g or gm	gram(s)	ppb	parts per billion
g/t	grams per metric tonne	ppm	parts per million
>	greater than	%	percent(age)
ha	hectare(s)	TRIM	Terrain Resource Information Management
NSR	Net Smelter Return		
GPS	Global Positioning System	VLF-EM	Very Low Frequency Electro Magnetic
kg	kilogram(s)	T	ton (2000 pounds or 977.2 kg)
km	kilometer(s)	QA	quality assurance
<	less than	QC	quality control
m	meter(s)	t	tonne (1000 kg or 2,204.6 pounds)
Ma	million years	1 gram	0.3215 troy oz
mg	milligram(s)	1 oz/Ton	28.22 gm/tonne
CIM	Canadian Institute of Mining and Metallurgy	1 troy oz	31.104 gm
CSA	Canadian Securities Administrators	RC	reverse circulation drill hole

3. Reliance on Other Experts

The B.C. Ministry of Mines and Energy's Mineral Titles Online ("MTO") facility has been relied upon to verify the tenure and status of the mineral claims held by Eureka. The author undertook a search on June 5, and June 24, 2015 of the tenure data on the British Columbia government's Mineral Titles Online website which confirms the geospatial locations of the claims boundaries. This is common practice in the mineral exploration industry in British Columbia to locate claim boundaries, since the advent of internet staking.

As of the date of this report, the authors are not aware of any material fact or material change with respect to the subject matter of this Technical Report that is not presented herein, or which the omission to disclose could make this report misleading.

All maps were made by ERSi Earth Resource Surveys Inc. ("ERSi") unless otherwise stated.

4. Property Description and Location

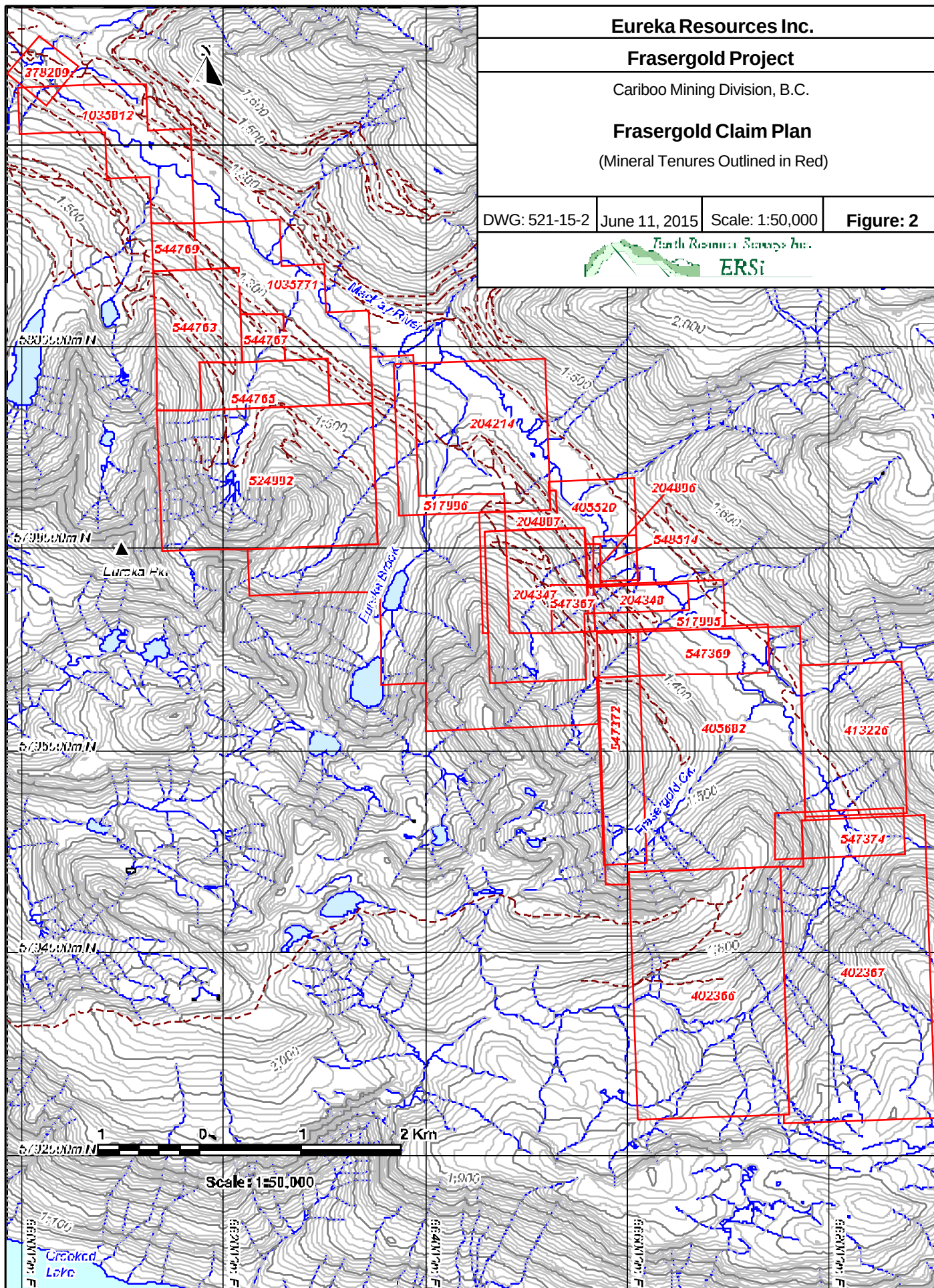
The Frasergold Property lies in the eastern Cariboo region of central British Columbia (Figure 1), approximately 100 km east of Williams Lake in the Cariboo Mining Division. The claims straddle the MacKay River valley and are centered at approximately 52° 18'N and 120° 35' W on National Topographic System (NTS) map sheet 093A07 (1:20,000 BCGS map sheets 093A027, 093A028, 093037 and 093A038) as shown on Figure 2.

The 25 claims included in the property are listed in Table 4.1. All of the claims are owned 100% by Eureka Resources Inc.

Table 4.1. Frasergold claim list.

Tenure	Claim Name	Issue Date	Good To Date	Area (ha)
204214	MAC	1979/Oct/19	2016/Oct/15	225.00
204347	KAY #10	1980/Sep/25	2016/Oct/15	150
204348	KAY #11	1980/Sep/25	2016/Oct/15	50
204887	MAC 9 FR.	1984/Jul/16	2016/Oct/15	25
204896	MAC 11 FR	1984/Jul/27	2016/Oct/15	25
378209	L-1	2000/Jun/18	2018/Aug/30	25
402366	KAY #10	2003/May/09	2016/Oct/15	375
402367	KAY #11	2003/May/09	2016/Oct/15	450
405520	J#1	2003/Oct/04	2016/Oct/15	100
405682	KAY #9	2003/Sep/26	2016/Oct/15	500
413226	J#2	2004/Aug/17	2016/Oct/15	150
517995	NUGGET	2005/Jul/18	2016/Oct/15	59.312
517996	IMPERIAL	2005/Jul/18	2016/Oct/15	494.312
524992	EUREKA	2006/Jan/10	2016/Oct/15	296.515
544763	EUREKA	2006/Nov/01	2016/Oct/15	98.8078
544765	MISSING	2006/Nov/01	2016/Oct/15	59.2926
544767	ADD ON	2006/Nov/01	2016/Oct/15	19.7624
544769	ANOTHER	2006/Nov/01	2016/Oct/15	19.755
547367	H#1	2006/Dec/14	2016/Oct/15	19.7712
547369	H#2	2006/Dec/14	2016/Oct/15	59.3169
547372	H#3	2006/Dec/14	2016/Oct/15	79.1099
547374	H#4	2006/Dec/14	2016/Oct/15	59.3374
548514	EUR #1	2007/Jan/03	2016/Oct/15	19.769
1035771	KK	2015/Apr/29	2016/Apr/29	138.3224
1035812	EXT	2015/Apr/30	2016/Apr/30	<u>118.5019</u>
			Total:	3616.89

In British Columbia, the owner of a mineral claim acquires the right to the minerals that were available at the time of the claim and location and as defined in the Mineral Tenure Act of British Columbia. Surface rights and placer rights are not included. Claims are valid for one year and the anniversary date is the annual occurrence of the date of record (the staking completion date of the claim). To maintain a claim in good standing the claim holder must, on or before the anniversary date of the claim, pay the prescribed



recording fee and either: (a) record the exploration and development work carried out on that claim during the current anniversary year; or (b) pay cash in lieu of work which is double the dollar value. The amount of work required in the first 2 years is \$5 per hectare, years 3 and 4 are \$10 per hectare, years 5 and 6 are \$15 per hectare, year 7 and beyond is \$20 per hectare. Only work and associated costs for the current anniversary year of the mineral claim may be applied toward that claim unit. If the value of work performed in a year exceeds the required minimum, the value of the excess work, in full year multiples can be applied to cover work requirements for that claim for additional years (subject to the regulations). A report detailing work done and expenditures must be filed with and approved by, the B.C. Ministry of Energy and Mines.

The Province of British Columbia owns surface rights over the property. Exploration permits must be obtained from the British Columbia Ministry of Energy, Mines and Petroleum Resources prior to carrying out mechanized exploration on the property. No such permit was required for the geochemical surveys completed in 2011 and 2015. Eureka's Exploration Permit MX-10-216 has been extended to April 14, 2017.

The author accessed British Columbia Mineral Titles Online on June 5th, and June 24th, 2015 to collect the claim information presented in Table 4.1.

There are no royalties, back in rights, environmental liabilities, and no known risks to undertake exploration, except for a 3% NSR to Southlands Mining Corp., dated September 22, 1989. The NSR can be purchased for \$2.6 Million (adjusted annually to reflect changes in the Consumer Price Index) and is only payable after a mining operation has recovered 100% of its capital costs.

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Frasergold property is accessed by road from highway 97 near 150 Mile House to the village of Horsefly. All weather gravel and logging roads extend 55 kilometers to the east along the Horsefly River to MacKay River. A branch logging road to the southeast enters the MacKay River valley and reaches the central area of the property after a distance of approximately 10 km. Large areas of the northeastern and eastern claims have been logged, providing improved outcrop exposure and a network of access trails, most of which have been decommissioned. Access to other areas of the property is very limited.

The upland area along the Eureka Peak ridge is an environmentally sensitive alpine region and with restricted access requiring permission from the Ministry of Environment. In February of 2009 the area was placed within the B.C. Government's Mountain Caribou Recovery Implementation Plan and is off-limits for logging and road building.

The area lies within the Columbia Highland ecoregion, specifically the Quesnel Highland Ecoregion (Ecosystem Information Section, B.C. Ministry of Environment¹).

This is a transitional highland area, intermediate between the lower level plateaus to the west and the higher rugged mountains to the east. Here there are remnants of a highly dissected plateau of moderate relief, which rise gradually from west to east. Glaciers covered all of this area moving north-westward to coalesce with north flowing ice from the Fraser Plateau. The glaciers resulted in greatly rounded summits, but cirque basins have developed on north sides. In this area the ecoregion is drained to the west: via the Fraser River by the, Cariboo, Matthew, Little, Roaring and upper Horsefly streams.

Precipitation is higher here than in the Shuswap Highland Ecoregion to the south. Easterly flowing moist Pacific air rising over this area drops considerable moisture as rain in the summer and snow in the winter. In addition cold, dense Arctic air can build up against the western margin or invade into the valleys and mountains giving intense cold for extended periods. Wet Interior Cedar – Hemlock forests dominate all the valleys and lower slopes; colder Engelmann Spruce – Subalpine Fir forest dominate the upper slopes including the lower mountain summits. Alpine occurs on the highest ridges and mountains in the middle and eastern sections of this ecoregion.

The Frasergold property occurs on the western flank of the Cariboo Mountain Range. Topography is moderately steep in the northwest portion of the property and steeper in the southeast portion. Elevations range from approximately 1,200 to 2,100 meters. Vegetation along the MacKay River valley consists of commercial spruce and balsam with thick underbrush. Rock exposure is sparse, being limited to steep and deeply incised creek gullies. Above 1,600 meters, forest cover becomes less dense and alpine vegetation is encountered at approximately 1,800 meter elevation. Large areas of the property have been logged within the past 20 years and second growth forests are maturing.

Climatic conditions are typical of the central interior of British Columbia. Average minimum low temperatures for January are -18°C and average maximum highs for July are +24°C. Frost free days last on average from mid-May to mid-August. Between May and September, precipitation at a low-elevation station is about 400 millimeters, almost twice that of Williams Lake, 100 kilometers to the west. During April, snow depths in the Quesnel Plateau (approximately 700 meters above sea level) are typically one to two meters.

The village of Horsefly is a supply centre for the local logging and ranching population and has readily available skilled labour, as well as board, lodging, fuel and other supply outlets. Field operations are generally conducted with crews lodged in Horsefly or in nearby fishing lodges. The electrical power grid does not currently extend east onto the Frasergold property. Year round work conditions for exploration activities are hampered only by snow accumulation, requiring snow clearing on the access roads and avalanche control along the steep terrain.

¹ <http://www.env.gov.bc.ca/ecology/ecoregions/index.html>

6. History

6.1. Prior to 2007

The first record of work being conducted in the vicinity of the Frasergold property for gold is in the late 1970s when Clifford E. Gunn began prospecting the area. He was attracted to the Frasergold Creek area by historic references to the placer gold potential of the region. In 1979 he staked the original claims in the area to cover a panned gold anomaly discovered in Frasergold Creek. Prospecting was completed in 1979 and from 1980 to 1982 the ground was optioned by Keron Holdings Ltd. and NCL Resources Ltd. A preliminary soil and rock geochemical survey was completed over the property and a geology map was produced. Results revealed a 10 kilometer long zone containing anomalous gold values from soil samples that was thought to have a stratigraphic control.

In 1983 Eureka acquired the property and optioned it to Amoco Canada Petroleum Co. Ltd. ("Amoco"). During 1983 and 1984 Amoco collected rock and soil geochemical samples, conducted limited electromagnetic and magnetic surveys, and drilled 14 diamond drill holes totalling 4,519 meters. Coarse visible gold was noted in 12 of the 14 drill holes and anomalous intersections had values ranging from 0.023 oz/T Au over 7.5 meters to 0.342 oz/T Au over 1.5 meters. Amoco terminated the option agreement at the end of these programs and returned the property to Eureka.

Eureka continued exploring the Frasergold property in 1985 and 1986 and completed further soil and rock chip geochemical sampling, trenching and bulk sampling, an I.P. survey, reverse circulation, diamond drilling and metallurgical testing. Limited reverse circulation drilling was completed in four holes totalling 406.5 meters, and eighteen diamond drill holes, totalling 2,021 meters were completed in three areas. Twelve of the 18 holes had sections with visible gold and anomalous values ranged from 0.057 oz/T over 39.0 meters (drill hole 86-2) to 1.311 oz/T Au over 1.5 meters (drill hole 86-18).

A surface bulk sampling program completed in 1985 consisted of selecting eight sites for excavation. A total of 56 samples were collected and analyzed for gold content by fire assay. One sample, 86-12-2A from the Jay Zone, was submitted to Coastech Research Inc. which milled the material and completed cyanidation testing on the sample. Results from the cyanidation work were compared to the standard fire assay analyses. The mean fire assay (FA) values from the 56 samples varied from 0.06 oz/T Au to 0.128 oz/T Au. Coastech split bulk sample 86-12-2A into 24 composites and completed cyanidation leach metallurgical work on the samples. Leishman and Campbell (1986) report that the results varied from 0.150 oz/T Au to 1.021 oz/T Au, with a weighted average of 0.479 oz/T Au. The gold content of bulk sample 86-12-2A was determined to be 0.137 oz/T Au (Marchant, 1985).

In 1987 Southlands Mining Corporation ("Southlands") undertook an option on the Frasergold property, with Eureka as operator. Southlands constructed and sampled eight trenches totalling 660 meters, and completed 21 reverse circulation holes totalling 1,710 meters.

In late 1987, Southlands optioned a portion of their interest to Sirius Resources Corp. ("Sirius"). Sirius completed 17 diamond drill holes totalling 1,536 meters, drilled 37 reverse circulation holes totalling 2,456 meters, and excavated 184 meters of underground workings to provide 524 tonnes of material for bulk sampling.

In the fall of 1988 Sirius completed work in the Eureka Peak zone, collecting 478 soil samples over a closely spaced grid, collecting 27 rock chip samples from hand trenches and drilling six diamond drill holes totalling 862 meters. Several short intervals of approximately one meter in length returned assays to 0.255 oz/T Au (Campbell, 1989).

In August 1989 a legal dispute between Eureka and Southlands over the validity of the option and joint venture agreement was resolved. During September, 1989, Eureka completed a program of underground channel sampling (284 samples), muck sampling (74 samples) from untested rounds, drill core sampling (297 samples) and re-logging of drill core and geological mapping of underground workings.

In 1990, Eureka entered into a joint venture agreement with Asarco Company of Canada Ltd. (Asarco). During the period 1990 and 1991, Asarco drilled 25 diamond drill holes totalling 4,687.2 meters, and 156 reverse circulation holes totalling 15,720 meters. Four 1.25 ton bulk samples were collected in 1990 for metallurgical testing by Bacon, Donaldson and Associates. The average composite grade of these bulks samples is 0.068 oz/T Au while preliminary tests indicated gold recoveries ranging from 87 to 92%.

The underground workings were lengthened by 114 meters in 1991. These workings produced 1,591 tons of material that was divided into nine lots for off-site milling. The estimated average grade of this material was 0.027 oz/T Au. Utilizing the drill hole and underground sample data, Campbell, et al (1991) state there is an "in-situ resource" of 3,396,970 tons at an average grade of 0.05 oz/T Au within the Main Zone portion of the Frasergold property. These estimations are not CIM compliant and therefore do not fulfill NI 43-101 reporting standards and should not be relied upon.

In their report, Campbell et al (1991) use the historic category of geological reserves which were developed based on sampling, assaying and geological interpretation with some constraints imposed by engineering considerations of minimum mining width and grade. They emphasize that this is not an estimate of "ore reserves", which require detailed engineering and cost estimation. The exploration work completed to provide data for the above resource estimation was conducted using then acceptable industry best practices by professional people and recognized laboratories. This work would require confirmation testing to determine the validity of the results reported. These estimations are not CIM compliant and therefore do not fulfill NI 43-101 reporting standards, and should not be relied upon. However the work provides relevant data on the Frasergold project and is provided from sources believed to be reliable.

The key parameters used by Campbell et al (1991) in their estimate include:

- establishing cross sections at 1:500 scale at 25 meter intervals through the zones of mineralization, with sections spaced 12.5 meters apart on the key workings of the Main Zone. All sections presented assays, geology and quartz content from each piercing drill hole from all drill campaigns;
- geological interpretation based on surface mapping and drill hole data to establish a geological framework and distribution of veins;
- zones of anomalous mineralization were interpreted from drill assays and quartz-rich zones of configurations on a hole to hole and section to section comparison basis;
- mineralized envelopes were then overlaid on the interpreted geology and mineralization sections.

Further key elements of the historical evaluation of Frasergold mineralization included:

- a cut-off grade of 0.02 oz/T Au was used in compositing assays within mineralized envelopes. A cutting factor of 0.30 oz/T Au was applied to all drill sample results prior to estimations;
- drill orientation was approximately perpendicular to the dip of the mineralized zone in the majority of drill holes. A minimum drill intersected width of 3 meters was used;
- mineralized envelopes were constructed to an elevation of 1425 meters, which is approximately 100 meters vertically below the depth of a majority of the drill holes;
- mineralized envelopes were drawn half the distance between holes when the holes were spaced less than 30 meters apart. With holes spaced greater than 30 meters apart, a maximum distance of 25 meters was used;
- volume estimations were based on lateral projection of the envelopes half the distance to the adjacent section with a specific gravity of 2.70 used for conversion to weight. The area of each mineralized envelope was digitized and the area Estimated electronically. The average grade based on the arithmetic mean of the cut assays weighted by length of sample was then applied to the individual envelopes.

The figures presented above for historical context do not conform to currently accepted CIM standards or NI 43-101 Standards of Disclosure for mineral exploration projects and should not be relied upon.

In January 1991, the mining, geological and geotechnical engineering firm James Askew Associates, Inc. of Englewood, Colorado was commissioned by Asarco to conduct a pre-feasibility study of the Frasergold project. This study does not conform to the current usage of a pre-feasibility study as defined by NI 43-101 and should not be relied upon. The Askew report does not take into account economic, mining, metallurgical, environmental, social or governmental factors. As part of this study, Askew completed "In-situ Reserves/Resources" for the project using hand drawn polygonal methods. The basis for drawing these mineralized envelopes was data collected by Asarco and others, which is believed to be reliable. Askew used a 0.03 oz/T Au cut-off with a minimum true width thickness of three meters. Assays greater than 0.60 oz/T Au were cut to 0.60 oz/T Au. Zones of gold mineralization were extended half-way to the adjacent section and were extended 75 meters down-dip. A specific gravity of 2.7 was used in the estimations.

Based on these parameters Askew (1991) summarized the gold mineralization at the Frasergold property as 6,612,675 tons of mineralized material at an average grade of 0.055 oz/T Au to represent 362,825 ounces of gold. Askew (1991) does not categorize the mineralized material due to “the comparatively small amount of geological and assay data for such a long strike length”. The volume and gold content estimates used by Askew (1991) do not conform to the “CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines”, issued in 2000 and modified with adoption of the “CIM Definition Standards – For Mineral Resources and Mineral Reserves” in 2005. The resource estimate provided by Askew (1991) does not use CIM compliant estimations and therefore do not fulfill NI 43-101 reporting standards and should not be relied upon.

Askew (1991) recommended the following: 1) that drilling be conducted on 25 meter sections with a fan of holes drilled on each section; 2) downhole surveys be conducted; 3) use of flocculent on the RC cuttings; 4) select two or three sites for underground exploration; and 5) provide on-site rock crushing facilities to reduce the handling of mineralized material. The Askew report is relevant to the current understanding of the Frasergold project as it indicates the extent of advanced exploration being conducted by Asarco on the project.

The above-mentioned resource calculation by James Askew Associates Ltd. for the Frasergold property was conducted by mining engineers, however the estimations were not conducted according to CIM standards and the resource was not signed by the engineer. The conclusions drawn in this report are reasonable for the level of exploration conducted on the Frasergold property and compare favourably with those of Campbell, et al (1990) for the Main Zone area. The Askew (1991) report is relevant to the current review of the Frasergold property as it provides an indication of the scope and depth of exploration conducted on the project.

The historic estimations are presented here for historic purposes only to provide the reader with a full understanding of the extent of previous exploration conducted on the property.

Between 1980 and 1994 it is estimated that \$8 million had been expended on the exploration of the Frasergold property. A total of 39,582 meters of drilling in 344 holes had been completed on the property, along with 294 meters of underground drifts to provide access for bulk sampling and metallurgical testing.

6.2. Historical Drill Results

Five key zones of mineralization have been identified along a 10 kilometer strike length of the sedimentary horizon, as shown in Figure 3 and summarized in Table 6.2.1 in relation to the local mine grid. Gold mineralization within these zones ranges from a minimum value of 0.005 oz/T Au to an arbitrary established upper level of 1.00 oz/T Au, with the highest reported gold grade being 7.35 oz/T Au. The zones are described as follows:

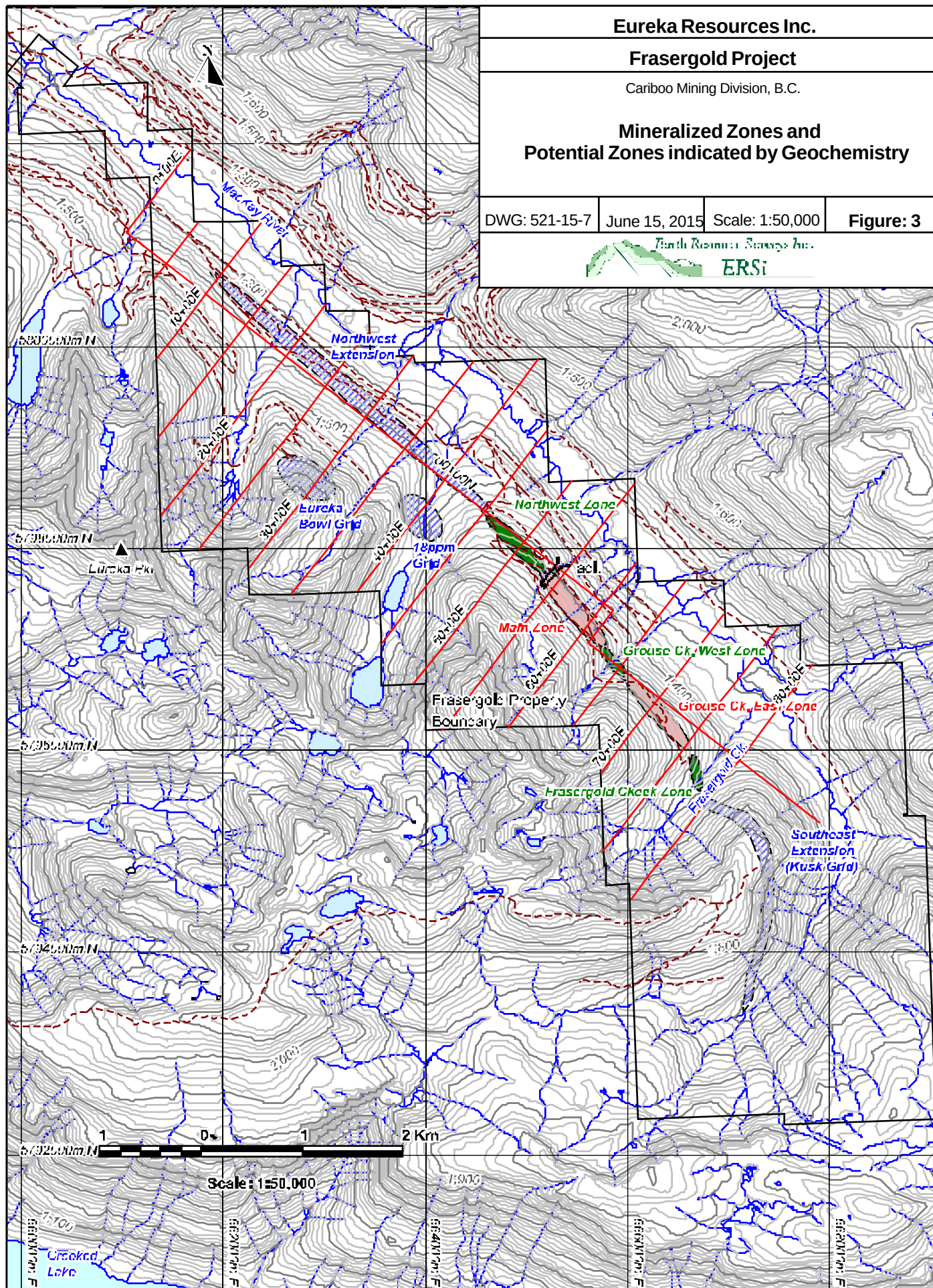


Table 6.2.1. Extent of mineralized zones in relation to Mine Grid.

Zone	Extent along strike of knotted phyllites
Historical Main Zone	55+25 to 60+50E
Grouse Creek West Zone	60+00 to 64+00E
Grouse Creek East Zone	64+00e to 72+50E
Frasergold Creek Zone	74+50 to 76+50E
Northwest Zone	44+00 to 52+00E

1. The Northwest Zone occurs between local grid co-ordinates 44+00E to 52+00E. Sixteen holes drilled in this zone indicate two lenticular zones of mineralization over widths of 10 meters, with a weighted average gold concentration of 1.04 g/t Au (0.037 oz/T Au).
2. The Main Zone is centrally located within the mineralized horizon, extending from local grid 55+25E southeast 525 meters to 60+50E. This zone has been explored by drilling and an underground drift. The thickness of the mineralized horizon varies from 7.5 meters to 30.2 meters with an average width of 15.6 meters. The weighted average gold concentration for this zone is estimated at 1.30 g/t Au (0.046 oz/T Au).
3. The Grouse Creek West Zone lies on the west side of Grouse Creek and is the southeast continuation of the Main Zone. The mineralized zone extends over an approximate 400 meter length from 60+00E to 64+00E. The average width of the mineralization is 10 meters based on drill intercept, and has a weighted average gold concentration of 1.24 g/t Au (0.044 oz/T Au).
4. The Grouse Creek East Zone extends from 64+00E to 72+50E and occurs on the east side of Grouse Creek. The 850 meter strike length of this zone has been tested to a vertical depth of 120 meters. The average width of the mineralized zone is 18.3 meters with a weighted average concentration of 1.33 g/t Au (0.047 oz/T Au).
5. The Frasergold Creek Zone extends from approximately 74+50E to 76+50E and has been tested with five drill holes. The 200 meter length of mineralization averages 16.8 meters wide with a gold concentration of 1.47 g/t Au (0.052 oz/T Au).

6.3. Recent Programs

6.3.1 2007 to 2008

The work program conducted by Hawthorne in 2007 included legal surveys, airborne photogrammetric mapping and generation of color orthophotos, trench sampling, underground channel sampling, adit rehabilitation, underground bulk sampling and a drill program laid out to test five previously outlined zones of interest, including the Northwest Zone, Main Zone, the Grouse Creek West Zone, the Grouse Creek East Zone and the Frasergold Zone. A total of 16 HQ core size diamond drill holes totalling 3,615 meters were drilled over a period of 3 ½ months, with an average depth of 226 meters.

In 2008 Hawthorne drilled an additional 58 NQ core size diamond drill holes totalling 10,414 meters, primarily in the Main Zone (Figure 3). Geochemical silt, soil and rock sampling was performed at a number of locations distal to the Main Zone.

The results of these recent programs completed by Hawthorne are summarized in Section 9.2 (Results of 2007 Programs) and Section 9.4 (Results of 2008 Programs).

6.3.2 2011

In 2011 Teslin undertook a geochemical sampling program over three gridded areas; the Kusk Grid between Frasersgold Ck. and the upper MacKay River, Eureka Bowl Grid in the vicinity of the Northwest Zone and the 18ppm Au Grid in the lower section of Eureka Brook. In total 27 line km were sampled resulting in 565 soil samples, 7 rock grab samples and 6 silt samples. The results are summarized in Section 9.5 (Results of 2011 Program).

6.3.3 2015

In the spring of 2015 Eureka completed 4 line km of geochemical soil sampling in the area of the 18ppm Au Grid resulting in 77 soil samples. The results are also summarized in Section 9.6 (Results of 2015 Program).

7. Geological Setting and Mineralization

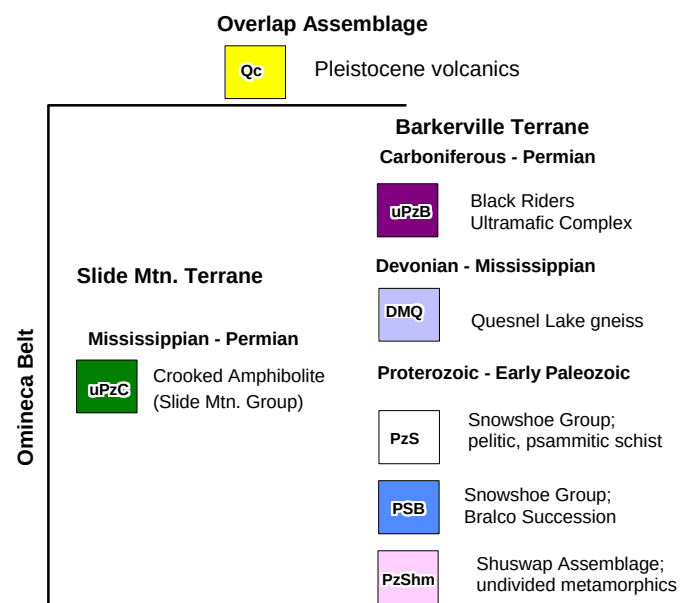
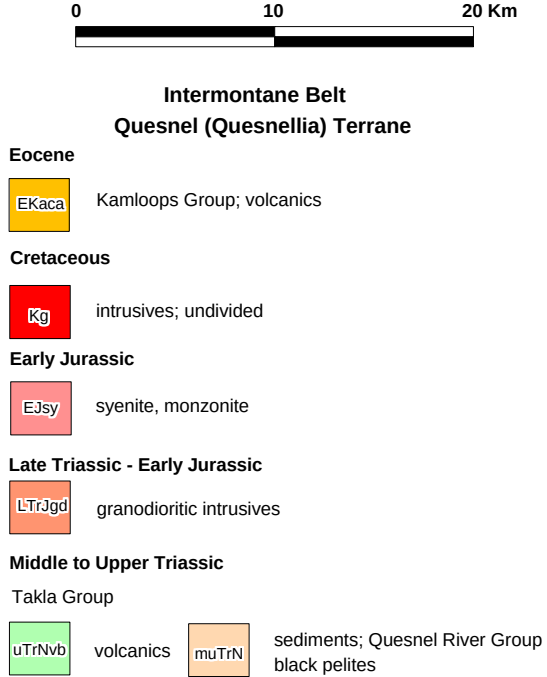
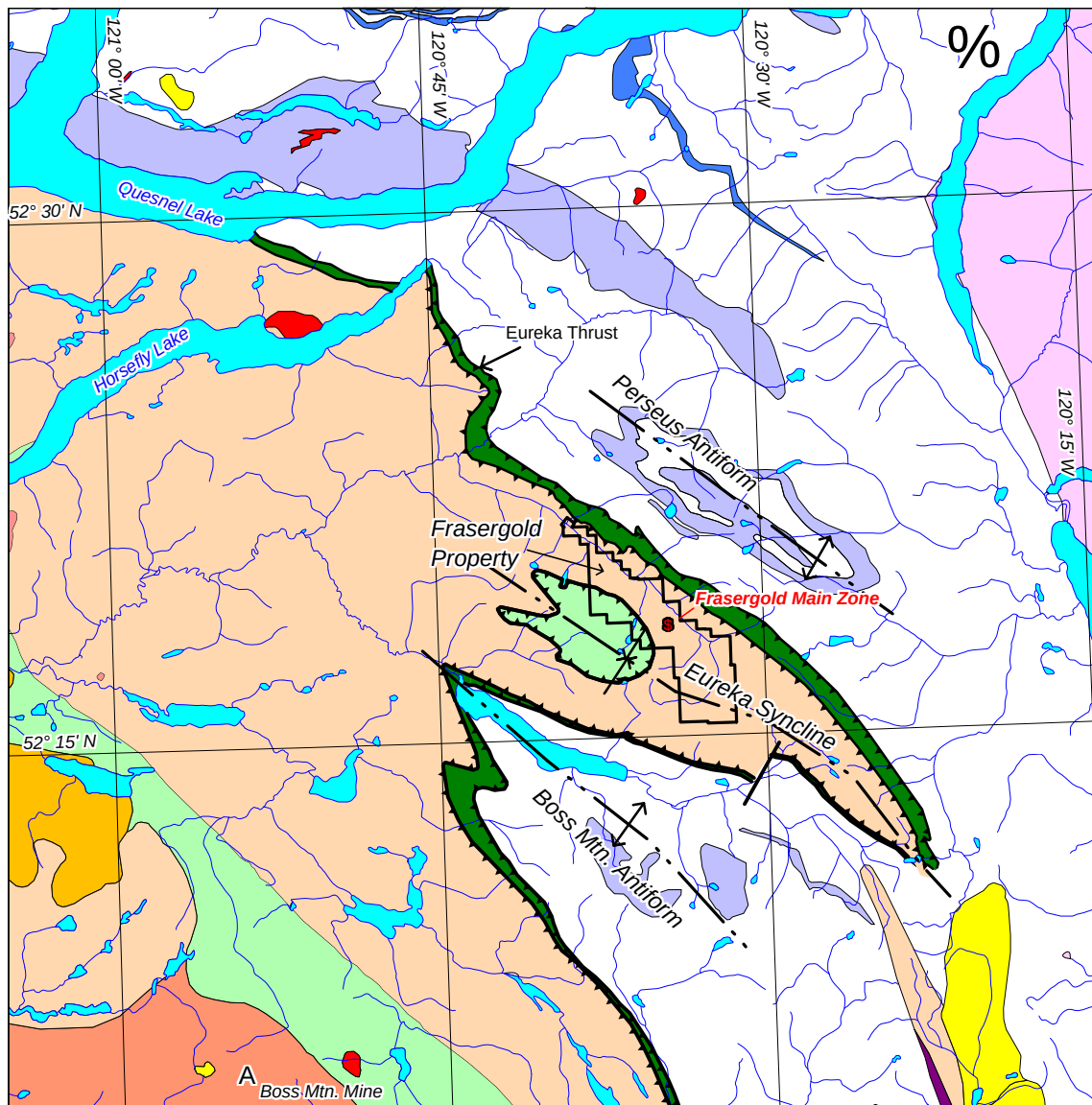
7.1. Regional Geology

The Frasersgold property straddles the boundary between two major tectonic belts of the Canadian Cordillera: the Omineca Tectonic Belt lies to the east of the property and the Intermontane Belt on the west and central portions of the property. Three regional tectonostratigraphic terranes are present; Kootenay, Slide Mountain and Quesnellia terranes. The Slide Mountain and Quesnellia terranes form part of the Intermontane Belt which has been accreted eastward onto the Kootenay terrane of the Omineca Belt. The Eureka Thrust makes the tectonic boundary between these two Belts.

In the project area, Figure 4, the Omineca Tectonic Belt is represented by Hadrynian to early Paleozoic quartz-mica schists and gneisses of the Snowshoe Group. These make up part of the Kootenay terrane; pericratonic, intensely deformed, variably metamorphosed rocks which appear to be stratigraphically related to ancestral North America. The Omineca Tectonic Belt is noted for its prevalence of gold and tungsten mineral occurrences such as the Barkerville gold mining camp to the north.

Pennsylvanian and Permian amphibolite, chlorite schist and chlorite-epidote schist make up the Crooked Amphibolite which structurally overlies the Snowshoe Group metasedimentary rocks above the Eureka Thrust. The Crooked Amphibolite is part of the Slide Mountain terrane of the Intermontane Belt. In general, rocks of the Slide Mountain terrane are oceanic marginal basin volcanics and sediments of Devonian to late Triassic age.

Overlying the Crooked Amphibolite are sediments of the middle to late Triassic Quesnel River Group and volcanics of the late to middle Triassic Takla Group. The Quesnel River and Takla Groups make up part of the Quesnellia terrane of the Intermontane Belt. Rocks of the Quesnel River Group are upper Triassic



Regional Geology

DWG: 521-15-5

15/06/2015

Figure: 4

and lower Jurassic arc volcanics, volcanoclastics and comagmatic intrusive rocks overlain by Jurassic arc-derived clastics (Monger et al, 1982). The Quesnellia terrane is noted for a prevalence of copper, gold and molybdenum mines and showings, such as those at Highland Valley, Boss Mountain, QR and Cariboo Bell - Mt. Polley (Panteleyev et al, 1996).

Jurassic and Cretaceous granite, granodiorite, quartz monzonite, syenodiorite and diorite intrude into rocks of the Omineca and Intermontane Belts. Small exposures of Tertiary and recent olivine basalt are also present in the region.

The dominant structures in the region are the northwest trending, shallowly plunging, Eureka Syncline and Perseus Anticline (Campbell, 1971). South of the Eureka Syncline lies the Boss Mountain anticline, which is similar in many respects to the Perseus anticline (Campbell, 1989). The Eureka syncline displays well developed, northeast striking, near vertical extension joints clearly manifested in the drainage pattern. Southeast of the project area, towards the nose of the syncline, the syncline becomes overturned to the southwest with axial planes dipping steeply northeast. In addition, northeast of the MacKay River the northeast limb is overturned to the southwest, although the syncline is upright in the area of the property.

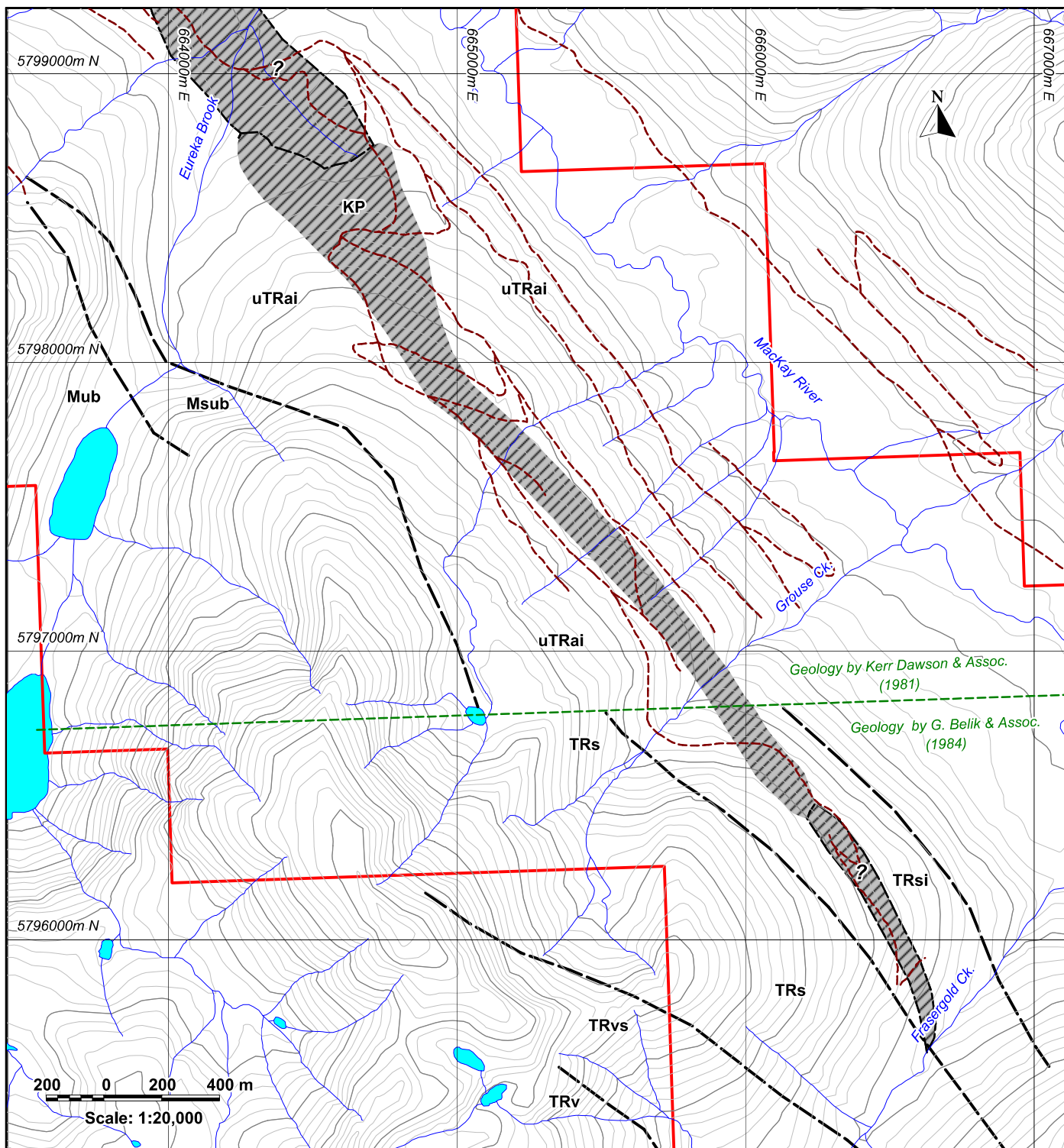
Basic volcanic rocks of the Takla Group occupy the core of the Eureka Syncline where basalt, augite porphyry flows, tuffs and volcanic breccias have been metamorphosed to a low grade. The contact with the underlying sediments of the Quesnel River Group has been described as a fault (Bloodgood, 1987).

Regional dynamothermal metamorphism affected all of the pre-Tertiary rocks in the area. The lowest grades are seen along the Horsefly River road where clastic textures are preserved. In the Eureka Syncline, the metamorphic grade of all units increases towards the Perseus and Boss Mountain anticlines. Large areas reach medium grades of metamorphism (amphibolite facies) and some rocks in the cores of the anticlines reach the kyanite-staurolite-fibrolite zone and are associated with pegmatites (Campbell, 1971). The age of the folding and metamorphism is considered to be Jurassic to early Cretaceous.

The northwest trending MacKay River valley marks a major zone of vertical or near vertical fracturing. Here the upper Triassic Quesnel River Group is sandwiched between two more competent units; younger intrusives and volcanoclastics to the south and older amphibolites, schists and gneisses to the north and east. In order to accommodate the transition from upright to overturned limb, shearing and faulting have been concentrated in the incompetent phyllite units striking along the valley (Campbell, 1989).

7.2. Property Geology

The geology of the Frasersgold property has been mapped in by Kerr Dawson & Associates (1981), G. Belik & Associates, 1984, Read (1988), Campbell (1989), Kerr and Campbell (1990) and Campbell et al (1991) and is shown in Figure 5. The following discussion is based on this information.



Geology by Kerr Dawson & Assoc. (1981)

KP - knotted phyllite, black phyllite with abundant carbonate porphyroblasts
uTRai - black phyllite, greenschist, quartz sericite schist
Mub - coarse grained pyroxene plagioclase sill
Msub - highly sheared, marginal phase of sill

Geology by G. Belik & Assoc. (1984)

TRv - greenstone, breccia tuff
TRvs - interbedded greenschist, black phyllite and quartzite
TRs - black phyllite with minor limestone and sandstone
TRsi - black phyllite with abundant iron carbonate knots

Eureka Resources Inc.

Frasergold Project

Cariboo Mining Division, B.C.

Property Geology

DWG: 521-15-6

June 15, 2015

Scale 1:20,000

Figure: 5

Earth Resource Surveys Inc.
ERSi

The property is located within the middle to upper Triassic Quesnel River Group and is underlain by a thick sequence of dark grey to black; lustrous phyllites with iron-bearing carbonate porphyroblasts, locally referred to as the “knotted phyllite”. The knotted phyllite contains minor intercalations of grey phyllite, thin lenses and layers of light grey massive to phyllitic siltstone and thin lenses of grey silty limestone. Underlying this sequence is a grey silty and locally calcareous phyllite. At the base of the knotted phyllite unit in the Northwest extension area of the Frasergold property lies a unit of pale carbonate-quartz-sericite-chlorite schist which are possibly metavolcaniclastics. The rock units described below occur in descending stratigraphic order.

7.2.1 Silty limestone

Medium grey, vaguely bedded silty limestone occurs as lenses in all rock types except the silty calcareous phyllite where it lies within a few meters of the top. Lenses of the grey silty limestone are generally less than one meter thick. The medium grey color and weakly bedded nature distinguish this unit from the silty calcareous phyllite.

7.2.2 Siliceous siltstone

These rocks are distinguished from other units by their light grey color and coarse sandy texture. Thin sections have determined that the protolith was most likely a quartz-rich sandstone or quartzite. The original bedding features are distinct, defined by thin grey porphyroblastic and non-porphyroblastic phyllite partings. Carbonate is commonly absent. The main horizons occur as erratic lenses within the knotted phyllite with thicknesses ranging from one to 25 meters.

7.2.3 Grey phyllite

Medium to dark grey in color, this unit is distinguished from the knotted phyllite by the lack of carbonate porphyroblasts. In drill core the apparent thickness ranges from one to 23 meters but averages less than five meters. The thinness and lenticular nature of this unit have resulted in an apparent lack of continuity even between closely spaced drill holes. The distribution of this unit, combined with that of the siliceous sediment, divides the knotted phyllite into upper and lower packages.

7.2.4 Knotted phyllite

This unit is medium to dark grey, weakly to moderately laminated phyllite with 10 to 20% Fe-carbonate porphyroblasts varying in size from two millimeters to two centimeters. The porphyroblasts are elongated within the foliation planes and lie in a groundmass of muscovite, quartz, colorless carbonate, minor albite and graphite. This unit is at least 200 meters thick and has been divided by Read (1988) into two packages: a lower package, 40 to 75 meters thick, which lies between the silty calcareous phyllite and a zone rich in lenses and layers of grey phyllite and siliceous metasediment; and an upper package of more than 75 meters thickness which lies above the zone rich in lenses and layers of grey phyllite and siliceous

metasediment. Surface weathering gives this rock unit a distinct brown mottling texture. In fresh rock the texture is recognized but is not as easily distinguished as it is on a weathering surface.

7.2.5 Carbonate-quartz-sericite-chlorite schist

About 30 to 40 meters of carbonate-quartz-sericite-chlorite schist lie between the knotted phyllite and silty, calcareous phyllite in the Northwest Extension. The rock unit is light green in color, is well banded to finely laminated and has a coarse granular aspect, locally resembling quartz pebble tuffs. White coarse grain clasts of carbonate, probably dolomite, are present in a matrix of sericite and clinocllore and quartz is present as fine grain inclusions in the carbonate grains and also in fine laminations. These rocks are considered by Campbell (AR 15715) to be the metamorphosed product of impure calcareous sediments and by others to be metamorphosed volcanoclastics.

7.2.6 Silty, calcareous phyllite

This rock is typically a light grey to dark grey color, is well-laminated and has calcareous horizons to 30 centimeters in thickness. The calcareous sections which comprise a quarter to a third of the unit distinguish this unit from the grey phyllite. Drill core and surface exposures indicate a thickness of at least 25 meters with no base exposed.

7.3. Structural Geology

Gold mineralization at the Frasergold property is hosted by a fine-grained turbiditic sequence that is dominated by black carbonaceous phyllite with local thin interbeds of metasiltstone, and more rarely, fine grained metasandstone. The sequence forms part of the Middle to Late Triassic age Nicola Group of the Quesnel Terrane that lies directly on the Crooked Amphibolite, which in turn is structurally emplaced on the underlying Barkerville Terrane by the Eureka Thrust (Panteleyev et al., 1996; Bloodgood, 1987, 1990). All of these features are affected by the regional, northwest trending Eureka syncline, an open, structurally late fold, the axis of which passes beneath Eureka Peak to the southwest of the deposit. The deposit lies on the northeast limb of the syncline. Bedding (S_0) in the Frasergold deposit area dips mainly moderately to shallowly to the southwest, although varies locally to northeast dips due to the effects of D_2 folds (Rhys, 2007). Regional mapping indicates that the overall sequence is upright, although some early, relatively minor isoclinal folds are present. The Frasergold deposit occurs on the northeast limb of the regional Eureka Syncline (Bloodgood, 1987, 1990).

Regionally, two dominant phases of deformation affect rocks in the Quesnel Terrane. These are equivalent to the principal D_1 and D_2 deformation events recognized at Frasergold. Phase 1 structures (D_1) produced a penetrative slaty to phyllitic cleavage (S_1) that is axial planar to generally east to northeast verging, tight to isoclinal, generally northwest trending F_1 folds and shear zones. The phase 1 (D_1) event is associated with emplacement of the Nicola Group rocks in the Quesnel Terrane onto the Barkerville Terrane along the Eureka Thrust (Panteleyev et al., 1996; Rees; 1987; Bloodgood, 1987). D_1

was accompanied by, and locally outlasted by, peak regional metamorphism. Phase 2 structures regionally include the Eureka Peak syncline, which openly refolds both the earlier S_1 foliation and associated folds and the D_1 Eureka Thrust (Bloodgood, 1987). Phase 2 structures regionally correspond with the D_2 event documented here, and the associated S_2 foliation at Frasergold, which is axial planar to the Eureka Syncline (Bloodgood, 1987, 1990, Rhys, 2007). Late north to northeast trending crenulation cleavage (S_3) and kink bands formed later and are also locally evident in outcrops at Frasergold.

7.4. Quartz Veins and Gold Mineralization

Gold mineralization on the Frasergold property occurs within, or is spatially associated with, sets of white quartz + Fe-carbonate + muscovite + pyrite veins that are developed in the knotted Fe-carbonate porphyroblastic carbonaceous phyllite unit. The veins form complex sets and networks that are developed in concentrated zones several meters to tens of meters wide. Zones of veining are generally stratabound and overall have shallow to moderate westerly dips which are usually parallel to S_0/S_1 . Apart from the presence of the carbonate porphyroblasts in the knotted phyllite unit and widespread Fe-carbonate alteration of siltstone and sandstone lamina which extend well outside the immediate areas of veining, no visually obvious alteration affects the host phyllitic sediments at Frasergold in areas of gold mineralization, although slightly elevated disseminated pyrite content was observed near some veins (Rhys, 2007).

Two general styles of veining are recognized in mineralized areas at Frasergold; ribbon veins and dilational veins (Rhys, 2007).

Ribbon veins refer to sets of approximately S_0/S_1 parallel quartz veins and veinlets which vary in width between stringers to veins up to 0.3 m thick that are traceable generally in crosscuts for several meters. These are by far the most abundant style of veining in the underground workings and surface exposures. They comprise blocky, recrystallized white quartz with minor Fe-carbonate. Muscovite may form narrow silver to grey colored selvages on vein margins. These veins form subparallel sets that are affected both by D_1 and D_2 strain. The veins are often transposed and boudinaged in the plane of S_1 , locally with the development of internal S_1 -parallel stylolites. Like S_1 , the veins are affected by F_2 folds, and vary in orientation across F_2 fold limbs.

Dilation veins refer to larger, 15-50 cm thick quartz-carbonate veins that are developed at moderate to high angles to S_0/S_1 . These veins generally have higher Fe-carbonate content than the smaller concordant ribbon veins but contain the same blocky recrystallized white quartz fill. Yellow-brown, coarse, blocky Fe-carbonate typically occurs as clots, bands and selvages on veins and contains disseminated pyrite. Although thicker than the narrower and more abundant concordant veins, these veins are also discontinuous and may terminate laterally or vertically. Rare fibrous quartz-Fe-carbonate textures on vein margins that are indicative of dilational carbonate-quartz growth at high angles to vein walls and sharp vein contacts with wall rock suggest dilational opening (Rhys, 2007), not replacement of wall rock textures

as has previously been suggested (Campbell et al, 1991). These thicker veins are openly folded by folds with S_2 foliation axial planar. Veins of this style are reported by Campbell et al. (1991) to be of higher grade than the S_1 foliation parallel veins, probably due to their generally higher Fe-carbonate-pyrite content.

The concordant S_0/S_1 ribbon veins and the thicker, discordant veins join one another without crosscutting relationships, have the same mineralogy and are equally as deformed, suggesting that they are all part of a single vein generation resulting from a single veining episode (Rhys et al, 2009). Studies are currently ongoing to further elucidate the structural controls of the vein system and its gold mineralization.

8. Deposit type

Gold mineralization at the Frasergold property is classified as an orogenic lode-gold deposit type. Lode-gold deposits generally occur in metamorphic terranes delineated by a regional fault system. The deposits are often regularly spaced along a zone hundreds of kilometers long, although economic ores may only be present over a short distance. Many of the most productive ore bodies of this type exceed one kilometer in vertical extent with little change in gold grade with depth (Goldfarb et al, 1986).

There is a spatial association of this type of gold deposit with greenschist grade of metamorphism, possibly due to the large fluid volume required during the amphibolite and/or greenschist transition (Goldfarb et al, 2005). First order faults provide conduits for massive volumes of auriferous fluids to migrate through the rocks while second and third order faults are the sites of mineral deposition. World-class gold deposits in metamorphic terranes are generally 2 to 10 kilometers long, contain 2 - 5% sulphide minerals and have gold to silver ratios from approximately 5:1 to 10:1. Examples of this deposit type include Kirkland Lake, Hollinger-McIntyre, Campbell-Red Lake and Bralorne-Pioneer camps.

Table 7.1 outlines the characteristics of the Frasergold deposits as well as they are known. These characteristics place the Frasergold mineralization in the class of orogenic gold deposits as defined by Groves et al (1998) and reviewed by Goldfarb et al (2001 and 2005) and Groves et al (2003). This class of gold deposits has been described and reported on at length by numerous authors. The deposit type has worldwide significance, accounting for 12 of the 32 gold camps in the world having production or resources exceeding 600 tonnes of gold or approximately 19 million ounces (Rhys and Lewis, 2004).

There are at least two other similar gold occurrences along the eastern margin of the Quesnellia terrane in central British Columbia; at Spanish Mountain (Minfile No. 093A043), about 54 km to the northwest, and at Hixon, (MEMPR Minfile Nos. 093G014 and 093G015), about 180 km northwest of the Frasergold property.

The Spanish Mountain occurrence is currently being explored and developed by Spanish Mountain Gold Ltd. The main mineralized zones consist of gold-bearing veins and stockworks in a pyritic and graphitic shaley siltstone. This occurrence differs in at least two respects from that at Frasergold; one, is that

strong carbonate-silica-pyrite alteration, described as listwanitic by Bloodgood (1990), is pervasive. The second is that mineralization probably post-dated the regional metamorphism (Bloodgood, 1990).

The occurrences at Hixon consist of gold-bearing quartz veins and stringers cutting highly altered quartz-sericite schist. The Minfile No. 093G015 occurrence, known as 'Quesnel Quartz', produced 6,438 grams gold and 8,553 grams silver from 2,048 tonnes mined in the period 1932 to 1939 (MEMPR Minfile database). This deposit is associated with a highly sheared and hydrothermally altered, northwest trending zone in which greenstones are in contact with quartz-sericite schists of the Upper Triassic to Lower Jurassic Takla Group. The alteration is described as carbonatization (MEMPR Minfile database).

Table 8.1 Characteristics of the Frasergold Property

Parameter or Feature	Characteristics of Frasergold gold mineralization
host rocks	turbidite with minor volcanoclastic component (Quesnel River Group)
age of host rocks	Upper Triassic
metamorphism	greenschist facies of classic Barrovian-type facies series (argillite, phyllite, siltite, fine grained psammite)
age of metamorphism	Jurassic - early Cretaceous
tectonic setting	within Quesnellia terrane proximal to boundary with Omineca terrane
local structure	situated on upright limb of regional syncline (Eureka syncline)
tectonics	predominant structure, the Eureka syncline, is compressive with interformational slip, shear and boudinage subparallel to the compositional and/or penetrative foliation evident on its limb. This is manifested in the ribbon-vein zone(s) that comprise the Frasergold zone of mineralization.
spatially associated intrusive rocks	no intrusive units that are spatially related to the Frasergold mineralized belt are recognized. The nearest intrusive is a small stock of granodioritic composition of possible Cretaceous age in the core of the Eureka syncline, ~ 1 to 2 km southwest of the Frasergold zone
deposit mineralogy	quartz-dominant vein sets or system with ~2 - 7% predominantly iron sulphides (mainly pyrite with subordinate pyrrhotite) and up to 20% carbonate minerals. The sulphides and carbonates are commonly segregated on the margins of quartz veins. Native gold is not uncommon. Lack of association of gold with other precious, base or toxic (As, Sb, Hg) metals.
deposit structure	vein sets are controlled by structures developed largely contemporaneous to the regional metamorphism and deformation
alteration	hydrothermal alteration adjacent to the mineralized vein sets in the form of bleaching or development of phyllosilicates is not recognized

9. Exploration

Exploration activities conducted by other companies are described in detail in Section 6.1, Prior to 2007 and Section 6.2, Historical Drill Results. In October, 2006 Hawthorne entered into an Option Agreement with Eureka on the Frasergold property and undertook two exploration programs in 2007 and 2008.

Details of the sampling method and approach are contained in Section 11 (Sample Preparation, Analyses and Security).

9.1. 2007 Program

From January to April, Hawthorne management began planning for completion of an airborne magnetic, electromagnetic and radiometric geophysical survey over the Frasergold property by reviewing property boundaries, geological structure and potential geophysical survey companies.

During the spring of 2007, Hawthorne began construction of a large 30-person exploration camp, utilizing a compound of trailers for accommodation, drilling a water well for drinking and camp water, constructing a sewage field, and water lines from the well. The sewage system included a septic system for all waste water. The camp facility consisted of accommodations for up to 30 people within a trailer unit. Other buildings included a large core logging and core cutting facility with the geology office attached, generator buildings to house two large generators, a large garage for heavy equipment, and smaller ancillary buildings to store tools and equipment. A large fuel system consisted of a series of above ground fuel storage tanks with concrete berms in place for safety.

Hawthorne exploration crews and geologists first arrived on site in mid June 2007 and worked throughout the season until mid December. Their initial work entailed a physical review of the property and ground-truthing historical drill holes, along with reconnaissance soil and rock sampling and completion of minor trenching programs. The crew also reviewed the historical underground workings of the Main Zone and oversaw the rehabilitation of the workings, including rock bolting. Rehabilitation of the adit was performed by D.R. Gunning, P.Eng. of D.R. Gunning Consulting, Langley, B.C. and C. Blagdon (miner/shiftboss) of Goldbridge, B.C.

On completion of the rehabilitation work, the crews took channel samples throughout the workings and supervised underground bulk sampling. The bulk samples were collected in large pails and sent to Process Research Associates (PRA), while channel samples were sent to International Plasma Labs Ltd. (IPL) in Richmond, B.C.

In addition to the support staff, four heavy equipment contracting companies were used at various times throughout the exploration program, including:

- 1. G. & S. Logging Co. Ltd., which supplied a 2054 Excavator and 850j Caterpillar.

- 2. Harpers Lake Development Ltd., which assisted with building drill access roads, hauling fill for the roads and digging trenches.
- 3. Can-Co Contracting, which hauled gravel and rock for road construction.
- 4. Horsefly Bay Construction Ltd., which provided a CAT 312 Excavator, D7 CAT and Dump Truck for drill access road construction, drill pad construction, hauling fill for the roads and snow clearing late in the year.

In early September SCS Diamond Drilling of Merritt, B.C. used one diamond drill to conduct a drill program on the property with the program running from September 3 through to December 10 and consisting of sixteen (16) HQ diamond drill holes.

In conjunction with the drilling, Aeroquest International Advanced Airborne Geophysics completed a helicopter-borne geophysical survey on the property, with the program running from September 6 through to the 13.

Rathbone & Goodrich, B.C. & Canada Land Surveyors ("Rathbone"), based in Williams Lake, performed a number of survey related tasks. This included establishing sufficient ground control to convert the historic work in the local Mine Grid and in NAD27 to NAD83 coordinates, and establishing ground control points and aerial photo target points for a subsequent aerial topographic survey.

Eagle Mapping Services Ltd. of Port Coquitlam, B.C. acquired high resolution aerial photographs and generated orthophotos and a topographic map with 5m contour interval over most of the property, and 1m contour interval over the primary exploration area of the known mineralized zones.

9.2. Results of 2007 Programs

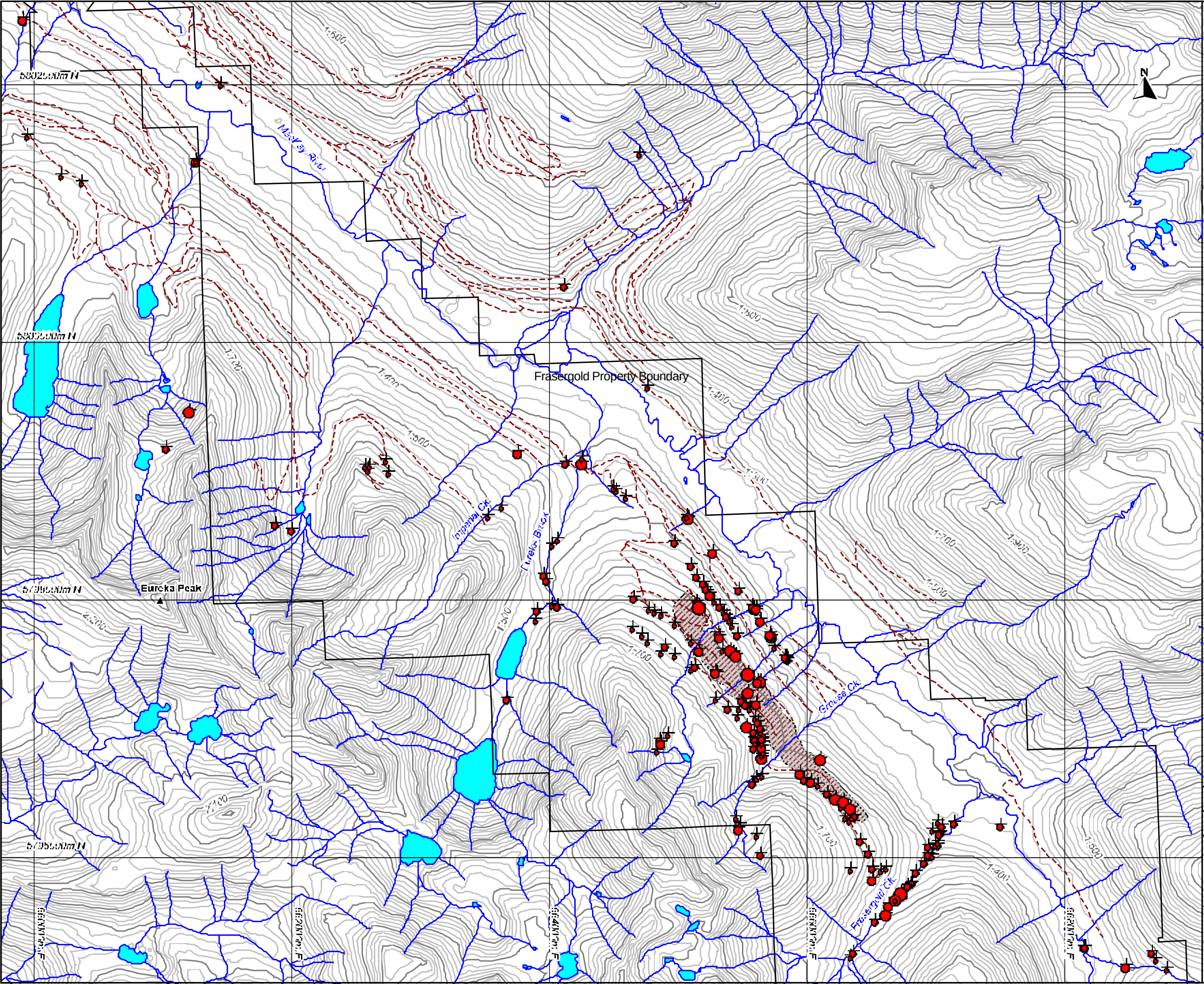
9.2.1 Surface Rock Sampling

Figure 6 is a plot of gold values (ppm) in the surface rock samples which are described in Appendix I. Of the 198 rock samples, only 10 reported in excess of 0.20 g/t Au. All of these were knotted phyllites with quartz veining. The maximum value reported was sample 135844 located at the northwest end of the known limits of the knotted phyllite unit and which reported 5.17 g/t Au.

9.2.2 Trenching

Seven trenches, totaling 345 meters in length, were selected based on historical drill hole data and slope stability. These were dug to identify structure and lithology, with the view of replicating or verifying historical drill data, as well as to provide for new potential areas for diamond drill hole exploration. The trench locations are shown Figure 7. The trench sample descriptions and assay results of these trench samples are listed in Appendix II.

The trenches were dug to varying depths of 0.5 - 1.5 meters, depending on local overburden thickness. The location, elevation, length, depth, width, approximate material removed from each trench, as well as total number of channel samples collected from each trench is as follows:



UTM 10 NAD83 Contour interval = 20m

Au (ppm)

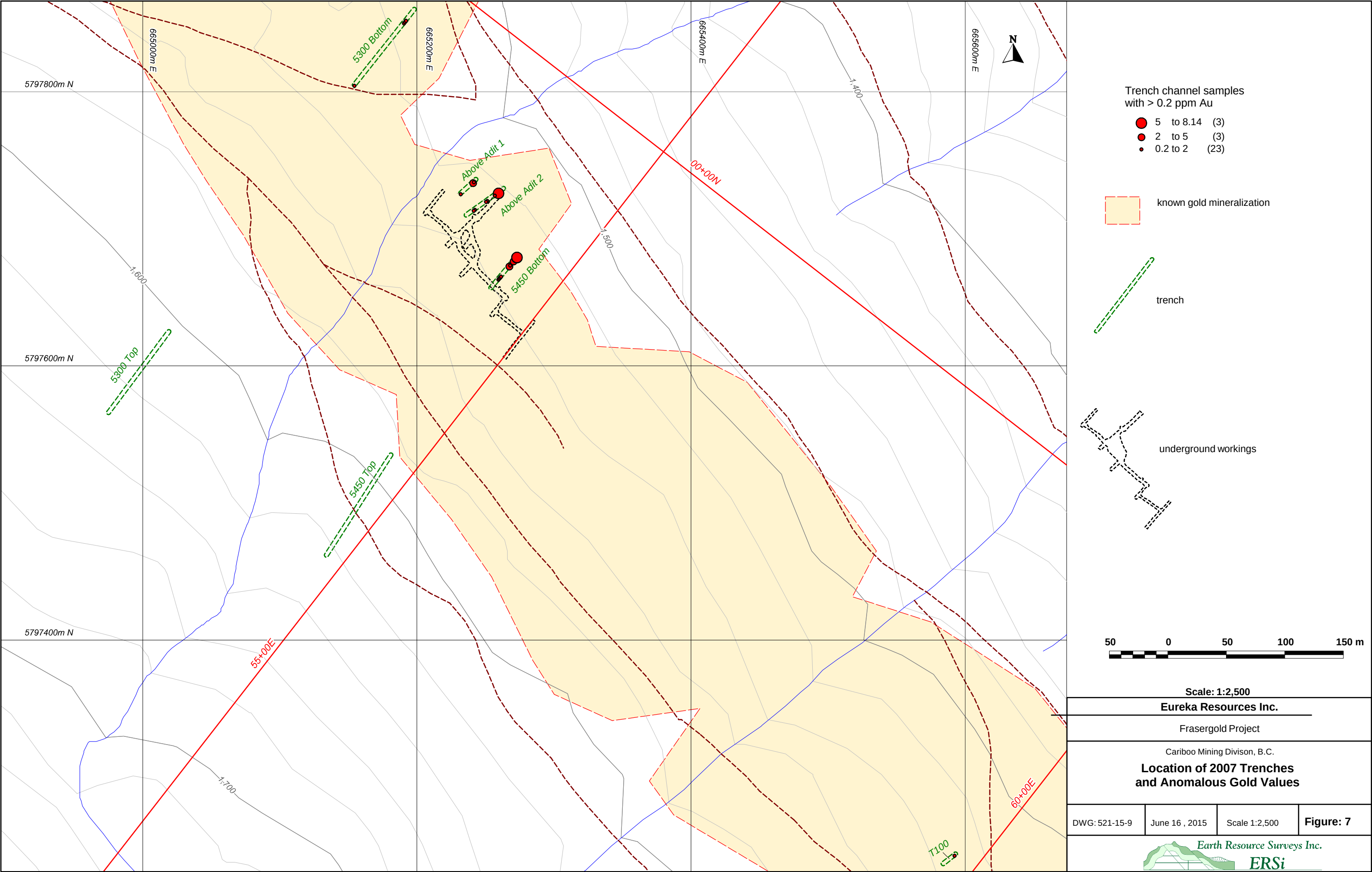
- 0.5 to 5.17 (3)
- 0.05 to 0.5 (17)
- 0.02 to 0.05 (23)
- 0.01 to 0.02 (73)
- 0 to 0.01 (82)

known gold mineralization

500 0 500 1000 1500 m

Scale: 1:30,000

Eureka Resources Inc.			
Frasergold Project			
Cariboo Mining Division, B.C.			
2007 Surface Rock Samples Au (ppm)			
DWG: 521-15-8	June 16, 2015	Scale 1:30,000	Figure: 6
			



UTM 10 NAD83 Contour interval = 20m

Trench 53+00 (Top), located from 5797566m N and 664975m E, elevation 1619.597m to 5797625m N and 665019m E, elevation 1594.017m, total length of 75 meters, average depth of 1.5 meters, average width of 5m, approximately 562.5 m³ of material removed, total number of channel samples collected was 74.

Trench 53+00 (Bottom), located from 5797805m N and 665154.4m E, elevation 1523.053m to 5797860m N and 665198.1m E, elevation 1502.73m, total length of 70 meters, average depth of 1.5 meters, and average width of 5m, approximately 525m³ of material removed, total number of channel samples collected was 57.

Trench Above Adit 1, located from 5797726m N and 665232m E, elevation 1519.689m to 5797736m N and 665243.5m E, elevation 1515.907m, total length of 30 meters, average depth of 0.5 meters, and average width of 35 meters, approximately 212.5m³ of material removed, total number of channel samples collected was 16.

Trench Above Adit 2, located from 5797710m N and 665236m E, elevation 1522.376m to 5797729m N and 665256.9m E, elevation 1513.976m, total length of 30 meters, average depth of 0.5 meters, and average width of 35 meters, approximately 212.5m³ of material removed, total number of channel samples collected was 29.

Trench 54+50 (Top), located from 5797462m N and 665134m E, elevation 1607.143 to 5797535m N and 665180.8m E, elevation 1581.312m, total length of 90 meters, average depth of 1.5 meters, and average width of 5m, approximately 675m³ of material removed, total number of channel samples collected was 45.

Trench 54+50 (Bottom), located from 5797657m N and 665254m E, elevation 1532.644m to 5797680m N and 665273m E, elevation 1519.563m, total length of 35 meters, average depth of 1.5 meters, and average width of 5m, approximately 262.5m³ of material removed, total number of channel samples collected was 31.

Trench T100, located from 5797237m N and 665584m E, elevation 1524.633m to 5797244m N and 665593m E, elevation 1519.976m, total length of 15 meters, average depth of 0.5 meters, average width of 3m, approximately 22.5m³ of material removed, total number of channel samples collected was 16.

All of the samples were of knotted phyllite. Of the 269 samples 29 reported in excess of 0.20 g/t Au and these are highlighted in Figure 7. The highest value reported was 8.13 g/t Au collected in the trench above the adit entrance.

9.2.3 Underground Channel Sampling

Following the initial rehabilitation of the underground workings, the exploration crew laid out a well defined underground sampling site program, using a predetermined location at the portal entrance to begin measurements from. From this predetermined measured location, a total of 115 sample sites were

chained off along the walls of the workings at approximately every two meters, depending on safety and underground conditions as well as access. A total of 211 underground channel samples were collected from the 115 sites, located in Figure 8.

Appendix III lists the assay results of these samples. Two conventional fire assays and a single screened total metallic assay procedures were performed on each of the samples. Figure 9 compares the average of the two fire assays versus the screened total metallic assay. Except for one high grade sample (average fire assay 95.305 g/t Au vs screened total metallic assay 37.77 g/t Au) the assay results approach a 1:1 relationship.

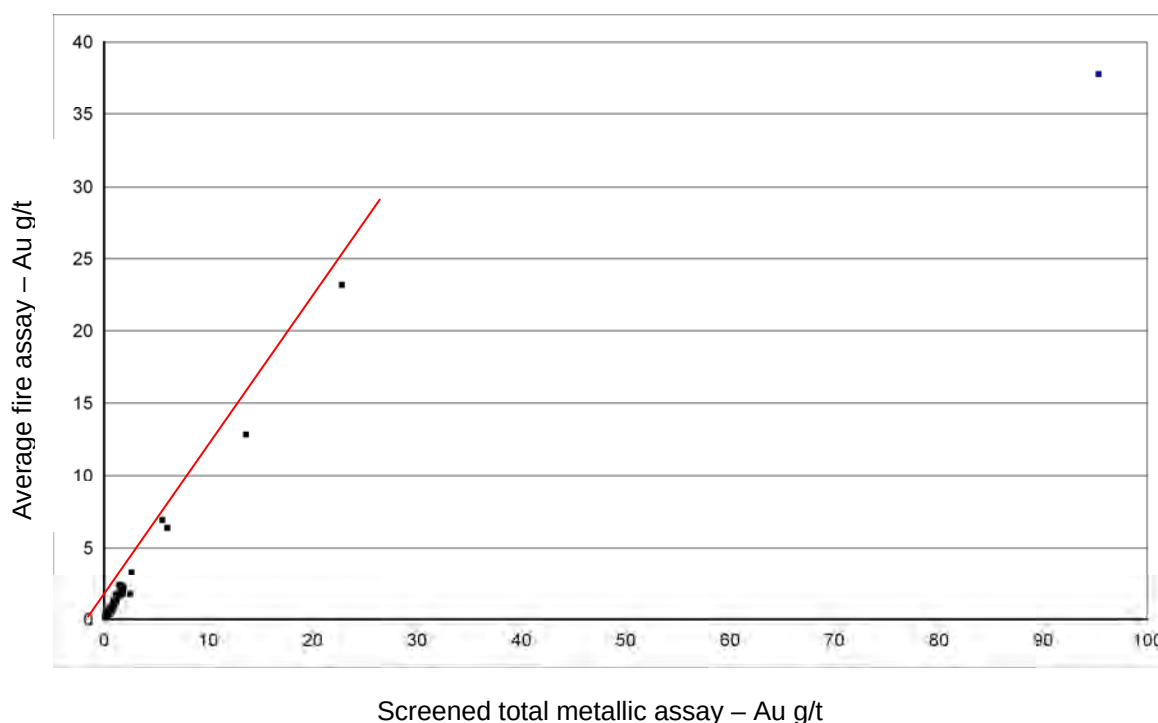
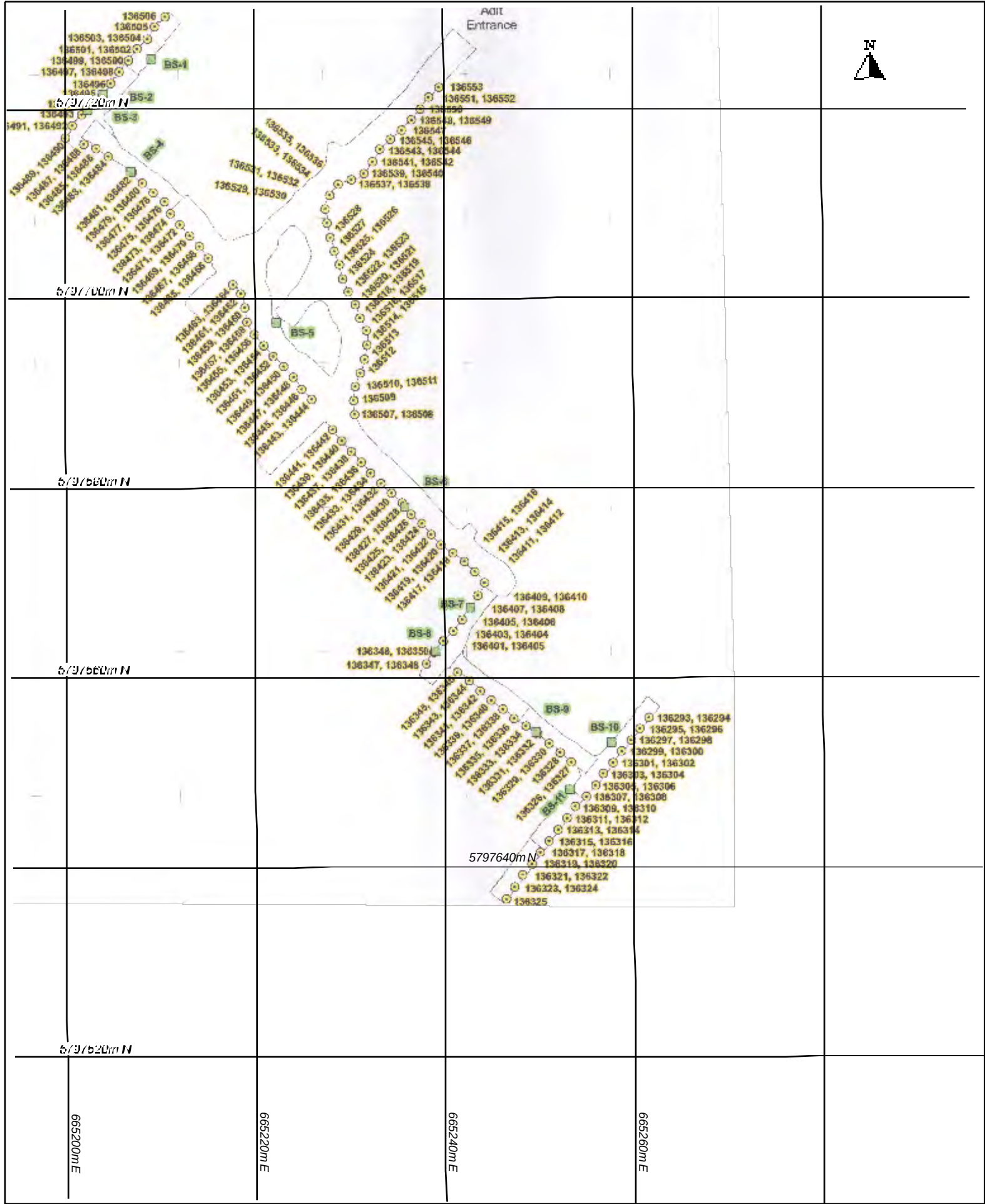


Figure 9. Underground channel samples; average fire assay vs screened total metallic assay, Au g/t

The results of the channel sampling within the underground workings to validate the results obtained from surface drilling, including a number of high grade gold samples. The results indicate good continuity of gold mineralization along strike to the northwest on an azimuth of approximately 310°, with the width of the zone of interest being approximately 30 meters.



underground adit outline



2007 bulk sample



2007 channel sample



Eureka Resources Inc.			
Frasergold Project			
Cariboo Mining Division, B.C.			
2007 Underground Work Bulk Sampling and Channel Sampling			
DWG: 521-15-10	June 17, 2015	Scale 1:500	Figure: 8

9.2.4 Underground Bulk Sampling

Eleven bulk samples were collected from the underground workings, distributed throughout the underground workings and recorded as BS-1 through BS-11. The location of the bulk sample sites were recorded with regard to adit reference, as well as in NAD83 coordinates and shown in Figure 8.

The objective of the bulk sampling analysis was to determine the average head grade through the mineralized zone exposed in the Main Zone adit. Two analytical techniques were used; cyanide leach assays performed on 8 samples (Table 9.3.4a) and metallic Au assaying performed on 11 samples (Table 9.3.4b).

Table 9.3.4a Assay Results of 2007 Bulk Sampling - Head Grade by Cyanide Leaching Method

Sample	Sub-sample	Weight		Slurry	Au Assay			Head Grade (Au g/t)
		Slurry (g)	Solids (g)	Density % solid	PLS (mg/L)	mg	Solids (g/t)	
BS1	CutA	732	291	39.73	0.02	0.0088	0.04	0.07
	CutB	733.41	288	39.26	0.02	0.0089	0.04	0.07
	CutC	757.89	300	39.64	0.02	0.0091	0.04	0.07
	Subtotal		879		0.02	0.0269		
	Total	126700	50099	39.54				0.07
BS2	CutA	743.94	353	47.45	1.14	0.4457	0.10	1.36
	CutB	729.75	346	47.37	1.15	0.4417	0.11	1.39
	CutC	770.71	362	46.91	1.16	0.4746	0.14	1.45
	Subtotal		1060		1.15	1.3620		
	Total	103300	48804	47.24				1.40
BS3	CutA	716.88	220	30.75	0.1	0.0496	0.38	0.61
	CutB	682.1	211	30.86	0.1	0.0472	0.39	0.61
	CutC	739.78	228	30.85	0.11	0.0563	0.43	0.68
	Subtotal		659		0.10	0.1531		
	Total	127100	39172	30.82				0.63
BS4	CutA	790.94	341	43.07	2.84	1.2789	0.99	4.74
	CutB	796.68	343	43.05	2.86	1.2976	1.00	4.78
	CutC	761.93	328	43.11	3.04	1.3178	1.00	5.01
	Subtotal		1012		2.91	3.8943		
	Total	112500	48459	43.07				4.84
BS5	CutA	769.12	288	37.49	0.59	0.2837	4.61	5.59
	CutB	801.65	301	37.59	0.64	0.3202	4.34	5.40
	CutC	790.28	296	37.50	0.65	0.3210	4.23	5.31
	Subtotal		886		0.63	0.9249		
	Total	129600	48635	37.53				5.43
BS6	CutA	738.25	294	39.79	0.24	0.1067	0.42	0.78

Sample	Sub-sample	Weight		Slurry	Au Assay			Head Grade (Au g/t)
		Slurry (g)	Solids (g)	Density % solid	PLS (mg/L)	mg	Solids (g/t)	
	CutB	796.43	317	39.86	0.25	0.1198	0.40	0.78
	CutC	776.72	307	39.57	0.23	0.1080	0.39	0.74
	Subtotal		919		0.24	0.3344		
	Total	122750	48778	39.74				0.77
BS7	CutA	833.37	332	39.81	0.12	0.0602	0.62	0.80
	CutB	822.72	325	39.47	0.12	0.0598	0.64	0.82
	CutC	846.38	334	39.41	0.12	0.0615	0.61	0.79
	Subtotal		990		0.12	0.1815		
	Total	122350	48407	39.56				0.81
BS8	CutA	817.15	306	37.48	0.02	0.0102	0.04	0.07
	CutB	813.18	305	37.50	0.02	0.0102	0.04	0.07
	CutC	821.89	307	37.36	0.02	0.0103	0.04	0.07
	Subtotal		918		0.02	0.0307		
	Total	121350	45444	37.45				0.07
BS9	CutA	792.9	297	37.42				0.00
	CutB	796.66	296	37.12				0.00
	CutC	789.41	298	37.70				0.00
	Subtotal		890		0.00	0.0000		
	Total	122650	45889	37.41				0.00
BS10	CutA	829.56	321	38.69				0.00
	CutB	816.56	318	38.95				0.00
	CutC	830.4	321	38.70				0.00
	Subtotal		960		0.00	0.0000		
	Total	122400	47469	38.78				0.00
BS11	CutA	787.8	297	37.64				0.00
	CutB	785.37	297	37.86				0.00
	CutC	793.45	299	37.69				0.00
	Subtotal		893		0.00	0.0000		
	Total	129700	48936	37.73				0.00

Table 9.3.4b Assay Results of 2007 Bulk Sampling - Head Grade by Metallic Assay

Sample	Screen Tyler Mesh	Weight (g)	Au	
			g/t	mg
BS1	+150	2.60	0.01	0.000
	-150A	302.56	0.07	0.021
	-150B	355.01	0.03	0.011
	-150C	329.78	0.06	0.020
	-150 subtotal	987.35	0.05	0.052
	Total	989.95	0.05	0.052
BS2	+150	2.85	7.12	0.020
	-150A	293.50	0.64	0.188
	-150B	302.75	1.00	0.303
	-150C	381.15	0.95	0.362
	-150 subtotal	977.40	0.87	0.853
	Total	980.25	0.89	0.873
BS3	+150	18.66	2.02	0.038
	-150A	258.88	1.95	0.505
	-150B	313.65	1.56	0.489
	-150C	397.63	1.93	0.767
	-150 subtotal	970.16	1.82	1.762
	Total	988.82	1.82	1.799
BS4	+150	5.86	211.32	1.238
	-150A	336.77	5.06	1.704
	-150B	318.77	5.18	1.651
	-150C	287.76	4.78	1.375
	-150 subtotal	943.30	5.02	4.731
	Total	949.16	6.29	5.969
BS5	+150	4.74	115.77	0.549
	-150A	328.34	7.34	2.410
	-150B	353.80	8.88	3.142
	-150C	290.44	6.53	1.897
	-150 subtotal	972.58	7.66	7.448
	Total	977.32	8.18	7.997
BS6	+150	19.30	6.46	0.125
	-150A	286.90	0.49	0.141
	-150B	309.57	0.53	0.164
	-150C	340.80	0.52	0.177
	-150 subtotal	937.27	0.51	0.482
	Total	956.57	0.63	0.607

Sample	Screen Tyler Mesh	Weight (g)	Au	
			g/t	mg
BS7	+150	19.34	17.03	0.329
	-150A	292.14	0.89	0.260
	-150B	300.92	0.82	0.247
	-150C	358.95	0.79	0.284
	-150 subtotal	952.01	0.83	0.790
	Total	971.35	1.15	1.120
BS8	+150	13.13	0.01	0.000
	-150A	254.57	0.02	0.005
	-150B	365.76	0.01	0.004
	-150C	352.62	0.02	0.007
	-150 subtotal	972.95	0.02	0.016
	Total	986.08	0.02	0.016
BS9	+150	20.03	0.02	0.000
	-150A	283.48	0.06	0.017
	-150B	392.48	0.10	0.039
	-150C	345.94	0.04	0.014
	-150 subtotal	1021.90	0.07	0.070
	Total	1041.93	0.07	0.070
BS10	+150	22.04	2.56	0.056
	-150A	206.88	0.69	0.143
	-150B	382.50	0.68	0.260
	-150C	379.43	0.72	0.273
	-150 subtotal	968.81	0.70	0.676
	Total	990.85	0.74	0.732
BS11	+150	27.08	0.07	0.002
	-150A	287.17	0.09	0.026
	-150B	341.98	0.09	0.031
	-150C	319.78	0.08	0.026
	-150 subtotal	948.93	0.09	0.082
	Total	976.01	0.09	0.084

The underground bulk sampling program proved very useful and essentially validated previous bulk sampling work that had been historically conducted in the same workings in 1987 and 1991. Bulk sampling in 1991 reported an average grade of 0.027 ounces Au/T equivalent to 0.9257 g/t Au. The cyanide analysis of the 2007 bulk sampling provided an average grade of 1.2745 g/t Au, whereas the screened metallic analysis provided a grade of 1.8118 g/t Au. The difference in results is likely due to a

number of factors, including the different analytical techniques, the 'nugget' effect within the veins and wall rock material collected, as well as the sample size used to complete the two different analytical techniques.

9.3. 2008 Program

The 2008 year began the compilation and digitally archiving all formerly collected data. Reviews of the data demonstrated that it would be prudent to conduct an assay result confirmation study to determine if there was a benefit in conducting full screen metallic assays on formerly fire assayed samples and, if so, what procedure should be invoked. The study was conducted on small, specific mineralized intersections within drill holes where mineralization would be expected. A total of 213 samples were re-run using a screen metallic fire assay technique. The study confirmed that gold values were accurately represented by previous sampling procedures and determined there was minimal benefit in re-assaying specific intervals using a screen fire assay method.

The 2008 field program was initiated on May 15, 2008 with the crew mobilizing into camp and preparing for the drill program. SCS Drilling Ltd. of Merritt, B.C. mobilized two diamond drills onto the property on May 28 and began drilling shortly thereafter. The two drills operated until July 17, while only a single drilled continued until program completion on August 6. SCS demobilized both drills and ancillary equipment on August 8. In addition to the drill program, soil sampling was undertaken in five areas. The trailer camp was demobilized in early September, 2009. Split core from both the 2007 and 2008 programs was labeled and stacked within a large metal storage shed on the property and secured.

In addition to Hawthorne's geological staff contracting companies were used at various times throughout the exploration program, including:

- G. & S. Logging Co. Ltd. provided a 2054 Excavator, Packer and 850j CAT for snowplowing and road clearing
- Rathbone and Goodrich
- EBA Environmental

Early in March 2008 Rathbone & Goodrich ("Rathbone") of Williams Lake were contracted to assist Hawthorne with correcting several inconsistencies in the historical data for Frasergold. A preliminary field trip was conducted on May 7 to located 2007 drill collars which were not surveyed in the 2007 field season due to heavy snowfall. On June 8 Rathbone & Goodrich surveyed several 2007 drill collars allowing access to the Main Zone for future drilling. They also surveyed three 2008 drill holes and attempted to establish some baseline points within the Main Zone area which were used to assist with locating future 2008 drill targets. On June 15, Rathbone returned to survey several additional grid points as well as update any completed 2008 drill holes. On July 9, 10 and 14, Rathbone & Goodrich returned to camp and surveyed approximately forty 2008 drill holes and placed pickets as collar markers for

proposed 2008 drill hole locations. August 24 was the final field visit by Rathbone & Goodrich for the 2008 season in which they completed surveying on all remaining 2008 drill hole collars.

9.4. Results of 2008 Programs

9.4.1 Soil Sampling

The locations of the 2008 soil sampling grids are shown in Figure 10. Five areas were sampled; a single line along the upper margin of a clear-cut on the north side of the MacKay River in the southeast part of the property, three areas to the northwest of the Main Zone (Eureka Bowl 1, Eureka Bowl 2 and the NW Extension) and the alpine meadows northeast of Crooked Lake.

The only area where soil sampling returned with Au values of interest lay within the NW Extension grid directly northwest of the known mineralized zone. Here four samples reported more than 1 ppm Au. This area is interpreted to be underlain by the knotted phyllite unit.

9.4.2 Silt Sampling

The locations of silt samples collected in 2008 are shown in Figure 11. The only silts reporting values more than the 0.30 ppm Au were those proximal to the known mineralized zone and from two unnamed creeks northwest of the property (Campbell, 2009).

9.4.3 Rock Sampling

A number of samples of black phyllite with quartz veining were collected in 2008, located in Figures 12 and 13. With a few exceptions the only samples reporting more than 0.05 ppm Au were proximal to the known surface mineralization shown in Figure 13 (Campbell, 2009).

9.5. Results of 2011 Program

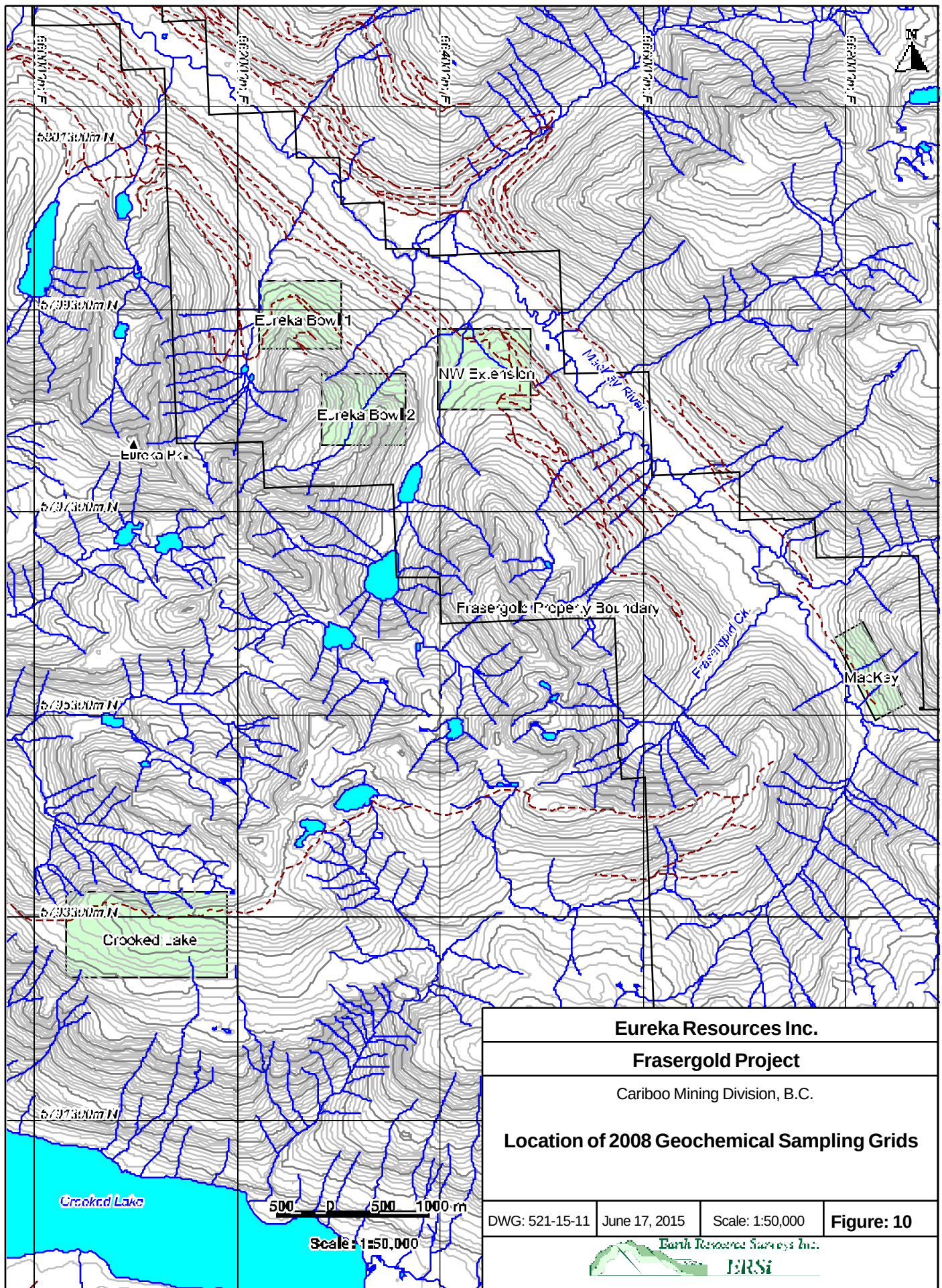
Gold analyses for soil, silt and rocks collected from the three grids Teslin sampled in 2011 are shown in Figure 14 (Kusk grid) and Figure 15 (Eureka Bowl and 18ppm Au grids) from Whitehead (2011).

In the central part of the Kusk grid, on the easterly facing slopes above the MacKay River, lies an area with anomalous gold values in soils warranting further exploration.

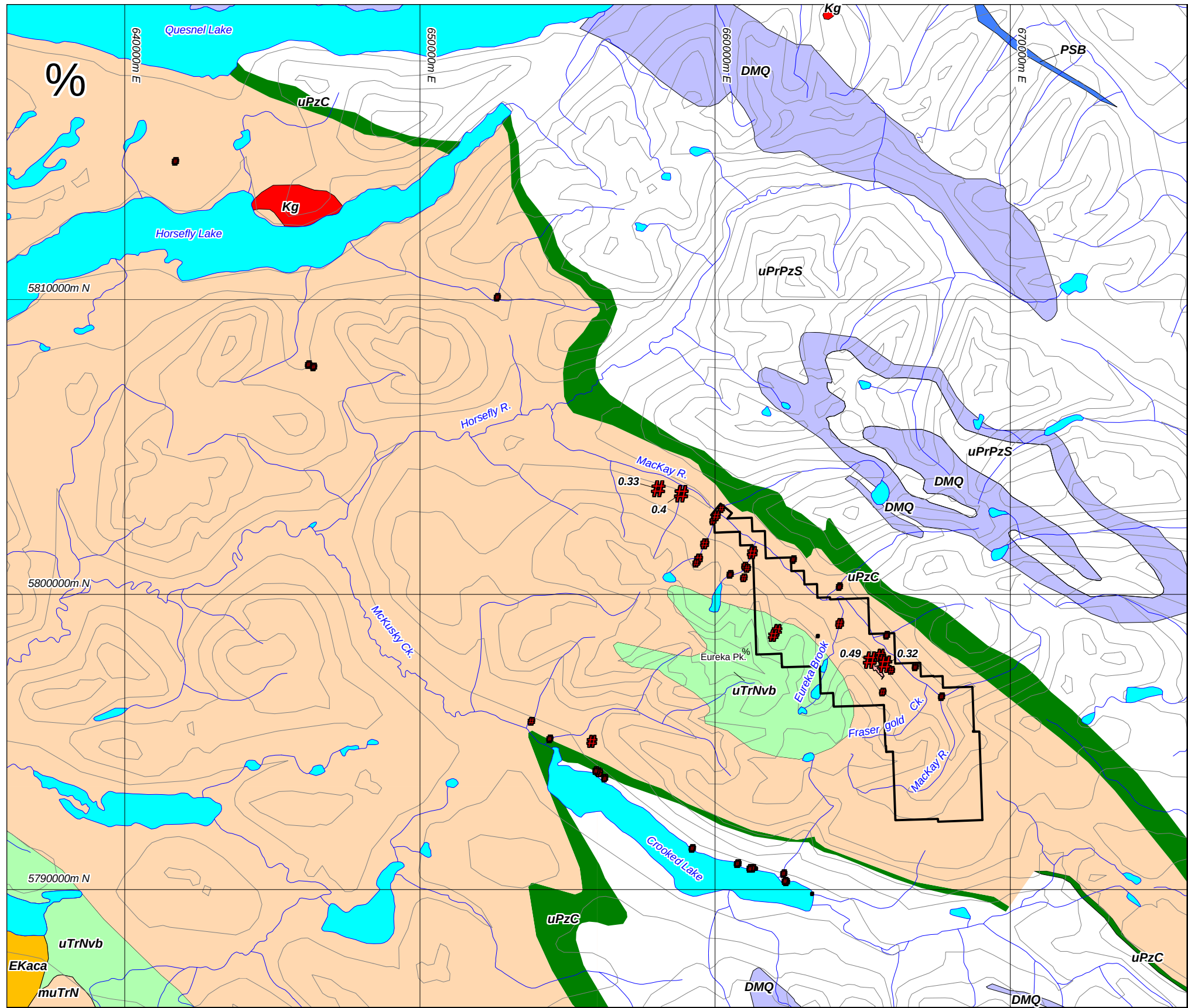
The gold in soils anomaly on the upper part of the Eureka Bowl grid is considered to be related to copper-gold mineralization identified in earlier exploration programs. The elevated gold in soil values in the central part of the 18 ppm grid are thought to be indicative of the mineralization in the knotted phyllite unit that lies directly to the southeast; i.e. in the NW Extension Zone.

9.6. Results of 2015 Program

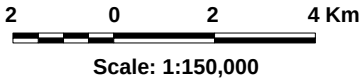
Figure 16 shows the 18 ppm grid extension and the results of the gold in soils from Whitehead (2015). The 18 ppm grid to the south is also shown with the same graduated symbols as those in Figure 15 for



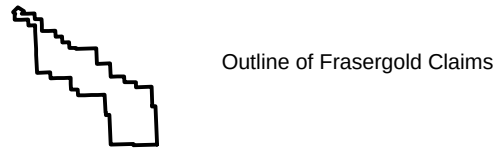
UTM 10 NAD83 Contour interval = 20m



UTM 10 NAD83
Contour interval = 200m



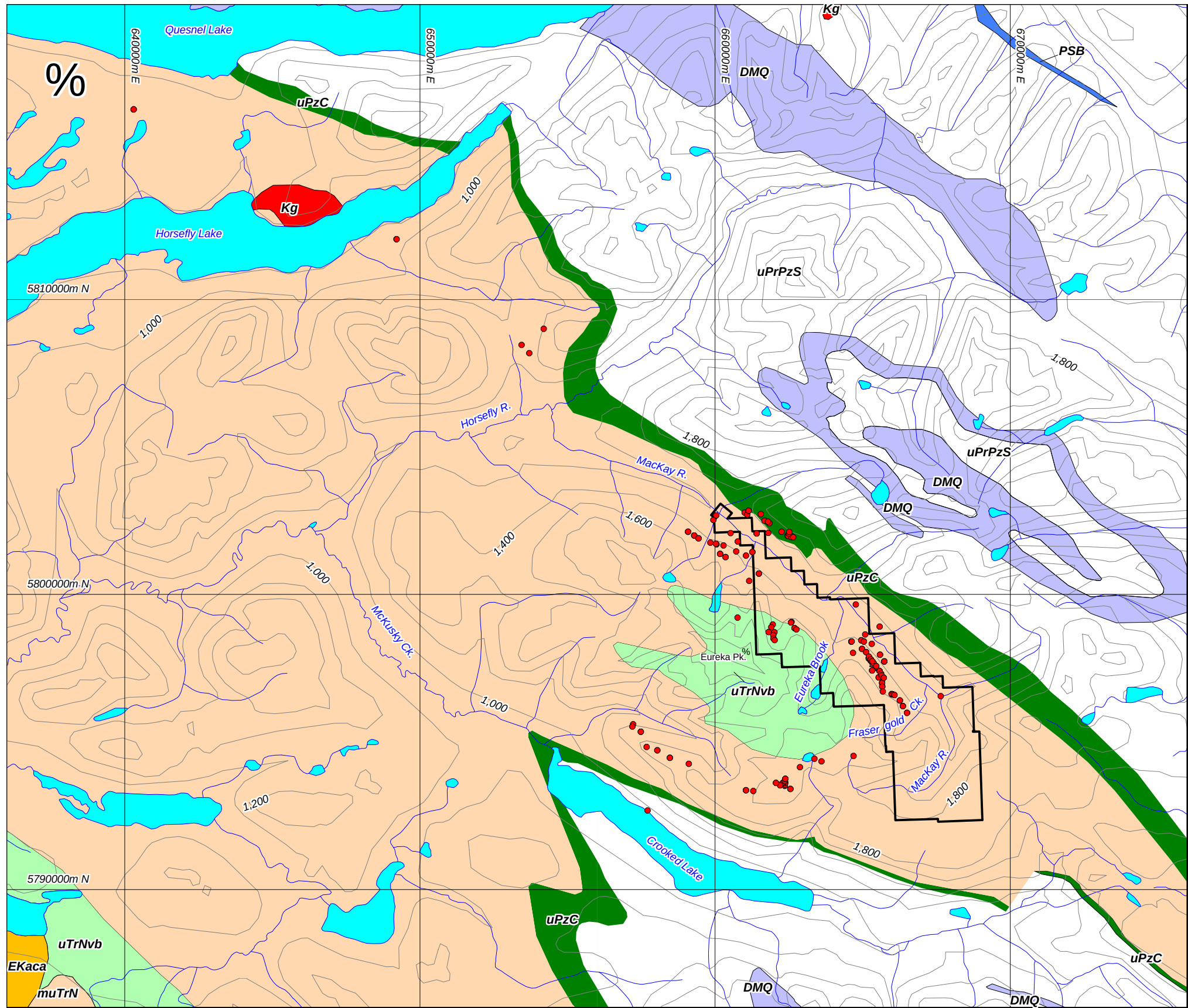
Rock Units	
EKaca	Eocene Kamloops Group
Kg	Cretaceous intrusives, undivided
uTrNvb	Upper Triassic Nicola Group; volcanics
muTrN	Middle Triassic to Upper Triassic Nicola Group; sediments
uPzC	Upper Paleozoic Crooked Amphibolite
PSB	Permian Snowshoe Group - Bralco Succession
DMQ	Devonian to Mississippian Quesnel Lake Gneiss
uPrPzS	Upper Proterozoic to Paleozoic Snowshoe Group



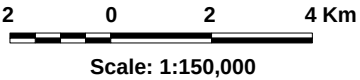
underground Frasergold mineralization projected to surface

Au in silts (ppm)		
	0.25 to 0.49	(4)
	0.06 to 0.25	(5)
	0.04 to 0.06	(5)
	0.02 to 0.04	(11)
	0.01 to 0.02	(21)
	all others	(3)

Eureka Resources Inc.			
Frasergold Project			
Cariboo Mining Division, B.C.			
2008 Silt Samples Au (ppm)			
DWG: 521-15-12	June 18, 2015	Scale 1:150,000	Figure: 11

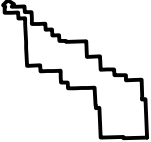


UTM 10 NAD83
Contour interval = 200m



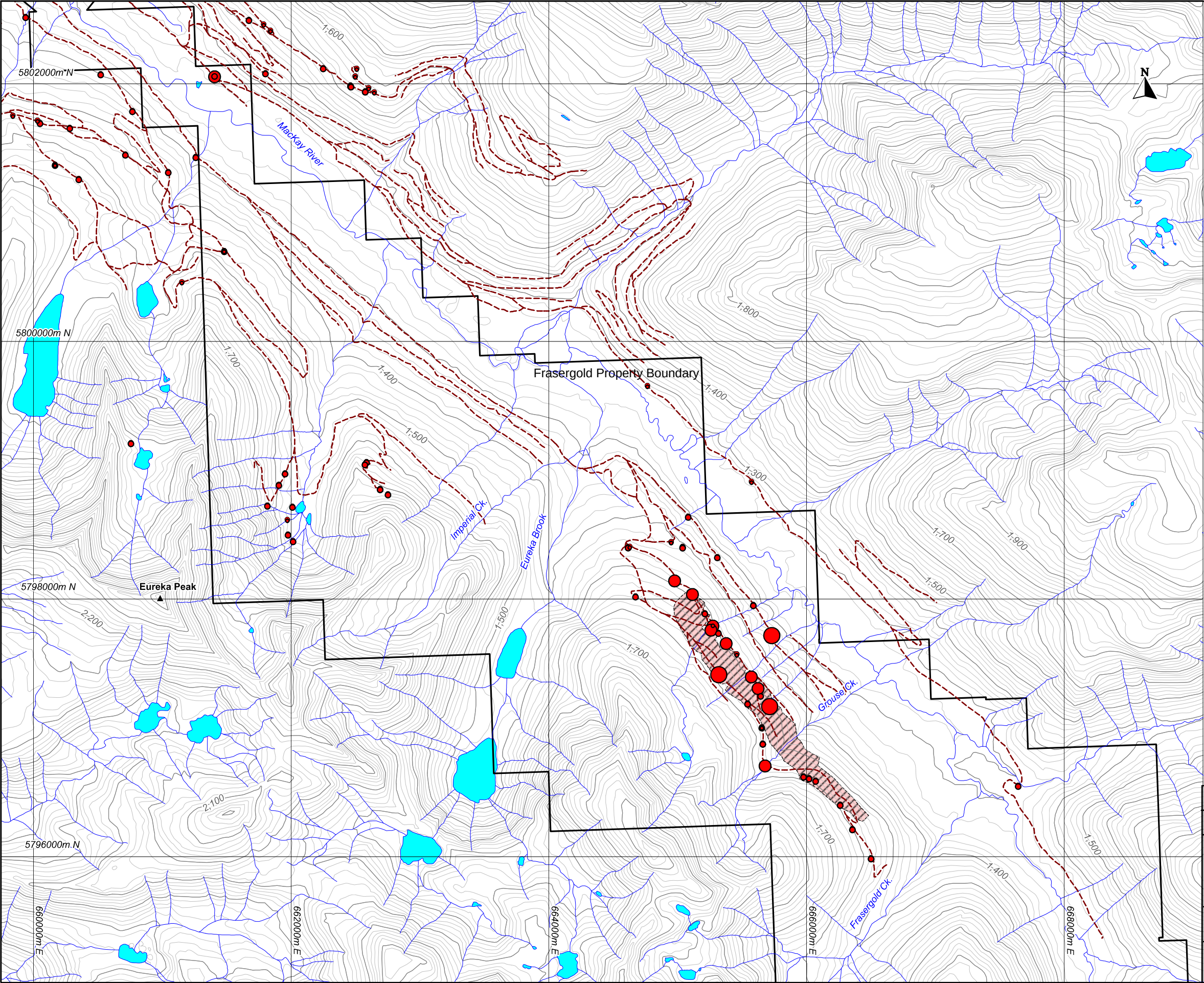
Rock Units

EKaca	Eocene Kamloops Group
Kg	Cretaceous intrusives, undivided
uTrNvb	Upper Triassic Nicola Group; volcanics
muTrN	Middle Triassic to Upper Triassic Nicola Group; sediments
uPzC	Upper Paleozoic Crooked Amphibolite
PSB	Permian Snowshoe Group - Bralco Succession
DMQ	Devonian to Mississippian Quesnel Lake Gneiss
uPrPzS	Upper Proterozoic to Paleozoic Snowshoe Group

 Outline of Frasersgold property

 2008 rock sample location

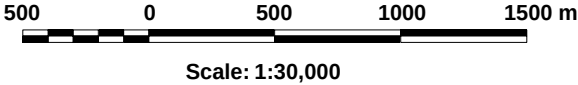
Eureka Resources Inc.			
Frasergold Project			
Cariboo Mining Division, B.C.			
2008 Rock Sample Locations - 1			
DWG: 521-15-13	June 18, 2015	Scale 1:150,000	Figure: 12
			



2008 Rock Samples
Au (ppm)

- 0.5 to 1.48 (3)
- 0.05 to 0.5 (11)
- 0.01 to 0.05 (86)
- 0.01 to 0.01 (63)

known gold mineralization



Eureka Resources Inc.

Frasergold Project

Cariboo Mining Division, B.C.

2008 Surface Rock Samples - 2
Au (ppm)

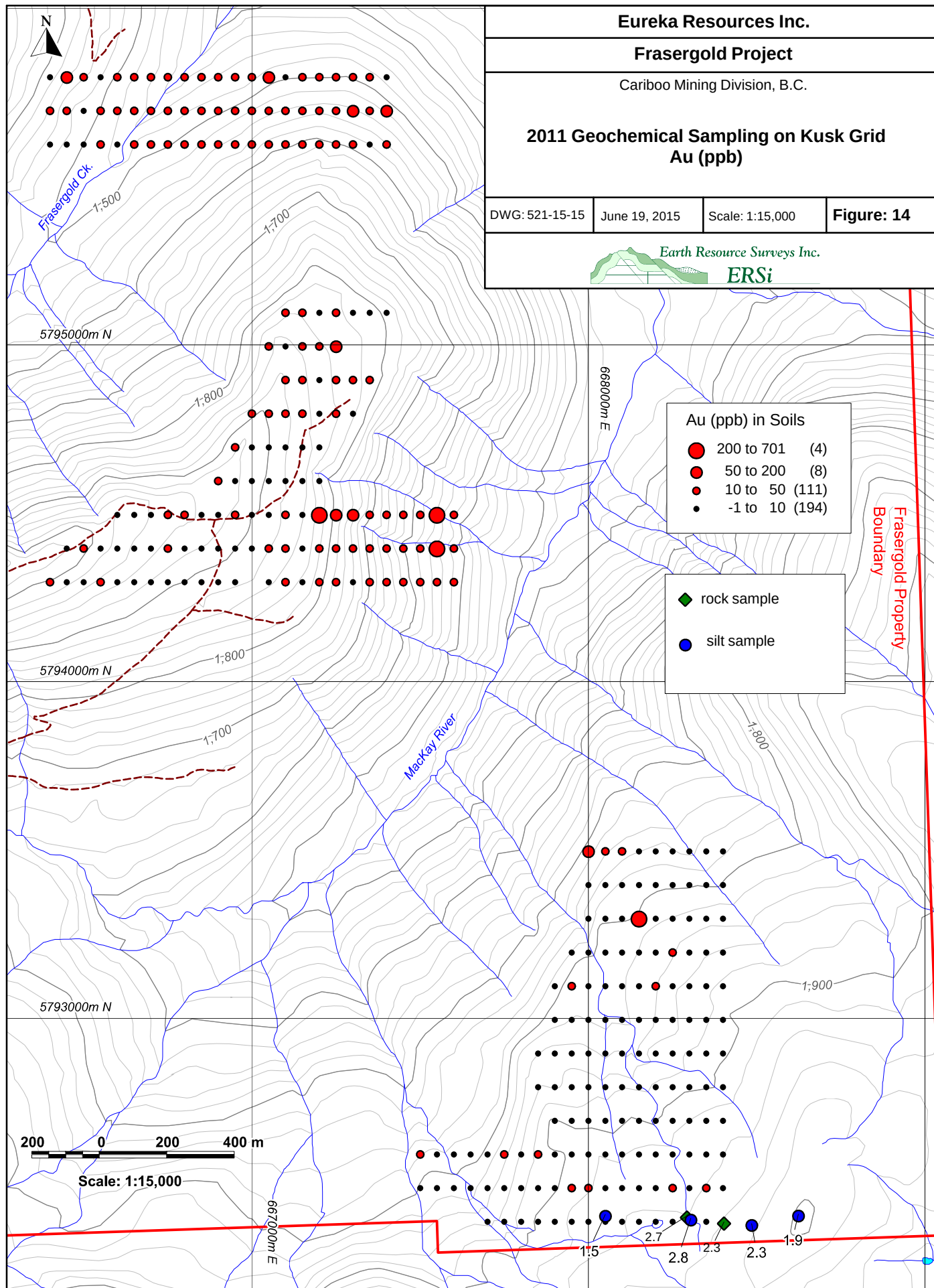
DWG: 521-15-14

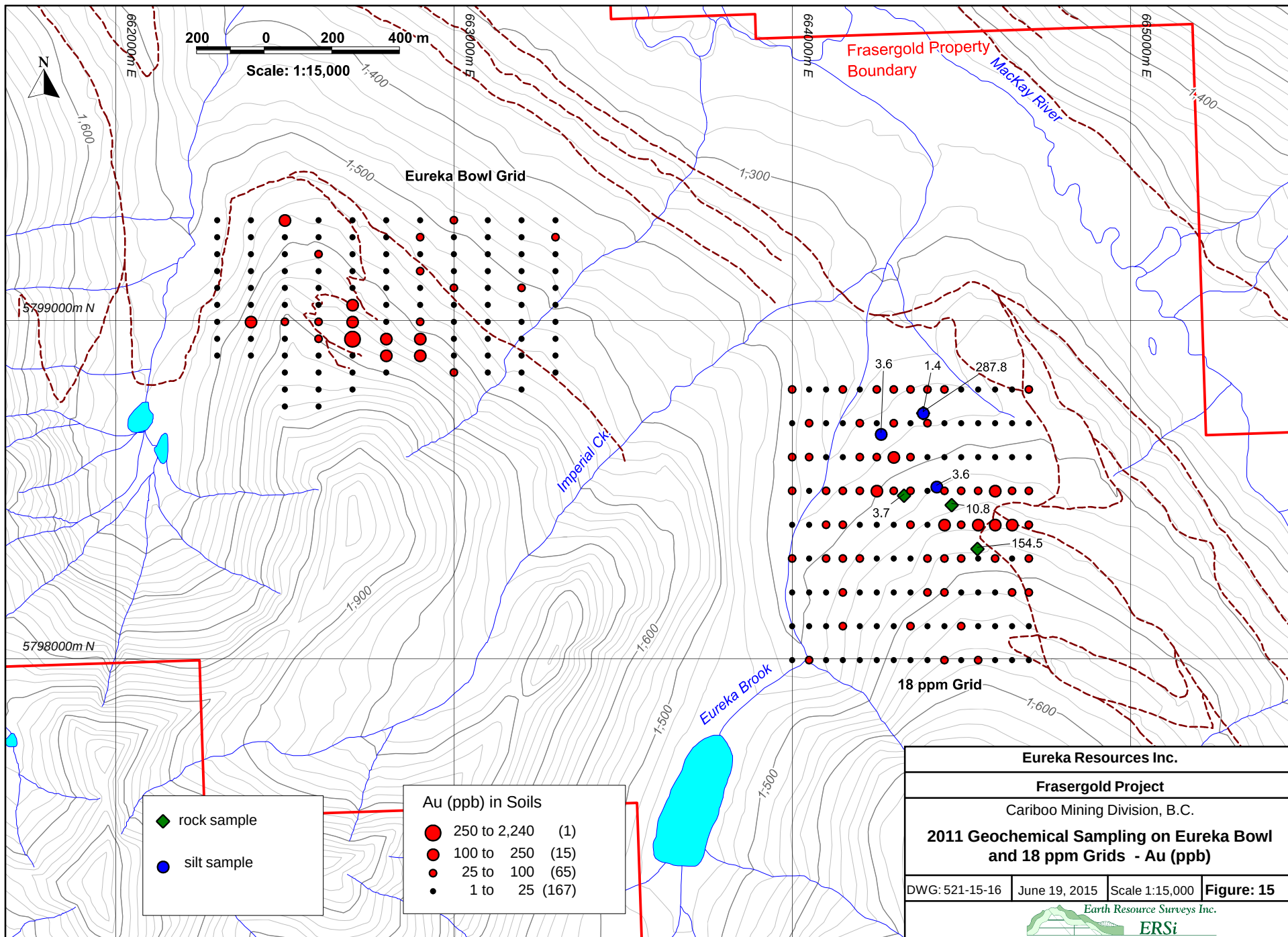
June 18, 2015

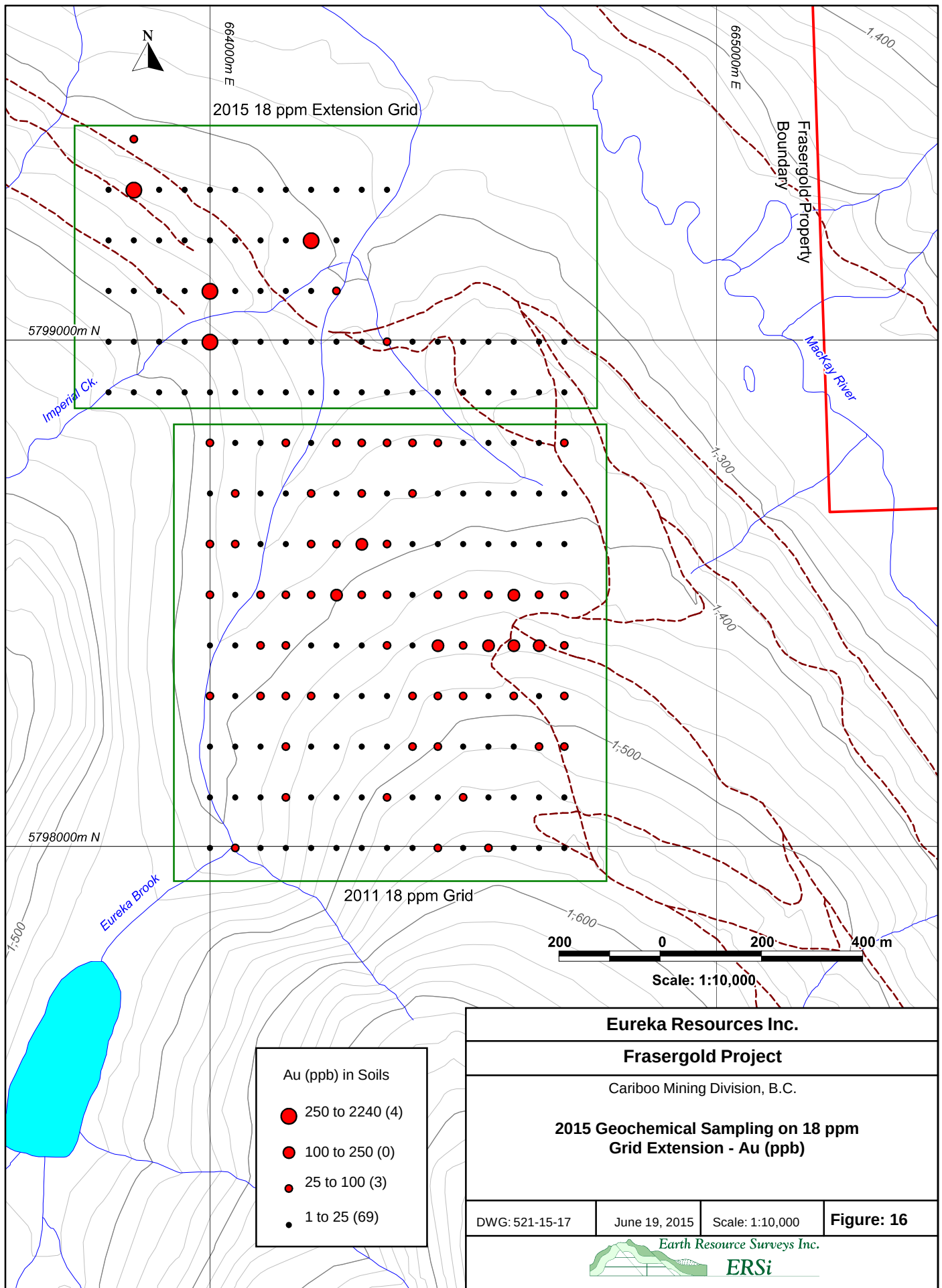
Scale 1:30,000

Figure: 13









reference. Three of the four soil samples with anomalous gold values in excess of 250 ppb are proximal to outcrops along Imperial Ck.

10. Drilling

A total of 328 drill holes and 35,967 meters were drilled on the Frasergold property between 1981 and 1994 (Figure 17). Most of the holes were shallow and concentrated on defining the known mineralized areas. Drilling has been conducted along the known 10 kilometer strike length of the mineralized zone, but drill hole density is sparse outward from the key zones.

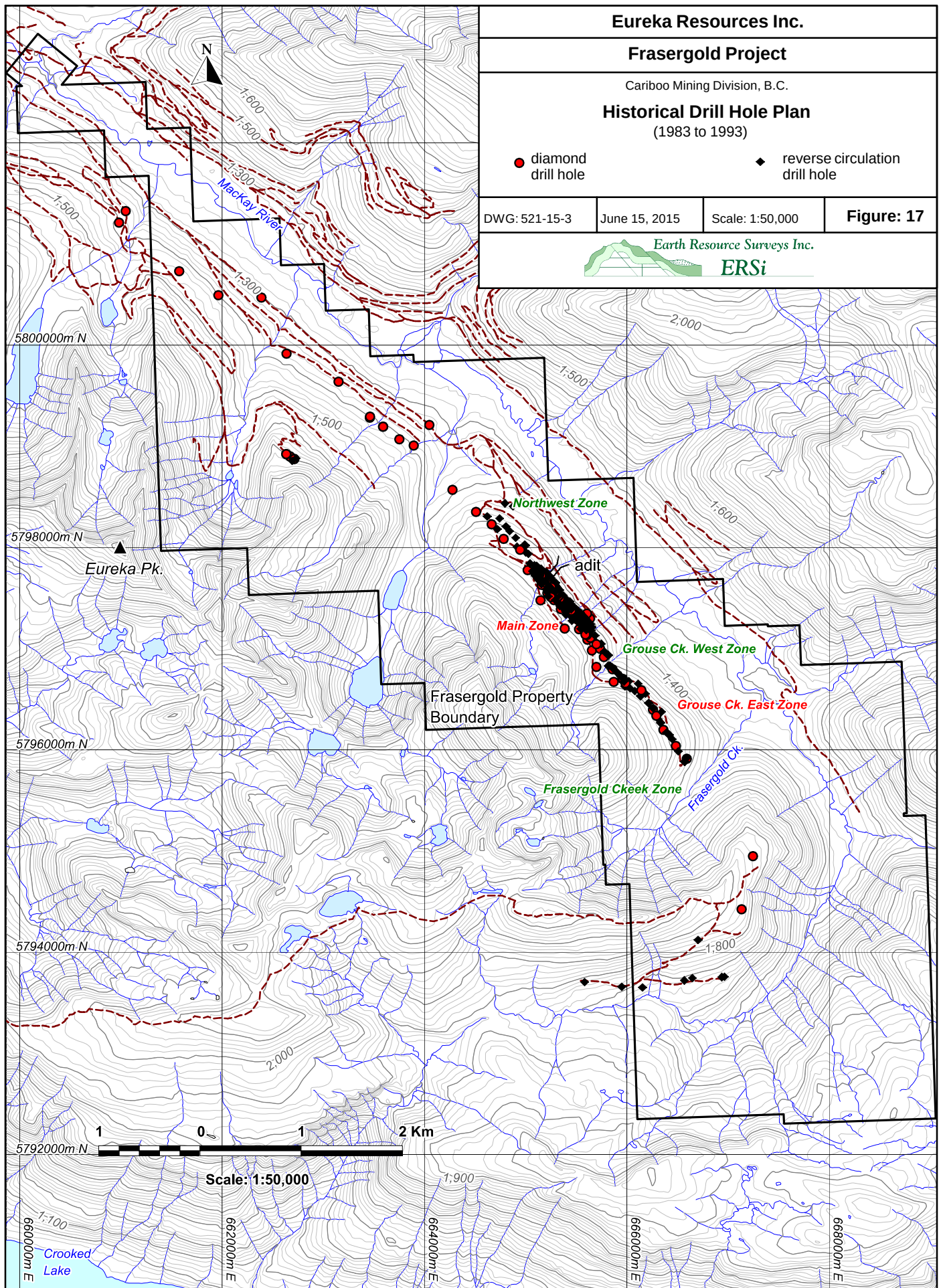
In 2007, Hawthorne conducted a major exploration program on the property. The 2007 drill program was laid out to test four previously defined zones of interest; including the Main Zone, the Grouse Creek West Zone, the Grouse Creek East Zone and the Frasergold Zone. A total of 16 HQ core size diamond drill holes totalling 3,615 meters were drilled over a period of three and a half months, with an average depth of 226 meters. For due diligence purposes ten drill holes were drilled with the remaining six drill holes completed as in-fill holes on the Main Zone. All drill holes were drilled to greater depths than historical drill holes to test potential for additional zones and horizons of interest. A total of 12 holes were drilled on the Main Zone: one drill hole was completed on the Grouse Creek West Zone, one drill hole was completed on the Grouse Creek East Zone and two drill holes were completed on the Frasergold Zone.

The 2008 drill program included 58 NQ diameter diamond drill holes for a total of 10,413.29 meters. The objective of the drill program was focussed on adding resources within the Main zone of the property. Drilling within this zone has specific objectives, those being to test for mineralization down dip, up dip and along strike in both directions as well as tightening spacing between former drilling to add confidence when modelling the deposit for resource estimate purposes.

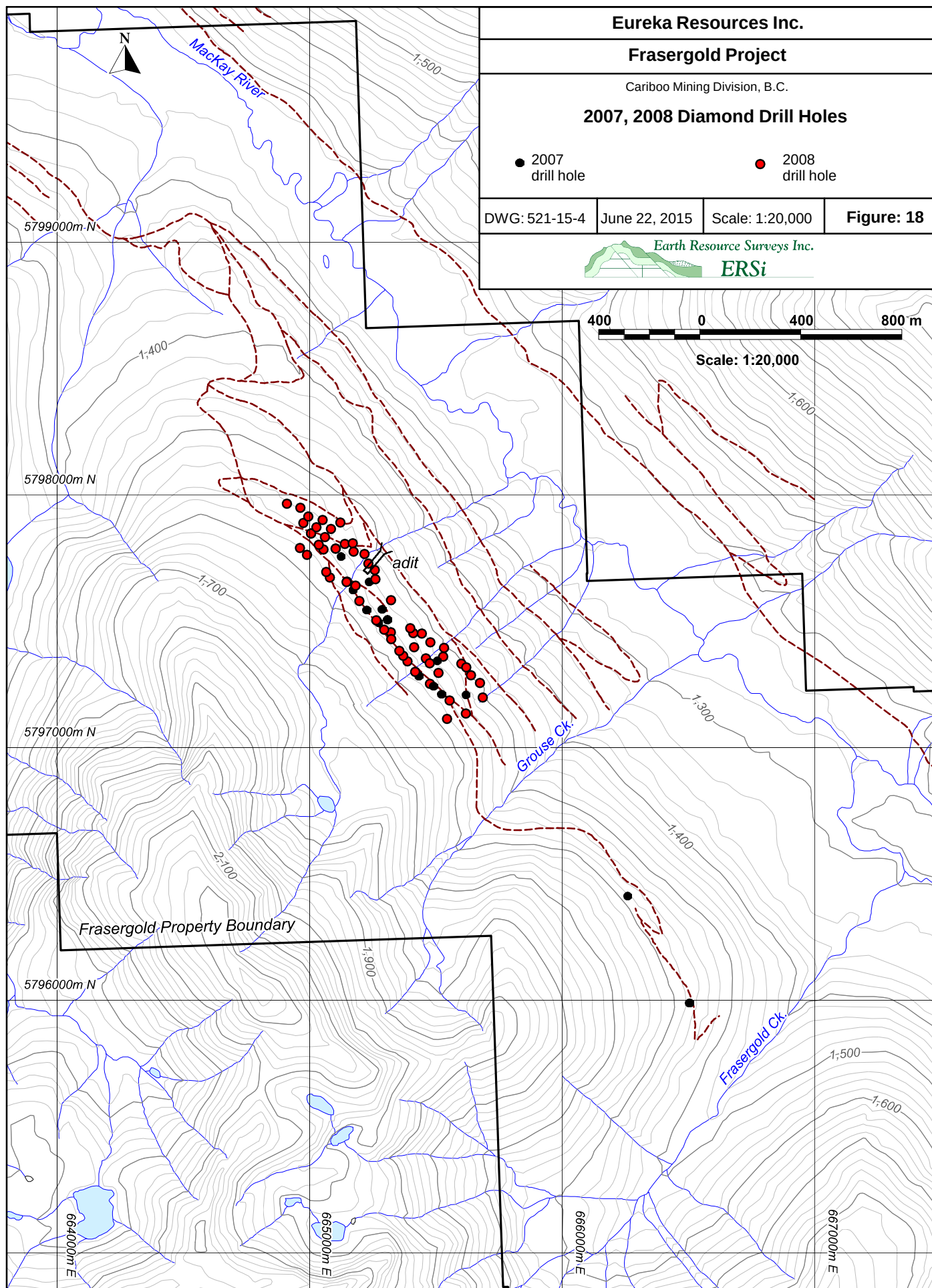
Table 10.1 lists the drill holes completed in 2007 and 2008 by Hawthorne which are located in Figure 18.

Table 10.1 2007 and 2008 Drill Hole Data

Zone	Drill Hole	Type	Easting	Northing	Elev (m)	Dip	Azimuth	Total Depth (m)
Main	07295	DDH	665123.512	5797758.32	1541.336	-62	45	227.38
Main	07296	DDH	665234.225	5797656.93	1537.503	-55	45	215.19
Main	07297	DDH	665287.188	5797548.34	1545.377	-62	45	218.23
Main	07298	DDH	665308.459	5797507.77	1549.58	-61	45	227.38
Main	07299	DDH	665505.705	5797344.65	1522.901	-61	45	212.45
Main	07300	DDH	665522.564	5797212.7	1556.203	-65	42	216.15
Grouse West	07301	DDH	665619.131	5797209.39	1517.378	-61	45	209.09
Main	07302	DDH	665491.239	5797243.87	1557.418	-60	40	233.78
Grouse East	07303	DDH	666258.663	5796412.66	1574.646	-66	45	236.83
Frasergold	07304A	DDH	666505.13	5795990.07	1564.51	-59	45	154.84



UTM 10 NAD83 Contour interval = 20m



UTM 10 NAD83 Contour interval = 20m

Zone	Drill Hole	Type	Easting	Northing	Elev (m)	Dip	Azimuth	Total Depth (m)
Frasergold	07304B	DDH	666504.094	5795989.19	1564.446	-59	45	262.43
Main	07305	DDH	665272.159	5797496.88	1563.44	-59	45	244.75
Main	07306	DDH	665226.265	5797546.58	1565.213	-65	45	267.31
Main	07307	DDH	665431.421	5797284.59	1564.37	-54	40	248.72
Main	07308	DDH	665172.04	5797625.64	1561.978	-65	45	239.88
Main	07309	DDH	665172.04	5797625.64	1561.978	-50	45	200.56
Main	08310	DDH	665147.644	5797663	1558.954	-60	38	53.95
Main	08311	DDH	665054.052	5797790.56	1555.643	-50	42	273.41
Main	08312	DDH	665145.587	5797661.57	1559.272	-60	38	255.12
Main	08313	DDH	665035.267	5797809.33	1555.104	-50	42	288.65
Main	08314	DDH	665079.487	5797679.73	1571.337	-65	40	233.79
Main	08315	DDH	665139.163	5797812.38	1525.509	-60	40	178.61
Main	08316	DDH	665170.015	5797814.74	1518.391	-60	40	130.30
Main	08317	DDH	665263.128	5797510.51	1563.614	-62	42	251.76
Main	08318	DDH	665181.49	5797646.96	1554.449	-50	43	224.33
Main	08319	DDH	665386.641	5797347.24	1561.309	-64	40	203.30
Main	08320	DDH	665320.993	5797462.52	1555.361	-56	40	239.57
Main	08321	DDH	665370.08	5797369.13	1560.11	-62	40	203.30
Main	08322	DDH	665355.01	5797389	1559.716	-62	40	252.07
Main	08323	DDH	665196.521	5797586.32	1565.28	-60	41	233.78
Main	08324	DDH	665475.677	5797258.59	1558.653	-59	38	196.90
Main	08325	DDH	665064.958	5797701.67	1571.487	-65	38	267.00
Main	08326	DDH	665553.542	5797192.22	1550.511	-64	36	195.68
Main	08327	DDH	665617.894	5797141.32	1538.631	-55	36	224.33
Main	08328	DDH	665060.513	5797839.37	1538.594	-50	45	206.40
Main	08329	DDH	665294.577	5797472.46	1561.868	-60	36	230.79
Main	08330	DDH	665084.032	5797871.72	1525.079	-50	38	200.25
Main	08331	DDH	665322.669	5797435.29	1561.701	-58	36	226.16
Main	08332	DDH	665417.79	5797305.75	1563.047	-58	36	206.04
Main	08333	DDH	665121.261	5797896.76	1510.92	-50	38	129.89
Main	08334	DDH	665321.556	5797589.69	1525.528	-55	38	193.85
NW Ext.	08335	DDH	665050.987	5797907.24	1524.749	-50	38	203.00
Main	08336	DDH	665216.683	5797772.74	1515.883	-60	37	81.38
Main	08337	DDH	665257.19	5797709.54	1517.622	-50	35	129.54
Main	08338	DDH	665231.585	5797734.65	1519.827	-50	35	127.10
Main	08339	DDH	665173.754	5797782.18	1523.672	-45	35	108.81
West Grouse	08340	DDH	665684.948	5797204.8	1493.802	-55	35	145.08
Main	08341	DDH	665102.454	5797794.08	1538.095	-62	35	51.21
West Grouse	08342	DDH	665543.589	5797119.93	1571.546	-60	35	199.95

Zone	Drill Hole	Type	Easting	Northing	Elev (m)	Dip	Azimuth	Total Depth (m)
Main	08343	DDH	665102.253	5797793.47	1538.222	-62	35	148.13
Main	08344	DDH	665443.464	5797457.33	1512.014	-49	35	126.19
NW Ext.	08345	DDH	664988.91	5797769.06	1569.43	-50	35	291.69
Main	08346	DDH	665409.937	5797459.05	1523.72	-53	37	169.77
Main	08347	DDH	665398.097	5797478.46	1523.465	-48	37	145.39
NW Ext.	08348	DDH	665005.114	5797854.23	1547.574	-51	34	69.19
Main	08349	DDH	665413.497	5797403.39	1538.172	-57	35	172.51
NW Ext.	08350	DDH	665005.108	5797854.15	1547.56	-51	34	230.43
Main	08351	DDH	665460.117	5797359.21	1532.972	-58	35	157.28
Main	08352	DDH	665477.491	5797423.15	1505.924	-57	35	111.56
NW Ext.	08353	DDH	664974.369	5797894.97	1543.312	-50	35	200.25
Main	08354	DDH	665528.442	5797366.26	1508.385	-50	35	151.18
Main	08355	DDH	665531.554	5797401.05	1494.808	-50	35	102.72
Main	08356	DDH	665600.034	5797337.85	1493.531	-65	36	96.62
Main	08357	DDH	665259.673	5797672.57	1524.772	-50	35	154.53
Main	08358	DDH	665638.159	5797292.7	1492.277	-57	36	99.36
West Grouse	08359	DDH	665674.025	5797262.27	1484.348	-52	36	84.43
Main	08360	DDH	665620.012	5797324.22	1490.09	-60	36	141.43
Main	08361	DDH	665475.312	5797339.83	1536.17	-60	35	145.08
Main	08362	DDH	665509.897	5797302.01	1533.619	-60	35	151.18
NW Ext.	08363	DDH	665025.823	5797878.81	1536.92	-50	35	221.28
NW Ext.	08364	DDH	664961.325	5797796.31	1569.022	-50	35	282.55
NW Ext.	08365	DDH	664962.369	5797955.67	1531.543	-50	35	206.35
NW Ext.	08366	DDH	664908.992	5797971.98	1533.888	-50	35	206.35
NW Ext.	08367	DDH	664994.225	5797921.13	1534.798	-50	35	203.00

11. Sample Preparation, Analyses and Security

11.1. 1984 to 1994 Programs

No sample material from the previous programs is available to conduct check or confirmation assays.

Sample preparation, quality control measures, sample splitting, analytical procedures, security measures for exploration programs during this period are summarized below, from Campbell et al, 1991 in a report for Asarco Inc. and Eureka Resources Inc.

11.1.2 Reverse Circulation Drilling

The reverse circulation drilling on the Frasergold property utilized a technique whereby compressed air was injected down the hole through the outer circumference of the drill pipe to the hammer which transmitted a high energy shock to the bit face. The resulting chips were carried into an inner sample

recovery tube, thereby avoiding any sample contamination from cuttings uphole. At the end of every 10ft (3.1m) drill rod the hole was flushed out.

A sample interval is 1.5m long from a 5.5" diameter hole having a theoretical weight of 61.6kg. Two opposing chambers in a rotary splitter provided a 1/8 split which was collected for assay. The assay sample (1/8 split) was collected in a specially treated, 13" x 26", 400 mesh Bone Velour Micro Por bag held in place under the splitter. The seams on the bag are double stitched with a polyester lock stitch. The bags have a rolled seam with a nylon draw cord for closure and hanging.

The entire sample (cuttings and water) was contained in the Micro Por bag under ideal conditions. Specially designed drying racks at the drill site were used to hang the assay bags for periods of up to 24 hours. When the assay samples were adequately drained at the drill site they were transported to camp where they were hung again to drip dry before placing in 100lb rice bags for transport. Samples were shipped on a weekly basis to the laboratory of Bondar-Clegg in North Vancouver, B.C. in a chartered 5 ton truck.

11.1.3 Underground Channel Samples

Sampling was carried out with electric chisels producing channels averaging 3-5cm wide and 1-4cm deep. The samples weighed between 3 and 15kg with an average weight of about 10kg. Wherever possible the samples were taken perpendicular to veins or vein zones. Where this was not possible, due to complexity or lack of veining, the samples were taken horizontally or vertically.

Individual samples were placed in heavy duty poly bags, sealed with twist ties before being placed in 100lb rice bags for transport to Bondar-Clegg.

11.1.4 Underground Bulk Sampling

In 1990 Asarco Inc. supervised the collection of four bulk samples, each about 1 metric tonne, from the main drift and crosscut. The samples were shipped to Bacon Donaldson and Associates Ltd. for preliminary metallurgical testing.

The mining program in 1991 was contracted to Beaton Engineering Ltd. and Vicore Mining Developments Ltd. of Vancouver, B.C. Drifts and crosscuts totalling 114m in length were mined resulting in 9 "Lots" which were then shipped on B-train transport trucks to Nesmont Precious Metals Corp. in Vancouver, B.C. by Kootenay Machinery and Transport Ltd. All shipments were weighed on certified scales upon arrival at the mill site.

11.1.5 Diamond Drilling

All core was cut in half with a rock saw or hand splitter. Half of each cut sample was placed in a plastic sample bag with an assay tag and stored in 100lb rice bags for transport to Bondar-Clegg.

11.1.6 Security

At no time did the primary author of this report, who was on site for many of the programs during this period, ever have any doubt as to the integrity of the sampling, sample preparation, sample storage, note keeping and the competence of those people involved in the exploration activities. This author has participated in all programs on the Frasersgold property since 1984 and has had full confidence in the data collection and processing during these programs.

11.1.7 Drill Sample Analyses

The great majority of drill samples were analyzed by Bondar-Clegg in North Vancouver, B.C. Both fire assays and metallic sieve analyses were performed by this very well known, established and certified laboratory.

The procedure for the fire assay was as follows:

6. one assay ton (29.166 grams) of homogenous pulp is weighed into a fireclay crucible and fluxed with lead oxide, litharge, soda, silica, borax glass and flour
7. the samples are fused at 1950°F until a clear melt is obtained
8. the 30-40 gram button that is produced is then separated from the slag and heated in a cupellation furnace to separate the lead from the noble metals.
9. the precious metal beads are then dissolved in aqua-regia which is then analyzed using atomic absorption
10. in the case of high grade samples, greater than 0.2 oz per ton, the precious metal bead is parted in dilute HNO₃ acid to dissolve the silver and the remaining gold is weighed.

The procedure for metallic sieve analysis was as follows:

1. the entire sample is crushed to 10 mesh
2. a representative split, weighing 200-300 grams, of the minus -10 mesh material is taken with a Jones riffle splitter
3. the minus -10 mesh split is pulverized with a ring puck type pulverizer and classified with a 150 mesh screen
4. coarse particulate gold is physically concentrated into one size fraction, the + 150 mesh fraction.
5. the gold content of the +150 mesh fraction is determined by the fire assay procedure outlined above
6. the gold content of the minus - 150 mesh fraction is determined by routine atomic absorption analysis
7. the gold content of the +150 mesh fraction is then mathematically redistributed over the original sample using a weighted average calculation

11.1.8 Analytical Quality Control Procedures

Bondar-Clegg routinely ran a duplicate analysis for 2 out of each batch of 24 samples as well as a standard sample analysis. All samples that were analyzed over 0.2 oz per ton on the original fusion were

run again to verify the results. Certified standards and in-house pulp standards as well as synthetic solution standards were run with each report or batch of samples.

11.2. 2007 Program

11.2.1 Rock Outcrop Sampling

During the surface geological mapping and sampling program geologists described rock samples in as much detail as possible. The location of the sample sites was determined with hand-held GPS units and reference to topographical maps or orthophotos. Sample sites were recorded and samples weighing approximately 4 kg were placed in individual plastic sample bags along with their corresponding sample tag. The notes and data in the field were then transferred from the paper records into a digital database.

Sample preparation and analyses of rock outcrop samples was as follows:

8. crush entire sample to minus (-)10 mesh, mix and riffle out 1.0 kg
9. pulverize the 1.0 kg sample to minus (-)150 mesh and screen
10. weigh and fire assay (FA/AAS) the entire +150 fraction
11. mix and split three identical 1 assay-tonne samples from the minus (-)150 fraction for fire assay
12. all Au assays to be 1 assay-tonne fire assay followed by gravimetric finish
13. average the three minus (-)150 mesh assays, combine with +150 mesh assay on a weighted basis to determine Estimated average gold value
14. report all four assays and weighted average gold value

11.2.2 Trench Sampling

In the course of the trench sampling exploration crews cut two narrow parallel channels with a portable diamond saw along the bottom of the sill of the trench. Each narrow channel was approximately 2-4 mm wide, the two parallel channels were spaced equally apart (approximately 2.5" or 63.5mm) along the entire length of the channels, with the material between each channel meant to represent the width of an HQ diamond drill hole. The intervening rock sample material between the two narrow channels was then carefully and entirely chipped out in 1 to 1.5 meter interval lengths, with the chipped material carefully bagged and marked for assaying. The approximate weight of the trench samples was 4 kg. All trench channel samples were sent to IPL.

Sample preparation and analyses of trench samples was as follows:

1. crush entire sample to minus (-)10 mesh, mix and riffle out 1.0 kg
2. pulverize the 1.0 kg sample to minus (-)150 mesh and screen
3. weigh and fire assay (FA/AAS) the entire +150 fraction.
4. mix and split three identical 1 assay-tonne samples from the minus (-)150 fraction for fire assay
5. all Au assays to be 1 assay-tonne fire assay followed by gravimetric finish

6. average the three minus (-)150 mesh assays, combine with +150 mesh assay on a weighted basis to determine Estimated average Au value
7. report all four assays and weighted average gold value

11.2.3 Underground Channel Sampling

At each predetermined channel sample site the exploration crews used portable rotary saws with diamond blades to cut two narrow parallel channels starting from near the sill and cut to near the back. Each cut channel ranged from approximately one to two meters in total length. Each narrow cut channel measured approximately 2-4 mm in width, with the two parallel channels spaced equally apart (approximately 2.5" or 63.5mm) along the entire length of the channels, with the material between each channel meant to represent the width of an HQ diamond drill hole. The intervening rock sample material between the two narrow channels was carefully and entirely chipped out in 0.38m to 1.14 meter interval lengths starting at the top of the 1 to 2 meter interval, with the chipped material carefully collected on plastic tarps placed on the sill of the workings. On average, each sample was 0.8m in length. At sites where the channel was two meters in length, the bottom one meter was chipped out separately after the first one meter had been collected. At 96 of the 115 sample sites two samples were collected. The remaining 19 channels sample sites were less than 1 meter in length and only one sample was collected. At each site the chip material was carefully bagged and marked for assaying; all channel samples were sent to IPL.

Sample preparation and analyses of the underground channel samples was as follows:

1. crush entire sample to minus (-)10 mesh, mix and riffle out 1.0 kg
2. pulverize the 1.0 kg sample to minus (-)150 mesh and screen
3. weigh and fire assay (FA/AAS) the entire +150 fraction
4. mix and split three identical 1 assay-tonne samples from the minus (-)150 fraction for fire assay
5. all Au assays to be 1 assay tonne fire assay followed by gravimetric finish
6. average the three minus (-)150 mesh assays, combine with +150 mesh assay on a weighted basis to determine Estimated average Au value
7. report all four assays and weighted average gold value

11.2.4 Underground Bulk Sampling

The location of the underground bulk sample sites were recorded with regard to adit reference, as well as in NAD83 coordinates and elevation. The samples were carefully identified lithologically and where possible structural notes were also taken. The samples were acquired by chipping from the walls over an approximate area of 1.5 m x 1 m, using a stopper or small chipping hammer. The sample material at each site was collected in 6 to 10 rice bags and removed from the workings using a wheelbarrow. It was then transferred to white 20 liter pails with each pail numbered and marked with respect to the bulk

sample site; each site bulk sample weighed approximately 200 kg in the pails. The samples were then shipped to Process Research Associates ("PRA") in Richmond, British Columbia for analysis.

Analysis of the underground bulk samples by PRA followed their recommended procedures for determining the head grade. Two procedures were followed for each bulk sample; a cyanide leaching method and a metallic gold assay procedure.

The cyanide leach process involved:

1. splitting out a 50 kg sample from each individual bulk sample which was then ground, crushed and heavily cyanided
2. the cyanide liquid, which is estimated by PRA to recover 95-97% of the gold, was then fire assayed
3. the residue from the leaching process, which is estimated by PRA to recover 3 - 5% of the gold, was then filtered, dried, well mixed and assayed in duplicate
4. the average weighted head grade was then estimated

The metallic sieve procedure involved:

1. splitting out a 50.0 kg sample from each individual bulk sample
2. crush the entire sample to minus (-)10 mesh, mix and riffle out 1.0 kg
3. pulverizing the 1.0 kg sample to minus (-)150 mesh and screening
4. weigh and fire assay the entire +150 fraction
5. mix and split out three identical 1 assay-tonne samples from the minus (-)150 fraction for fire assay
6. all gold assays were for 1 assay-tonne followed by gravimetric finish
7. average the three minus (-)150 mesh assays, combine with the +150 mesh assay on a weighted basis to determine estimated head grade
8. report all four (4) assays and the weighted head grade

11.2.5 Diamond Drilling and Sampling

The core sampling protocol followed in the 2007 and 2008 programs were as follows:

1. convert all drilling blocks from Imperial to metric
2. label each box with drillhole number, box number and the contained core interval using a 1"x 4" buttersoft tag or Dymo aluminum tapewriter
3. calculate core recoveries and rock-quotient-density measurements (RQD)
4. log core for rock type, alteration, structure and mineralization
5. Lay out sample intervals using a 1"x 3" buttersoft with colored tagging and sample tag
6. with a black felt marker, write on the top left-hand corner of each box, the hole number, box number and starting interval
7. on the bottom right corner, mark the finishing interval. This was in large, clear letters for photographing
8. photograph the core, with all tags, markers, etc. in place before splitting

9. mark the cut lines on the core with a felt marker
10. split core using the diamond saw and assemble samples according to the intervals
11. write sample number on inner bag, double bag all samples, and place the sample tag between the two bags
12. place up to five consecutively numbered samples in a white 'rice sack'
13. each 'rice sack' is to be consecutively numbered and sealed with a randomly numbered security tag
14. the last 'rice sack' in the shipment is to contain copy No.1 of the IPL sample submission form
15. the 'rice sacks' were transported to Williams Lake where they are placed on pallets, shrink wrapped and stored until being shipped to Vancouver to International Plasma Lab Ltd.
16. specific gravity determinations were run on approximately 100 representative whole core samples by the Mining Engineering Department at UBC. These 8" to 10" cores were collected throughout the drilling program, identified, tagged and held until the end of the drilling season

Sample preparation and analysis of the core samples collected in 2007 was as follows:

1. dry sample
2. crush entire sample to minus (-)10 mesh, mix and riffle out 1.0 kg
3. pulverize the 1.0 kg sample to minus (-)150 mesh and screen
4. weigh and fire assay the entire +150 mesh fraction
5. mix and split three identical 1 assay-tonne samples from the minus (-)150 m fraction for fire assay
6. all gold assays were for 1 assay-tonne followed by gravimetric finish
7. average the three minus (-)150 mesh assays, combine with the +150 mesh assay on a weighted basis to determine Estimated head grade
8. report all four assays and the weighted head grade

Due to the slow process of completing the screened metallic analysis on all core samples, a review of the procedures was undertaken by management and IPL, and a decision was made to modify the core sample analysis in order to speed up the assaying process without materially affecting the overall results. Consequently starting in late October 2007 all core samples submitted to IPL were first assayed by conventional fire assay procedures prior to having screened metallic analysis completed on all samples. Starting in early November a new protocol was enacted whereby a threshold was set for fire assay values at which time the IPL automatically completed screened metallic assays on all samples where the original fire assay were equal to or greater than 0.1 ppm (1 gm/Tonne – I prefer using 1.0 g/t Au). Any samples less than 0.1 ppm did not have screened metallic analysis completed.

Core samples collected during the 2008 program were prepared and analysed by IPL as follows:

1. the entire 1.0 to 5.0 kg core samples were crushed with >85% passing -2mm or minus (-) 10 mesh
2. a 1000 gram sample was then 'riffle split' from the crushed sample
3. 1000 gram split sample was then pulverized to >85% passing 75 microns or minus (-)200 mesh

4. the 1000 gram pulverized sample was weighed and a 150 gram representative sub-sample pulp was collected by taking a number of scoops at right angles through the homogenized 1000 gram split sample and leaving behind an 850 gram bulk Master Pulp/Fine Reject fraction to be stored at IPL
5. the 150 gram representative sub-sample, referred to as the working pulp and was placed in a small envelope for analysis
6. the 150 gram sub-sample was sub-sampled for analysis by taking 2 to 3 scoops from different places in the small envelope containing the sample
7. 50 gram samples were used for each Au analysis by fire assay and AAS. The same 50 gram sample was used for 30 element ICP analysis. This consisted of four acid "near total" digestion by HF-HNO₃-HClO₄ digestion, HCl leach and ICPAES
8. all samples reporting more than 0.10 ppm Au by fire assay were re-assayed using the screened metallics procedure described above for the 2007 core samples

11.3. 2008 Exploration Program

11.3.1 Soil Sampling

Soil samples, weighing approximately 1 kg, were collected primarily from the B Horizon and placed into Kraft paper bags together with a sample tag. Brief description notes were made and locations determined with hand-held GPS units.

11.3.2 Silt Sampling

Conventional silt sampling practices were followed. Approximately 1 kg of fine grained material was placed into Kraft paper bags which were then air dried. Brief description notes were made and locations determined with a hand-held GPS.

11.3.3 Rock Sampling

One to two kilograms of fresh rock was sampled and placed into a plastic sample bag together with a sample tag. Brief description notes were made and locations determined with a hand-held GPS.

11.3.4 Diamond Drilling

The 2008 drill program included 58 NQ2 diameter diamond drill holes for a total of 10,413.29 meters. The objective of the drill program was focussed on adding resources within the Main zone of the property. Drilling within this zone has specific objectives, those being to test for mineralization down dip, up dip and along strike in both directions as well as tightening spacing between former drilling to add confidence when modelling the deposit for resource estimate purposes.

The core sampling protocol followed in the 2008 program was the same as that described for the 2007 program (Section 11.2.5).

12. Data Verification

This report draws much information from work completed prior to the implementation of National Instrument 43-101. In the opinion of the authors, the exploration programs as described in the reviewed reports were conducted to then-accepted industry standards.

The authors were provided unencumbered access to all available data known for the Frasergold property. The majority of the data was in paper format and included copies of reports prepared by the geological consultants and company representatives that oversaw the exploration work on the property. Included in this data were some, but not all, of the original signed analytical certificates. A comparison of the geochemical database with the certificates did not identify any data input errors.

Digital data provided for the Frasergold property consisted primarily of drill collar survey co-ordinates. These points were shown to correspond accurately with the hand drafted drill plans. A total 35,967 meters in 328 drill holes were completed on the Frasergold property between 1981 and 1994, with an additional 16 drill holes completed and totalling 3,615 meters in 2007, and 58 drill holes completed and totalling 10,414m in 2008.

No new check assay or sample evaluation work was conducted by the authors on the historic drill samples due to their lack of availability. No historic samples exist on which to perform this work. However, the specific companies involved at Frasergold and the number of companies involved adds confidence to this database.

12.1. 2007 Program

12.1.1 2007 Standards, Blanks and Duplicate Samples

The accepted analytical values for blanks and low, - medium- and high grade standards used in the 2007 quality control program are as follows:

Low	0.77 ± 0.06 g/t Au	GS-P7A
Medium	2.03 ± 0.12 g/t Au	GS-2B
High	3.58 ± 0.31 g/t Au	GS-3C
Blank	<0.01 g/t Au	

The +/- error quoted is the "between lab" two standard deviations. All the standards were prepared by CDN Resource Laboratories Ltd. of Delta, B.C. according to a well defined, rigorous procedure. The quoted errors for the standards are very different from those quoted for the wide variety of internationally available standards and are described as follows:

"The mean and standard deviation for all data was calculated. Outliers were defined as samples beyond the mean +2 standard deviations from all the data. These outliers were removed from the data and a new mean

and standard deviation was determined. This method is different from that used by Government agencies in that the actual 'between-laboratory' standard deviation is used in the estimations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Certified Limits published on other standards."

A standard or blank sample was inserted into the drill core samples according to the following:

Every 20th sample – standard sample (one low, medium or high in random order)

Every 80th sample – blank sample

Every 30th sample – duplicate core sample (see below for description)

Based on this methodology, it was important to physically count 20 actual core samples before inserting the next standard, and to physically count all core samples before inserting the next standard and/or blank.

At the point standards were inserted, core logging geologists used a 1"x 3" buttersoft and a different colored piece of tagging, and placed the small standard sample bag and sample tag in the core box.

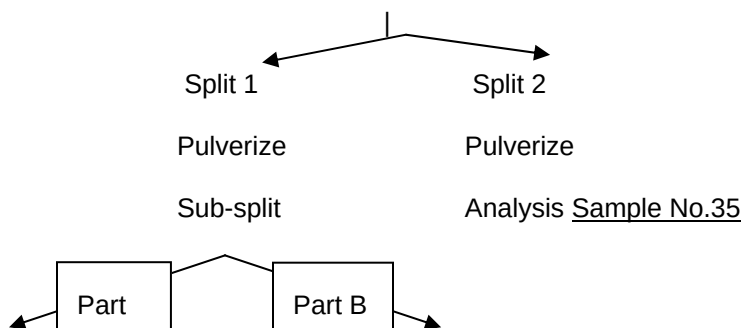
The 'duplicate core sample' process was done in conjunction with the various QA/QC standards and blanks inserted. In the case of the 'duplicate core sample', a second 1"x 3" buttersoft, tagging and sample tag illustrating the same interval on it was placed in the core box. For this 'duplicate sample', the original split half and corresponding ¼ split were used to make up two identical samples in sequence. All three 'duplicate' sequential sample tags and an explanatory sheet (with filled-in sample numbers) were placed in with the bag of the 1/4 of the split core. By placing them all together, it alerted the sample preparation person at IPL that something quite different was required for that sample.

The following outlines the numbering sequence and required analyses to be used for the duplicate core sample, using an actual example. It should be noted that, in this particular set of duplicate samples, 3/4 of the core is used.

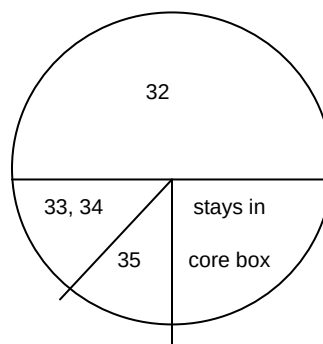
1. First half of core: crush, prepare and assay as routine: Sample No. 32
2. Second half of core: diamond saw in half to produce two ¼ core sections. Place the first part of the ¼ core back in core box.
3. Second part (1/4) of core: crush and split
4. First half of crushed sample is split into Parts A and B.
5. Part A of pulverized pulp for analysis: Sample No. 33
6. Part B of pulverized pulp for analysis: Sample No. 34
7. Second half of crushed sample for analysis: Sample No. 35

This is shown schematically as:

Second Part (1/4) of Split Drill Core ----- → Crush & Split



Analysis Sample No.33 Analysis Sample No. 34



The QA/QC Check Assay portion of the 2007 program was performed once all assay samples had been finished by IPL. On completion of the assay program, Hawthorne Gold sent a list of specific samples (1 in every 20 core samples) in which IPL split out a 2 kg subsample from the original crushed reject and sent this to ALS Chemex for Metallic Sieve Au analysis. ALS Chemex split the sample and returned one half to IPL for blind sample re-assay.

12.1.2 Results of 2007 Blanks and Standard Analyses

Results of 42 consecutive analyses of blank material were documented in an in-house account of QC results for the 2007 sampling/analytical work. Forty-one of these analyses are near or below the detection limit as expected. A single value returned as 3.8 g/t Au, a value that is close to the high standard used. It is likely that a high grade standard was inadvertently tagged as a 'blank'.

Plots of successive analyses for the three standards are shown in Figures 19, 20 and 21. Results for the 41 low-grade standard values are reasonable and acceptable (Figure 19). The single low outlier value on the plot for GS-2B (Figure 20) is likely a result of mislabelling of a low-grade standard as a medium grade standard. The two low outlier values on the diagram for GS-3C (Figure 21) appear to represent two blanks that were inadvertently labelled as high-grade standards.

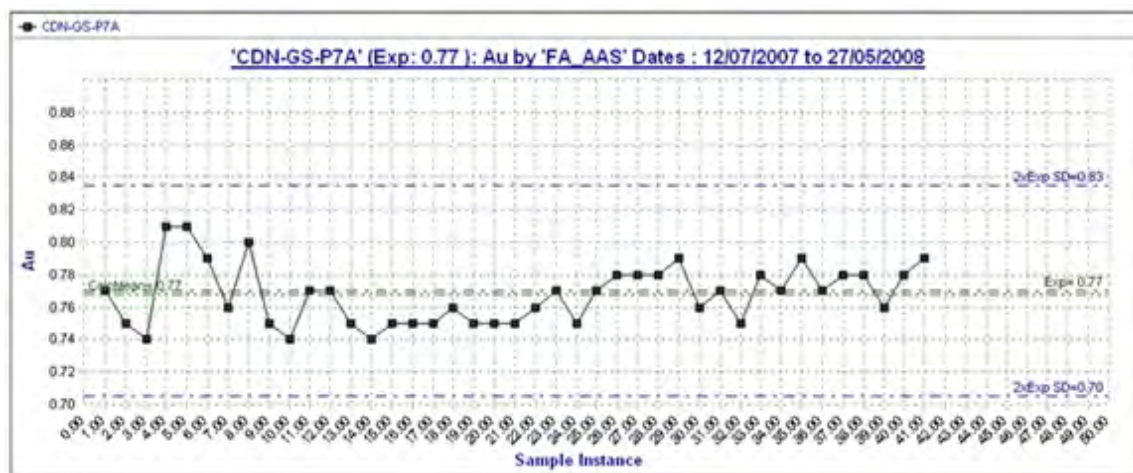


Figure 19. Successive analyses by IPL of low grade standard, GS-P7A, obtained during the 2007 sampling/assaying program. Extreme dashed lines mark 95% error range quoted by CDN Resource Laboratories Ltd.

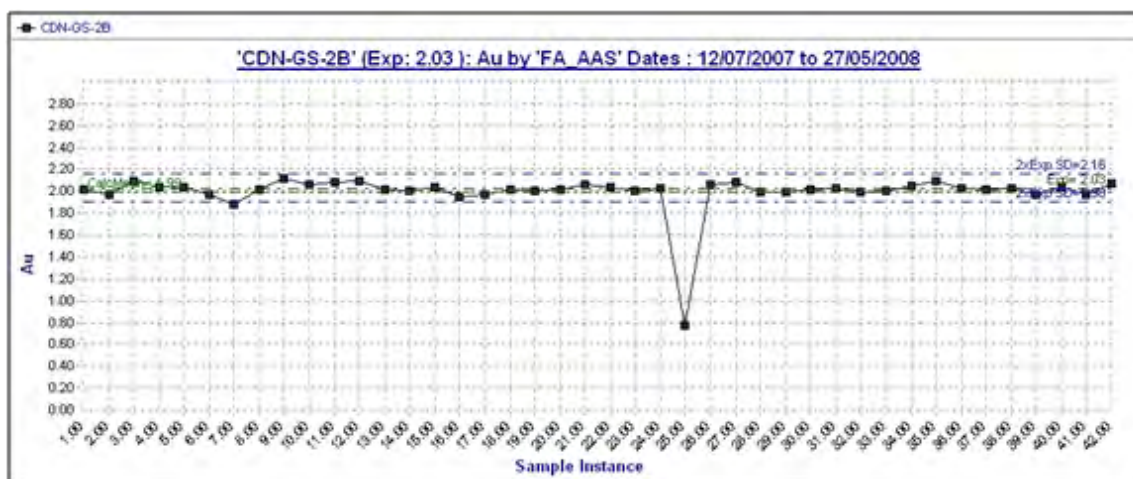


Figure 20. Successive analyses by IPL of medium grade standard, GS-2B, obtained during the 2007 sampling/assaying program. Extreme dashed lines mark 95% error range quoted by CDN Resource Laboratories Ltd.

The analytical data for the blank and standards shows acceptable analytical quality and absence of bias. Four of the 167 analyses (2.4%) appear to represent mislabelling when blanks and standards were inserted into the analytical sequence.

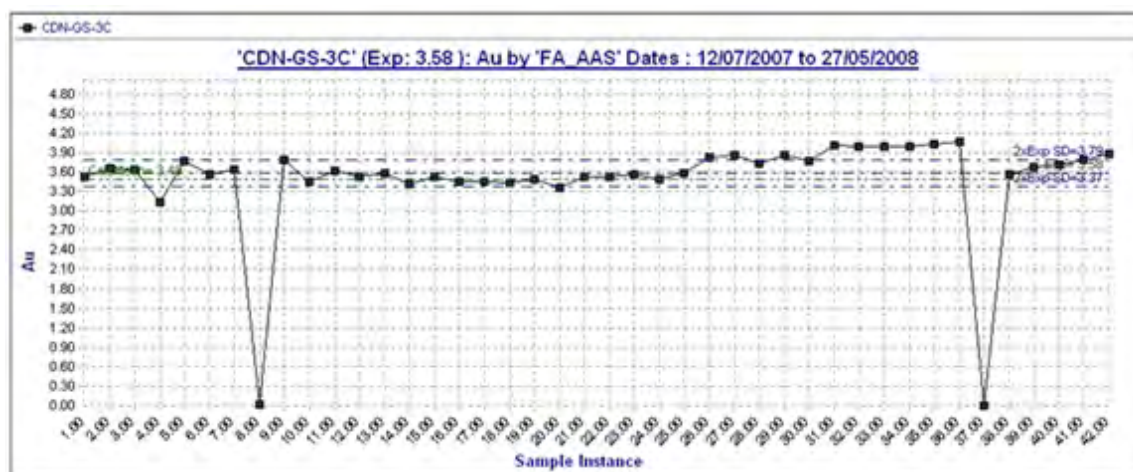


Figure 21. Successive analyses by IPL of high-grade standard, GS-3C, obtained during the 2007 sampling/assaying program. Extreme dashed lines mark 95% error range quoted by CDN Resource Laboratories Ltd.

12.1.3 2007 IPL Original Analyses versus ALS Chemex Check Analyses

Check analyses were obtained for 202 original samples through re-analysis of pulps by ALS Chemex. One of these samples produced an obvious outlier that was removed from the data. The remaining 201 pairs of data are plotted on Figure 22.

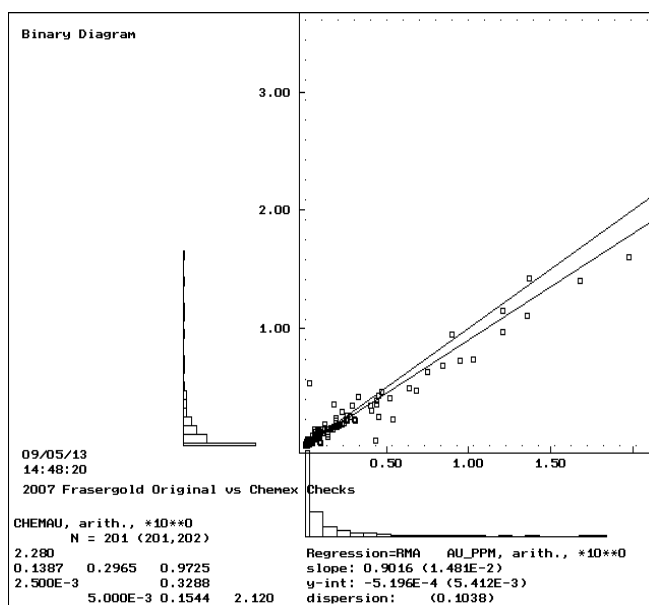


Figure 22. IPL original gold value (Au ppm) versus ALS Chemex corresponding check value (CHEMAU) for 2007 data. Values in ppm (g/t). N = 201. The upper line is $y = x$; the lower line is the reduced major axis fit to the data (RMA), the parameters of which are given in the diagram.

There are two important features shown in Figure 22. The first is that there is a consistent 10 percent proportional bias between the results from the two labs, with IPL averaging high relative to ALS Chemex. This is evident in the slope value of 0.90 for the RMA line. The available data do not make evident if one or both labs are in significant error. This result is surprising because of acceptable results reported by IPL for the three standards throughout the period of their analyses. It is possible that during the short period when ALS Chemex did their analyses, the lab was reporting values slightly too low. These results should be rechecked.

The second important feature of the diagram is the relatively limited scatter about the RMA line and the uniformity of dispersion for values higher than about 0.5 g/t Au. Of course, none of the check values are for original data greater than 2 g/t Au, a limitation of the data, and no information exists concerning possible bias of values greater than 2.0 g/t. Considering the somewhat poorer internal repeatability for pulps shown by IPL (see discussion of field duplicates), the limited dispersion of Figure 11 is surprising, particularly since the IPL analyses were done intermittently over a long period of time whereas the ALS Chemex analyses were done over a relatively short time interval after the main sampling/analytical program. In general, monitoring by an external lab should be done concurrent with the main analyses of the sampling/analytical program.

12.1.4 Field Duplicate Analyses, 2007

The 'field duplicates' for the 2007 program are not strictly field duplicates because the original values are half-core and the 'duplicates' are quarter-core. Consequently, a comparison will overestimate the average total error in the data. In order to make a fairer comparison a single value for the three values for each quarter-core 'duplicate' has been generated. This procedure to some extent counteracts the increased dispersion of values to be expected from quarter-core data. The average was derived by first averaging the two values obtained on the same pulp and then combining that average with the third value obtained for a separate prep sample.

A plot of 'field duplicate' values is shown in Figure 23, which displays very wide dispersion (expected in gold deposits with a high nugget effect) and a superficial bias.

The bias is minimal if only data less than 2.0 g/t Au are considered. Difficulty in reproducing high values is characteristic of deposits with a high nugget effect and can arise in part because the figure compares half-core data with quarter-core data. Consequently, the bias is considered to be more apparent than real and to be superficial, related to the general difficulty in reproducing high grade samples in a high nugget effect gold deposit.

The wide dispersion of values shown in Figure 23 reflects total error in the data, that is the sum of sampling error, preparation error and analytical error, and provides insight into the quality of individual analyses that will eventually be used for resource estimation purposes. The value for dispersion about the RMA line allows an estimation of the average error (standard deviation) in the data, 0.59 g/t Au in this

case. If five samples are combined into a single composite, the error is reduced substantially to 0.264 g/t Au. Similarly, if four such composites are used to estimate a block, the error is further reduced to 0.13 g/t Au. Of course these estimates are based on idealized estimations and are possibly overestimated because they are based on a half-core to quarter-core comparison. Nevertheless, the results give a guideline to the minimum amount of data required to obtain acceptable quality in block estimation.

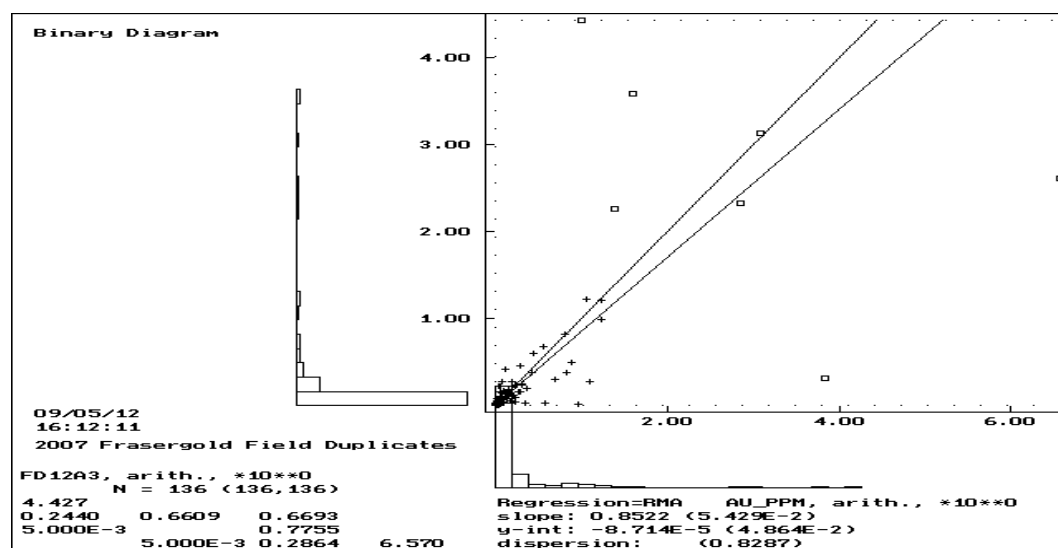


Figure 23. IPL analyses of original sample (AU_PPM) versus “field duplicate” (FD12A3). Upper line is $y = x$; lower line is reduced major axis best fit to data. Apparent bias is reduced if values greater than 2 g/t (open squares) are removed.

12.1.5 Preparation Duplicate Analyses, 2007

Two comparisons of preparation duplicates can be made with the available data, each of the two pulp analyses (from ‘field duplicate’) can be compared with the third analysis of the quarter-core which is based on reject material. One of these comparisons is shown in Figure 24. Comparable results were obtained for the other comparison. If two values greater than 4.0 g/t Au are removed, the RMA line and the $y = x$ line are essentially coincident. The dispersion of values is significantly less than in the case of total error (Figure 23) and is substantially more than for analytical error (Figure 25).

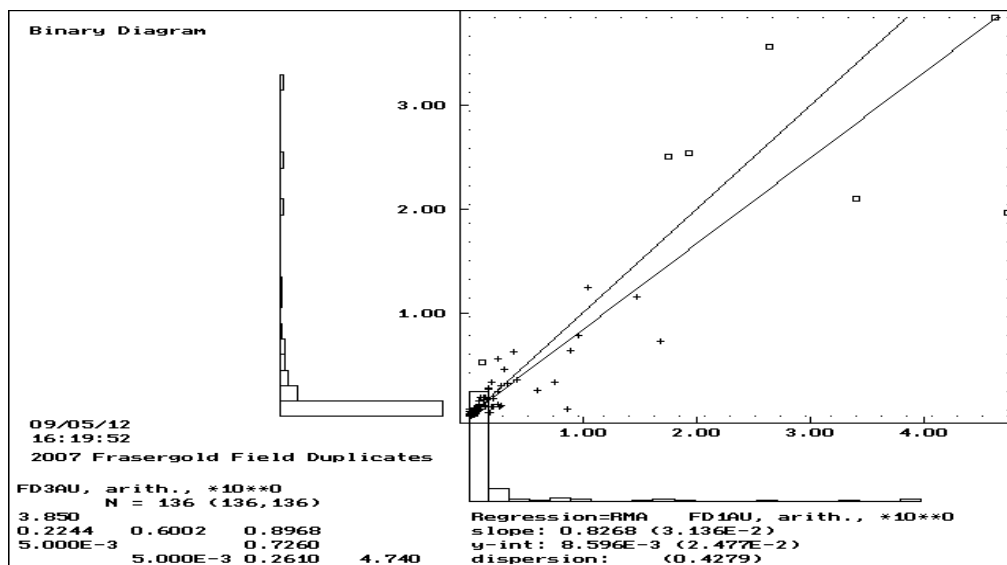


Figure 24. IPL analyses of first pulp (from 'field duplicate') versus third analysis representing separate reject material. Upper line is $y = x$; lower line is RMA fit to the data. With two values greater than 4.0 g/t Au removed, the RMA and $y = x$ lines are essentially coincident.

12.1.6 IPL Reproducibility of Pulp Analyses, 2007

Duplicate pulp analyses were done by IPL as part of the analytical program dealing with 'field duplicates'. These are shown in Figure 25. The data have a relatively large dispersion for repeat pulp analyses for various subsets, indicating problems with reproducibility over the entire range of values. The spread of data points is also large relative to the plot of Figure 22 comparing IPL analyses and those by ALS Chemex on the same pulps.

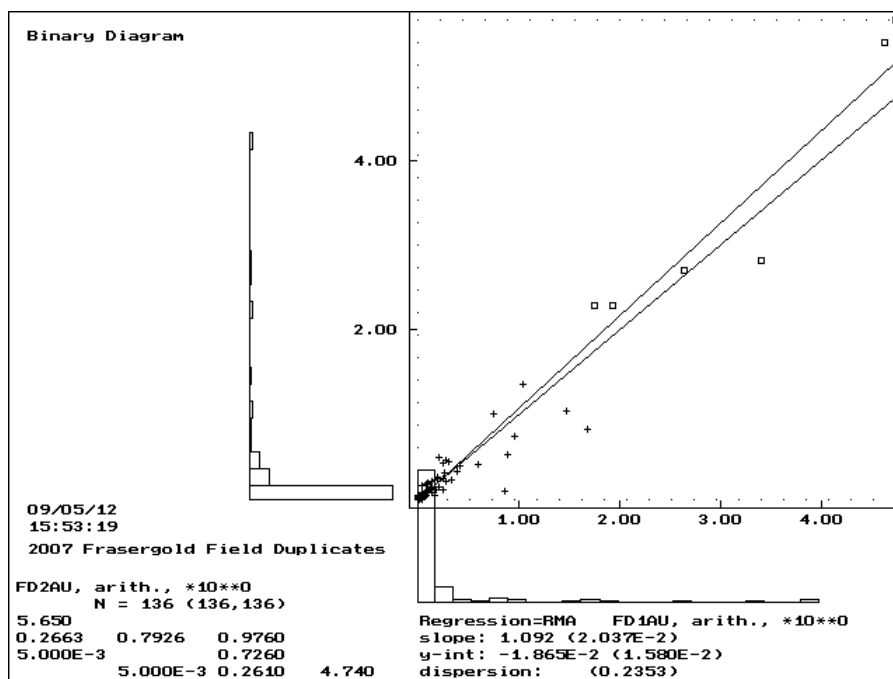


Figure 25. IPL duplicate analyses of same pulp (from 'field duplicate' data). Data in "g/t Au". Upper line is RMA line, greatly influenced by two values greater than 4.0 g/t Au. With those two values removed the RMA line essentially coincides with the lower line $y = x$.

12.1.7 Sources of Errors, 2007

The available data allows a very crude assessment of the magnitude of errors from various sources. In general, errors are additive as variances (squares of standard deviations) as follows:

$$s_t^2 = s_s^2 + s_p^2 + s_a^2$$

where s_t^2 = total error

s_s^2 = sampling error

s_p^2 = sample preparation error, and

s_a^2 = analytical error

Error Source	Dispersion	Std. Deviation	(Std. Deviation) ²
Analytical	0.2353	0.1793	0.03213
Preparation plus average analytical error	0.4403	0.3113	0.09693
Total Error	0.8287	0.586	0.3434

From the above listing, a rough estimate of the magnitude of various errors can be determined as follows:

$$(\text{Analytical error})^2 = 0.03213 \text{ or } 9.4\%$$

$$(\text{Preparation error})^2 = .09693 - .03213 = .06480 \text{ or } 18.9\%$$

$$(\text{Sampling error})^2 = 0.3434 - 0.09693 = 0.24647 \text{ or } 71.7\%$$

The various absolute errors are high because the deposit has a high nugget effect. Relative magnitude of errors (roughly 10% for analysis, 20% for preparation and 70% for sampling) is typical of high nugget effect gold deposits.

12.2. 2008 Program

12.2.1 2008 Standards, Blanks and Duplicate Sample Preparation

Duplicate samples were collected every 20 samples. The following sample bag was left empty but with a designated sample number and known as the check assay sample. At IPL the designated preparation duplicate sample was crushed into three samples; two were retained by IPL for pulverizing and duplicate assaying, while the third crushed sample was sent to ALS Chemex Laboratories (ALS Chemex) in North

Vancouver. The third crushed sample was pulverized at ALS Chemex and assayed as the check assay sample.

The preparation duplicate and check assay samples were prepared during milling at IPL. During the crushing stage, the duplicate sample was riffle or rotary split into three (3) 1000 gram crushed sub-sample portions. Two of the three (3) crushed 1000 gram samples were pulverized and homogenized at IPL as above, while the third (3rd) 1000 gram crushed sample was sent to ALS without being pulverized. The first two (2) samples were analyzed as above as the 'preparation original' and the 'preparation duplicate'. The third (3rd) of the three (3) crushed samples, the check assay sample, was sent to the ALS Chemex for similar analysis.

1. At the crushing stage, the sample is riffle or rotary split into three (3) 1000 gram crushed sub-sample portions.
2. Two of the three (3) crushed 500 gram samples are then pulverized and homogenized at IPL as above, while the third (3rd) crushed sample was sent to ALS Chemex without being pulverized.
3. The first two (2) samples are analyzed as above as the 'preparation original' and the 'preparation duplicate'.
4. The third (3rd) of the three (3) crushed samples are sent to the ALS Chemex for similar analysis and is defined as the 'check assay'.
5. The crusher is cleaned after every batch using barren material.
6. The pulverizer is cleaned after every batch using barren material.

Where field duplicates are taken, each represents $\frac{1}{4}$ core; where duplicate and check assay samples were taken, they represent crushed material that is riffled to three samples, two for analyses by IPL and one for analysis by ALS Chemex.

The field duplicates will over estimate the total error in the non-duplicated data because the field duplicates are $\frac{1}{4}$ -core rather than $\frac{1}{2}$ -core for the great bulk of analyses. The preparation duplicates will permit an evaluation of the combined preparation and assaying error by IPL. Check assays by ALS Chemex can be compared with the IPL preparation duplicates to monitor for bias. No data exist from the QC protocol to provide an independent monitor of IPL analytical precision for the 2008 program.

12.2.2 Results of 2008 Blanks and Standard Analyses

Blanks and standards used for quality control in 2008 are listed below. A change in standards was necessitated part way through the program because of depletion of standards. Quoted limits are two interlab standard deviations according to CDN Resource Laboratories Ltd., who prepared the standards.

Reference #	Standard	Au g/t
CDN-GS-P7A	Low	0.77 +/- 0.06
CDN-GS-2B	Medium	2.03 +/- 0.12
CDN-GS-3C	High	3.58 +/- 0.31
CDN-CGS-11	Low2	0.73 +/- 0.068
CDN-GS-2C	Medium2	2.06 +/- 0.15
CDN-GS-3D	High2	3.41 +/- 0.25
CDN-BL-3	Blank	<0.01

Hole 08310 to 08356 used the first group of Standards.

Hole 08357 to 08367 used the second group of Standards.

12.2.3 2008 IPL Analyses versus Check Analyses by ALS Chemex

Approximately every 20th sample analyzed by IPL was also sent to ALS Chemex as a monitor on bias. Check samples are separates from the initially crushed sample so represent preparation duplicates. The results are plotted in Figure 26 (522 lower grade values) and Figure 27 (9 higher grade values). For the lower grade values (less than 2 g/t) the data indicate a slight bias with ALS Chemex reporting about 8 percent higher, on average, than IPL. Note the very large dispersion of data for higher grade

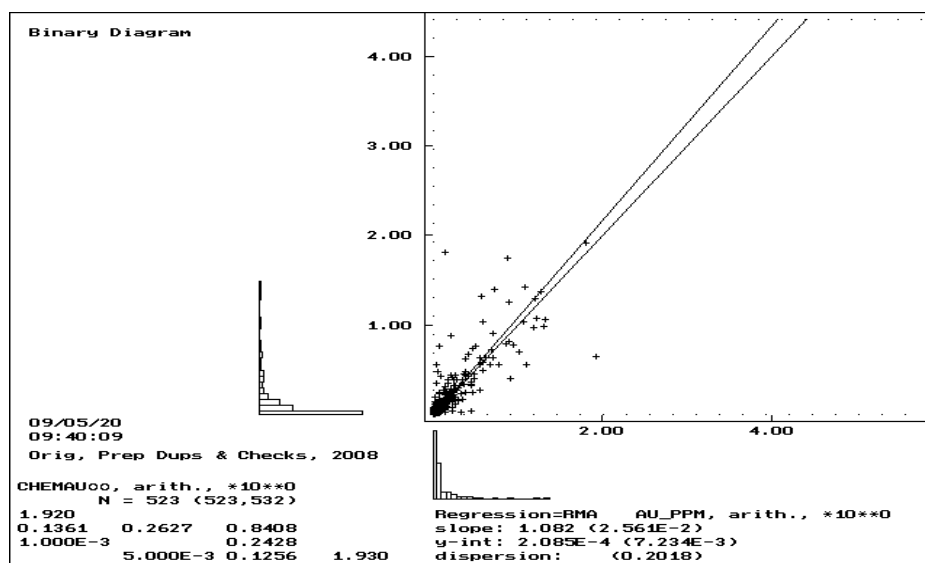


Figure 26. Plot of 523 paired analyses by IPL and ALS Chemex for the 2008 sampling/assaying program for values less than 2 g/t. The RMA line shown is essentially coincident with the y = x line.

relative to the dispersion for lower grade values, an indication of the relative difficulty of reproducing high grade values. The dispersion of lower grade values is substantially more than for the 2007 data.

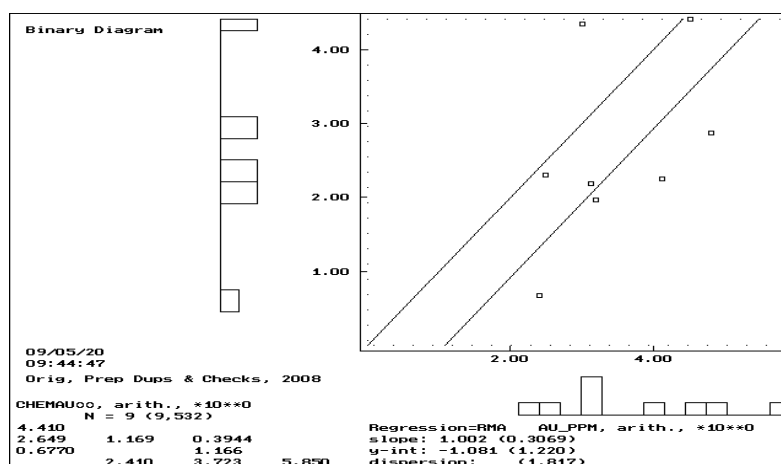


Figure 27. Plot of 9 paired values greater than 2 g/t Au, analyzed by both IPL and ALS Chemex. The upper line is $y = x$; the lower line is the RMA fit to the data. The large scatter is typical of high nugget effect gold.

12.2.4 2008 Field Duplicates

Approximately every 20th half core sample assay was paired with a facing quarter core sample assay and referred to as a field duplicate. The two are rough duplicates because the original value is based on ½-core whereas the 'duplicate' is based on ¼-core. The data are plotted on Figure 28, minus three very high values (greater than 15.0 g/t Au). This plot of 531 data pairs suggests significant bias between the two sets of data. However, if only data below 2.0 g/t Au are considered, as illustrated in Figure 29, the results below 2.0 g/t Au are seen to be unbiased. Values greater than 2.0 g/t Au are difficult to reproduce.

This plot represents total error in the data and even though it is probably somewhat of an overestimate, it is less than the total error contained in the 2007 data.

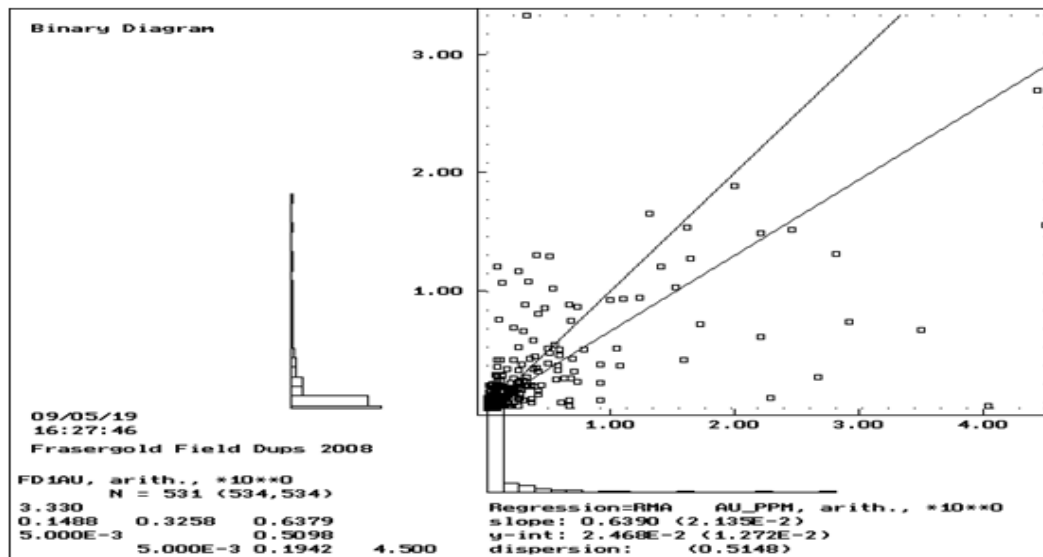


Figure 28. Plot of 503 IPL 'field duplicate' values for the 2008 sampling/assaying program. Top line is $y = x$; bottom line is RMA fitted to the data. See text for explanation.

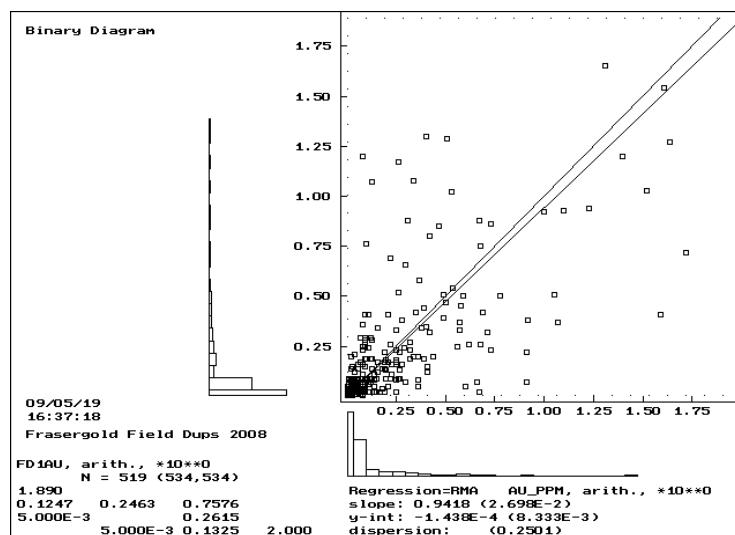


Figure 29. Plot of IPL field duplicate data values less than 2 g/t Au, 2008 data. Scatter is unbiased and is much less than for comparable 2007 data.

12.2.5 2008 Preparation Duplicates

Preparation duplicates by IPL are plotted on Figures 30 and 31 for lower (<2.0 g/t Au) and higher (>2.0 g/t Au) grades respectively. The data demonstrate the unbiased nature of the lower grade values (Figure 30) and the difficulty in reproducing higher grade values (Figure 31).

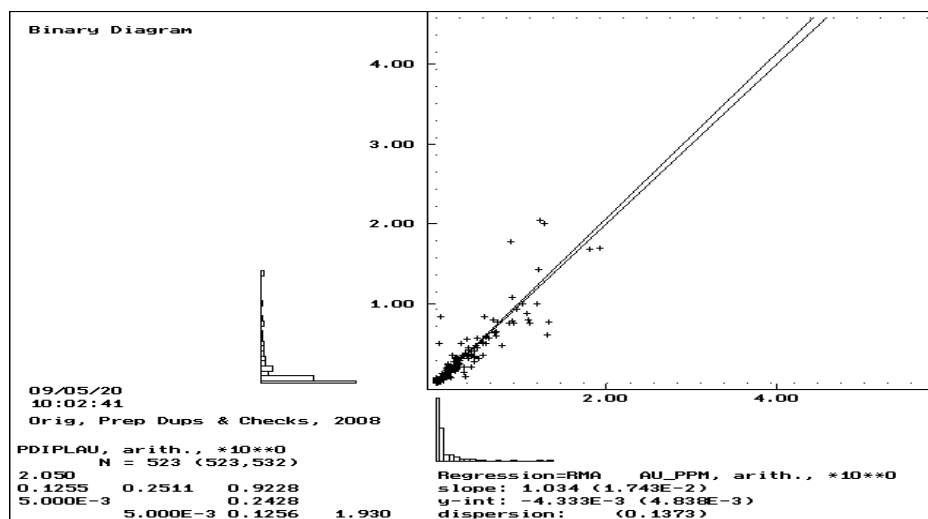


Figure 30. Sample preparation duplicate analyses by IPL for data less than 2.0 g/t Au. Lower line is $y = x$; upper line is RMA line fitted to the data. The two lines are statistically equivalent. Dispersion about the line is substantially less than for corresponding 2007 data.

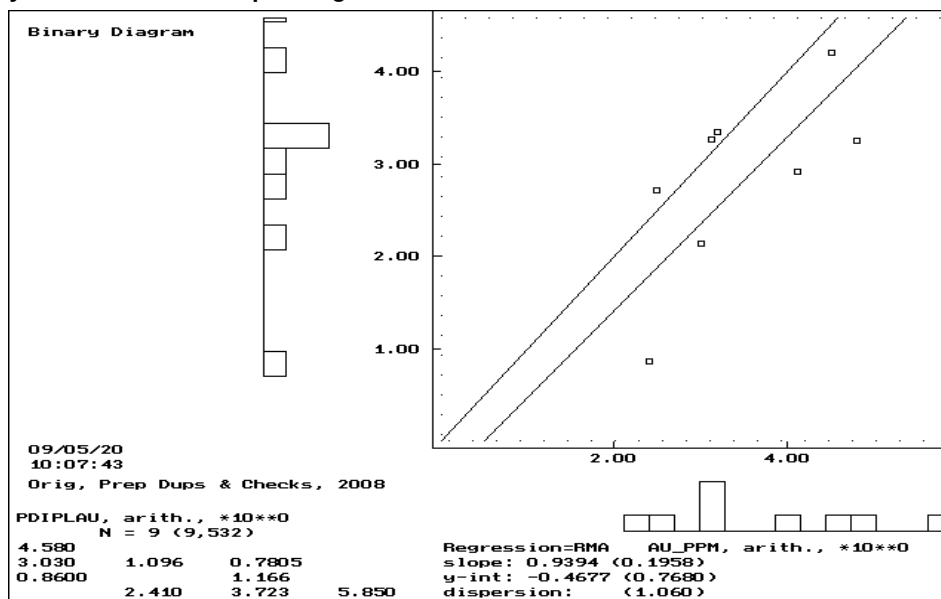


Figure 31. Sample preparation duplicate analyses greater than 2.0 g/t Au by IPL. Upper line is $y = x$; lower line is RMA fit to the data. The disposition of values to the right of the $y = x$ line indicates the difficulty of reproducing higher grades, even within the same lab.

12.3. Quality Assurance and Quality Control Conclusions

Quality control data for the 2007 and 2008 drilling programs on the Frasergold property show that the data are of adequate quality upon which to base a resource estimate. The bulk of the analyses are below 2.0 g/t Au and show no significant global bias. These lower grade values have a moderately large dispersion characteristic of gold deposits with a high nugget effect. Values in excess of about 2.0g/t Au are reproduced poorly, perhaps in part because the duplicates are based on $\frac{1}{4}$ core samples rather than $\frac{1}{2}$ core samples like the original analyses.

Monitoring analyses by ALS Chemex show small proportional departures from IPL data but are compensating over the two years. IPL replicate analyses of standards and blanks indicates an acceptable level of accuracy. Nevertheless, field duplicates and check samples with values greater than 2.0 g/t Au should be repeated because in general, original analyses are higher than those obtained from the duplicated samples.

The distribution of errors is roughly 10% analytical error, 20% sample preparation error and 70% sampling error based on 2007 data. Absolute values of each of these sources of error are relatively high and all could probably be reduced somewhat by taking larger samples, refining the sample reduction protocol and more care in preparation and sampling of pulps. Nevertheless, errors in analyses for samples from a high nugget effect deposit, such as Frasergold, will still be relatively high. Consequently, block estimates will require a substantial number of such samples in order to provide estimates of reasonable quality. For example, if 5.0 m composites are used for block estimation, then several such composites will be required for a block estimate.

Future work should use a sampling protocol patterned after that used in 2007, with the exception that somewhat fewer field duplicate samples are required than were taken. Both years and field duplicates should be equivalent to original samples. Moreover, it is particularly useful if preparation duplicates are taken from a sample duplicate and duplicate pulps from one of the preparation duplicates are also analyzed. This produces a coherent data set with which to evaluate sources of error. All sampling involved in the protocol should be analyzed in the normal routine of analytical work rather than having some of the analyses done months later. This will ensure that the quality control data reflect the operating conditions by samplers and lab personnel at the time the majority of data are obtained.

13. Mineral Processing And Metallurgical Testing

13.1. 1984 to 1993 Programs

Previous operators had conducted preliminary test work in order to determine general sampling and gold measurement methods. The following discussion is based on historical metallurgical work conducted on the Frasergold mineralization.

In 1985, Eureka provided Coastech Research Inc. ("Coastech") with a 700 kg bulk sample to determine the gold content of the sample and a reasonable extraction method for preliminary metallurgical evaluation (Marchant, 1985). Coastech crushed the entire bulk sample to less than 1.3 cm and following a procedure of crushing, riffle splitting and quartering to ensure proper mixing., An approximate 500 kg subsample was extracted. This sample was subjected to gravity separation methods of jigging and tabling in order to concentrate gold and gold-bearing rock. The concentrate was then cyanide leached in a stirred reactor and the gold content determined. Composites of the sample were prepared and submitted to three independent analytical laboratories and analyzed for gold content by fire assay. A metallic screening process was applied at one laboratory (Min-En Labs).

The 510 kg bulk sample produced 8.5 kg of concentrate. The concentrate was leached in cyanide with a pH of 11.0 for 48 hours. The leach solution was then placed in activated charcoal, dried and the gold content measured by fire assay techniques. Marchant, 1985 reports, "The results of this work indicated that the gold content of the bulk sample as received is 4.70 g/t Au (0.137 oz/T Au)." The primary author considers the information provided accurately represents the preliminary study of mineralized material from the Frasergold property.

In June, 1990, Asarco submitted approximately 4,500 kg of mineralized material to Bacon, Donaldson and Associates Ltd. of Richmond, B.C. in order to conduct preliminary metallurgical testing of Frasergold mineralization. The sample material was supplied as four composite samples with gold grades of 1.62, 2.53, 2.15 and 2.90 g/t Au, resulting in a estimated average grade of 2.16 g/t Au.

Three primary testing methods were employed by Bacon, Donaldson and Associates Ltd. on the material: whole ore cyanidation, gravity concentration, and flotation.

Coarse material (minus 3/8" and minus 1 1/2") bottle roll cyanide tests yielded 22 to 40% gold recovery after 96 hours with high cyanide consumption rates of 2.2 kg NaCN/tonne of material. Fine grind (66% passing 200 mesh) cyanide tests resulted in gold extractions of 79% and 82% after 48 hours of leaching with cyanide consumption of 3.5 kg/tonne of mineralized material.

Gravity concentration tests were conducted using four 45 kg samples. Jigging of the material resulted in 7 - 20% of the measured gold recovered in the concentrate, with gold observed in the jig concentrates.

Five 45 kg batch flotation tests were conducted with gold recovery ranging from 87 to 92%. The flotation tests were conducted at the same grind as the fine leach tests (66% passing 200 mesh). Bacon, Donaldson and Associates concluded that the higher gold recoveries indicate the gold is associated with sulphide mineralization and that the gold is sufficiently coarse to require a gravity concentration stage prior to flotation.

In addition, Bacon, Donaldson and Associates Ltd. determined a Bond Work Index of 11.4 kwh/tonne

The primary author has reviewed copies of the original reports and test data for the 1985 and 1990 test work completed for Eureka and Asarco. This information is historical in nature and is provided to indicate the advanced level of exploration conducted on the Frasergold property.

13.2. 2007 and 2008 Programs

Hawthorne did not conduct any metallurgical testing of mineralized material on the Frasergold property during their exploration programs.

14. Mineral Resource Estimates

14.1. Mineral Resource And Mineral Reserve Estimates

Historically, at least three estimations of gold content have been made by engineering firms for previous operators of the project. This information is presented in Sections 4 (History) and 14 (Mineral Processing and Metallurgical Testing). These historic estimates do not conform to the current "CIM Estimation of Mineral Resources and Reserves Best Practices Guidelines", issued in 2003 and modified last with adoption of the "CIM Definition - Standards for Mineral Resources and Mineral Reserves" in 2014 and therefore are not in compliance with Canadian National Instrument 43-101 standards for disclosure, and should not be relied upon. However, the authors believe that sufficient original data of acceptable quality exists and based on underground channel sampling, underground bulk sampling and 74 surface diamond drill holes completed in 2007 and 2008 (see Sections 11.2.3 to 11.2.5), this historic data could be utilized in the preparation of a mineral resource estimate. Drilling to date is insufficient to adequately assess the entire 10 kilometer strike length of the Frasergold mineralized zone.

In the spring of 2009 G. Giroux completed a resource estimation of the Frasergold property based on the historical and recent 2007-08 drilling in the Main Zone. Appendix IV lists the drill holes used in the estimate. Trench data and underground channel samples were not used in the estimation.

As no further drilling has been completed since this 2009 estimate was completed it is still current and valid.

14.2. Data Analysis

Table 14.2 Summary of drilling on Frasergold Property.

Year	Company	DDH Holes		RC Holes	
		No. Holes	Total Length (m)	No. Holes	Total Length (m)
1983	Eureka Resources Inc.	5	1,644		
1984	Eureka Resources Inc.	10	2,875		
1985	Eureka Resources Inc.	2	677		
1986	Eureka Resources Inc.	20	2,021	4	388
1987	Eureka Resources Inc.	5	469	21	1,707
1988	Eureka Resources Inc.	18	1,924	37	2,458
1990	Eureka Resources Inc.	26	4,684	43	4,334
1991	Eureka Resources Inc.			117	11,461
1993	Eureka Resources Inc.			20	1,020
2007	Hawthorne Gold Corp.	16	3,615		
2008	Hawthorne Gold Corp.	58	10,414		
	Totals	160	28,323	242	21,368

Exploration by Eureka Resources identified difficulties in higher grade gold analysis using a standard gold fire assay. The coarse nature of gold particles produced a significant nugget effect. For the 2007 and 2008 drill program completed by Hawthorne routine fire assay samples using a one assay tonne sample. The sampling method and analytical procedures are described in Sections 9 (Sampling Method and Approach) and 10 (Sample Preparation, Analyses and Security).

Where the original fire assay for gold differed from the screen fire assay the screen fire assay was used. A total of 2,496 assays had different values produced by the two methods. The mean grade from these 2,496 screen fire assays was 1.142 g/t Au compared to 1.106 g/t Au from the original fire assay. Figure 32 below shows a scatter plot of the samples with different results by the two methods. While the screen fire assay makes more sense as all coarse gold should be recovered the results are very similar with the best-fit regression line shown in red which is very close to the equal value line shown in black and the highest sample is responsible for this. A log transformed scatter plot (Figure 33) shows the same data with no bias indicated and the data clustering about an equal value line.

SCREEN FIRE ASSAY VS FIRE ASSAY FOR AU

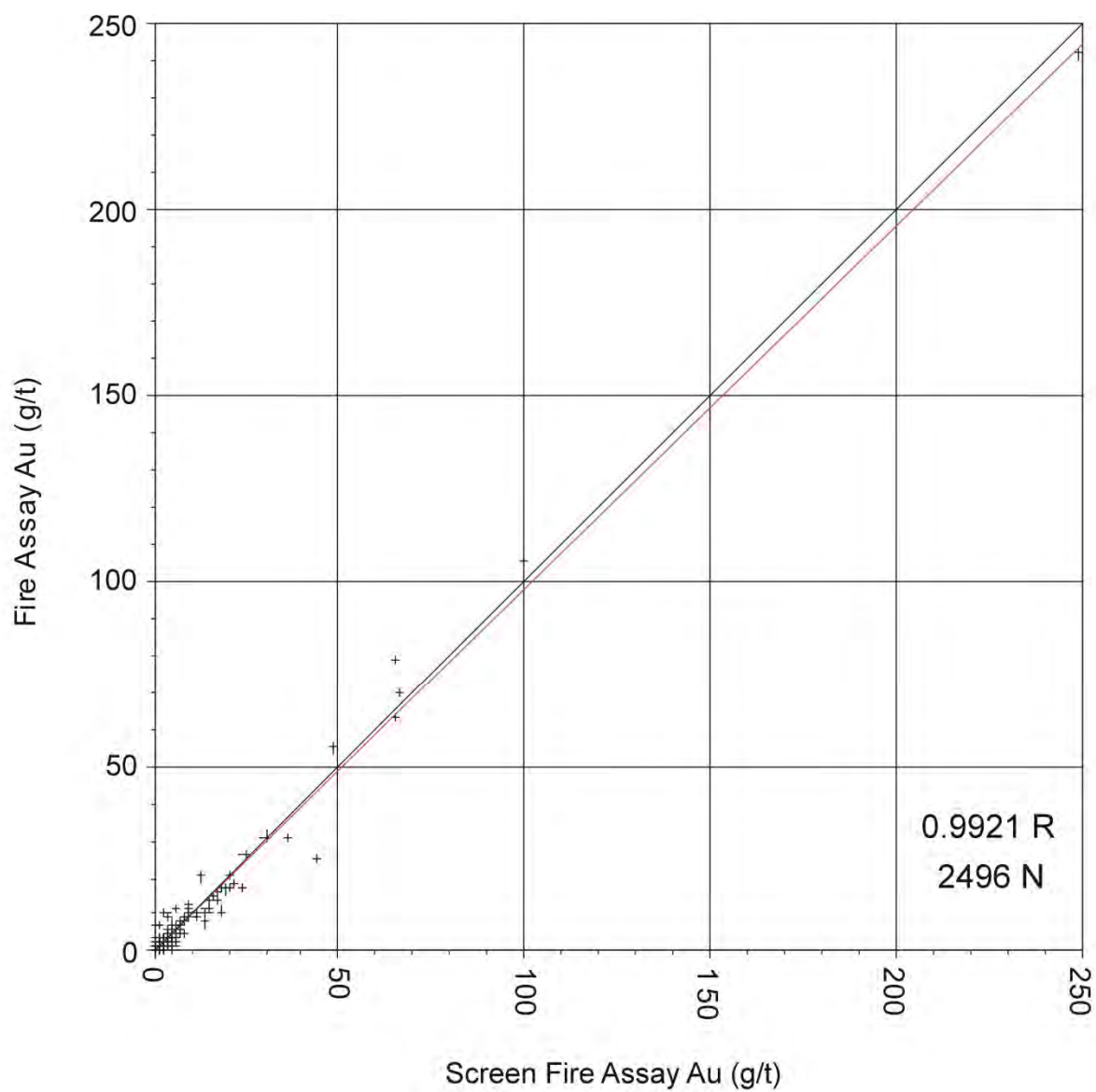


Figure 32. Scatter plot showing screen fire assay and regular fire assay Au values.

SCREEN FIRE ASSAY VS FIRE ASSAY FOR AU

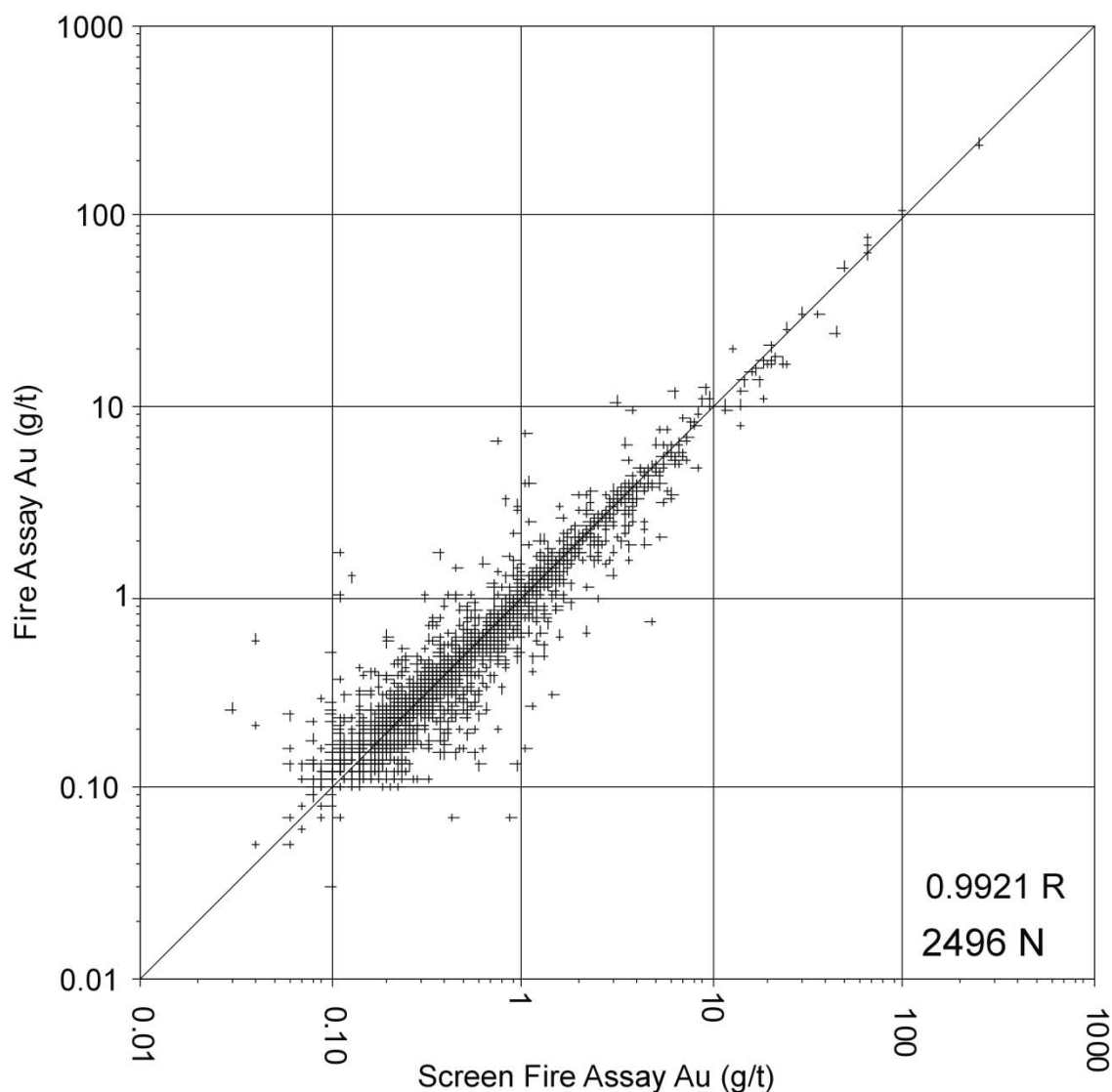


Figure 33. Lognormal scatter plot showing screen fire assay and regular fire assay Au values.

14.3. Geologic Model

Geologists from Hawthorne Gold produced a 3-dimensional solid that constrained a higher grade style of mineralization within the Main Zone. The solid was built in Vulcan from cross section interpretations. A lower grade envelope surrounding this zone was built to constrain grades outside the high grade solid. The model and resource estimate included the Main Zone mineralization and exploration holes extending

to the north-west (NW Zone) along strike and to the south (SE Zone). Figure 34 below shows the Main Zone with the higher grade core in red, the lower grade envelope in dark grey and the surface topography in light grey.

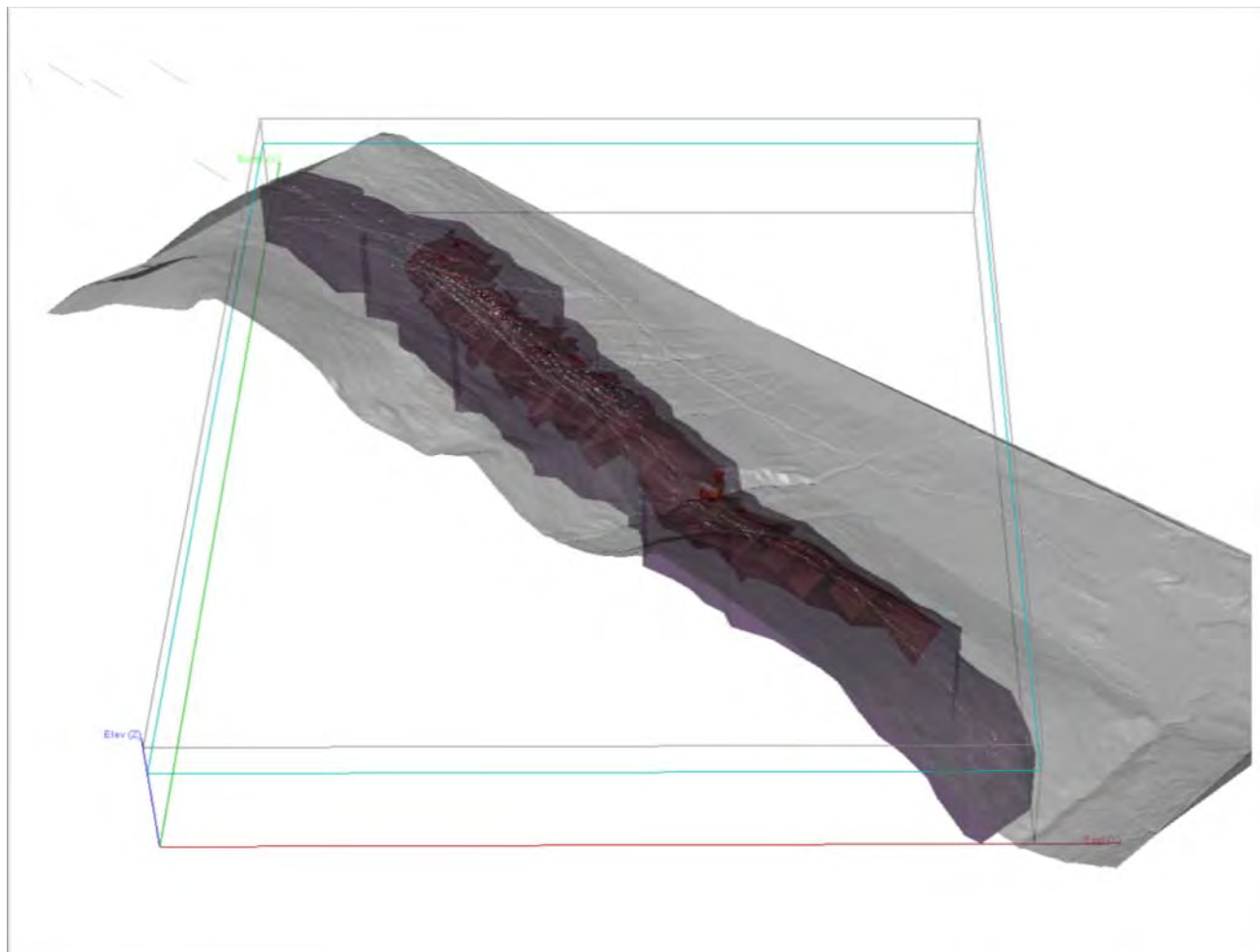


Figure 34. Isometric view looking north showing the High grade core zone in red with the dark grey lower grade envelope around it and the light gray showing surface topography.

14.4. Diamond Drilling versus RC Drilling

Due to the different drilling techniques and the coarse nature of the gold the RC results are compared to the diamond drill results within similar volumes. The comparison was made within the higher grade solid and within the surrounding lower grade envelope.

Figure 35 shows the location of assays in both the low grade envelope and high grade core of the Main Zone. Samples in the high grade zone are colour coded in red and magenta for diamond drill samples and RC samples respectively while samples in the low grade zone are coloured green and blue.

Figures 36 and 37 show log-normal cumulative distribution curves for gold within the low and high grade areas colour coded by drill type. Within the low grade envelope there appears to be a slight bias with RC

Eureka Resources Inc.

Frasergold Project

Cariboo Mining Division, B.C.

Location of assays in the low-grade envelope
and high-grade core of the Main Zone

DWG: 521-15-18

June 23, 2015

Scale: 1:12,500

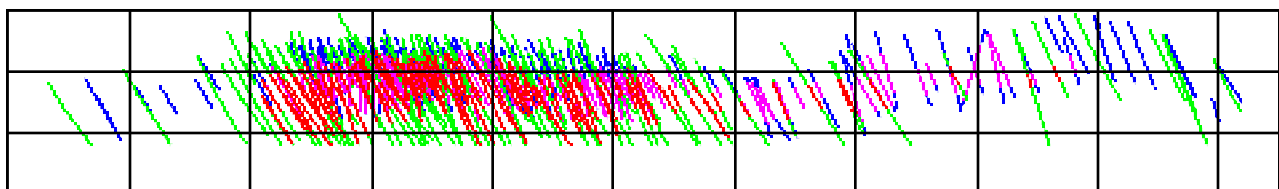
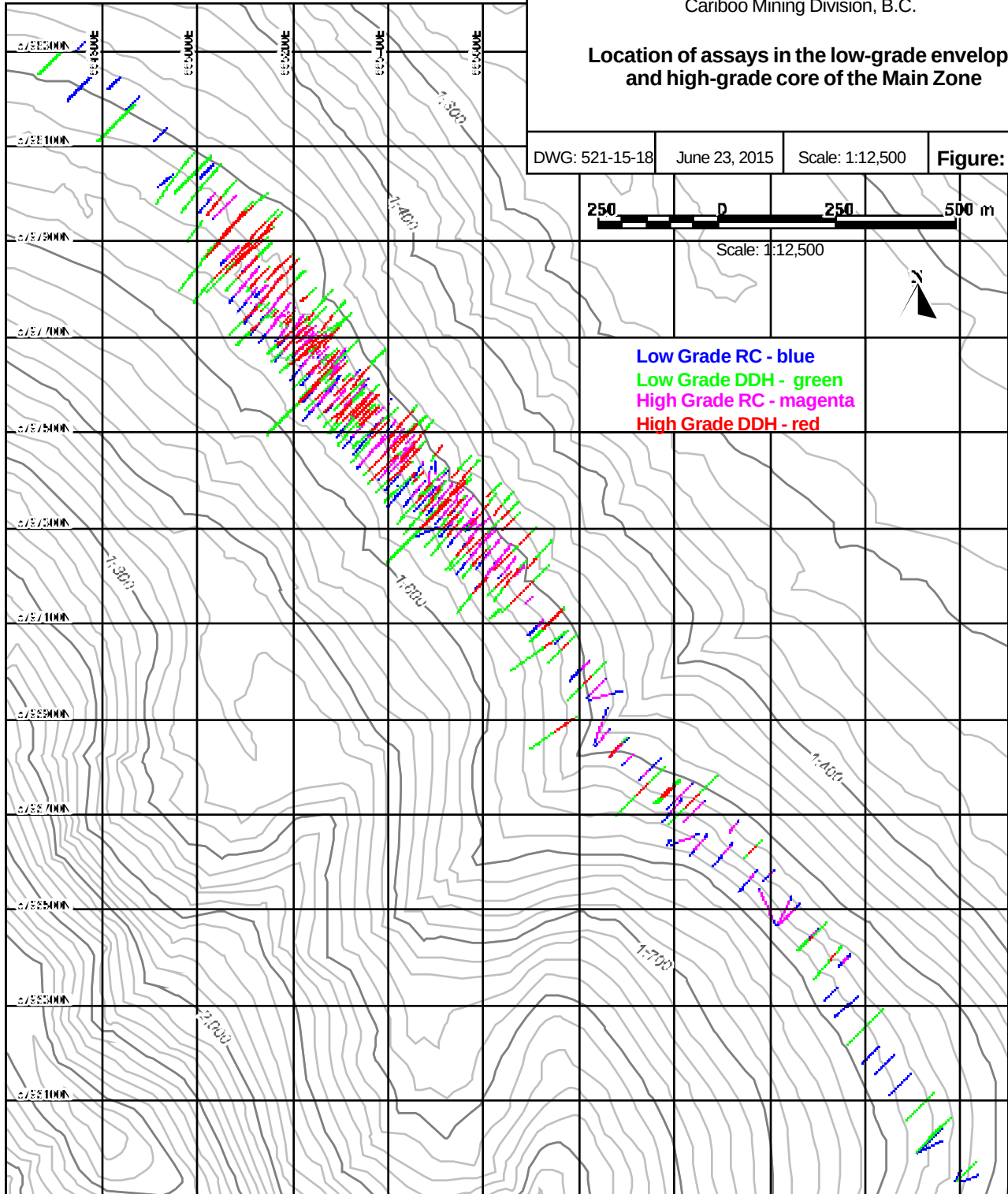
Figure: 35

250 0 250 500 m

Scale: 1:12,500



Low Grade RC - blue
Low Grade DDH - green
High Grade RC - magenta
High Grade DDH - red



UTM 10 NAD83 Contour interval = 20m

samples overestimating gold relative to diamond drilling above about 0.006 g/t. The mean value of 4,895 gold assays from RC drill results is 0.218 g/t compared to the mean of 8,935 gold assays from diamond drill core of 0.152 g/t.

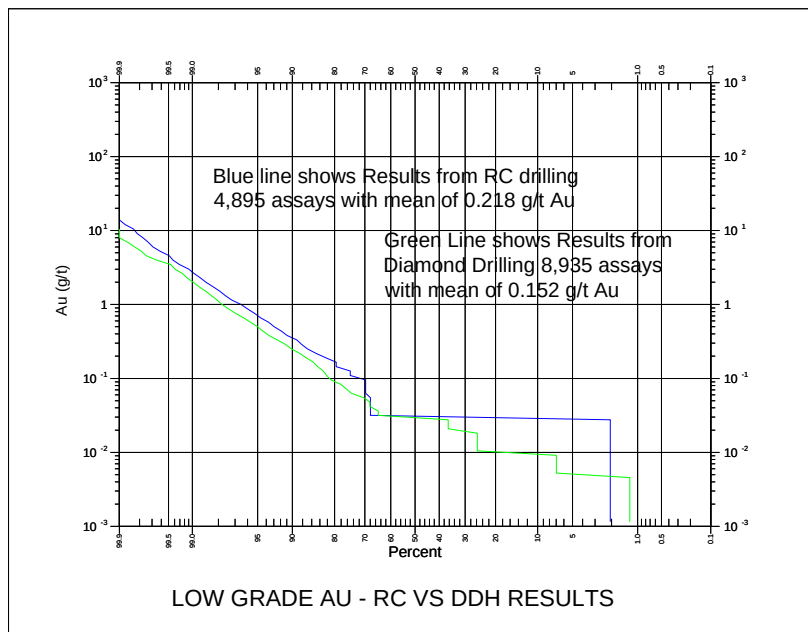


Figure 36. Lognormal cumulative distribution of gold within low grade envelope, RC vs DDH results.

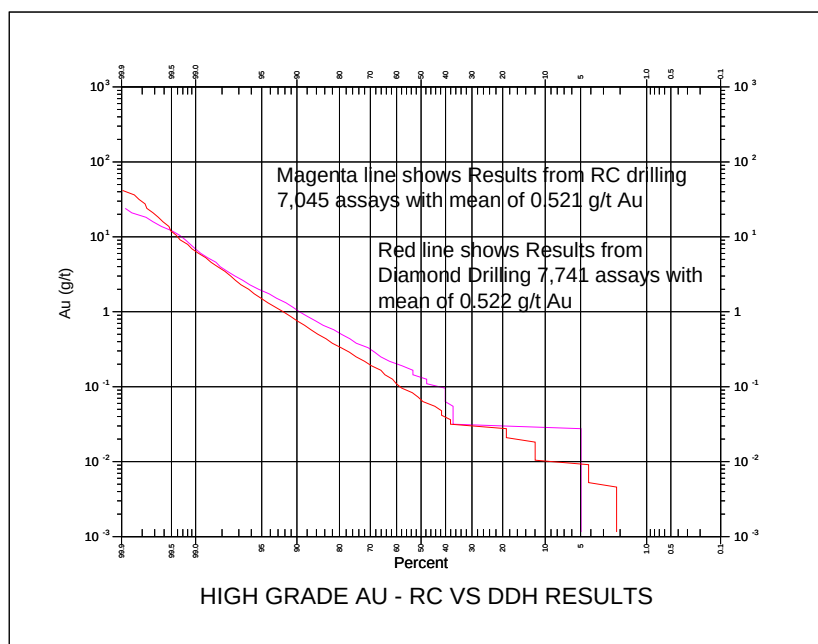


Figure 37. Lognormal Cumulative Frequency Plots for Gold in High Grade and Low Grade Zones showing results from RC and Diamond Drilling.

Within the high grade core zone the comparison is better with no bias indicated and the mean values much closer with 7,045 RC assays at 0.521 g/t Au compared to 7,741 DDH assays at 0.522 g/t Au. In general the bias on low grade is small and the data is sufficiently similar to use both styles of drill results in the resource estimation.

Using the geologic 3D solid model, drill holes were compared to the solid shapes and then point at which the drill holes entered and left each solid was recorded. Based on these intersections individual assays were tagged with a code LG or HG for low grade envelope and high grade core respectively. The statistics for gold within the two domains are tabulated below.

Table 14.4.1 Summary of Au assay statistics sorted by Domain

	Low Grade Envelope	High Grade Core
Number of Assays	15,896	14,947
Mean Value Au (g/t)	0.163	0.517
Standard Deviation Au (g/t)	1.420	3.558
Minimum Value Au (g/t)	0.001	0.001
Maximum Value Au (g/t)	116.88	252.00
Coefficient of Variation	8.69	6.88

Both domains have erratic high values resulting in high standard deviations and resulting high coefficients of variation. Both domains showed highly skewed gold distributions and lognormal cumulative distribution plots were used to partition out multiple overlapping lognormal populations.

Lognormal cumulative probability plots were produced for Au in both the mineralized zones and the individual plots are shown in Appendix V. Each variable showed multiple overlapping lognormal populations. In each case the individual populations were partitioned out with the mean grade and proportion of the total population summarized in the following tables (individual populations shown on plots as open circles). The multiple overlapping populations for each variable result in different grade distributions and capping should be applied at different levels in each case. In most cases a small proportion of high grades are present and could, at this level of drill information, be considered erratic.

Table 14.4.2 Summary of gold populations present in High Grade Core Domain

Population	Mean Au (g/t)	Percentage of Total Data	Number of Assays
1	250.80	0.01 %	2
2	34.42	0.17 %	25
3	16.35	0.37 %	55
4	7.26	0.75 %	112
5	2.96	2.09 %	312
6	0.49	25.23 %	3,771
7	0.05	71.38 %	10,670

Within the high grade core domain population 1 is clearly erratic and should be capped at 2 standard deviations above the mean of population 2. A total of 2 assays were capped at 70 g/t. Populations 2, 3 and 4 represent mineralized vein material that is very difficult to model and predict. This material is not considered erratic outliers based on the amount of samples present (1.3% of total). To avoid undue smearing of these populations, however, an Indicator Kriging approach was employed. The photograph below, taken from the underground workings, shows the problems attached to estimating high grades. A drill hole could pass through this zone and depending on location could give many different estimations of grade in this area. Populations 5 and 6 represent the wider spread low grade mineralization present between the veins perhaps. Finally, population 7 represents the background mineralization within the zone.

**Photo 1. View from underground of the mineralized vein swarms**

Within the lower grade envelope the top 3 populations, representing a combined 0.12% of the total samples, are considered erratic, isolated high grades and are capped at 2 standard deviations above the mean of population 4. A total of 21 gold assays are capped at 7.9 g/t.

Table 14.4.3 Summary of gold populations present in Low Grade Envelope Domain

Population	Mean Au (g/t)	Percentage of Total Data	Number of Assays
1	102.4	0.02 %	3
2	19.74	0.05 %	8
3	13.04	0.05 %	8
4	3.02	1.19 %	189
5	0.31	13.86 %	2,203
6	0.07	14.02 %	2,229
7	0.02	70.87 %	11,256

The results of capping are shown below in Table 14.4.4.

Table 14.4.4 Summary of capped Au assay statistics sorted by Domain

	Low Grade Envelope	High Grade Core
Number of Assays	15,896	14,947
Mean Value Au (g/t)	0.141	0.493
Standard Deviation Au (g/t)	0.506	2.218
Minimum Value Au (g/t)	0.001	0.001
Maximum Value Au (g/t)	7.90	70.00
Coefficient of Variation	3.59	4.50

14.5. Composites

Uniform downhole composites 5 m in length were produced to honour the domain boundaries. Intervals at domain boundaries less than 2.5 m were combined with adjoining samples while those greater than or equal to 2.5 m were left. This produced a composite file of uniform support 5 ± 2.5 m.

The distribution of gold composites within the higher grade core zone indicated multiple populations (see Appendix V). The 3 highest grade populations were assumed to represent the vein style mineralization while the lower three populations represented the more prevalent disseminated style mineralization. A threshold of 1.6 g/t would separate these two groups. The strategy for modeling and estimation was to model the highest three populations using an indicator approach.

IND = 0 if comp Au < 1.6 g/t. Disseminated Au

= 1 if comp Au $\geq$ 1.6 g/t. Vein Style Au

The approach was to model the outer envelope separately from the inner core zone for gold. The inner higher grade core zone was then modeled in two stages. Material less than 1.6 g/t was modeled and

estimated as disseminated style mineralization while material greater than or equal to 1.6 g/t Au was modeled and estimated using an indicator approach.

Table 14.5.1 Summary of Au composite statistics sorted by Domain

	Low Grade Envelope Au (g/t)	High Grade Core	
		Vein Style Au (g/t)	Disseminated Style Au (g/t)
Number of Assays	4,722	207	3,865
Mean Value Au (g/t)	0.126	3.686	0.272
Standard Deviation Au (g/t)	0.296	2.426	0.316
Minimum Value Au (g/t)	0.001	1.603	0.001
Maximum Value Au (g/t)	3.65	14.009	1.598
Coefficient of Variation	2.35	0.66	1.16

14.6. Variography

Pairwise relative semivariograms were produced for gold within the outer envelope, inner core background and inner core high grade. Gold within the outer envelope showed a geometric anisotropy with longest continuity of 180 m along azimuth 135° and second longest continuity along azimuth 45° dipping -45° to the northeast. Gold within the inner core representing the background mineralization showed longest continuity of 160 m also along azimuth 135° with the second longest continuity of 60 m along azimuth 225° dipping -45° to the southwest. Finally the zero/one indicator was modeled within the inner core zone with the longest direction of continuity found along azimuth 135°. In all cases nested spherical models were fit to the data.

The models are included as Appendix V and the parameters of the models are tabulated below.

Table 14.6.1 Summary of semivariogram parameters for Au

Domain	Azimuth	Dip	C0	C1	C2	Short Range (m)	Long Range (m)
Low Grade Envelope Au	135	0	0.30	0.32	0.33	12.0	180.0
	45	-45	0.30	0.32	0.33	25.0	100.0
	225	-45	0.30	0.32	0.33	5.0	40.0
Core Zone Background Au	135	0	0.40	0.22	0.20	20.0	160.0
	45	-45	0.40	0.22	0.20	10.0	30.0
	225	-45	0.40	0.22	0.20	8.0	60.0
Core Zone High Grade Indicator	135	0	1.70	0.13	0.12	10.0	100.0
	45	0	1.70	0.13	0.12	20.0	40.0
	0	-90	1.70	0.13	0.12	5.0	40.0

14.7. Block Model

A rotated block model with block dimensions of 10 x 10 x 5 m was built to cover the mineralized zone solids. The block model origin is as follows:

Lower Left Corner

666489.63 East	Column Size – 10 m	70 columns
5795666.996 North	Row Size – 10 m	340 rows

Top of Model

1740 Elevation	Level Size – 5 m	95 levels
----------------	------------------	-----------

Rotation – x axis rotated 40 degrees counter clockwise

The northwest and southeast extensions of the main zone were also estimated and two additional models were created to cover these areas. These block models had the following origins.

Lower Left Corner of NW Extension

663614.713 East	Column Size – 10 m	160 columns
5797693.039 North	Row Size – 10 m	460 rows

Top of Model

2050 Elevation	Level Size – 5 m	196 levels
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Rotation – X axis rotated 40 degrees counter clockwise

Lower Left Corner of SE Extension

665940.131 East	Column Size – 10 m	300 columns
5792741.553 North	Row Size – 10 m	189 rows

Top of Model

2050 Elevation	Level Size – 5 m	196 levels
----------------	------------------	------------

Rotation – X axis rotated 40 degrees counter clockwise

For each block, the percentage of the block below surface topography and the percentage within each of the solids were recorded. The topography used was provided by Eagle Mapping Ltd. and had 1 m contours.

An isometric view of the three estimated block models is shown in Figure 38.

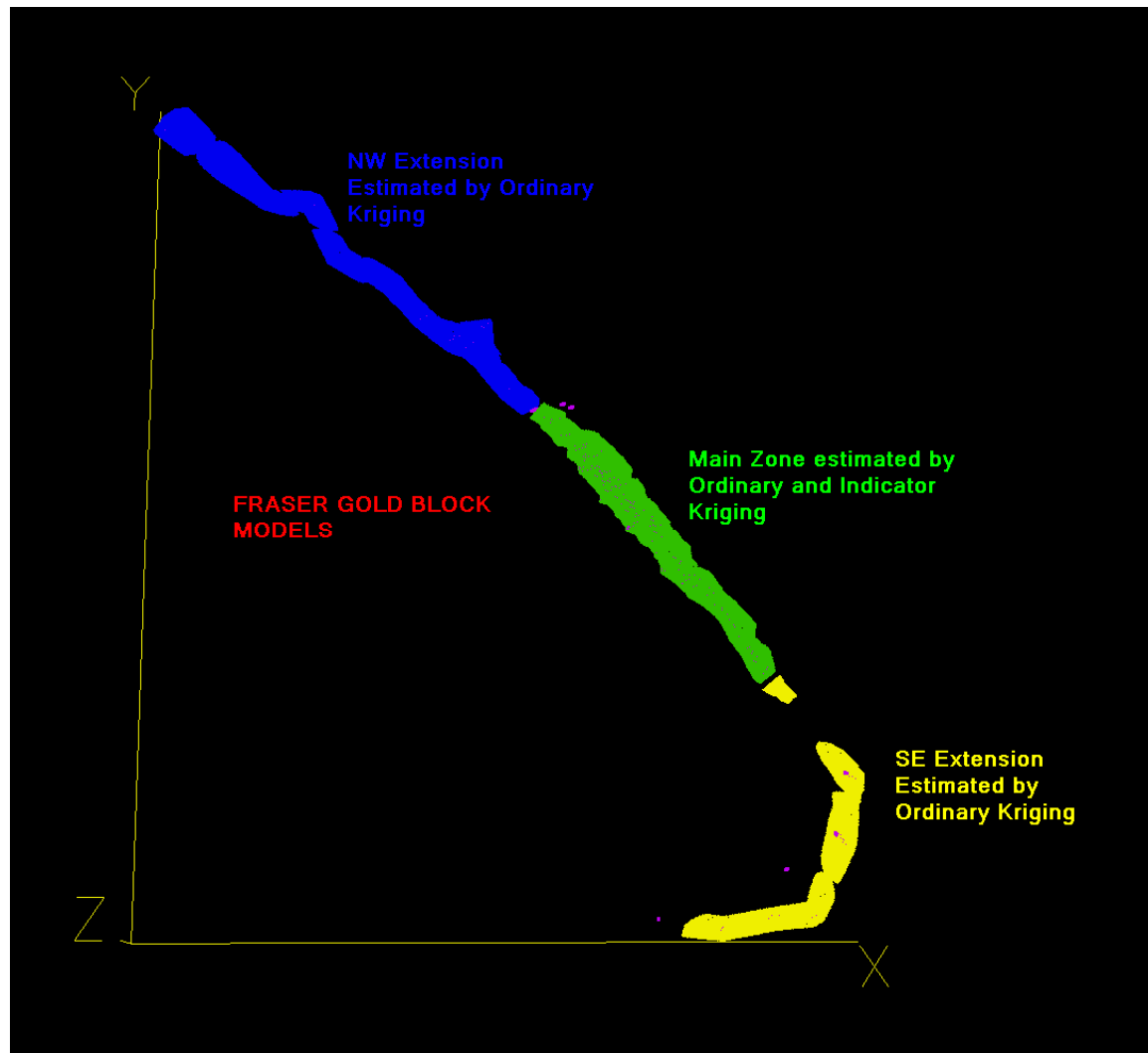


Figure 38. Isometric view of the three estimated block models.

14.8. Bulk Density

A total of 128 specific gravity determinations were made in 2007 from pieces of drill core in a variety of rock types. The method used was as follows:

$$\text{Specific Gravity} = \frac{\text{Weight of Sample in air (g)}}{(\text{Weight in Air (g)} - \text{Weight in Water (g)})}$$

The results are presented in Appendix VI and summarized below. There appears to be no correlation between high grades and high specific gravities and very little difference within the various rock types, so for this resource estimation the average of the 128 measurements, a value of 2.82 was used.

Table 14.8.1 Average specific gravity of Frasergold rock types

Lithology Code	Lithology	Number of Samples	Average SG
CBP	carbonaceous black phyllite	10	2.76
KP	knotted phyllite	72	2.83
SLST	siltstone	3	2.77
BBP	black banded phyllite	40	2.81
QV	quartz vein	1	2.63
LS	limestone	1	2.77
P	phyllite	1	2.81
Average		128	2.82

14.9. Grade Interpolation

Gold grades were interpolated into blocks using ordinary kriging in a number of steps. First a grade for the low grade envelope mineralization was estimated into all blocks with some percentage within the low grade envelope using composites outside the higher grade core.

For blocks with some percentage within the higher grade core zone, a three step procedure was completed. The first step involved kriging the 0-1 indicator to determine the percentage of high grade that might be present within any block in the core zone. The next step was to estimate the grade of this high grade component by kriging all composites greater than or equal to 1.6 g/t Au within the high grade core (HG-AU). Finally the grade for the disseminated mineralization within the high grade core (DISS-AU) was estimated by kriging all composites within the high grade core less than 1.6 g/t Au. A weighted average was produced to represent the grade of the portion of each block within the high grade core.

$$\text{HG Core Au} = (\text{KRIGED IND} * \text{HG_AU}) + ((1 - \text{KRIGED IND}) * \text{DISS AU})$$

A gold grade for the block was then a weighted average of the two zones.

$$\text{TOT_AU} = (\% \text{LG} * \text{LG_AU}) + (\% \text{HG} * \text{HG Core Au}) / (\% \text{LG} + \% \text{HG})$$

In all cases ordinary kriging was completed in a series of passes with expanding search ellipses. The first pass used ellipse dimensions equal to $\frac{1}{4}$ of the semivariogram range in each of the three principal directions. A minimum of 4 composites were required within this ellipse centered on the block to estimate a given block. For blocks not estimated in Pass 1 a second pass using $\frac{1}{2}$ the range of the semivariogram was completed. A third pass using the full range and a fourth pass using twice the range were completed on un-estimated blocks. In all cases if more than 12 composites were found in any given search, the closest 12 were used.

Due to different search ellipses, blocks within the HG Core zone not estimated for the Indicator were assigned a zero value. Blocks with a HG Indicator estimated but not estimated for HG_Au were assigned the average of the HG population, a value of 3.69 g/t Au.

For the estimation of the NW and SE extensions only the low grade envelope composites were used. The area to the northwest of the main mineralized zone was estimated by ordinary kriging using the low grade envelope semivariogram and search ellipses in a similar manner to the main zone estimate. The southeast extension was estimated in three stages varying the strike of the search ellipse and semivariogram to fit the bend around the fold nose. Again only low grade envelope composites were used and the kriging method was as described above.

14.10. Classification

Based on the study herein reported, delineated mineralization of the Frasergold Deposit is classified as a resource according to the following definitions from National Instrument 43-101 and from CIM (2014):

"In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council on May 10, 2014, as those definitions may be amended."

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

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase "reasonable prospects for economic extraction" implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cut-off grade and

geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

The terms Measured, Indicated and Inferred are defined by CIM (2014) as follows:

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

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 estimated with sufficient confidence to allow the

application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

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

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

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

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

Geologic continuity is not easily quantified. It is based on confidence determined from surface mapping, drill hole logging and underground mapping that a reasonable understanding of what constrains

mineralization exists. Grade continuity can be quantified by the semivariogram. Blocks within the well drilled central portion of the deposit and estimated during Pass 1 (1/4 the range of the semivariogram) were classified as measured. Blocks estimated in Pass 2 were classified as Indicated. All other blocks estimated were classified as Inferred. The results are tabulated by class below. **At this time no economic studies have been completed and as a result the economic cut-off for this deposit is unknown.** The nearest analogous deposit with an economic study completed is Spanish Mt. Gold (Koffybert et al, Tetra Tech PEA, 2012). A gold cut-off of 0.20 g/t Au for Spanish Mt. was determined as the minimum grade of mineralized material within the pit that is sufficient to cover the cost of mining and milling, G & A, stockpile handling, stockpile maintenance and the incremental haul costs from the waste dump to the crusher. A base case gold price of \$1,350 per oz. was used in this study. Within the designed pit at Spanish Mt. a total of 167 million tonnes averaging .48 g/t were classified as M+I while an additional 3.7 million tonnes averaging 0.49 g/t Au were classified Inferred.

A cut-off of 0.5 g/t is highlighted as a possible cut-off at Frasergold for extraction by open pit methods, considering the smaller tonnage and lower current gold price of \$1,100 per oz.

Table 14.10.1 Measured resource, Frasergold Main Zone

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Au (grams)	Au Ounces
0.30	11,470,000	0.595	6,800,000	219,000
0.40	7,980,000	0.703	5,600,000	180,000
0.50	5,600,000	0.812	4,500,000	145,000
0.60	4,130,000	0.907	3,700,000	119,000
0.70	3,020,000	1.003	3,000,000	96,000
0.80	2,240,000	1.093	2,400,000	77,000
0.90	1,650,000	1.181	1,900,000	61,000
1.00	1,160,000	1.279	1,500,000	48,000

Table 14.10.2 Indicated resource, Frasergold Main Zone

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Au (grams)	Au Ounces
0.30	22,610,000	0.540	12,200,000	392,000
0.40	14,310,000	0.653	9,300,000	299,000
0.50	9,570,000	0.755	7,200,000	231,000
0.60	6,530,000	0.852	5,600,000	180,000
0.70	4,450,000	0.948	4,200,000	135,000
0.80	2,910,000	1.055	3,100,000	100,000
0.90	1,910,000	1.164	2,200,000	71,000
1.00	1,270,000	1.274	1,600,000	51,000

Table 14.10.3 Measured plus Indicated resource, Frasergold Main Zone

Au Cut-off	Tonnes > Cut-off	Grade > Cut-off		
(g/t)	(tonnes)	Au (g/t)	Au (grams)	Au Ounces
0.30	34,080,000	0.559	19,100,000	614,000
0.40	22,290,000	0.671	15,000,000	482,000
0.50	15,170,000	0.776	11,800,000	379,000
0.60	10,660,000	0.874	9,300,000	299,000
0.70	7,470,000	0.970	7,200,000	231,000
0.80	5,150,000	1.071	5,500,000	177,000
0.90	3,560,000	1.172	4,200,000	135,000
1.00	2,420,000	1.277	3,100,000	100,000

Table 14.10.4 Inferred resource, Frasergold Main Zone

Au Cut-off	Tonnes > Cut-off	Grade > Cut-off		
(g/t)	(tonnes)	Au (g/t)	Au (grams)	Au Ounces
0.30	26,530,000	0.473	12,500,000	402,000
0.40	14,940,000	0.571	8,500,000	273,000
0.50	8,270,000	0.670	5,500,000	177,000
0.60	4,510,000	0.774	3,500,000	113,000
0.70	2,580,000	0.874	2,300,000	74,000
0.80	1,260,000	1.005	1,300,000	42,000
0.90	790,000	1.098	900,000	29,000
1.00	440,000	1.225	500,000	16,000

Table 14.10.5 Inferred resource, Frasergold NW Extension

Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Au (grams)	Au Ounces
0.30	45,790,000	0.538	24,600,000	791,000
0.40	30,850,000	0.629	19,400,000	624,000
0.50	19,180,000	0.740	14,200,000	457,000
0.60	14,200,000	0.813	11,500,000	370,000
0.70	10,360,000	0.873	9,000,000	289,000
0.80	8,840,000	0.894	7,900,000	254,000
0.90	210,000	0.947	200,000	6,000
1.00	10,000	1.194	10,000	300

Table 14.10.6 Inferred resource, Frasergold SE Extension

Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Au (grams)	Au Ounces
0.40	77,000	0.546	42,000	1,400
0.50	43,000	0.632	27,000	900
0.60	41,000	0.635	26,000	800
0.70	1,000	0.704	1,000	30

Table 14.10.7 Summary of Frasergold resource at a 0.5 g/t Au cut-off

Zone	Class	Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
				Au (g/t)	Au (grams)	Au Ounces
Main	Measured	0.50	5,600,000	0.812	4,500,000	145,000
Main	Indicated	0.50	9,570,000	0.755	7,200,000	231,000
Main	M+I	0.50	15,170,000	0.776	11,800,000	376,000
Main	Inferred	0.50	8,270,000	0.670	5,500,000	177,000
NW	Inferred	0.50	19,180,000	0.740	14,200,000	457,000
SE	Inferred	0.50	43,000	0.632	27,000	900
Total	Inferred	0.50	27,493,000	0.718	19,727,000	634,900

23. Adjacent Properties

There are no mines in the immediate vicinity of the Frasergold property. The closest operating mine is Imperial Metal's Mount Polley copper-gold porphyry deposit located 84 kilometers to the northwest. Numerous gold and copper prospects are located throughout the region, including the Woodjam property 15 kilometers south of Horsefly, Spanish Mountain 60 kilometers to the northwest and the QR past producing mine site 90 kilometers to the northwest. None are immediately adjacent to the property but occur within a similar geologic setting and share some mineralogical attributes with Frasergold.

24. Other Relevant Data And Information

The authors of this report were provided with free and clear access to all data pertaining to the Frasergold property. The authors are not aware of any other relevant data or information pertaining to the Frasergold property that should be included in this report.

25. Interpretations And Conclusions

The results from the 2008 soil and silt sampling program demonstrated the best potential northwest of the mineralized Main zone. The anomalous gold values are interpreted to be derived from upslope in that area thus adding credence to gold mineralization continuing along strike towards the northwest of the main zone.

The results of the channel sampling within the underground working validated and verified results obtained from historical surface drilling, including a number of high grade gold samples. The results indicated good continuity of gold mineralization along strike to the northwest at approximately 310° azimuth, with the width of the zone of interest being approximately 30 meters. As a result further drilling and sampling is recommended to potentially prove up a resource for the Frasergold zone, and further exploration work should be planned to follow-up on the high grade samples encountered in the underground workings that may be correlated along strike with surface drilling intercepts. Because the quartz veins frequently boudinage as noted from underground observations and historical mapping, some veins can expand substantially in size. Consequently there is potential that high grade intercepts of mineable widths underground could be intercepted and proven with additional close spaced surface drilling.

Drilling was conducted at a density with the Main zone as recommended by Gary Giroux with the aim to bring the current drilling towards indicated and inferred status. The results of the diamond drill program continued to demonstrate the structurally complex nature of the deposit. Results of the 2008 drilling indicate the following:

1. Gold mineralization appears to be more continuous along strike than downdip or between drillholes on the same cross section.
2. Gold mineralization appears partially be controlled by S_0 and S_1 structures.
3. Weak gold mineralized ribbon veins interspersed with sporadic high grade gold ribbon veins are possibly due to intersection of fold noses.
4. Continuity along strike can be seen more consistently with high grade intersections whereas the low grade typically demonstrates continuity less consistently.
5. Weaker gold grades may be due to drillholes intersecting veins along fold limbs and not within fold nose regions.

6. There may possibly be more than one controlling mineralizing structural event other than S_1 .
7. Mineralization is observed to be contained within "stacked" zones; some of the zones are defined by ribbon veining while others are devoid of ribbon veining.

26. Recommendations

The previous geochemical sampling and drilling programs on the property have demonstrated mineralization continues to depth thus increasing the previous defined width of the Main Zone. Drilling has also demonstrated mineralization continues down dip in particular areas along the strike of the Main zone and that potential continues to exist along strike which is further demonstrated by soil grid results emanating from a soil sampling program conducted over 1 km from the Main Zone along strike towards Northwest. This combined with the fact that Main zone is the only area within the property that has had been drill tested the following exploration program work is recommended:

Additional drilling is recommended, targeting step out regions along strike of the Frasergold Main zone at tighter spacing in order to potentially assist with the structural interpretation of the zone and add resource. Additional drilling will assist with understanding the structural controls which are currently poorly understood. Continued detailed and consistent logging is required in order to capture as much structural information as possible, such as vein density, overall quartz percentage and structural measurements, which may assist with future exploration programs and understanding the complex structural controls within this region.

Additional soil sampling should be conducted along both the strike of the Main zone and of areas within the property demonstrating prospective geophysical anomalies. It would also be advised that increased sampling density would be prudent on some low sample density historical grids as well as complimenting this sampling with ICP analysis which was not conducted in the past.

A two stage program is recommended. Additional resource may be developed along the southeast and northwest projections of the main zone and within the 18ppm and Eureka Bowl grid areas. The first stage is continued soil sampling, road and drill site construction, and a 6,000m diamond drill program at an estimated cost of \$2,000,000. 1,500m of this drilling could be conducted in 2015, the remaining 4,500m completed in 2016. Contingent upon favorable results from the Stage I drilling, additional drilling would be recommended at a cost of about \$2,500,000.

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28. Certificates and Consent of Authors

These certificates apply to the technical report entitled '**Frasergold Exploration Project, Cariboo Mining Division, B.C. Latitude 52° 18'N Longitude 120° 35' W.**

- I, K. Vincent Campbell, Ph.D., P.Geo. of 6599 Millar Road, Horsefly BC V0L 1L0 (PO Box 271) do hereby certify that I am responsible for the preparation of the technical report titled '**Frasergold Exploration Project, Cariboo Mining Division**' dated July 20th, 2015 and amended July 27, 2015.
- I am an independent consulting geologist.
- I graduated with a degree of Bachelor of Science, Honours Geology, from the University of British Columbia in 1966, a degree of Master of Science, Geology, from the University of Washington in 1969 and a degree of Doctor of Philosophy, Geology, from the University of Washington in 1971.
- I have worked as a geologist for over 46 years since my graduation from university.
- I am a member of the Association of Professional Engineers and Geoscientists of B.C. (Registration #19411).
- I have read the definition of a "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI-43-101.
- My relevant experience with respect to the Frasergold exploration project includes over 33 years in mineral exploration and geological mapping in North America, South America and Africa. Over my working career I have carried out over 500 exploration projects ranging from "grass-roots" projects to well advanced drilling projects and am well versed in geochemical exploration techniques, photo interpretation and remote sensing, structural analysis, interpretation of geophysics and geological mapping both on a highly detailed scale and on a regional scale. I undertook regional mapping of the Crooked Lake and Eureka Peak area of the Cariboo Mountains over the period 1963 to 1971 during summer field programs for the Geological Survey of Canada. From 1984 to 1991 I was involved in various geological investigations of the Frasergold property for Amoco Petroleum and Eureka Resources, Inc. I visited the Frasergold property a number of times during the 2008 field season and conducted a property-wide rock and silt sampling program during the 2009 field season.
- I last visited the property on June 13th, 2015 to assess road conditions.
- I am responsible for Sections 1 to 13 inclusive and Sections 23 to 27 inclusive of this report including the title of this technical report dated July 20, 2015 and amended July 27, 2015.
- I am independent of Eureka Resources Inc. as defined by Section 1.5 of the Instrument. I have read National Instrument 43-101 and the sections for which I am responsible in this Technical Report have been prepared in compliance with National Instrument 43-101 and Form 43-101F1. My relationship to Eureka Resources Inc. is as a geological consultant. I have no anticipation of acquiring or receiving any shares or interest in the company.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I consent to the filing of the Technical Report with any stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible to the public.

- I am not aware of any information or omission of such information that would make this Technical Report misleading. This Technical Report contains available scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 27th day of July, 2015.



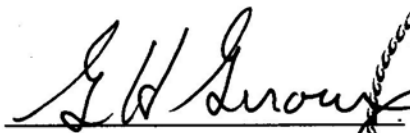
K. Vincent Campbell, Ph.D., P.Geo.

Signature of Qualified Person

I, Gary H. Giroux, P.Eng., do hereby certify that:

- I am a consulting geological engineer with offices at 1215-675 West Hastings Street, Vancouver, British Columbia, Canada V6B 1N2.
- I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc., both in Geological Engineering.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.
- I have practised my profession continuously since 1970. I have had over 39 years' experience estimating mineral resources. I have worked on numerous bulk gold deposits including Spanish Mt., Livengood, La India, Kisladag and Brewery Creek.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am responsible for the preparation of Section 14.0 of the technical report titled '**Frasergold Exploration Project, Cariboo Mining Division, B.C. Latitude 52° 18'N Longitude 120° 35'**' and dated July 20, 2015 and amended July 27, 2015 (the "Technical Report") relating to the Frasergold project. I have visited the property on September 10-11, 2008.
- I have had prior involvement with the property completing a preliminary geostatistical evaluation in 1992 for Eureka.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
- I have read National Instrument 43-101 and Form 43-101F1, and my portion of the Technical Report has been prepared in compliance with that instrument and form.

Dated this 27th Day of July, 2015.



Print name of Qualified Person



APPENDIX I

2007 Rock Sample Locations and Assay Values

Appendix I. 2007 Rock Sample Locations and Assay Values

Sample ID	UTME	UTMN	Elevation (m)	Au (ppm)	Description
135782	665651	5796648	1574	0.005	No major quartz veins, outcrop approximately 3m in height and 3m wide. Sample taken along and perpendicular to strike
135783	665617	5796621	1601	0.005	20m long outcrop (up creek) that looks like volcanics. Massive, greenish, very siliceous, anhedral to subhedral, 1-2% disseminated pyrite visible. Quartz veins cutting through along similar orientation as KP strike (130 degrees) up to 30cm thick. Possible intrusive sill or dyke, resembles dyke from FG56
135784	665578	5796581	1611	0.010	Knotted Phyllite. Approximately 30% Quartz.
135772	664905	5797648	1621	0.010	Knotted Phyllite. Slightly graphitic with 10-15% Quartz. Abundant Chlorite
135764	665460	5797735	1435	0.010	Knotted Phyllite. <1% Quartz.
135765	665397	5797847	1428	0.005	Knotted Phyllite. Approximately 15% Quartz.
135766	665354	5797911	1437	0.005	Grey Phyllite
135767	665288	5798003	1435	0.005	Grey Phyllite. Millimeter laminations of whitish carbonate.
135768	665231	5798069	1434	0.005	Grey Phyllite. Graphitic with occasional knots.
135769	665199	5798135	1432	0.010	Knotted Phyllite
135770	665109	5798272	1410	0.010	Knotted Phyllite. Graphitic with occasional pyrite stringer
135771	664726	5797736	0	0.005	Grey Phyllite
135773	665133	5797487	1605	0.010	Knotted Phyllite. <1% Quartz.
135774	665471	5797169	1590	0.005	Knotted Phyllite. Slightly graphitic with approximately 15% Quartz.
135775	665608	5797201	1526	0.030	Knotted Phyllite. Slight graphitic with <1% Quartz.
135776	665551	5797295	1522	0.210	Knotted Phyllite. Approximately 15-20% Quartz.
135777	664920	5796966	1803	0.005	Knotted Phyllite, Grey Phyllite, Siltstone with 30% Quartz
135778	664864	5796917	1824	0.010	Knotted Phyllite, Grey Phyllite, Siltstone with 30% Quartz
135779	664867	5796899	1854	0.020	Metavolcanics
135780	664836	5796834	1840	0.005	Knotted and Grey Phyllite. Approximately 25% Quartz.
135796	665417	5797812	1429	0.005	Grey Phyllite. Approximately 5% quartz. Graphitic at quartz margins. Minor sulphides.
135797	665378	5797885	1432	0.010	Grey Phyllite. <2% Quartz. Sulphides occurring in deformed areas. Haematite weathering.
135798	665326	5797957	1439	0.010	Grey phyllite with some weathered carbonates. Graphitic sheen. Quartz veining ~10% outcrop. Veins mm to 5cm.
135799	665255	5798049	1437	0.020	Grey Phyllite. Graphitic. Approximately 25% quartz.
135800	665219	5798094	1433	0.010	Knotted Phyllite. Sulphide stringers. <5% quartz. Haematite weathering.
135801	665149	5798194	1435	0.010	Knotted Phyllite. Approximately 20% quartz.
135804	664648	5797789	1626	0.005	Knotted Phyllite with approximately 10% quartz
135805	664766	5797684	1641	0.005	Knotted Phyllite with approximately 15% quartz veins and boudins. Sulphides in quartz margins
135806	664866	5797603	1638	0.005	Knotted Phyllite with approximately 10% quartz
135807	664975	5797572	1628	0.005	Knotted Phyllite with approximately < 5% quartz
135808	665107	5797468	1618	0.005	Knotted Phyllite with approximately 15% quartz

Sample ID	UTME	UTMN	Elevation (m)	Au (ppm)	Description
135809	665294	5797234	1613	0.005	Knotted Phyllite with approximately 30% quartz. Sulphides in quartz margins
135810	665390	5797167	1622	0.010	Knotted Phyllite with approximately 20% quartz. Some hematite weathering
135811	665468	5797099	1613	0.005	Knotted Phyllite with approximately < 2% quartz
135812	665542	5797021	1601	0.050	Knotted Phyllite with approximately 30% quartz. Chlorite weathering on quartz veins
135813	665589	5796985	1595	0.010	Knotted Phyllite with approximately < 5% quartz
135814	665593	5796933	1604	0.010	Knotted Phyllite with approximately 50% quartz, more graphitic near vein margins. Chlorite weathering on quartz veins.
135815	665597	5796924	1606	0.010	Knotted Phyllite with approximately < 1% quartz
135816	665599	5796896	1603	0.005	Knotted Phyllite with approximately 25% quartz
135817	665588	5796855	1598	0.010	Knotted Phyllite with approximately 30% quartz, very graphitic in quartz fold margins
135818	661875	5798597	1610	0.010	Volcanics. Green siliceous matrix with some feldspar clasts, unknown black mineral, some sulphides (bornite?)
135819	664063	5798477	1366	0.005	Knotted Phyllite
135820	664025	5798435	1388	0.005	Knotted Phyllite with approximately < 5% quartz
135821	663959	5798200	1448	0.010	Siliceous Siltstone, some quartz veining, chlorite alteration, sulphide stringers
135822	663980	5798163	1410	0.010	Metavolcanics? Large phenocrysts, greenish matrix, calcite veining
135859	664068	5797956	1423	0.010	Siliceous siltstone. Highly weathered. 15% quartz veining. Chloride in quartz veining.
135860	664027	5797967	1411	0.005	Green siliceous metavolcanics. Disseminated, cubes, pyrite.
135861	663909	5797928	1443	0.010	Green siliceous metavolcanics. Disseminated, cubes, pyrite.
135862	663895	5797855	1546	0.005	Volcanics. Green matrix with hornblende phenocrysts. Some quartz. Disseminated sulphides. Massive, some jointing.
135863	663673	5797233	1463	0.010	Green (chlorite?) siliceous unit. 5% pyrite, stringers and cubes. Apple green in places.
135880	664257	5799070	1326	0.050	Grey phyllite. O/C is rusty and shiny on weathered surface. Fresh surface is grey with no sulphides. O/C continues down creek for 10-15m.
135881	664262	5799112	1302	0.005	Grey phyllite with disseminated sulphides and quartz veins (2-15 cm) along bedding.
135882	664525	5798861	1388	0.005	Grey phyllite. Minor knots/carbonates that are heavily weathered with a pink/green tarnish on some surfaces. Minor sulphide stringers.
135883	664505	5798876	1378	0.005	Grey phyllite similar to last o/c except sulphides are more abundant (2-3%) and appear disseminated and as stringers.
135884	664591	5798803	1394	0.005	Grey phyllite appears siliceous in some areas, heavily weathered, and have minor sulphides (disseminated)
135885	665312	5797750	1491	0.010	Knotted Phyllite. No quartz or sulphides.
135886	665330	5797726	1492	0.030	KP with quartz veins. Veins are ~2cm - 4 cm thick, milky white with orange oxidation, and no sulphides apparent.
135887	665345	5797617	1544	0.005	Knotted Phyllite. Minor quartz (veinlets) and no sulphides.
135888	665430	5797600	1476	0.320	KP with quartz veins that are 15 cm tick, appears along bedding, milky white with orange oxidation. No apparent

Sample ID	UTME	UTMN	Elevation (m)	Au (ppm)	Description
					sulphides.
135889	665545	5797444	1459	2.440	Knotted Phyllite. No quartz and sulphides.
135890	665624	5797379	1459	0.030	Knotted Phyllite. No quartz and sulphides.
135891	665452	5796322	1782	0.005	Grey Phyllite. Abundant quartz (along bedding & crosscutting) and no sulphides
135892	665472	5796229	1737	0.020	KP appears shaley with mm scale carbonates and a sleek graphiti surface. Quartz laminae along bedding that are sometimes microfolded, quartz is creamy with orange oxidation. Weathered surface of KP is rusty. Also, phyllite as a wavy appearance and there are quartz veins that crosscut the unit.
135893	665641	5796027	1828	0.010	Knotted phyllite (looks like shaley grey phyllite) with small carbonates. O/C is masive. Quartz is milky white with orange oxidation and range mm to 3 cm. Occasional quartz is along bedding and crosscuts, also appears as microfolds.
135876	663757	5799153	0	0.020	KP with abundant knots. Quartz veins (~2cm) along bedding that are creamy white and as orange oxidation. Minor sulphides. KP continues up creek 20m.
135877	663629	5798730	0	0.005	KP. No apparent quartz veins, however there are quartz laminations on fresh surface. No apparent sulphides.
135878	663523	5798657	1532	0.005	Feldspar Porphyry. No quartz or sulphides.
135879	664127	5799074	0	0.010	Grey phyllite. Weathered surface is rusty and apears shiny. Quartz veins present are milky white. Sulphides (py) are present
135894	666998	5796109	1754	0.005	Grey Phyllite. 3-5% sulphides (py, stringers & disseminated) & no quartz.
135895	667022	5796131	1336	0.005	Grey Phyllite. Minor quartz and 1% sulphides (py)
135896	666941	5796091	1347	0.010	Knotted Phyllite. 2% sulphides (disseminated py)
135897	666982	5796023	1417	0.005	Knotted Phyllite. 15-20% quartz and 1-2% sulphides (disseminated py)
135898	666914	5795967	1417	0.010	Knotted Phyllite. 30% quartz and no sulphides.
135912	666729	5795738	1444	1.290	Knotted Phyllite. No quartz and <1% sulphides (disseminated py)
135913	666691	5795677	1447	0.050	70% quartz (massive) and 5% sulphides (py, patchy & disseminated)
135914	666691	5795677	1447	0.005	Knotted Phyllite. 20% quartz (along bedding) and 1% quartz.
135915	666338	5795242	1512	0.005	Knotted Phyllite. Quartz veins (15 cm) and no sulphides.
135916	666475	5796045	1567	0.010	Knotted Phyllite. Minor quartz & no sulphides.
135917	666347	5795913	0	0.005	Knotted Phyllite. No quartz or sulphides.
135918	666161	5796505	1603	0.010	Knotted Phyllite. <1% quartz & no sulphides.
135919	666089	5796571	1600	0.005	Knotted phyllite. No quartz or sulphides.
135920	665985	5796613	1597	0.010	Knotted Phyllite. No quartz or sulphides.
135921	661448	5802015	1253	0.005	Grey Phyllite. Minor quartz (along bedding) & 1% sulphides (py). CHANGE CO-ORDINATES - INCORRECT
135781	662005	5798554	1524	0.010	Augite Porphyry
135899	664116	5800446	1347	0.010	Grey Phyllite. < 10% Quartz. Pyrite up to 5% disseminated and in mm stringers.
135900	666635	5795639	1466	0.030	Knotted Phyllite. Slightly graphitic with approximately 25% Quartz.

Sample ID	UTME	UTMN	Elevation (m)	Au (ppm)	Description
135901	666613	5795564	1470	0.060	Knotted Phyllite. <10% Quartz. Slightly graphitic.
135902	666538	5795510	1478	0.010	Knotted Phyllite
135903	666364	5795269	1488	0.010	Grey Phyllite, slightly graphitic. Pyrite cubes up to 5%.
135904	666510	5795923	1563	0.010	Knotted Phyllite
135905	666413	5796140	1593	0.010	Knotted Phyllite
135906	666295	5796321	1618	0.005	Knotted Phyllite. < 5% Quartz.
135907	666264	5796397	1607	0.005	Knotted Phyllite. Approximately 30% Quartz.
135908	666036	5796602	1598	0.030	Knotted Phyllite. Approximately 20% Quartz.
135864	665412	5797615	1481	0.280	Knotted phyllite. 10% quartz veining. Veins approx. 5cm thick. Pyrite stringers in margins.
135865	665456	5797572	1481	0.270	Knotted phyllite. 5% quartz as boudin <15cm.
135866	665647	5797372	1467	0.070	Knotted phyllite. Deformed with quartz veining. 20% quartz.
135867	665483	5796238	1760	0.005	Siliceous siltstone. Small lens, 1.5x1.5m exposed, surrounded by grey phyllite. 5% quartz veining.
135868	665485	5796262	1769	0.005	Knotted phyllite, small porphyroblasts (knots). Graphitic. 20-30% quartz veining.
135869	665607	5796184	1779	0.005	Black phyllite. 35-45% quartz. Quartz boudins.
135870	667143	5796280	1326	0.010	Grey phyllite. Small, dirty quartz veins, with phyllite material. Pyrite stringers, minor.
135871	667015	5796101	1364	0.010	Grey phyllite. Dark grey. 2% pyrite, stringers and cubes.
135872	666973	5796218	1418	0.005	Siltstone, siliceous. Pyrite, 1-2%, stringers and disseminated. Location +/- 90m
135873	666946	5796035	1422	0.010	Knotted phyllite. 5-10% quartz., as narrow quartz veins. 5-10%
135874	666855	5795892	1420	0.010	Knotted phyllite. 10% quartz veining. Veins mm scale to 5cm. 2% pyrite, disseminated.
135875	666819	5795809	1414	0.010	Knotted phyllite. 10-15% quartz, approx 20cm wide. 2-3% disseminated pyrite.
135922	666803	5795800	1423	0.005	Knotted phyllite. Quartz veining; large (50-100cm) veins showing pinch and swell, veining filling jointing, thin veining parallel to bedding. 1% pyrite disseminated near quartz margins.
135923	666783	5795784	1454	0.010	Siliceous siltstone. 20% quartz veining, mm scale, parallel to bedding. Disseminated pyrite 2-3%. Chlorite.
135924	666620	5795928	1540	0.005	Knotted phyllite. Weathered knots. <5% quartz veining.
135925	666580	5795904	1530	0.005	Knotted phyllite. <5% quartz veining.
135926	666517	5795840	1538	0.020	Knotted phyllite. 50% quartz veining. Veins 2-10cm wide. Weathered.
135927	664697	5801476	1629	0.005	Siliceous biotite shist. 10% quartz veining, 1-5cm wide. Some larger qtz veins with internal shist material.
135928	668477	5795160	1390	0.020	Grey phyllite. Very hard to break, very blocky. O/C is siliceous, 3-5% sulphides that appear silver and they are finely disseminated.
135929	668682	5795269	1355	0.010	Very siliceous grey phyllite with finely disseminated sulphides as well as stringers, 5-7%. 10% quartz veinlets throughout. Also some minor orange carbonates apparent.
135930	665953	5796668	1542	0.020	Knotted Phyllite, approximately 1% quartz, no sulphides
135932	666222	5796469	1525	0.270	Knotted Phyllite/Grey Phyllite with 20% quartz

Sample ID	UTME	UTMN	Elevation (m)	Au (ppm)	Description
135933	666289	5796441	1550	0.050	Knotted Phyllite, approximately 10% quartz
135935	666375	5796329	1563	0.010	Knotted Phyllite
135937	666330	5796301	1587	0.005	Knotted Phyllite. <1% Quartz.
135938	660368	5801246	1559	0.005	Phyllite with banded appearance. Minor quartz (< 1%)
135939	660216	5801303	1575	0.005	Volcanics. Massive, greenish grey, very siliceous, minor calcite, some areas with 10-15% biotite, minor quartz
135940	659954	5801612	1497	0.005	Grey Phyllite with 1% finely disseminated sulphides
135941	662733	5799092	1612	0.005	Vocanics? Mafic, very dark green and black minerals
135942	662755	5798996	1637	0.005	Volcanic Tuff. Massive, grey, mineralized sections cross cutting light green tuff with black crystals
135943	662755	5798996	1637	0.005	Vocanics? Mafic, very dark green and black minerals. Appears to be a flow later than tuff above
135944	662606	5799052	1670	0.005	Volcanics. Mafic, very fine grained
135945	662572	5799053	1670	0.005	Volcanics. Possibly Amphibolite. Massive, dark grey, very fine grained, mafic
135946	662591	5799014	1674	0.005	Volcanics? Abundant mica (biotite), occasional deformed quartz veins, green and black
135947	668709	5795212	1389	0.005	Grey Phyllite/Siltstone. Dark grey to black, thin millimeter stringers of sulphides (5%), siliceous
135948	668799	5795146	1454	0.000	Grey Phyllite/Siltstone with approximately 20% quartz. Dark grey to black, thin millimeter stringers of sulphides (5%), siliceous
135931	666105	5796770	1540	0.050	Knotted Phyllite with approximately 15% quartz
135934	666340	5796390	1557	0.210	Knotted Phyllite with approximately 15% quartz
135936	666350	5796315	1570	0.005	Knotted Phyllite with approximately 10% quartz
135802	664763	5799664	1271	0.005	Grey Phyllite. Disseminated sulphides. Approximately 10% quartz.
135803	664763	5799664	1271	0.005	Grey Phyllite. Disseminated sulphides. Approximately 10% quartz.
135850	667021	5796284	NR	0.005	Milky to white colored massive quartz flow, 2-5% limonite along fractures, fractured, grey phyllite in-filled, no sulphides
135849	665472	5798084	1337	0.010	Knotted Phyllite with trace pyrite, chalcopyrite and malachite
135823	665855	5797548	1343	0.010	Knotted Phyllite with Grey Phyllite. < 1% Sulphides
135824	665848	5797555	1343	0.010	Knotted Phyllite/Grey Phyllite, massive milky colored quartz veins. < 2% Sulphides with euhedral pyrite
135825	665844	5797563	1344	0.010	Knotted Phyllite predominately with trace sulphides
135826	665835	5797570	1345	0.010	Knotted Phyllite, no quartz veining, trace sulphides
135827	665753	5797639	1348	0.005	Knotted Phyllite with trace sulphides
135828	665726	5797720	1342	0.180	Knotted Phyllite, moderately silicified, 30% quartz, < 1% sulphides
135829	665726	5797720	1342	0.010	Knotted Phyllite with trace sulphides
135830	665717	5797746	1339	0.005	Knotted Phyllite, laminated/banded, 1-2% sulphides, 2% quartz veins, hematite and chlorite alteration
135831	665717	5797746	1339	0.020	Knotted Phyllite and calcareous Grey Phyllite, no quartz veins
135851	667048	5796255	NR	0.010	Grey Phyllite, no sulphides, strong limonite coating

Sample ID	UTME	UTMN	Elevation (m)	Au (ppm)	Description
135832	665648	5797853	1335	0.020	Knotted Phyllite, dark grey, moderately silicified, 10% quartz veining, 1% sulphides
135833	665596	5797939	1332	0.010	Knotted Phyllite, 2-5% quartz veins, < 1% sulphides
135834	665594	5797944	1340	0.010	Knotted Phyllite, moderately silicified, moderately folded, < 1% quartz, < 1% sulphides
135835	665589	5797951	1331	0.020	Knotted Phyllite, hematite < 5%, limonite 2%, 1-2% pyrite, malachite < 1% and several massive quartz veins
135836	665580	5797956	1329	0.005	Knotted Phyllite/Grey Phyllite with 50% quartz
135837	665578	5797959	1332	0.010	Knotted Phyllite with quartz and sulphides in trace up to 1%
135838	665275	5798376	1327	0.020	Knotted Phyllite with < 1% sulphides
135839	665078	5798645	1308	0.210	Knotted Phyllite with < 1% sulphides
135840	665068	5798659	1308	0.005	Knotted Phyllite with < 0.5% sulphides
135841	665610	5797921	1330	0.010	Knotted Phyllite with < 0.5% sulphides
135852	667506	5796253	NR	0.010	Grey Phyllite and quartz, no sulphides
135842	664976	5798457	1504	0.010	Knotted Phyllite/calcareous Grey Phyllite with 50% quartz
135843	665146	5797987	1490	0.010	Grey Phyllite/Knotted Phyllite with trace sulphides
135844	665164	5797955	1492	5.170	Grey Phyllite/Knotted Phyllite with trace sulphides
135845	661259	5801419	1286	0.030	Grey Phyllite with scattered pyrite throughout
135846	661259	5801419	1286	0.005	Grey Phyllite with scattered pyrite throughout, massive quartz veins
135847	659921	5802523	1402	0.020	Knotted Phyllite
135853	667042	5796203	NR	0.010	Massive quartz, some phyllite, no sulphides
135854	667027	5796185	NR	0.010	Quartz with some grey phyllite
135855	668153	5795316	NR	0.010	Quartz with some grey phyllite, no sulphides
135856	668158	5795316	NR	0.010	Quartz with some grey phyllite, some limonite staining
135857	668163	5795316	NR	0.010	Quartz and grey phyllite, no sulphides
135848	665607	5797943	1332	0.030	Knotted Phyllite with trace pyrite, chalcopryite and malachite
135751	665656	5796821	1566	0.010	Knotted Phyllite. Shear zone, broken KP with <5% Quartz.
135760	665167	5797620	1566	0.020	Knotted Phyllite. < 5% Quartz.
135761	664978	5797819	1557	0.005	Knotted Phyllite.
135762	664818	5797911	1567	0.005	Knotted Phyllite.
135763	664774	5797940	1564	0.005	Knotted Phyllite. Approximately 50% Quartz.
135752	665657	5796871	1563	0.020	Knotted Phyllite. Slightly graphitic with abundant Quartz up to 65%.
135753	665658	5796923	1562	0.010	Knotted Phyllite. Approximately 40% Quartz.
135754	665657	5796952	1561	0.020	Knotted Phyllite. Approximately 50% Quartz.
135755	665642	5796987	1567	0.005	Knotted Phyllite
135756	665626	5797055	1557	0.010	Knotted Phyllite. <1% Quartz.
135757	665558	5797167	1556	0.005	Knotted Phyllite with high graphite content. Approximately 50% Quartz.
135758	665498	5797228	1560	0.010	Knotted Phyllite. Approximately 60% Quartz
135759	665309	5797438	1562	0.005	Knotted Phyllite

Sample ID	UTME	UTMN	Elevation (m)	Au (ppm)	Description
135791	665604	5797089	1565	0.005	Knotted Phyllite. Graphitic with chlorite in quartz and quartz margins. Approximately
135792	665532	5797199	1557	0.010	Knotted Phyllite. Approximately 20% quartz, 2-5cm veins. Graphitic with chlorite in quartz.
135858	665294	5797452	1562	0.040	Knotted Phyllite with approximately 30% quartz
135793	665107	5797679	1568	0.005	Knotted Phyllite. <5% Quartz.
135794	664870	5797890	1561	0.005	Knotted Phyllite. Approximately 20% quartz. Pyrite in quartz margins. Haematite weathering.
135795	664656	5798022	1562	0.010	Knotted Phyllite. <5% Quartz. Pyrite in quartz margin.
135785	665653	5796786	1575	0.060	Knotted Phyllite. Approximately 20% Quartz.
135786	665653	5796799	1569	0.005	Knotted Phyllite. Approximately 10-20% Quartz.
135787	665660	5796822	1563	0.005	Knotted Phyllite. Quartz veins <5cm. Pyrite stringers.
135788	665659	5796903	1580	0.005	Knotted Phyllite. Quartz veins cutting bedding. Approximately 5% Quartz.
135789	665659	5796935	1562	0.010	Knotted Phyllite. Folded Quartz vein. Pyrite cubes, disseminated.
135790	665646	5797017	1566	0.010	Knotted Phyllite. Graphitic between Quartz veins. Veins 5-60cm
136363	661210	5799468	1774	0.060	
136364	661028	5799181	1886	0.010	
136365	661028	5799181	1886	0.010	

APPENDIX II

Frasergold Trench Sampling Program 2007

Locations of Samples and Sample Descriptions

Appendix II. Frasergold Trench Sampling Program 2007

Locations of Samples and Sample Descriptions

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
135949	5300 Trench Top	1.20	2.20	664975	5797566	1627	Knotted Phyllite with < 5% quartz, one small vein	0.005
135950	5300 Trench Top	2.20	3.20				Knotted Phyllite with < 5% quartz, one small vein	0.005
135951	5300 Trench Top	3.20	4.20				Knotted Phyllite with < 2% quartz	0.005
135952	5300 Trench Top	4.20	5.20				Knotted Phyllite	0.005
135953	5300 Trench Top	5.20	6.20				Knotted Phyllite with 15% quartz, one large vein	0.005
135954	5300 Trench Top	6.20	7.20				Knotted Phyllite with < 2% quartz	0.005
135955	5300 Trench Top	7.20	8.20				Knotted Phyllite with 15% quartz	0.005
135956	5300 Trench Top	8.20	9.20				Knotted Phyllite with 5% quartz, one small vein	0.005
135957	5300 Trench Top	9.20	10.20				Knotted Phyllite	0.005
135958	5300 Trench Top	10.20	11.20				Knotted Phyllite with 20% quartz	0.005
135959	5300 Trench Top	11.20	12.20				Knotted Phyllite with 20% quartz	0.01
135960	5300 Trench Top	12.20	13.20	664986	5797572	1621	Knotted Phyllite with 15% quartz	0.005
135961	5300 Trench Top	13.20	14.20				Knotted Phyllite with 5-10% quartz	0.005
135962	5300 Trench	14.20	15.20				Knotted Phyllite with 5%	0.005

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
	Top						quartz	
135963	5300 Trench Top	15.20	16.20				Knotted Phyllite with 25% quartz	0.005
135964	5300 Trench Top	16.85	17.85				Knotted Phyllite with 25% quartz, graphitic around vein margins, abundant chlorite	0.005
135965	5300 Trench Top	17.85	18.85				Knotted Phyllite with < 5% quartz	0.005
135966	5300 Trench Top	18.85	19.85				Knotted Phyllite with < 1% quartz	0.005
135967	5300 Trench Top	19.85	20.85				Knotted Phyllite	0.005
135968	5300 Trench Top	20.85	21.85				Knotted Phyllite with 20% quartz	0.005
135969	5300 Trench Top	21.85	22.85				Knotted Phyllite with < 5% quartz	0.005
135970	5300 Trench Top	22.85	23.85	664993	5797580	1620	Knotted Phyllite	0.005
135971	5300 Trench Top	23.85	24.85				Knotted Phyllite with 25% quartz	0.005
135972	5300 Trench Top	24.85	25.85				Knotted Phyllite with 25% quartz	0.005
135973	5300 Trench Top	25.85	26.85				Knotted Phyllite with < 5% quartz	0.005
135974	5300 Trench Top	26.85	27.85				Knotted Phyllite with < 5% quartz	0.005
135975	5300 Trench Top	27.85	28.85				Knotted Phyllite with 20% quartz	0.005
135976	5300 Trench Top	31.05	32.05				Knotted Phyllite with < 1% quartz	0.005
135977	5300 Trench	32.05	33.05				Knotted Phyllite with	0.01

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
	Top						50% quartz	
135978	5300 Trench Top	33.05	34.05				Knotted Phyllite with < 1% quartz	0.005
135979	5300 Trench Top	34.05	35.05				Knotted Phyllite with < 1% quartz	0.005
135980	5300 Trench Top	35.05	36.05	664995	5797591	1612	Knotted Phyllite with < 1% quartz	0.005
135981	5300 Trench Top	36.05	37.05				Knotted Phyllite	0.005
135982	5300 Trench Top	37.05	38.05				Knotted Phyllite with < 1% quartz	0.01
135983	5300 Trench Top	38.05	39.05				Knotted Phyllite with < 1% quartz	0.01
135984	5300 Trench Top	40.95	41.95				Knotted Phyllite with 5% quartz	0.01
135985	5300 Trench Top	41.95	42.95				Knotted Phyllite with 10% quartz	0.005
135986	5300 Trench Top	42.95	43.95				Knotted Phyllite with 15% quartz	0.005
135987	5300 Trench Top	44.45	45.45				Knotted Phyllite	0.01
135988	5300 Trench Top	45.45	46.45				Knotted Phyllite with < 1% quartz	0.005
135989	5300 Trench Top	50.05	51.05				Knotted Phyllite with 20% quartz	0.005
135990	5300 Trench Top	51.05	52.05	665004	5797603	1610	Knotted Phyllite with 10% quartz	0.005
135991	5300 Trench Top	52.05	53.05				Knotted Phyllite with 30% quartz	0.005
135992	5300 Trench Top	53.05	54.05				Knotted Phyllite with 5% quartz	0.005
135993	5300 Trench	54.05	55.05				Knotted Phyllite with	0.005

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
	Top						10% quartz	
135994	5300 Trench Top	55.05	56.05				Knotted Phyllite with < 1% quartz	0.005
135995	5300 Trench Top	56.05	57.05				Knotted Phyllite with < 3% quartz	0.005
135996	5300 Trench Top	57.05	58.05				Knotted Phyllite with < 1% quartz	0.005
135997	5300 Trench Top	58.65	59.65				Knotted Phyllite with 35% quartz	0.005
135998	5300 Trench Top	59.65	60.65				Knotted Phyllite with 35% quartz, abundant chlorite within quartz veins. Some hematite replacing carbonate knots	0.005
135999	5300 Trench Top	63.95	64.95				Knotted Phyllite with 50% quartz	0.01
136000	5300 Trench Top	64.95	65.95	665008	5797609	1597	Knotted Phyllite with 50% quartz	0.005
136001	5300 Trench Top	67.95	68.95				Knotted Phyllite with 70% quartz	0.005
136002	5300 Trench Top	70.30	71.30				Knotted Phyllite with 15% quartz	0.005
136003	5300 Trench Top	71.30	72.30				Knotted Phyllite with 15% quartz	0.01
136004	5300 Trench Top	72.30	73.30				Knotted Phyllite with 40% quartz	0.005
136005	5300 Trench Top	73.30	74.30				Knotted Phyllite with 40% quartz	0.01
136006	5300 Trench Top	74.30	75.30				Knotted Phyllite with 25% quartz	0.005
136007	5300 Trench Top	76.60	77.60				Knotted Phyllite with 50% quartz	0.005

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136008	5300 Trench Top	77.60	78.60				Knotted Phyllite with 50% quartz	0.005
136009	5300 Trench Top	78.60	79.60				Knotted Phyllite with < 5% quartz	0.005
136010	5300 Trench Top	79.60	80.60	665017	5797624	1591	Knotted Phyllite with 5% quartz	0.005
136011	5300 Trench Top	80.60	81.60				Knotted Phyllite with < 1% quartz	0.005
136012	5300 Trench Top	81.60	82.60				Knotted Phyllite with < 1% quartz	0.005
136013	5300 Trench Top	82.60	83.60				Knotted Phyllite with < 1% quartz	0.005
136014	5300 Trench Top	83.60	84.60				Knotted Phyllite with < 1% quartz	0.005
136015	5300 Trench Top	84.60	85.60				Knotted Phyllite with 5-10% quartz	0.005
136016	5300 Trench Top	85.60	86.60				Knotted Phyllite with 15-20% quartz	0.005
136017	5300 Trench Top	86.60	87.60				Knotted Phyllite with < 1% quartz	0.005
136018	5300 Trench Top	87.60	88.60				Knotted Phyllite with < 1% quartz	0.005
136019	5300 Trench Top	88.60	89.60				Knotted Phyllite with < 1% quartz	0.005
136020	5300 Trench Top	89.60	90.60				Knotted Phyllite with < 1% quartz	0.005
136021	5300 Trench Top	90.60	91.60				Knotted Phyllite with < 1% quartz	0.005
136022	5300 Trench Top	99.00	100.00	665028	5797637	1585	Knotted Phyllite with < 5% quartz	0.005
136023	Above Adit 1	0.00	1.00	665232	5797726	1527	Knotted Phyllite with 20% quartz; vein	0.34

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
							margins more graphitic	
136024	Above Adit 1	1.00	2.00				Knotted Phyllite with <15% quartz; chlorite in quartz vein margins	0.02
136025	Above Adit 1	2.00	3.00				Knotted Phyllite with < 1% quartz	0.02
136026	Above Adit 1	3.00	4.00				Knotted Phyllite with <35% quartz boudins and veinlets	0.03
136027	Above Adit 1	4.00	5.00				Knotted Phyllite with 10% quartz	0.04
136028	Above Adit 1	5.00	6.00				Knotted Phyllite with 50% quartz; some chlorite	0.02
136029	Above Adit 1	6.00	7.00				Knotted Phyllite with 40% quartz	0.01
136030	Above Adit 1	7.00	8.00				Knotted Phyllite with 10% quartz; chlorite in quartz; graphitic along vein margins	0.02
136031	Above Adit 1	8.00	9.00				Knotted Phyllite with < 1% quartz	0.01
136032	Above Adit 1	9.00	10.00				Knotted Phyllite with < 1% quartz	0.01
136033	Above Adit 1	10.00	11.00				Knotted Phyllite with 15% quartz	0.01
136034	Above Adit 1	11.00	12.00				Knotted Phyllite with 10% quartz	0.01
136035	Above Adit 1	12.00	13.00				Knotted Phyllite with < 1% quartz	0.03
136036	Above Adit 1	13.00	14.00				Knotted Phyllite with 10-15% quartz	2.14

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136037	Above Adit 1	14.00	15.00				Knotted Phyllite with 50-60% quartz	0.48
136038	Above Adit 1	15.00	16.00	665250	5797733	1520	Knotted Phyllite with < 1% quartz	0.01
136039	5450 Trench Bottom	0.00	1.00	665254	5797657	1535	Knotted Phyllite	0.01
136040	5450 Trench Bottom	1.00	2.00				Knotted Phyllite with < 1% quartz	0.01
136041	5450 Trench Bottom	2.00	3.00				Knotted Phyllite with < 1% quartz	0.01
136042	5450 Trench Bottom	3.00	4.00				Knotted Phyllite with < 1% quartz	0.01
136043	5450 Trench Bottom	4.00	5.00				Knotted Phyllite with < 1% quartz	0.01
136044	5450 Trench Bottom	5.00	6.00				Knotted Phyllite	0.11
136045	5450 Trench Bottom	6.00	7.00				Knotted Phyllite	0.04
136046	5450 Trench Bottom	7.00	8.00				Knotted Phyllite	0.2
136047	5450 Trench Bottom	8.00	9.00				Knotted Phyllite with < 1% quartz	0.38
136048	5450 Trench Bottom	9.00	10.00				Knotted Phyllite with < 1% quartz	0.64
136049	5450 Trench Bottom	10.00	11.00				Knotted Phyllite with < 3% quartz	1.08
136050	5450 Trench Bottom	11.00	12.00	665261	5797663	1528	Knotted Phyllite	0.04
136051	5450 Trench Bottom	12.00	13.00				Knotted Phyllite	0.04
136052	5450 Trench Bottom	13.00	14.00				Knotted Phyllite	0.09

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136053	5450 Trench Bottom	14.00	15.00				Knotted Phyllite	0.08
136054	5450 Trench Bottom	15.00	16.00				Knotted Phyllite with < 1% quartz	0.07
136055	5450 Trench Bottom	16.00	17.00				Knotted Phyllite	0.05
136056	5450 Trench Bottom	17.00	18.00				Knotted Phyllite	0.05
136057	5450 Trench Bottom	18.00	19.00				Knotted Phyllite	0.03
136058	5450 Trench Bottom	19.00	20.00				Knotted Phyllite with 5% quartz	0.01
136059	5450 Trench Bottom	20.00	21.00				Knotted Phyllite with 50-60% quartz	3.28
136060	5450 Trench Bottom	21.00	22.00	665266	5797673	1526	Knotted Phyllite with < 1% quartz	0.41
136061	5450 Trench Bottom	22.00	23.00				Knotted Phyllite with < 1% quartz	0.06
136062	5450 Trench Bottom	23.00	24.00				Knotted Phyllite with 25% quartz	0.15
136063	5450 Trench Bottom	24.00	25.00				Knotted Phyllite with 50% quartz	0.43
136064	5450 Trench Bottom	25.00	26.00				Knotted Phyllite with 25% quartz	4.12
136065	5450 Trench Bottom	26.00	27.00				Knotted Phyllite with 30% quartz	0.7
136066	5450 Trench Bottom	27.00	28.00				Knotted Phyllite with < 5% quartz	0.91
136067	5450 Trench Bottom	28.00	29.00				Knotted Phyllite with 45% quartz	0.47
136068	5450 Trench Bottom	29.00	30.00				Knotted Phyllite with 25% quartz	7.04

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136069	5450 Trench Bottom	30.00	31.00	665273	5797680	1522	Knotted Phyllite with < 1% quartz	7.15
136070	Above Adit 2	0.00	1.00	665236	5797710	1528	Knotted Phyllite	0.03
136071	Above Adit 2	1.00	2.00				Knotted Phyllite	0.01
136072	Above Adit 2	2.00	3.00				Knotted Phyllite	0.14
136073	Above Adit 2	3.00	4.00				Knotted Phyllite with < 3% quartz	1.28
136074	Above Adit 2	4.00	5.00				Knotted Phyllite with 60% quartz	8.13
136075	Above Adit 2	5.00	6.00				Knotted Phyllite with 40% quartz	0.09
136076	Above Adit 2	6.00	7.00				Knotted Phyllite with < 5% quartz	0.58
136077	Above Adit 2	7.00	8.00				Knotted Phyllite	0.52
136078	Above Adit 2	8.00	9.00				Knotted Phyllite with 5% quartz	0.07
136079	Above Adit 2	9.00	10.00				Knotted Phyllite	0.04
136080	Above Adit 2	10.00	11.00				Knotted Phyllite with 30% quartz	0.01
136081	Above Adit 2	11.00	12.00				Knotted Phyllite with 25% quartz	0.03
136082	Above Adit 2	12.00	13.00				Knotted Phyllite with 20% quartz	0.08
136083	Above Adit 2	13.00	14.00				Knotted Phyllite with < 5% quartz	0.23
136084	Above Adit 2	14.00	15.00				Knotted Phyllite with 15-20% quartz	0.54
136085	Above Adit 2	15.00	16.00				Knotted Phyllite	0.17
136086	Above Adit 2	16.00	17.00				Knotted Phyllite	0.02
136087	Above Adit 2	17.00	18.00				Knotted Phyllite	0.01

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136088	Above Adit 2	18.00	19.00				Knotted Phyllite	0.01
136089	Above Adit 2	19.00	20.00				Knotted Phyllite with 50% quartz	0.1
136090	Above Adit 2	20.00	21.00				Knotted Phyllite with 50% quartz	0.01
136091	Above Adit 2	21.00	22.00				Knotted Phyllite	0.01
136092	Above Adit 2	22.00	23.00				Knotted Phyllite with < 1% quartz	0.02
136093	Above Adit 2	23.00	24.00				Knotted Phyllite with 10% quartz	0.58
136094	Above Adit 2	24.00	25.00				Knotted Phyllite	0.21
136095	Above Adit 2	25.00	26.00				Knotted Phyllite with 5-10% quartz	0.13
136096	Above Adit 2	26.00	27.00				Knotted Phyllite	0.01
136097	Above Adit 2	27.00	28.00				Knotted Phyllite	0.005
136098	Above Adit 2	28.00	29.00	665255	5797727	1517	Knotted Phyllite	0.01
136099	5450 Trench Top	0.00	1.00	665134	5797462	1610	Knotted Phyllite with 20% quartz	0.01
136100	5450 Trench Top	1.00	2.00				Knotted Phyllite with 20% quartz	0.005
136101	5450 Trench Top	2.00	3.00				Knotted Phyllite	0.005
136102	5450 Trench Top	3.00	4.00				Knotted Phyllite with 10% quartz, siltstone lenses	0.005
136103	5450 Trench Top	4.00	5.00				Knotted Phyllite with 5% quartz, pyrite	0.01
136104	5450 Trench Top	5.00	6.00				Knotted Phyllite with 25% quartz, pyrite	0.01
136105	5450 Trench Top	6.00	7.00				Knotted Phyllite with 10% quartz	0.005

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136106	5450 Trench Top	7.00	8.00				Knotted Phyllite with 50% quartz	0.01
136107	5450 Trench Top	8.00	9.00				Knotted Phyllite with 15% quartz	0.005
136108	5450 Trench Top	9.00	10.00				Knotted Phyllite with 10% quartz	0.01
136109	5450 Trench Top	10.00	11.00	665148	5797475	1610	Knotted Phyllite	0.005
136110	5450 Trench Top	11.00	12.00				Knotted Phyllite	0.02
136111	5450 Trench Top	12.00	13.00				Knotted Phyllite	0.01
136112	5450 Trench Top	13.00	14.00				Knotted Phyllite	0.01
136113	5450 Trench Top	14.00	15.00				Knotted Phyllite	0.01
136114	5450 Trench Top	15.00	16.00				Knotted Phyllite	0.005
136115	5450 Trench Top	16.00	17.00				Knotted Phyllite	0.005
136116	5450 Trench Top	17.00	18.00				Knotted Phyllite	0.01
136117	5450 Trench Top	18.00	19.00				Knotted Phyllite	0.01
136118	5450 Trench Top	19.00	20.00				Knotted Phyllite	0.005
136119	5450 Trench Top	20.00	21.00	665153	5797487	1600	Knotted Phyllite	0.005
136120	5450 Trench Top	21.00	22.00				Knotted Phyllite	0.01
136121	5450 Trench Top	22.00	23.00				Knotted Phyllite	0.01

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136122	5450 Trench Top	23.00	24.00				Knotted Phyllite	0.005
136123	5450 Trench Top	24.00	25.00				Knotted Phyllite	0.01
136124	5450 Trench Top	25.00	26.00				Knotted Phyllite	0.01
136125	5450 Trench Top	26.00	27.00				Knotted Phyllite	0.005
136126	5450 Trench Top	27.00	28.00				Knotted Phyllite	0.01
136127	5450 Trench Top	28.00	29.00				Knotted Phyllite	0.01
136128	5450 Trench Top	29.00	30.00				Knotted Phyllite	0.005
136129	5450 Trench Top	30.00	31.00	665172	5797517	1594	Knotted Phyllite	0.005
136130	5450 Trench Top	31.00	32.00				Knotted Phyllite	0.005
136131	5450 Trench Top	32.00	33.00				Knotted Phyllite	0.005
136132	5450 Trench Top	33.00	34.00				Knotted Phyllite with 1% quartz	0.005
136133	5450 Trench Top	34.00	35.00				Knotted Phyllite with 15% quartz	0.005
136134	5450 Trench Top	35.00	36.00				Knotted Phyllite	0.005
136135	5450 Trench Top	36.00	37.00				Knotted Phyllite	0.005
136136	5450 Trench Top	37.00	38.00				Knotted Phyllite	0.005
136137	5450 Trench Top	38.00	39.00				Knotted Phyllite	0.005

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136138	5450 Trench Top	39.00	40.00				Knotted Phyllite	0.005
136139	5450 Trench Top	40.00	41.00	665181	5797530	1588	Knotted Phyllite with 10% quartz, pyrite	0.01
136140	5450 Trench Top	41.00	42.00				Knotted Phyllite	0.01
136141	5450 Trench Top	42.00	43.00				Knotted Phyllite with 20% quartz	0.005
136142	5450 Trench Top	43.00	44.00				Knotted Phyllite with 15% quartz, pyrite	0.01
136143	5450 Trench Top	44.00	45.00	665181	5797535	1584	Knotted Phyllite with 15% quartz	0.02
136151	5300 Trench Bottom	0.00	1.00	665163	5797804	1527	Knotted Phyllite	0.39
136152	5300 Trench Bottom	1.00	2.00				Knotted Phyllite	0.02
136153	5300 Trench Bottom	2.00	3.00				Knotted Phyllite	0.04
136154	5300 Trench Bottom	3.00	4.00				Knotted Phyllite	0.01
136155	5300 Trench Bottom	4.00	5.00	665163	5797808	1525	Knotted Phyllite	0.05
136156	5300 Trench Bottom	5.00	6.00				Knotted Phyllite	0.06
136157	5300 Trench Bottom	6.00	7.00				Knotted Phyllite with 5% quartz	0.05
136158	5300 Trench Bottom	7.00	8.00				Knotted Phyllite	0.01
136159	5300 Trench Bottom	8.00	9.00				Knotted Phyllite	0.02
136160	5300 Trench Bottom	9.00	10.00	665162	5797818	1520	Knotted Phyllite with 10% quartz	0.03

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136161	5300 Trench Bottom	10.00	11.00				Knotted Phyllite	0.005
136162	5300 Trench Bottom	11.00	12.00				Knotted Phyllite	0.01
136163	5300 Trench Bottom	12.00	13.00				Knotted Phyllite	0.005
136164	5300 Trench Bottom	13.00	14.00				Knotted Phyllite	0.04
136165	5300 Trench Bottom	14.00	15.00	665172	5797835	1537	Knotted Phyllite	0.01
136166	5300 Trench Bottom	15.00	16.00				Knotted Phyllite	0.04
136167	5300 Trench Bottom	16.00	17.00				Knotted Phyllite	0.01
136168	5300 Trench Bottom	17.00	18.00				Knotted Phyllite	0.05
136169	5300 Trench Bottom	18.00	19.00				Knotted Phyllite	0.01
136170	5300 Trench Bottom	19.00	20.00	665172	5797824	1516	Knotted Phyllite	0.005
136171	5300 Trench Bottom	20.00	21.00				Knotted Phyllite	0.01
136172	5300 Trench Bottom	21.00	22.00				Knotted Phyllite	0.01
136173	5300 Trench Bottom	22.00	23.00				Knotted Phyllite	0.02
136174	5300 Trench Bottom	23.00	24.00				Knotted Phyllite	0.02
136175	5300 Trench Bottom	24.00	25.00	665171	5797828	1516	Knotted Phyllite	0.03
136176	5300 Trench Bottom	25.00	26.00				Knotted Phyllite	0.02

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136177	5300 Trench Bottom	26.00	27.00				Knotted Phyllite	0.04
136178	5300 Trench Bottom	27.00	28.00				Knotted Phyllite	0.16
136179	5300 Trench Bottom	28.00	29.00				Knotted Phyllite with 15% quartz	0.06
136180	5300 Trench Bottom	29.00	30.00	665173	5797833	1516	Knotted Phyllite with 5% quartz	0.07
136181	5300 Trench Bottom	34.00	35.00				Knotted Phyllite	0.01
136182	5300 Trench Bottom	35.00	36.00				Knotted Phyllite with 10% quartz	0.03
136183	5300 Trench Bottom	36.00	37.00				Knotted Phyllite with 15% quartz	0.08
136184	5300 Trench Bottom	37.00	38.00				Knotted Phyllite	0.06
136185	5300 Trench Bottom	38.00	39.00	665182	5797841	1515	Knotted Phyllite	0.005
136186	5300 Trench Bottom	39.00	40.00				Knotted Phyllite with 5% quartz	0.005
136187	5300 Trench Bottom	40.00	41.00				Knotted Phyllite with 50% quartz	0.005
136188	5300 Trench Bottom	41.00	42.00				Knotted Phyllite with 10% quartz	0.005
136189	5300 Trench Bottom	42.00	43.00				Knotted Phyllite with 20% quartz	0.005
136190	5300 Trench Bottom	43.00	44.00	665176	5797844	1508	Knotted Phyllite	0.02
136191	5300 Trench Bottom	44.00	45.00				Knotted Phyllite	0.005
136192	5300 Trench Bottom	45.00	46.00				Knotted Phyllite	0.005

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
136193	5300 Trench Bottom	46.00	47.00				Knotted Phyllite	0.01
136194	5300 Trench Bottom	50.00	51.00				Knotted Phyllite	0.04
136195	5300 Trench Bottom	51.00	52.00	665186	5797850	1513	Knotted Phyllite	0.06
136196	5300 Trench Bottom	52.00	53.00				Knotted Phyllite	0.01
136197	5300 Trench Bottom	53.00	54.00				Knotted Phyllite	0.02
136198	5300 Trench Bottom	54.00	55.00				Knotted Phyllite	0.06
136199	5300 Trench Bottom	55.00	56.00				Knotted Phyllite with 25% quartz	0.23
136200	5300 Trench Bottom	56.00	57.00	665191	5797853	1509	Knotted Phyllite	0.23
136201	5300 Trench Bottom	57.00	58.00				Knotted Phyllite with 5% quartz	0.29
136202	5300 Trench Bottom	58.00	59.00				Knotted Phyllite with 25% quartz	0.005
136203	5300 Trench Bottom	59.00	60.00				Knotted Phyllite with 20% quartz	0.005
136204	5300 Trench Bottom	60.00	61.00				Knotted Phyllite with 10% quartz	0.01
136205	5300 Trench Bottom	61.00	62.00	665198	5797860	1513	Knotted Phyllite	0.06
136206	5300 Trench Bottom	62.00	63.00				Knotted Phyllite with 15% quartz	0.09
136207	5300 Trench Bottom	63.00	64.00				Knotted Phyllite with 5% quartz	0.02
136208	T100 Trench	0.00	1.00	665584	5797237	1529	Knotted Phyllite	0.01
136209	T100 Trench	1.00	2.00				Knotted Phyllite with	0.01

Sample #	Field ID	From	To	Easting NAD83	Northing NAD83	Elevation (m)	Description	SFA Au (g/t)
							<1% quartz	
136210	T100 Trench	2.00	3.00				Knotted Phyllite	0.01
136211	T100 Trench	3.00	4.00				Knotted Phyllite	0.01
136212	T100 Trench	4.00	5.00				Knotted Phyllite	0.01
136213	T100 Trench	5.00	6.00	665587	5797242	1529	Knotted Phyllite	0.01
136214	T100 Trench	6.00	7.00				Knotted Phyllite with <2% quartz	0.01
136215	T100 Trench	7.00	8.00				Knotted Phyllite	0.01
136216	T100 Trench	8.00	9.00				Knotted Phyllite	0.01
136217	T100 Trench	9.00	10.00				Knotted Phyllite	0.01
136218	T100 Trench	10.00	11.00	665593	5797244	1525	Knotted Phyllite	0.01
136219	T100 Trench	11.00	12.00				Knotted Phyllite	0.01
136220	T100 Trench	12.00	13.00				Knotted Phyllite with 10% quartz	0.01
136221	T100 Trench	13.00	14.00				Knotted Phyllite	0.09
136222	T100 Trench	14.00	15.00				Knotted Phyllite	0.23
136223	T100 Trench	15.00	16.00	665593	5797244	1522	Knotted Phyllite	0.01

APPENDIX III

Fire Assay and Screened Metallic Assay Values of 2007 Underground Channel Samples

Appendix III. Fire Assay and Screened Metallic Assay Values of 2007 Underground Channel Samples

Fire Assay						Screened Metallic Au Total (g/t)
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	
136293	Rock	4.9	0.50	0.54	0.52	0.45
136294	Rock	2.3	0.20	0.18	0.19	0.34
136295	Rock	4.7	0.41	0.43	0.42	0.36
136296	Rock	2.5	2.53	2.77	2.65	3.33
136297	Rock	3.6	0.10	0.08	0.09	0.07
136298	Rock	3.3	0.33	0.32	0.325	0.34
136299	Rock	3.4	0.16	0.20	0.18	0.16
136300	Rock	2.9	0.08	0.06	0.07	
136301	Rock	4.8	0.06	0.07	0.065	
136302	Rock	3.8	0.08	0.09	0.085	
136303	Rock	2.3	0.34	0.27	0.305	0.20
136304	Rock	2.9	0.34	0.33	0.335	0.34
136305	Rock	2.5	0.05	0.05	0.05	
136306	Rock	3.2	0.02	0.02	0.02	
136307	Rock	3.9	0.12	0.15	0.135	0.12
136308	Rock	4.1	0.03	0.04	0.035	
136309	Rock	3.0	0.04	0.03	0.035	
136310	Rock	1.6	0.03	0.04	0.035	
136311	Rock	4.2	0.28	0.24	0.26	0.39
136312	Rock	2.4	0.03	0.03	0.03	
136313	Rock	3.7	0.02	0.02	0.02	
136314	Rock	2.8	0.02	0.02	0.02	
136315	Rock	2.9	0.19	0.19	0.19	0.16
136316	Rock	2.6	1.79	1.21	1.5	1.87
136317	Rock	5.2	0.18	0.18	0.18	0.19
136318	Rock	1.5	0.92	0.87	0.895	0.76

Fire Assay						Screened
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	Metallic Au Total (g/t)
136319	Rock	6.2	0.73	0.59	0.66	0.70
136320	Rock	3.0	0.11	0.12	0.115	0.17
136321	Rock	3.8	0.01	0.01	0.01	
136322	Rock	2.3	0.08	0.05	0.065	
136323	Rock	4.5	0.01	0.01	0.01	
136324	Rock	2.9	0.26	0.32	0.29	0.30
136325	Rock	3.9	0.04	0.04	0.04	
136326	Rock	8.3	0.02	0.02	0.02	
136327	Rock	5.4	0.01	0.02	0.015	
136328	Rock	7.2	0.02	0.03	0.025	
136329	Rock	6.0	0.07	0.04	0.055	
136330	Rock	4.3	0.11	0.07	0.09	0.13
136331	Rock	5.5	0.01	0.01	0.01	
136332	Rock	5.3	0.05	0.06	0.055	
136333	Rock	4.8	0.01	0.01	0.01	
136334	Rock	3.2	0.10	0.08	0.09	0.08
136335	Rock	5.2	0.01	0.01	0.01	
136336	Rock	4.5	0.04	0.03	0.035	
136337	Rock	7.9	0.03	0.05	0.04	
136338	Rock	6.1	0.03	0.04	0.035	
136339	Rock	6.2	0.03	0.02	0.025	
136340	Rock	4.2	0.02	0.01	0.015	
136341	Rock	3.8	0.04	0.03	0.035	
136342	Rock	2.5	0.05	0.07	0.06	
136343	Rock	2.9	0.10	0.09	0.095	0.14
136344	Rock	3.7	0.05	0.07	0.06	

Fire Assay						Screened Metallic Au Total (g/t)
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	
136345	Rock	3.9	0.03	0.02	0.025	
136346	Rock	3.7	0.05	0.06	0.055	
136347	Rock	5.4	0.03	0.04	0.035	
136348	Rock	1.8	0.02	0.01	0.015	
136349	Rock	6.5	0.04	0.04	0.04	
136350	Rock	5.6	0.02	0.04	0.03	
136401	Rock	2.1	0.02	0.04	0.03	
136402	Rock	2.8	0.01	0.01	0.005	
136403	Rock	5.2	0.44	0.70	0.57	0.41
136404	Rock	2.7	0.03	0.04	0.035	
136405	Rock	3.9	0.05	0.03	0.04	
136406	Rock	3.7	0.02	0.02	0.02	0.08 - 2.77
136407	Rock	4.0	0.09	0.10	0.095	
136408	Rock	3.0	0.09	0.07	0.08	
136409	Rock	5.2	5.40	6.75	6.075	6.37
136410	Rock	4.1	1.49	1.95	1.72	2.33
136411	Rock	4.9	1.85	1.85	1.85	1.82
136412	Rock	4.4	0.67	0.70	0.685	0.71
136413	Rock	5.8	0.21	0.21	0.21	0.21
136414	Rock	1.4	0.65	0.34	0.495	0.36
136415	Rock	6.3	0.07	0.11	0.09	
136416	Rock	4.6	0.06	0.11	0.085	
136417	Rock	4.0	0.06	0.07	0.065	
136418	Rock	2.3	1.97	1.59	1.78	2.08
136419	Rock	3.6	0.25	0.25	0.25	0.31
136420	Rock	3.8	0.32	0.29	0.305	0.36

Fire Assay						Screened
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	Metallic Au Total (g/t)
136421	Rock	4.5	0.34	0.44	0.39	0.43
136422	Rock	1.5	0.51	0.37	0.44	0.51
136423	Rock	4.7	1.13	0.67	0.9	1.23
136424	Rock	2.7	0.24	0.42	0.33	0.67
136425	Rock	5.7	0.56	0.47	0.515	0.73
136426	Rock	4.7	0.24	0.30	0.27	0.28
136427	Rock	6.3	0.91	1.02	0.965	1.39
136428	Rock	3.0	0.40	0.42	0.41	0.45
136429	Rock	7.4	1.77	1.50	1.635	2.27
136430	Rock	3.3	1.00	0.84	0.92	1.22
136431	Rock	2.0	0.29	0.30	0.295	0.32
136432	Rock	2.5	1.13	0.98	1.055	1.20
136433	Rock	6.6	0.29	0.30	0.295	0.39
136434	Rock	4.1	0.11	0.14	0.125	0.10
136435	Rock	4.0	1.17	1.19	1.18	1.62
136436	Rock	4.6	0.16	0.16	0.16	0.25
136437	Rock	6.4	0.93	0.41	0.67	0.50
136438	Rock	5.6	0.03	0.05	0.04	
136439	Rock	2.3	1.11	0.42	0.765	0.51
136440	Rock	1.6	0.18	0.13	0.155	0.20
136441	Rock	5.9	1.22	1.12	1.17	1.76
136442	Rock	6.3	0.54	0.51	0.525	0.58
136443	Rock	6.6	0.09	0.05	0.07	
136444	Rock	5.3	0.04	0.04	0.04	
136445	Rock	5.7	1.47	1.45	1.46	2.41
136446	Rock	3.5	0.22	0.14	0.18	0.18

Fire Assay						Screened Metallic Au Total (g/t)
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	
136447	Rock	5.5	0.31	0.34	0.325	0.32
136448	Rock	5.2	0.06	0.06	0.06	
136449	Rock	3.5	0.06	0.05	0.055	
136450	Rock	5.2	0.02	0.02	0.02	
136451	Rock	6.3	0.13	0.16	0.145	0.15
136452	Rock	4.9	0.11	0.10	0.105	0.23
136453	Rock	5.3	0.02	0.01	0.015	
136454	Rock	3.6	0.08	0.06	0.07	
136455	Rock	5.4	0.11	0.12	0.115	0.15
136456	Rock	3.2	0.24	0.15	0.195	0.14
136457	Rock	4.6	0.16	0.35	0.255	0.17
136458	Rock	4.3	0.37	0.35	0.36	0.58
136459	Rock	9.1	0.49	0.45	0.47	0.52
136460	Rock	4.8	0.95	0.70	0.825	0.94
136461	Rock	6.6	0.29	0.25	0.27	0.29
136462	Rock	3.4	0.25	0.28	0.265	0.27
136463	Rock	4.0	0.51	0.45	0.48	0.54
136464	Rock	3.0	0.19	0.18	0.185	0.20
136465	Rock	6.5	1.57	1.50	1.535	1.64
136466	Rock	5.7	0.09	0.10	0.095	
136467	Rock	5.0	0.54	0.45	0.495	0.58
136468	Rock	5.1	0.76	0.75	0.755	0.73
136469	Rock	7.0	0.23	0.24	0.235	0.18
136470	Rock	6.1	0.29	0.24	0.265	0.25
136471	Rock	8.3	0.39	0.31	0.35	0.45
136472	Rock	5.8	0.34	0.26	0.3	0.21

Fire Assay						Screened
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	Metallic Au Total (g/t)
136473	Rock	5.3	0.21	0.18	0.195	0.18
136474	Rock	2.7	0.02	0.02	0.02	
136475	Rock	8.1	0.17	0.19	0.18	0.25
136476	Rock	6.4	0.53	0.65	0.59	0.56
136477	Rock	6.2	0.15	0.16	0.155	0.16
136478	Rock	2.6	0.11	0.12	0.115	0.14
136479	Rock	4.9	0.18	0.20	0.19	0.22
136480	Rock	2.6	0.54	0.57	0.555	0.78
136481	Rock	4.6	0.30	0.31	0.305	0.32
136482	Rock	4.1	2.03	3.02	2.525	1.77
136483	Rock	5.0	18.98	26.61	22.795	23.20
136484	Rock	5.0	1.00	1.02	1.01	1.04
136485	Rock	7.0	0.42	0.63	0.525	0.55
136486	Rock	2.1	0.55	0.56	0.555	0.76
136487	Rock	3.0	0.77	0.95	0.86	0.85
136488	Rock	--	0.04	0.05	0.045	
136489	Rock	4.2	0.61	0.52	0.565	0.72
136490	Rock	4.7	0.34	0.25	0.295	0.30
136491	Rock	2.6	0.44	0.35	0.395	0.49
136492	Rock	--	0.10	0.11	0.105	
136493	Rock	4.8	0.57	0.48	0.525	0.47
136494	Rock	3.6	1.43	2.29	1.86	2.25
136495	Rock	4.8	0.48	0.45	0.465	0.43
136496	Rock	5.9	0.30	0.34	0.32	0.48
136497	Rock	3.6	0.09	0.05	0.07	
136498	Rock	2.1	0.17	0.13	0.15	0.15

Fire Assay						Screened Metallic Au Total (g/t)
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	
136499	Rock	3.1	0.01	0.01	0.01	
136500	Rock	1.6	0.02	0.01	0.015	
136501	Rock	5.0	0.01	0.01	0.01	
136502	Rock	2.8	0.01	0.01	0.01	
136503	Rock	4.5	0.02	0.02	0.02	
136504	Rock	4.8	0.05	0.08	0.065	
136505	Rock	1.9	0.11	0.14	0.125	0.18
136506	Rock	6.7	0.03	0.05	0.04	
136507	Rock	4.0	0.19	0.29	0.24	0.30
136508	Rock	2.4	0.29	0.31	0.3	0.51
136509	Rock	3.4	0.33	0.42	0.375	0.46
136510	Rock	4.5	0.08	0.08	0.08	
136511	Rock	--	0.13	0.12	0.125	
136512	Rock	3.4	0.13	0.06	0.095	0.19
136513	Rock	9.4	1.59	1.64	1.615	2.02
136514	Rock	4.0	7.56	3.64	5.6	6.91
136515	Rock	2.9	16.14	11.09	13.615	12.83
136516	Rock	5.1	0.19	0.20	0.195	0.22
136517	Rock	1.5	0.27	0.24	0.255	0.44
136518	Rock	5.5	0.07	0.03	0.05	
136519	Rock	4.8	0.09	0.10	0.095	
136520	Rock	5.1	0.09	0.11	0.1	
136521	Rock	3.4	0.02	0.02	0.02	
136522	Rock	6.3	0.26	0.37	0.315	0.32
136523	Rock	3.3	0.10	0.10	0.1	
136524	Rock	4.5	0.74	0.71	0.725	0.85

Fire Assay						Screened Metallic Au Total (g/t)
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	
136525	Rock	2.8	0.63	0.74	0.685	0.76
136526	Rock	4.9	0.37	0.43	0.4	0.39
136527	Rock	6.4	1.12	1.31	1.215	1.20
136528	Rock	4.0	86.85	103.76	95.305	37.77
136529	Rock	3.0	0.20	0.15	0.175	0.31
136530	Rock	2.3	0.08	0.09	0.085	
136531	Rock	4.9	0.27	0.37	0.32	0.36
136532	Rock	2.3	0.03	0.03	0.03	
136533	Rock	5.9	0.23	0.21	0.22	0.28
136534	Rock	2.5	0.03	0.03	0.03	
136535	Rock	3.7	0.06	0.06	0.06	
136536	Rock	2.4	0.09	0.07	0.08	
136537	Rock	2.1	0.06	0.06	0.06	
136538	Rock	2.0	0.08	0.10	0.09	
136539	Rock	5.0	0.01	0.02	0.015	
136540	Rock	2.4	0.02	0.02	0.02	
136541	Rock	3.3	0.02	0.02	0.02	
136542	Rock	2.9	0.02	0.01	0.015	
136543	Rock	4.6	0.01	0.01	0.01	
136544	Rock	3.1	0.01	0.01	0.01	
136545	Rock	5.2	0.02	0.03	0.025	
136546	Rock	4.2	0.01	0.01	0.01	
136547	Rock	4.9	0.01	0.01	0.01	
136548	Rock	2.2	0.01	0.01	0.0075	
136549	Rock	3.4	0.01	0.01	0.01	
136550	Rock	5.6	0.08	0.07	0.075	

Fire Assay						Screened Metallic Au Total (g/t)
Sample	Type	Wt (kg)	Au (g/t)	Au (g/t)	Avg FA (g/t)	
136551	Rock	4.5	0.03	0.02	0.025	
136552	Rock	1.4	0.04	0.04	0.04	
136553	Rock	4.9	0.02	0.01	0.015	
RE 136293	Repeat	--	0.55	0.55		
RE 136312	Repeat	--	0.04	0.03		
RE 136332	Repeat	--	0.05	0.06		
RE 136401	Repeat	--	0.01	0.01		
RE 136421	Repeat	--	0.36	0.38		
RE 136440	Repeat	--	0.16	0.09		
RE 136460	Repeat	--	0.72	0.61		
RE 136479	Repeat	--	0.20	0.17		
RE 136499	Repeat	--	0.01	0.01		
RE 136518	Repeat	--	0.04	0.04		
RE 136538	Repeat	--	0.06	0.06		
Blank IPL	Blk IPL	--	<0.01	<0.01		
GS-1P5B	STD IPL	--	1.48	1.49		
GS-1P5B REF	STD IPL	--	1.46	1.46		
Minimum detection		0.1	0.01	0.01		0.01
Maximum detection		9999	5000	5000		5000
Method		Spec	FA/AAS	FA/AAS		FA/AAS

APPENDIX IV
DRILL HOLES USED IN RESOURCE ESTIMATE

Appendix IV Drill Holes Used In Resource Estimate

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
07304A	666505.13	5795990.07	1564.51	154.84	DDH
07304B	666504.09	5795989.19	1564.44	262.43	DDH
07295	665123.51	5797758.32	1541.33	227.38	DDH
07296	665234.22	5797656.93	1537.50	215.19	DDH
07297	665287.18	5797548.34	1545.37	218.23	DDH
07298	665308.45	5797507.77	1549.58	227.38	DDH
07299	665505.70	5797344.64	1522.90	212.45	DDH
07300	665522.56	5797212.70	1556.20	216.15	DDH
07301	665619.13	5797209.38	1517.37	209.09	DDH
07302	665491.23	5797243.86	1557.41	233.78	DDH
07303	666258.66	5796412.65	1574.64	236.83	DDH
07305	665272.15	5797496.87	1563.43	244.75	DDH
07306	665226.26	5797546.57	1565.21	267.31	DDH
07307	665431.42	5797284.59	1564.36	248.72	DDH
07308	665172.03	5797625.64	1561.97	239.88	DDH
07309	665172.03	5797625.64	1561.97	200.56	DDH
83001	665525.35	5797208.69	1555.87	328.30	DDH
83002	665211.58	5797566.60	1566.27	322.20	DDH
83003	665346.56	5797407.22	1559.97	324.90	DDH
83004	665017.63	5797791.90	1555.17	239.90	DDH
83005	665143.92	5797493.70	1597.77	428.90	DDH
08310	665147.64	5797663.00	1558.95	53.95	DDH
08311	665054.05	5797790.55	1555.64	273.41	DDH
08312	665145.58	5797661.56	1559.27	255.12	DDH
08313	665035.26	5797809.33	1555.10	288.65	DDH
08314	665079.48	5797679.73	1571.33	233.79	DDH

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
08315	665139.16	5797812.38	1525.50	178.61	DDH
08316	665170.01	5797814.74	1518.39	130.30	DDH
08317	665263.12	5797510.50	1563.61	251.76	DDH
08318	665181.49	5797646.96	1554.44	224.33	DDH
08319	665386.64	5797347.24	1561.30	203.30	DDH
08320	665320.99	5797462.51	1555.36	239.57	DDH
08321	665370.08	5797369.13	1560.11	203.30	DDH
08322	665355.01	5797389.00	1559.71	252.07	DDH
08323	665196.52	5797586.32	1565.28	233.78	DDH
08324	665475.67	5797258.59	1558.65	196.90	DDH
08325	665064.95	5797701.66	1571.48	267.00	DDH
08326	665553.54	5797192.22	1550.51	195.68	DDH
08327	665617.89	5797141.31	1538.63	224.33	DDH
08328	665060.51	5797839.36	1538.59	206.40	DDH
08329	665294.57	5797472.46	1561.86	230.79	DDH
08330	665084.03	5797871.72	1525.07	200.25	DDH
08331	665322.66	5797435.28	1561.70	226.16	DDH
08332	665417.79	5797305.74	1563.04	206.04	DDH
08333	665121.26	5797896.75	1510.91	129.80	DDH
08334	665321.55	5797589.68	1525.52	193.85	DDH
08335	665050.98	5797907.24	1524.74	203.00	DDH
08336	665216.68	5797772.74	1515.88	81.38	DDH
08337	665257.18	5797709.53	1517.62	129.54	DDH
08338	665231.58	5797734.65	1519.82	127.10	DDH
08339	665173.75	5797782.17	1523.67	108.81	DDH
08340	665684.94	5797204.79	1493.80	145.08	DDH
08341	665102.45	5797794.08	1538.09	51.21	DDH

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
08342	665543.58	5797119.93	1571.54	199.95	DDH
08343	665102.25	5797793.46	1538.22	148.13	DDH
08344	665443.46	5797457.32	1512.01	126.19	DDH
08345	664988.90	5797769.06	1569.43	291.69	DDH
08346	665409.93	5797459.05	1523.71	169.77	DDH
08347	665398.09	5797478.46	1523.46	145.39	DDH
08348	665005.11	5797854.22	1547.57	69.19	DDH
08349	665413.49	5797403.38	1538.17	172.51	DDH
08350	665005.10	5797854.14	1547.56	230.43	DDH
08351	665460.11	5797359.20	1532.97	157.28	DDH
08352	665477.49	5797423.15	1505.92	111.56	DDH
08353	664974.36	5797894.97	1543.31	200.25	DDH
08354	665528.44	5797366.25	1508.38	151.18	DDH
08355	665531.55	5797401.04	1494.80	102.72	DDH
08356	665600.03	5797337.84	1493.53	96.62	DDH
08357	665259.67	5797672.57	1524.77	154.53	DDH
08358	665638.15	5797292.70	1492.27	99.36	DDH
08359	665674.02	5797262.27	1484.34	84.43	DDH
08360	665620.01	5797324.21	1490.09	141.43	DDH
08361	665475.31	5797339.82	1536.17	145.08	DDH
08362	665509.89	5797302.00	1533.61	151.18	DDH
08363	665025.82	5797878.80	1536.92	221.28	DDH
08364	664961.32	5797796.30	1569.02	282.55	DDH
08365	664962.36	5797955.66	1531.54	206.35	DDH
08366	664908.99	5797971.98	1533.88	206.35	DDH
08367	664994.22	5797921.12	1534.79	203.00	DDH
84006	665608.81	5797104.52	1538.55	56.70	DDH

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
84006A	665637.29	5797123.06	1534.77	260.90	DDH
84007	665478.55	5797315.53	1541.87	275.80	DDH
84008	665276.92	5797495.41	1563.57	300.80	DDH
84009	665155.13	5797646.76	1560.47	484.60	DDH
84010	665771.48	5796933.47	1512.47	245.70	DDH
84011	665383.96	5797213.90	1612.97	412.40	DDH
84012	665984.15	5796676.07	1542.47	286.10	DDH
84013	665867.45	5796687.26	1564.95	312.70	DDH
84014	665652.91	5796998.00	1562.67	239.00	DDH
85DDH-1	667132.22	5794440.30	1782.98	446.54	DDH
85DDH-2	667245.21	5794965.73	1770.79	230.13	DDH
86002	665211.52	5797566.47	1566.05	148.50	RC
86002B	665211.52	5797566.47	1566.05	81.00	RC
86009	665155.08	5797646.62	1560.25	70.50	RC
86009B	665155.08	5797646.62	1560.25	88.00	RC
86015	665264.08	5797638.93	1533.77	78.60	DDH
86016	665303.81	5797606.01	1527.87	72.50	DDH
86017	665224.15	5797669.78	1535.65	106.30	DDH
86018	665224.15	5797669.78	1535.65	69.50	DDH
86019	665198.31	5797708.48	1535.27	75.50	DDH
86020	665152.79	5797738.58	1544.17	72.50	DDH
86021	665634.02	5797315.03	1489.67	87.80	DDH
86022	665603.84	5797362.95	1483.07	69.40	DDH
86023	665183.72	5797723.41	1535.05	90.80	DDH
86024	665206.69	5797687.35	1534.65	69.40	DDH
86025	665134.20	5797754.54	1544.17	42.00	DDH
86026	665134.20	5797754.54	1544.17	102.10	DDH

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
86027	665171.79	5797722.96	1536.87	69.40	DDH
86028	664274.90	5798586.51	1375.27	185.90	DDH
86029	663458.27	5799299.62	1392.17	19.20	DDH
86029A	663460.56	5799310.75	1391.95	200.20	DDH
86030	663891.80	5799023.82	1382.07	163.70	DDH
86031	663750.63	5799085.11	1395.27	185.30	DDH
86032	663589.34	5799210.90	1394.47	48.20	DDH
86032A	663589.37	5799210.86	1394.25	212.70	DDH
87024	665214.81	5797690.99	1534.87	72.00	RC
87033	665208.27	5797700.16	1535.27	73.00	RC
87034	665115.96	5797779.20	1539.17	70.50	RC
87035	665221.07	5797680.56	1535.57	72.00	RC
87036	665246.09	5797651.24	1536.67	69.00	RC
87037	665370.09	5797532.97	1522.77	82.50	RC
87038	665426.29	5797449.80	1520.77	91.50	RC
87039	665483.90	5797363.94	1524.67	79.50	RC
87040	665560.40	5797294.77	1519.57	81.00	RC
87041	665619.53	5797223.79	1514.87	90.00	RC
87042	665042.43	5797846.11	1542.97	98.00	RC
87043	664999.63	5797956.59	1525.87	93.00	RC
87044	664914.76	5798010.64	1524.57	70.50	RC
87045	665590.93	5797263.03	1516.57	81.00	RC
87046	665527.46	5797326.51	1522.07	90.00	RC
87047	665415.88	5797384.47	1541.37	15.00	RC
87047A	665420.32	5797380.57	1541.37	27.00	RC
87048	665370.28	5797463.91	1539.97	109.50	RC
87049	665308.23	5797556.21	1539.17	111.00	RC

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
87050	665267.58	5797614.44	1540.07	111.00	RC
87051	665399.52	5797413.44	1540.87	120.00	RC
87052	665335.35	5797509.24	1539.45	111.60	DDH
87053	665335.35	5797509.24	1539.45	99.40	DDH
87054	665560.40	5797294.77	1519.57	81.10	DDH
87055	665253.42	5797624.03	1539.45	85.30	DDH
87056	665253.42	5797624.03	1539.45	92.00	DDH
88001	662668.85	5798915.36	1684.87	160.93	DDH
88002	662668.85	5798915.36	1684.87	192.94	DDH
88003	662632.88	5798939.72	1687.07	153.31	DDH
88004	662632.88	5798939.72	1687.07	176.17	DDH
88005	662711.65	5798890.48	1682.57	98.45	DDH
88006	662711.65	5798890.48	1682.57	74.79	DDH
88057	665253.42	5797624.03	1539.45	88.70	DDH
88058	665287.44	5797576.56	1538.95	91.40	DDH
88059	665287.44	5797576.56	1538.95	87.20	DDH
88060	665321.11	5797533.82	1538.95	88.10	DDH
88061	665321.11	5797533.82	1538.95	90.20	DDH
88062	665269.67	5797604.40	1539.45	93.00	DDH
88063	665269.67	5797604.40	1539.45	72.50	DDH
88064	665448.72	5797392.76	1526.55	75.00	DDH
88065	665448.72	5797392.76	1526.55	74.70	DDH
88066	665427.00	5797442.43	1520.95	77.70	DDH
88067	665427.00	5797442.43	1520.95	75.30	DDH
88068	665248.74	5797526.77	1563.67	153.90	DDH
88069	665235.01	5797725.39	1519.97	76.20	RC
88070	665227.87	5797717.68	1523.27	38.10	RC

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
88071	665224.64	5797710.72	1525.17	89.90	RC
88072	665212.56	5797711.37	1528.17	97.50	RC
88073	665203.43	5797762.33	1521.77	97.50	RC
88074	665197.71	5797757.32	1523.67	96.00	RC
88075	665189.41	5797751.51	1526.47	97.50	RC
88076	665183.17	5797746.65	1528.67	94.50	RC
88077	665162.95	5797736.93	1537.87	96.00	RC
88078	665242.84	5797658.63	1536.47	74.70	RC
88079	665276.04	5797621.68	1541.07	72.50	RC
88080	665267.99	5797612.16	1539.77	105.20	RC
88081	665250.88	5797668.40	1527.27	56.40	RC
88082	665257.65	5797671.38	1526.17	47.20	RC
88083	665289.46	5797634.50	1524.77	59.47	RC
88084	665282.18	5797625.58	1531.87	74.70	RC
88085	665291.63	5797567.70	1539.37	73.20	RC
88086	665295.98	5797573.12	1539.07	67.10	RC
88087	665300.00	5797577.49	1537.77	62.50	RC
88088	665320.16	5797525.64	1541.27	80.70	RC
88089	665325.16	5797530.72	1539.37	64.00	RC
88090	665335.60	5797533.27	1534.67	47.20	RC
88091	665338.10	5797545.44	1531.77	30.50	RC
88092	665310.86	5797587.57	1529.87	22.90	RC
88093	665214.83	5797690.97	1534.97	24.40	RC
88094	665111.10	5797772.14	1540.67	70.10	RC
88095	665117.58	5797778.80	1539.37	56.40	RC
88096	665221.17	5797673.62	1536.07	83.80	RC
88097	665238.60	5797693.34	1526.87	47.20	RC

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
88098	665246.42	5797702.55	1522.57	17.70	RC
88099	665275.77	5797656.62	1524.57	36.60	RC
88100	665271.08	5797652.95	1525.37	48.80	RC
88101	665293.70	5797604.94	1529.67	38.10	RC
88102	665255.82	5797636.15	1534.27	80.70	RC
88103	665281.80	5797592.24	1539.57	64.00	RC
88104	664795.23	5798440.67	1419.17	84.70	RC
88105	664874.37	5798410.82	1420.27	83.80	RC
90007	662730.37	5798887.44	1682.67	58.50	RC
90008	662692.01	5798860.58	1698.77	46.50	RC
90106	664943.35	5797993.35	1525.57	178.30	DDH
90107	664781.65	5798100.66	1518.07	224.90	DDH
90108	664662.17	5798245.20	1489.37	190.20	DDH
90109	664507.06	5798368.32	1456.07	178.40	DDH
90110	665600.93	5797150.28	1541.37	149.70	DDH
90110A	665588.30	5797155.19	1541.16	16.80	DDH
90111	665732.76	5797015.93	1524.77	150.60	DDH
90112	666141.68	5796604.98	1534.97	102.10	DDH
90113	666256.73	5796413.50	1573.97	55.10	DDH
90114	666358.95	5796214.29	1598.57	180.70	DDH
90115	666481.45	5796054.79	1574.27	149.40	DDH
90116	666589.83	5795928.59	1523.37	108.20	DDH
90117	666290.67	5796352.72	1586.67	161.50	DDH
90118	665855.29	5796811.70	1505.57	130.30	DDH
90118T	665858.69	5796812.09	1505.57	133.50	RC
90119	665953.59	5796717.61	1529.97	150.90	DDH
90119T	665957.24	5796717.15	1529.67	141.70	DDH

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
90120	662635.34	5799931.25	1364.27	236.20	DDH
90121	664617.61	5798311.86	1476.07	120.35	RC
90122	664718.12	5798186.37	1502.97	150.00	RC
90123	664740.42	5798293.35	1468.47	75.00	RC
90124	664804.74	5798211.16	1484.97	75.00	RC
90125	664841.87	5798167.20	1491.67	81.00	RC
90126	664899.81	5798100.34	1502.77	81.00	RC
90127	664996.76	5798025.16	1508.07	81.00	RC
90127A	664989.35	5798030.81	1508.27	79.50	RC
90128	665025.31	5797939.62	1525.17	124.50	RC
90129	665084.80	5797827.98	1536.77	148.50	RC
90130	665648.62	5797184.14	1514.07	90.00	RC
90131	665681.63	5797136.06	1517.87	52.50	RC
90132	665744.79	5797052.16	1515.57	75.00	RC
90133	665777.44	5796977.40	1507.77	100.50	RC
90134	665813.11	5796940.63	1499.27	109.50	RC
90135	665815.11	5796938.54	1499.17	136.50	RC
90136	665691.59	5797072.61	1532.37	85.50	RC
90137	666015.81	5796680.87	1533.07	120.00	RC
90138	666112.46	5796659.54	1521.07	64.50	RC
90139	666184.29	5796558.89	1542.27	89.50	RC
90140	666342.45	5796379.77	1559.07	89.00	RC
90141	666216.59	5796466.31	1569.38	127.50	RC
90142	666215.83	5796466.04	1569.92	139.50	RC
90143	666213.51	5796465.58	1570.67	151.50	RC
90144	666258.07	5796416.99	1574.47	126.00	RC
90145	666309.07	5796305.84	1595.47	76.50	RC

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
90146	666331.01	5796272.93	1597.67	121.50	RC
90147	666391.93	5796177.98	1592.87	121.50	RC
90147A	666392.59	5796178.56	1592.97	30.00	RC
90148	666449.04	5796109.96	1583.17	130.50	RC
90149	666417.57	5796152.24	1588.47	123.00	RC
90150	666506.95	5795989.15	1564.87	130.50	RC
90151	661962.02	5800511.01	1391.77	186.20	DDH
90152	661045.16	5801340.98	1363.97	352.40	DDH
90153	661573.69	5800746.17	1436.67	336.80	DDH
90154	660979.63	5801226.79	1384.87	263.30	DDH
90155	663149.24	5799654.98	1347.47	242.30	DDH
90156	662386.62	5800487.22	1294.77	218.50	DDH
90157	665696.00	5796835.87	1546.17	237.70	DDH
90158	665697.37	5797061.11	1532.37	144.80	DDH
90159	664046.78	5799226.41	1319.47	196.60	DDH
90160	666507.68	5795989.02	1564.87	118.50	RC
90161	666568.38	5795887.12	1523.67	130.50	RC
90161A	666568.38	5795887.12	1523.67	34.50	RC
90162	666589.36	5795925.98	1523.45	90.00	RC
90163	665879.57	5796791.15	1512.57	88.50	RC
90164	665913.06	5796759.09	1523.57	141.00	RC
90165	665975.49	5796702.16	1533.47	141.50	RC
90165A	665936.65	5796734.42	1527.87	45.00	RC
91019T	665195.01	5797711.73	1535.47	63.00	RC
91050T	665264.43	5797617.43	1539.67	105.00	RC
91166	665160.67	5797646.79	1560.57	145.50	RC
91167	665249.74	5797600.61	1548.77	114.00	RC

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
91167A	665250.79	5797600.09	1548.87	47.50	RC
91168	665274.50	5797630.86	1532.87	90.00	RC
91169	665285.65	5797644.00	1524.17	66.00	RC
91170	665209.32	5797730.84	1526.67	37.50	RC
91171	665166.18	5797675.15	1553.47	115.00	RC
91172	665295.02	5797657.62	1517.97	41.50	RC
91173	665182.26	5797695.24	1543.37	87.00	RC
91174	665230.71	5797614.93	1550.17	114.75	RC
91175	665213.13	5797632.84	1551.27	132.00	RC
91176	665196.31	5797652.53	1551.77	118.50	RC
91177	665195.73	5797651.65	1552.17	125.00	RC
91178	665177.16	5797667.67	1552.47	105.00	RC
91179	665077.35	5797784.29	1551.37	82.50	RC
91180	665095.48	5797766.85	1550.57	90.00	RC
91181	665110.39	5797748.78	1549.37	108.00	RC
91182	665127.13	5797725.97	1552.77	99.00	RC
91183	665142.00	5797704.71	1552.17	96.00	RC
91184	665159.53	5797686.29	1553.77	108.00	RC
91185	665215.55	5797595.76	1558.37	157.50	RC
91186	665189.31	5797602.67	1564.97	162.00	RC
91187	665125.43	5797683.99	1559.57	132.00	RC
91187A	665125.99	5797684.71	1559.77	66.00	RC
91188	665110.94	5797705.65	1560.27	122.00	RC
91188A	665110.65	5797705.22	1560.17	33.50	RC
91189	665110.24	5797704.71	1560.37	138.00	RC
91190	665094.67	5797725.16	1560.07	141.00	RC
91191	665080.86	5797748.22	1558.87	115.00	RC

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
91192	665063.66	5797769.20	1558.27	105.80	RC
91193	665108.49	5797823.70	1532.37	27.00	RC
91194	665150.82	5797796.90	1526.47	60.00	RC
91195	665183.66	5797756.98	1526.97	34.50	RC
91196	665252.82	5797683.28	1525.57	45.00	RC
91197	665164.38	5797732.89	1538.37	69.00	RC
91198	665155.43	5797762.13	1535.17	57.00	RC
91199	665128.24	5797767.83	1539.47	81.00	RC
91200	665097.11	5797807.98	1537.67	96.00	RC
91201	665268.22	5797581.80	1547.27	119.00	RC
91202	665299.51	5797541.27	1544.97	121.50	RC
91203	665336.42	5797547.09	1534.27	81.00	RC
91204	665353.79	5797488.62	1540.67	87.00	RC
91205	665381.82	5797443.16	1540.87	105.00	RC
91206	665396.58	5797421.40	1541.17	117.00	RC
91207	665440.50	5797355.75	1543.97	111.00	RC
91208	665237.00	5797542.13	1565.07	114.00	RC
91209	665264.45	5797536.55	1558.67	150.00	RC
91210	665287.60	5797484.82	1563.17	132.00	RC
91211	665302.86	5797463.55	1562.77	147.00	RC
91212	665335.64	5797424.73	1560.97	145.50	RC
91213	665334.85	5797423.77	1560.97	158.00	RC
91214	665326.69	5797453.68	1556.97	127.50	RC
91215	665326.21	5797453.12	1557.17	121.50	RC
91216	665378.61	5797399.20	1552.67	140.00	RC
91217	665391.60	5797374.74	1552.27	140.00	RC
91218	665409.07	5797356.47	1552.07	152.00	RC

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
91219	665429.38	5797341.14	1551.37	146.50	RC
91220	665249.07	5797553.34	1560.57	147.50	RC
91221	665314.53	5797600.32	1527.47	63.30	RC
91222	665334.21	5797585.02	1523.67	60.00	RC
91223	665395.49	5797500.10	1521.47	51.00	RC
91224	665435.47	5797472.49	1512.77	45.00	RC
91225	665449.65	5797447.98	1512.67	54.00	RC
91226A	665462.55	5797424.33	1511.77	29.00	RC
91227A	665482.36	5797409.21	1508.07	16.50	RC
91228	665408.25	5797476.86	1521.07	66.00	RC
91229	665501.89	5797352.55	1523.17	84.00	RC
91230	665523.88	5797300.13	1530.47	100.50	RC
91231	665538.44	5797277.38	1531.27	108.00	RC
91232	665554.09	5797257.53	1531.27	115.50	RC
91233	665571.54	5797238.98	1529.77	119.00	RC
91234	665582.34	5797212.45	1532.07	120.00	RC
91235	665597.00	5797190.77	1532.07	118.50	RC
91236	665595.92	5797230.27	1523.97	91.50	RC
91237	665630.37	5797192.41	1519.17	105.00	RC
91238	665615.42	5797253.60	1512.67	56.00	RC
91239	665574.49	5797283.46	1518.77	91.50	RC
91240	665480.61	5797366.10	1525.27	106.50	RC
91241	665502.00	5797393.48	1505.67	63.00	RC
91242	665503.00	5797392.72	1506.37	76.00	RC
91243	665539.53	5797360.58	1508.07	44.50	RC
91244	665555.90	5797340.51	1508.57	59.50	RC
91245	665573.29	5797322.52	1507.75	41.00	RC

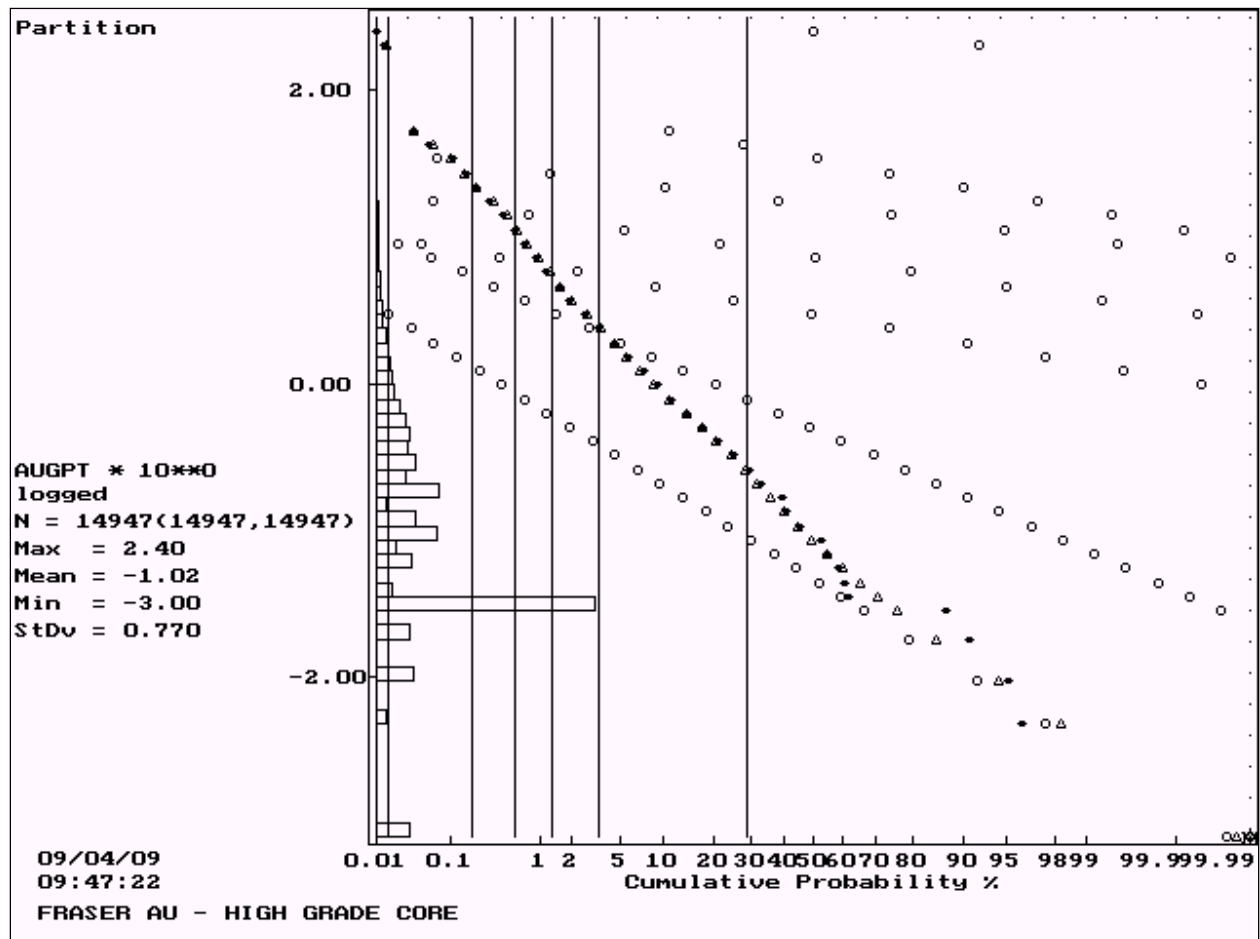
HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
91246	665592.20	5797299.07	1509.87	57.00	RC
91247	665606.15	5797283.15	1509.17	61.50	RC
91248	665640.80	5797245.56	1503.17	44.50	RC
91249	665649.88	5797217.55	1505.47	58.50	RC
91249A	665649.80	5797217.39	1505.25	20.00	RC
91250	665523.42	5797259.44	1543.67	130.50	RC
91251	665507.97	5797280.63	1542.17	130.50	RC
91252	665496.47	5797304.49	1540.37	127.50	RC
91253	665496.24	5797304.24	1540.37	133.50	RC
91254	665478.21	5797323.86	1541.97	140.00	RC
91255	665452.37	5797330.63	1546.77	143.00	RC
91256	665219.94	5797522.67	1574.07	109.00	RC
91257	665578.72	5797166.90	1546.37	152.00	RC
91258	665539.98	5797199.58	1554.27	152.00	RC
91259	666076.60	5796586.37	1570.47	145.50	RC
91260	666029.83	5796611.48	1572.37	148.50	RC
91261	665980.05	5796632.27	1572.47	42.00	RC
91261A	665979.77	5796632.06	1572.37	32.50	RC
91262	665980.23	5796631.13	1572.37	151.50	RC
91263	665827.85	5796836.33	1502.27	142.50	RC
91264	665828.71	5796835.74	1502.27	97.50	RC
91265A	665811.59	5796834.55	1502.67	76.00	RC
91266A	665851.48	5796826.03	1504.27	72.00	RC
91267A	665902.65	5796770.22	1502.95	25.90	RC
91268	665972.34	5796680.64	1542.67	160.50	RC
91269	666132.18	5796534.27	1577.67	161.00	RC
91270	665454.92	5797281.27	1562.67	127.50	RC

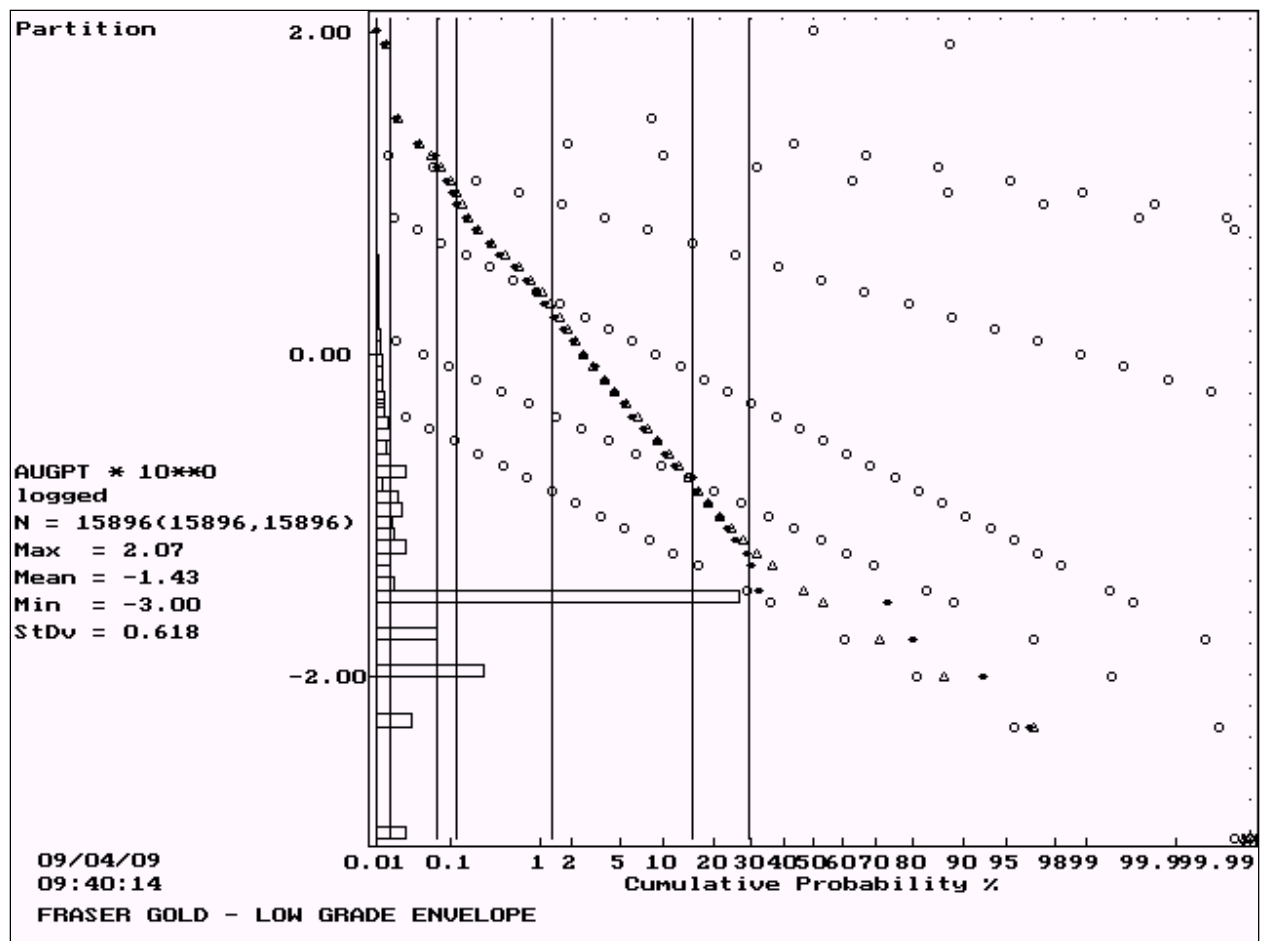
HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
91271	665395.14	5797338.84	1562.27	139.50	RC
91272	665220.80	5797561.67	1566.57	130.50	RC
91273	665449.10	5797394.88	1526.47	100.00	RC
91273A	665437.14	5797417.51	1525.27	26.00	RC
91274	665107.06	5797821.98	1532.37	41.70	RC
93275	665269.74	5797611.18	1540.64	60.00	RC
93276	665268.62	5797609.87	1540.63	60.00	RC
93277	665267.67	5797608.47	1540.52	60.00	RC
93278	665266.83	5797607.59	1540.54	52.50	RC
93279	665271.81	5797607.26	1540.74	60.00	RC
93280	665270.30	5797605.55	1540.73	60.00	RC
93281	665269.34	5797604.43	1540.74	60.00	RC
93282	665266.94	5797613.91	1540.61	60.00	RC
93283	665265.47	5797612.22	1540.57	60.00	RC
93284	665264.59	5797611.12	1540.63	60.00	RC
93285	665565.63	5797291.50	1520.56	45.00	RC
93286	665564.69	5797290.47	1520.46	45.00	RC
93287	665563.50	5797288.72	1520.44	40.50	RC
93288	665560.66	5797291.81	1520.21	40.50	RC
93289	665560.93	5797292.14	1520.49	45.00	RC
93290	665562.21	5797293.72	1520.61	42.00	RC
93291	665562.40	5797294.06	1520.67	42.00	RC
93292	665559.58	5797296.90	1520.53	45.00	RC
93293	665559.25	5797296.49	1520.17	42.00	RC
93294	665559.65	5797295.75	1520.20	40.50	RC
HK-1	666964.77	5793761.53	1660.01	32.00	RC
HK-2	666936.64	5793759.19	1662.68	118.26	RC

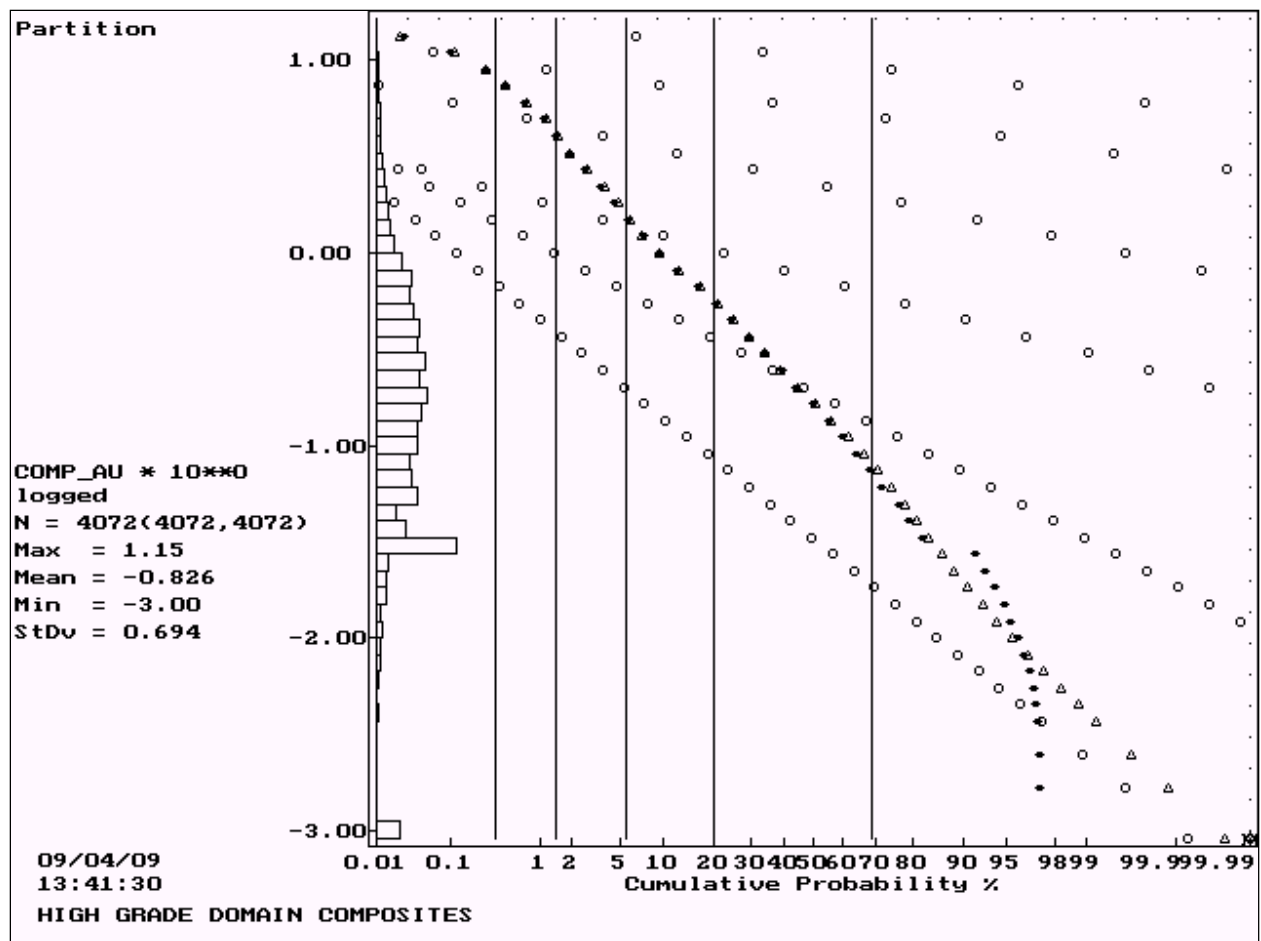
HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)	TYPE
HK-3	665947.40	5793663.08	1726.08	6.10	RC
HK-4	665580.54	5793708.79	1773.49	12.19	RC
HK-5	666152.52	5793654.87	1679.18	121.01	RC
HK-6	666702.70	5794125.00	1855.52	121.92	RC
HK-7	666647.32	5793751.02	1692.59	112.78	RC
HK-8	666564.90	5793725.42	1686.12	85.35	RC

APPENDIX V
LOGNORMAL CUMULATIVE FREQUENCY PLOTS FOR
AU IN HIGH GRADE CORE AND LOW GRADE ENVELOPE

**APPENDIX V Lognormal Cumulative Frequency Plots For
Au In High Grade Core And Low Grade Envelope**







APPENDIX VI

Specific Gravity Results

APPENDIX VI Specific Gravity Results

carbonaceous black
 CBP phyllite
 KP knotted phyllite
 SLST siltstone
 black banded
 BBP phyllite
 QV quartz vein
 LS limestone
 P phyllite

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
SG-01	07-304B	175.39	175.57	0.18	CBP		1291.90	827.32	2.781
SG-02	07-304B	197.35	197.6	0.25	BBP	high quartz	1392.75	899.94	2.826
SG-03	07-304B	212.25	212.43	0.18	BBP		1443.55	924.94	2.783
SG-04	07-304B	224.32	224.54	0.22	BBP		1425.45	908.04	2.755
SG-05	07-304B	255.9	256.1	0.2	BBP		1268.40	805.16	2.738
SG-06	07-303	9.36	9.55	0.19	KP		1250.90	805.60	2.809
SG-07	07-303	20.19	20.42	0.23	KP	slightly silty	1405.95	898.86	2.773

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
SG-08	07-303	30	30.2	0.2	KP		870.78	555.70	2.764
SG-09	07-303	39.55	39.73	0.18	KP		1030.86	643.68	2.662
SG-10	07-303	47.64	47.85	0.21	KP		1077.32	696.90	2.832
SG-11	07-303	55.85	56.08	0.23	KP		1336.75	865.28	2.835
SG-12	07-303	66.71	66.89	0.18	KP		1609.00	1043.52	2.845
SG-13	07-303	75.44	75.68	0.24	KP		1618.60	1049.37	2.843
SG-14	07-303	83.05	83.24	0.19	KP		1350.70	874.40	2.836
SG-15	07-303	91.74	91.93	0.19	KP		1332.85	860.56	2.822
SG-16	07-303	101.99	102.21	0.22	KP		1366.85	884.28	2.832
SG-17	07-303	108.45	108.65	0.2	KP		1347.60	874.68	2.850
SG-18	07-303	116.16	116.38	0.22	KP		1396.65	904.80	2.840
SG-19	07-303	126.54	126.78	0.24	CBP		1441.25	920.09	2.765
SG-20	07-303	139.37	139.61	0.24	CBP		1489.50	946.70	2.744
SG-21	07-303	150.02	150.24	0.22	CBP		1297.85	828.76	2.767
SG-22	07-303	158.2	158.38	0.18	CBP		1657.40	1056.58	2.759
SG-23	07-303	165.72	165.93	0.21	CBP		1356.05	868.24	2.780

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
	303								
SG-24	07-303	176.4	176.59	0.19	CBP		1191.48	758.38	2.751
SG-25	07-303	185.13	185.35	0.22	CBP		1417.90	904.64	2.763
SG-26	07-303	192.72	192.87	0.15	BBP		938.52	596.75	2.746
SG-27	07-303	201.52	201.72	0.2	BBP	40% quartz	1401.80	890.92	2.744
SG-28	07-303	210.73	210.89	0.16	BBP		971.28	616.58	2.738
SG-29	07-303	218.54	218.71	0.17	BBP		956.78	610.52	2.763
SG-30	07-303	227.29	227.51	0.22	BBP		1483.05	943.40	2.748
SG-31	07-303	236.45	236.61	0.16	BBP		1064.54	676.90	2.746
SG-32	07-297	6.95	7.17	0.22	KP		1043.76	669.98	2.792
SG-33	07-297	18.48	18.69	0.21	KP		1261.95	815.74	2.828
SG-34	07-297	26.87	27.05	0.18	KP	25% quartz	1196.10	775.34	2.843
SG-35	07-297	35.98	36.18	0.2	KP	high graphite; 30% quartz	1428.70	937.64	2.909
SG-36	07-297	45.29	45.43	0.14	KP		1129.10	732.04	2.844
SG-37	07-297	51.05	51.23	0.18	KP		1365.65	884.38	2.838
SG-38	07-	65.67	65.84	0.17	KP		1273.75	822.66	2.824

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
	297								
SG-39	07-297	85.23	85.4	0.17	KP		1235.05	803.84	2.864
SG-40	07-297	108.51	108.67	0.16	KP		1274.80	820.94	2.809
SG-41	07-297	123.75	123.91	0.16	BBP/KP	high silt content	1072.52	690.78	2.810
SG-42	07-297	138.12	138.29	0.17	BBP	40% quartz	1260.40	806.06	2.774
SG-43	07-297	153.22	153.36	0.14	KP		1102.56	714.82	2.844
SG-44	07-297	166.27	166.42	0.15	BBP		990.70	644.06	2.858
SG-45	07-297	182.26	182.4	0.14	BBP		1124.20	731.74	2.864
SG-46	07-297	200.25	200.4	0.15	CBP		1146.62	732.16	2.767
SG-47	07-301	14.82	14.98	0.16	KP		1227.20	794.30	2.835
SG-48	07-301	30.38	30.54	0.16	KP		1224.55	793.36	2.840
SG-49	07-301	45.79	45.99	0.2	KP		1360.45	875.04	2.803
SG-50	07-301	59.58	59.74	0.16	SLST		836.96	536.39	2.785
SG-51	07-301	76.48	76.64	0.16	KP		1327.90	870.64	2.904
SG-52	07-301	93.91	94.07	0.16	KP		1250.20	805.94	2.814
SG-53	07-301	106.21	106.39	0.18	KP		1386.85	898.58	2.840

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
SG-54	07-301	120.7	120.84	0.14	KP		1125.64	729.96	2.845
SG-55	07-301	139.12	139.29	0.17	BBP		1386.55	898.48	2.841
SG-56	07-301	161.94	162.14	0.2	BBP		1260.15	808.42	2.790
SG-57	07-301	170.71	170.89	0.18	BBP		1436.00	917.22	2.768
SG-58	07-301	178.61	178.77	0.16	CBP		1071.16	682.72	2.758
SG-59	07-301	186.75	186.96	0.21	BBP		1480.60	950.70	2.794
SG-60	07-301	203.12	203.31	0.19	BBP		1394.05	885.98	2.744
SG-61	07-298	17.95	18.12	0.17	KP		1286.08	835.70	2.856
SG-62	07-298	26.87	27.05	0.18	SLST		908.46	577.26	2.743
SG-63	07-298	35.36	35.52	0.16	QUARTZ		1100.86	682.94	2.634
SG-64	07-298	41.15	41.33	0.18	KP		857.76	555.00	2.833
SG-65	07-298	53.66	53.85	0.19	KP		1569.10	1024.90	2.883
SG-66	07-298	68.7	68.88	0.18	KP		1489.15	963.72	2.834
SG-67	07-298	78.68	78.87	0.19	KP		1646.10	1070.68	2.861
SG-68	07-298	87.98	88.19	0.21	KP		1710.15	1116.02	2.878
SG-69	07-	99.81	99.93	0.12	KP		1043.04	677.36	2.852

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
	298								
SG-70	07-298	113.14	113.3	0.16	KP/SLST		1259.35	800.48	2.744
SG-71	07-298	123.45	123.59	0.14	KP		1189.86	774.14	2.862
SG-72	07-298	135.64	135.77	0.13	KP		887.12	574.99	2.842
SG-73	07-298	149.88	150.06	0.18	BBP/KP		1430.55	936.58	2.896
SG-74	07-298	160.42	160.57	0.15	KP		1250.90	810.62	2.841
SG-75	07-298	169.02	169.22	0.2	KP		1492.30	958.68	2.797
SG-76	07-298	177.37	177.55	0.18	BBP/KP		1458.90	952.66	2.882
SG-77	07-298	189.68	189.84	0.16	BBP		1289.50	836.52	2.847
SG-78	07-298	199.04	199.18	0.14	BBP		1146.92	743.00	2.839
SG-79	07-298	212.44	212.64	0.2	BBP		1619.10	1037.20	2.782
SG-80	07-298	225.28	225.43	0.15	BBP		1217.10	773.10	2.741
SG-81	07-299	7.78	7.94	0.16	KP		1098.18	707.42	2.810
SG-82	07-299	20.15	20.33	0.18	KP		1530.45	992.14	2.843
SG-83	07-299	36.35	36.53	0.18	KP		1605.40	1046.92	2.875
SG-84	07-299	45.36	45.51	0.15	BBP		1028.52	660.18	2.792

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
SG-85	07-299	56.35	56.5	0.15	BBP		1371.10	885.16	2.822
SG-86	07-299	69.19	69.36	0.17	BBP		1407.10	913.92	2.853
SG-87	07-299	83.93	84.13	0.2	KP		1386.70	882.50	2.750
SG-88	07-299	99.1	99.33	0.23	KP		1543.85	983.92	2.757
SG-89	07-299	120.59	120.78	0.19	P		1349.45	869.70	2.813
SG-90	07-299	135.67	135.9	0.23	BBP		1510.70	987.18	2.886
SG-91	07-299	150.15	150.34	0.19	BBP		1440.65	919.48	2.764
SG-92	07-299	170	170.18	0.18	BBP		1210.50	772.78	2.765
SG-93	07-299	190.71	190.9	0.19	BBP		1421.45	910.72	2.783
SG-94	07-299	208.59	208.77	0.18	BBP		1388.05	890.32	2.789
SG-95	07-305	14.77	14.97	0.2	KP		1747.65	1134.46	2.850
SG-96	07-305	31.93	32.13	0.2	KP		1202.65	789.56	2.911
SG-97	07-305	42.63	42.83	0.2	BBP		1370.50	891.62	2.862
SG-98	07-305	59.86	60.05	0.19	KP		1441.35	931.56	2.827
SG-99	07-305	76.44	76.63	0.19	KP/BBP		1068.36	692.94	2.846
SG-100	07-	86.87	87.05	0.18	KP		1418.40	925.22	2.876

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
	305								
SG-101	07-305	100.56	100.73	0.17	KP		1471.75	952.28	2.833
SG-102	07-305	114.55	114.7	0.15	KP		1223.85	794.56	2.851
SG-103	07-305	134.07	134.22	0.15	KP		1350.80	876.94	2.851
SG-104	07-305	158.78	158.97	0.19	SLST/QV		1511.85	965.92	2.769
SG-105	07-305	174.77	174.93	0.16	KP		1455.15	943.54	2.844
SG-106	07-305	190.8	190.95	0.15	KP		1312.10	848.78	2.832
SG-107	07-305	205.85	206.02	0.17	KP		1395.40	908.18	2.864
SG-108	07-305	220.98	221.15	0.17	KP		1431.30	928.84	2.849
SG-109	07-305	237.45	237.6	0.15	BBP/KP		1358.25	887.38	2.885
SG-110	07-302	89.48	89.67	0.19	KP		1386.75	891.40	2.800
SG-111	07-302	101.35	101.53	0.18	KP		1295.65	840.32	2.846
SG-112	07-302	112.03	112.18	0.15	KP		1211.45	785.12	2.842
SG-113	07-302	124.62	124.77	0.15	KP		1006.10	648.48	2.813
SG-114	07-302	137.34	137.56	0.22	BBP		1349.25	870.06	2.816
SG-115	07-302	150.29	150.48	0.19	LS		1193.34	762.56	2.770

Sample #	Hole #	From (m)	To (m)	Length (m)	Unit	Comments	Dry Weight (g)	Bath Weight (g)	Specific Gravity (g/cc)
SG-116	07-302	163.31	163.48	0.17	KP		1145.00	740.14	2.828
SG-117	07-302	176.15	176.35	0.2	KP		1456.40	937.36	2.806
SG-118	07-302	196.6	196.83	0.23	KP		1569.25	1025.36	2.885
SG-119	07-302	204.27	204.48	0.21	KP		1287.00	838.28	2.868
SG-120	07-302	218.98	219.14	0.16	BBP		988.30	639.26	2.831
SG-121	07-302	228.31	228.54	0.23	BBP		1714.90	1111.34	2.841
SG-122	07-306	169.58	169.77	0.19	KP		1430.70	931.20	2.864
SG-123	07-306	180.98	181.17	0.19	KP/SLST	siliceous sediment with knots	1280.40	828.12	2.831
SG-124	07-306	192.25	192.45	0.2	KP		1400.65	900.24	2.799
SG-125	07-306	208.1	208.26	0.16	KP		992.76	640.38	2.817
SG-126	07-306	213.75	213.91	0.16	KP	10% quartz, sericite lenses	1305.80	846.86	2.845
SG-127	07-306	231.79	231.97	0.18	BBP	minor knots	1233.95	803.04	2.864
SG-128	07-306	239.88	240.06	0.18	BBP	minor knots	1389.45	904.94	2.868