REPORT

on the

FYRE LAKE PROPERTY

Watson Lake Mining District
Yukon Territory, Canada

Latitude: 61° 14' North   Longitude: 130° 30' West
N.T.S. 105 G/1 and 2

Mineral Claims:

KONA 43 to 46 (4 Mineral Claims)

FIRE 2 to 328 (66 Mineral Claims)

EMBER 1 to 99 (99 Mineral Claims)

(Total of 169 Yukon Quartz Mineral Claims)

- Prepared For -

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January 17, 2006

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Consulting Geologist

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SUMMARY

The Fyre Lake property is situated immediately east of Fire Lake in the Finlayson Lake area of the Watson Lake Mining District, southeastern Yukon Territory, Canada. It is comprised of 169 Yukon Quartz mineral claims, covering approximately 3,532 hectares (8,725 acres), that are currently entirely (100 percent) owned by Pacific Ridge Exploration Ltd. (formerly named 'Columbia Gold Mines Ltd.').

The mineral claim holdings are located immediately east and south of Fire Lake within the North River drainage, approximately 160 kilometres northwest of Watson Lake, 140 kilometres southeast of Ross River, or 250 kilometres east of Whitehorse. Their geographic coordinates are centred at latitude 61^{o} 14' North by longitude 130^{o} 30' West (N.T.S. 105 G/1, 2).

The property is underlain by the Devonian to Mississippian Grass Lakes succession; the oldest stratigraphy in the district belonging to the Yukon-Tanana Terrane (Murphy and Piercey, 1999b). It is comprised, in part, of a basal unit of non-carbonaceous, metasedimentary rocks and marble. This basal unit is overlain by rocks of the Fire Lake metavolcanic unit (~ 365-360 Ma) which hosts the Kona volcanogenic massive sulphide deposit and is comprised of mafic metavolcanic and metaintrusive rocks, and lesser felsic metavolcanic rocks, carbonaceous phyllite and quartzite, and marble. The Fire Lake metavolcanic unit is stratigraphically overlain by the Kudz Ze Kayah felsic metavolcanic unit (~360 Ma). This unit consists dominantly of felsic metavolcanic and carbonaceous metasedimentary rocks with lesser mafic metavolcanic and metaintrusive rocks, and felsic metaintrusive rocks.

The Kona volcanogenic massive sulphide deposit has been tested by 115 holes totalling 22,663 m or 74,354 ft. of diamond drilling. Most of this drilling was conducted along a 1,500-metre section of an apparent 2,100 strike length from near its surface exposure in Kona Creek southeasterly to the ridge between Kona and Outfitter's Creek drainage. The drill-tested mineralization occurs within an area approximately 1,500 m long by 250 m wide, trends at 130^{o}, and plunges 0^{o} to -20^{o} southeastwardly.

Two parallel zones of volcanogenic massive sulphide mineralization, East Kona and West Kona, comprise the Kona deposit; separated by an inferred reverse fault. The East Kona zone mineralization is 100 to 150 m wide, and consists of two massive to banded sulphide-bearing horizons (i.e. Upper and Lower East Kona) separated by 40 to 70 metres of chlorite schist. The Lower East Kona horizon has been divided into north and south portions separated by an apparent gap in the horizon. The northern portion is 3 to 16 metres thick and the southern portion is 2 to 11 metres thick. The Upper East Kona horizon averages thicknesses of 8 to 12 metres. These horizons consist mainly of pyrite with lesser pyrrhotite and chalcopyrite, local lenses of massive magnetite, and minor sphalerite.

The West Kona zone is inferred to be 75 to 125 m wide. The thickness of the mineralized horizon varies across this width from about 44 metres in the east to less than 1 metre at the western margin; the thickness also varies along strike. It includes mineralization that changes laterally from magnetite, pyrite and chalcopyrite in a siliceous matrix, through massive pyrite and lesser chalcopyrite, to massive pyrrhotite with minor pyrite and chalcopyrite. This mineralization occurs close to a stratigraphic contact between chlorite schist and overlying carbonaceous phyllite.

The metavolcanic rocks (i.e. chlorite schist) that host the Kona deposit have a distinct boninitic chemical signature, and the deposit has many lithological and mineralogical characteristics of Besshi-style volcanogenic massive sulphide mineralization. It is possible that the Kona volcanogenic massive sulphide mineralization was deposited in an active back-arc basin and was displaced by syn- and post-depositional faulting prior to regional tectonism and metamorphism.

The property was actively explored in 1996 and 1997 by Columbia Gold Mines Ltd. (now Pacific Ridge Exploration Ltd.); the last reported exploration activity on the subject claim holdings. This exploration work included: establishing three survey control grids, prospecting, geological mapping, silt, soil and rock geochemical sampling, geophysical surveying and diamond drilling. The results of this work discovered the buried portion of the Kona deposit and identified several coincident geochemical and geophysical anomalies along strike and parallel to the trend of the Kona deposit. These anomalies

indicate that the Kona deposit may continue for at least 600 metres, at a trend of 130^{o}, to the Outfitter's Creek drainage. These same anomalies also suggest that there may be similar volcanogenic massive sulphide mineralization northeast and southwest of the known mineralization hosted by fault-displaced stratigraphy paralleling the Kona deposit trend.

The results of a NI 43-101 compliant mineral resource study of the Kona deposit estimates an indicated mineral resource of 3.571 million tonnes grading 1.57% copper, 0.10% cobalt and 0.61 grams gold per tonne at a 1 percent copper cut-off grade. Its inferred mineral resource, at the same cut-off grade, is 5.361 million tonnes grading 1.48% copper, 0.08% cobalt and 0.53 grams gold per tonne. Zinc and silver grades were not calculated in this mineral resource estimate.

Preliminary metallurgical testwork results indicate that target recoveries of 90% for copper and 70% for gold, into a concentrate representing 7.5% of the feed and assaying 21% to 23% copper, 10 g gold/tonne and 0.08% cobalt, should be achieved from a mill feed grade of approximately 2% copper and 1.2 g gold/tonne (Melis Engineering, 1997). Subsequent scoping leach tests on cobalt-bearing pyrite concentrate indicated a 70% cobalt recovery in flotation and 95% recovery from pressure leaching with an overall cobalt recovery of possibly 65 to 70%. No cobalt minerals were identified during the microscopic examination of two samples but pyrite was identified as the main cobalt carrier. No visible gold has been observed but it occurs associated with pyrite grains as sub-microscopic and/or colloidal gold (Lakefield, 1997).

It is the writer's opinion that the Fyre Lake property has considerable merit and further exploration work is justified. Accordingly, the writer has recommended a 2006 exploration program of drill testing to evaluate additional inferred volcanogenic massive sulphide mineralization along and beside the trend of the Kona deposit, and to continue exploring for similar mineralization elsewhere within the property. The exploration work would include:

- compiling, correlating and interpreting available lithogeochemical data from previous exploration of the Kona deposit to identify any possible lithogeochemical trends that could be correlated with thicker or higher grade volcanogenic massive sulphide mineralization for later drill testing;

- 5,000 metres (~16,400 ft) of NQ2- and/or BQTK-core diamond drilling to continue assessing the Kona deposit and exploratory drilling to investigate any worthy exploration targets elsewhere on the property;

- additional metallurgical testing to possibly improve copper, cobalt and gold recoveries; and

- re-estimating the mineral resources of the Kona deposit.

The writer has estimated that the recommended exploration work will cost CAN $1,003,000.

INTRODUCTION and TERMS OF REFERENCE

The Fyre Lake volcanogenic massive sulphide copper-cobalt-gold (zinc, silver) property is situated immediately east of Fire (formerly 'Fyre') Lake in the Finlayson Lake area of the Watson Lake Mining District, southeastern Yukon Territory, Canada. It is comprised of 169 Yukon Quartz mineral claims that are entirely owned (100 percent) by Pacific Ridge Exploration Ltd. (formerly named 'Columbia Gold Mines Ltd.').

Terms of Reference

The writer was retained by Pacific Ridge Exploration Ltd. ("Pacific Ridge") in January, 2006 to prepare an independent technical report documenting past exploration work and any mineral resources, and, if justified, to propose future exploration work on the subject property. This report is intended to provide technical support for other documents to be filed with appropriate regulatory authorities.

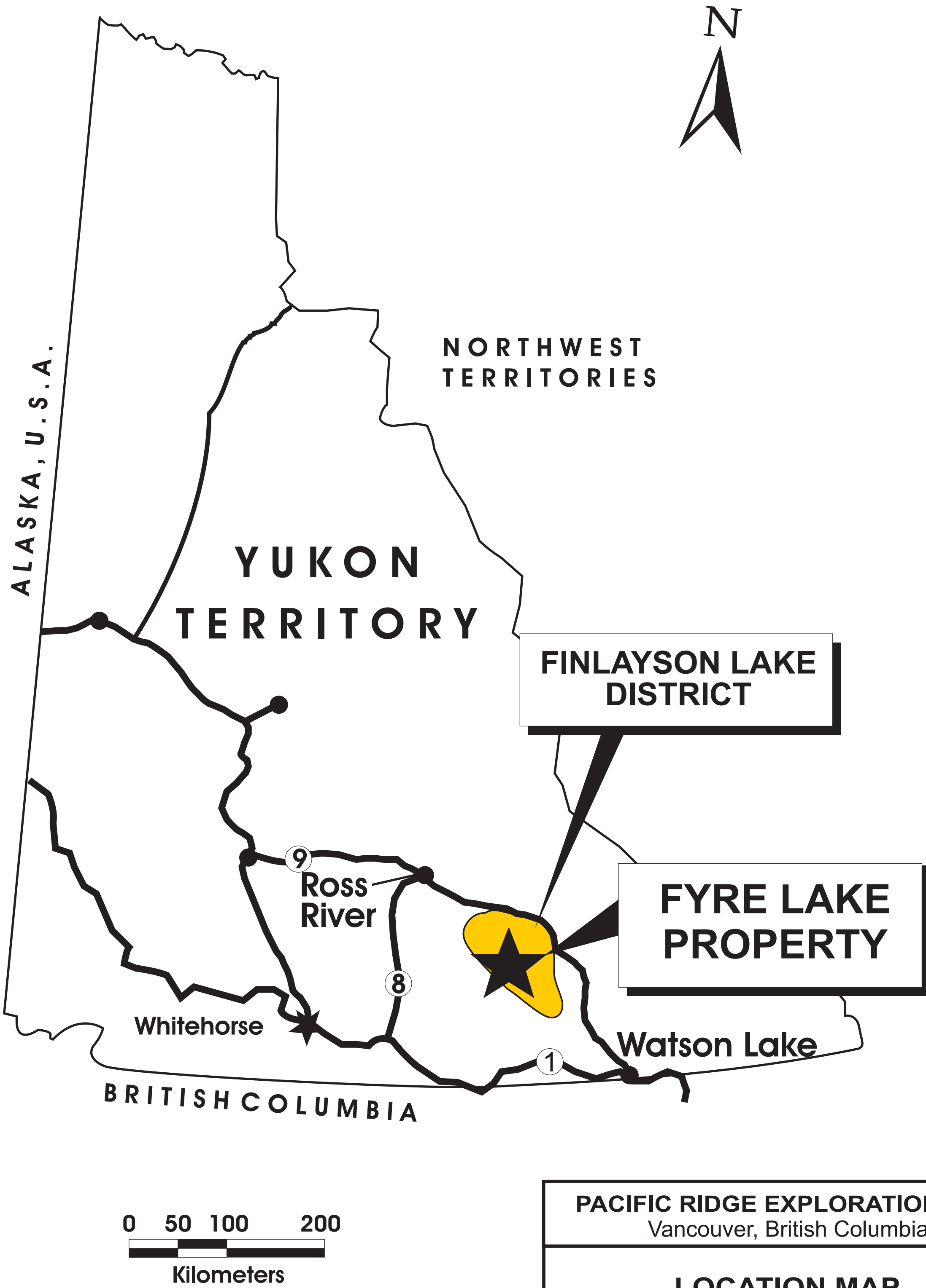
Sources of Information

This report is based upon: various reports and data published by the Geological Survey of Canada and Yukon Geology Program; unpublished reports and data made available to the writer by Pacific Ridge (2002, 2006); personal communications with Mr. W. J. Roberts (2006), an officer with Pacific Ridge; the writer's work experience on the subject property while conducting and directing field operations during the 1996 and 1997 exploration programs and 1999 reclamation work; and the writer's mineral resource estimate study undertaken in 2002.

Disclaimer

This report is for the sole use of Pacific Ridge. The writer has followed the professional procedures defined by National Instrument 43-101. This report is based upon published and unpublished information and data available to the writer since his last visit to the property in 1999; the last reported activity on the property. Much of the published information and data is derived from recent annual exploration reports and bulletins published by the Yukon Geology Program, a joint venture between the Department of Indian and Northern Affairs and the Department of Energy, Mines and Resources of the Yukon Government, summarizing the results of their geological studies in the Finlayson Lake area, Yukon.

Some of the unpublished data was authored or co-authored by the writer while he was contracted by Pacific Ridge (formerly 'Columbia Gold Mines Ltd.') Other unpublished information and data was reported by: J. Deighton and I. Foreman (1997), I. Foreman (1998), W. J. Roberts (2000, 2006), Melis Engineering Ltd. (1997), Lakefield Research Limited (1997a, b) and Kilborn Engineering Pacific Ltd. (1997). Messrs. John Deighton, P. Geo. and Ian Foreman, B.Sc. are experienced geologists and were both employed by Columbia Gold Mines Ltd. during the 1996 and 1997 exploration programs to log diamond drill core and supervise various aspects of the field work under the supervision of the writer. Mr. Wayne J. Roberts, P. Geo. is an officer and director of Pacific Ridge and managed all of the recent exploration work on the subject property. Melis Engineering Ltd. and Lakefield Research Limited were contracted in 1997 by Columbia Gold Mines Ltd. to conduct preliminary metallurgical testwork and mineralogical examinations. Kilborn Engineering Pacific Ltd. was also contracted to prepare an 'in-house' "Preliminary Economic Study" which investigated some of the mineral resource requirements for the Fyre Lake project.



To Accompany Report By J. D. Blanchflower, P. Geo., 2006

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

LOCATION MAP

Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 1

PROPERTY DESCRIPTION and LOCATION

Property Description and Location

The Fyre Lake property is comprised of 169 unsurveyed Yukon Quartz mineral claims, encompassing approximately 3,532 hectares (8,725 acres). The claims are located immediately east and south of Fire Lake within the North River drainage, approximately 160 kilometres northwest of Watson Lake, 140 kilometres southeast of Ross River, or 250 kilometres east of Whitehorse, in the Finlayson Lake area of southeastern Yukon Territory, Canada. Their geographic coordinates are centred at latitude 61^{o} 14' North by longitude 130^{o} 30' West (N.T.S. 105 G/1,2) in the Watson Lake Mining District. See Figure 1 accompanying this report for the location of the property.

In 1998 the Fyre Lake property was comprised of 415 contiguous Yukon Quartz mineral claims but over the intervening years peripheral and non-strategic mineral claims have been allowed to lapse by Pacific Ridge resulting in four non-contiguous groups of mineral claims covering the Kona volcanogenic massive sulphide deposit and its immediate inferred trend plus the existing campsite location. See Figures 2A and 2B of this report for the location and configuration of the mineral claims comprising the subject property. Table I of this report summarizes the pertinent Yukon Quartz mineral claim data.

Property Ownership and Proposed Option Agreement Terms

According to P. McLeod (2006), the mining recorder for the Watson Lake Mining District, the registered owners of the subject 169 Yukon Quartz mineral claims are either Columbia Gold Mines Ltd. or Pacific Ridge Exploration Ltd. Subsequent to the title search, the writer contacted Mr. W. Roberts, a director of Pacific Ridge, and received a copy of the 'Certificate of Change of Name' documenting the renaming of Columbia Gold Mines Ltd. to Pacific Ridge Exploration Ltd. on May 27, 1999. As of January 16, 2006 the registered owner on most of the subject claim documents had not been changed to reflect the corporation name change.

The property was subject to an underlying agreement between Pacific Ridge and Welcome Opportunities Ltd. ("Welcome Opportunities"), the original owner of the 'KONA' and 'FIRE' mineral claims. The four 'KONA 43' to 'KONA 46' mineral claims were staked in 1980 by Welcome North Mines Ltd., and the 'FIRE 2' to 'FIRE 133' mineral claims were staked in 1990 and 1991 by Placer Dome Exploration Limited on behalf of Welcome North Mines Ltd. These claims were subsequently transferred to Welcome Opportunities Ltd. When Columbia Gold Mines Ltd. acquired the property in 1995 the Columbia Gold Mines Ltd. – Welcome Opportunities Ltd. agreement allowed Columbia Gold Mines Ltd. to earn up to an eighty percent (80%) interest in the property by expending CDN $6.0 million on exploration work, including a pre-feasibility study, by December 31, 1999. Welcome Opportunities Ltd. could then elect to complete final feasibility, provide production financing and place the property into production to increase its interest from twenty percent (20%) to fifty-five percent (55%).

Based upon documents received from P. McLeod (2006), the 'EMBER 1 to 99' mineral claims are valid until December 3, 2006 and some of the 'FIRE' mineral claims expire on December 31, 2006. The mineral claims covering the existing campsite and known mineralization (i.e. FIRE 194 and 195 and KONA 43 to 46 respectively) expire in 2006 (see Table I of this report). Thus, Pacific Ridge must conduct assessable exploration work and file appropriate documents with the Watson Lake Mining Recorder prior to July 8, 2006 to maintain the current claim holdings. Annual assessment requirements are $100 per claim.

Reclamation and Permitting

In July 1999 Pacific Ridge, under the field supervision of the writer, removed all drilling and camp equipment, fuel, fuel barrels and other exploration equipment and supplies from the property. The existing 36-person camp on the eastern shore of Fire Lake was left standing and the logs used for drilling platform cribbing were piled for future burning near the site of the 1997 drilling work. Since that time a local guide/outfitter with hunting guide rights in the area has been intermittently utilizing the camp site to house guests (Roberts, 2006). The 1999 reclamation work by Pacific Ridge fulfilled then existing reclamation requirements with the Yukon Government provided the claims on which the campsite was constructed were maintained in good standing (Roberts, 2006). Since that time the Yukon Territory has adopted new exploration and reclamation guidelines, and governmental permitting will be required for any future exploration work.

Pacific Ridge should contact the appropriate Yukon Government agencies once future exploration plans are formalized to discuss possible reclamation, permitting and bonding requirements in accordance with their proposed program.

ACCESSIBILITY, INFRASTRUCTURE, PHYSIOGRAPHY, CLIMATE, VEGETATION, and LOCAL RESOURCES

Accessibility and Infrastructure

The property is readily accessible year-round with fixed wing aircraft, or helicopter support. Fire Lake is 8 kilometres long with excellent approaches that could be utilized by a variety of float- or ski-mounted fixed wing aircraft throughout the year. Charter fixed wing aircraft are available in Watson Lake, Whitehorse, and seasonally from Finlayson Lake near the Robert Campbell Highway. A variety light- and medium-lift helicopters are available for charter from several companies with permanent bases in Ross River, Watson Lake and/or Whitehorse (see Figure 1). The nearest road access is approximately 30 kilometres northeast at the 'Kudz Ze Kayah' property which is owned by Teck Cominco Limited. This seasonal gravel access road joins the Robert Campbell Highway at Finlayson Lake but has very restricted use and only with permission from Teck Cominco Limited and First Nations governing bodies.

Physiography, Climate and Vegetation

The property is situated regionally within the Yukon Plateau physiographic region in the Simpson Range of the eastern Pelly Mountains, approximately five kilometres northeast of the Tintina Trench. The Fire Lake area has linear open valleys and high rolling to craggy ridges and mountains. Topographic relief is moderate to locally high with elevations ranging from 1,100 m (3,609 ft) at Fire Lake to 1,900 m (6,234 ft) A.M.S.L. along the eastern ridge crests. A 2,351-metre high peak situated six kilometres north of the property is the highest mountain in the area. The various mineral showings are situated between elevations of 1,450 and 1,700 m A.M.S.L.

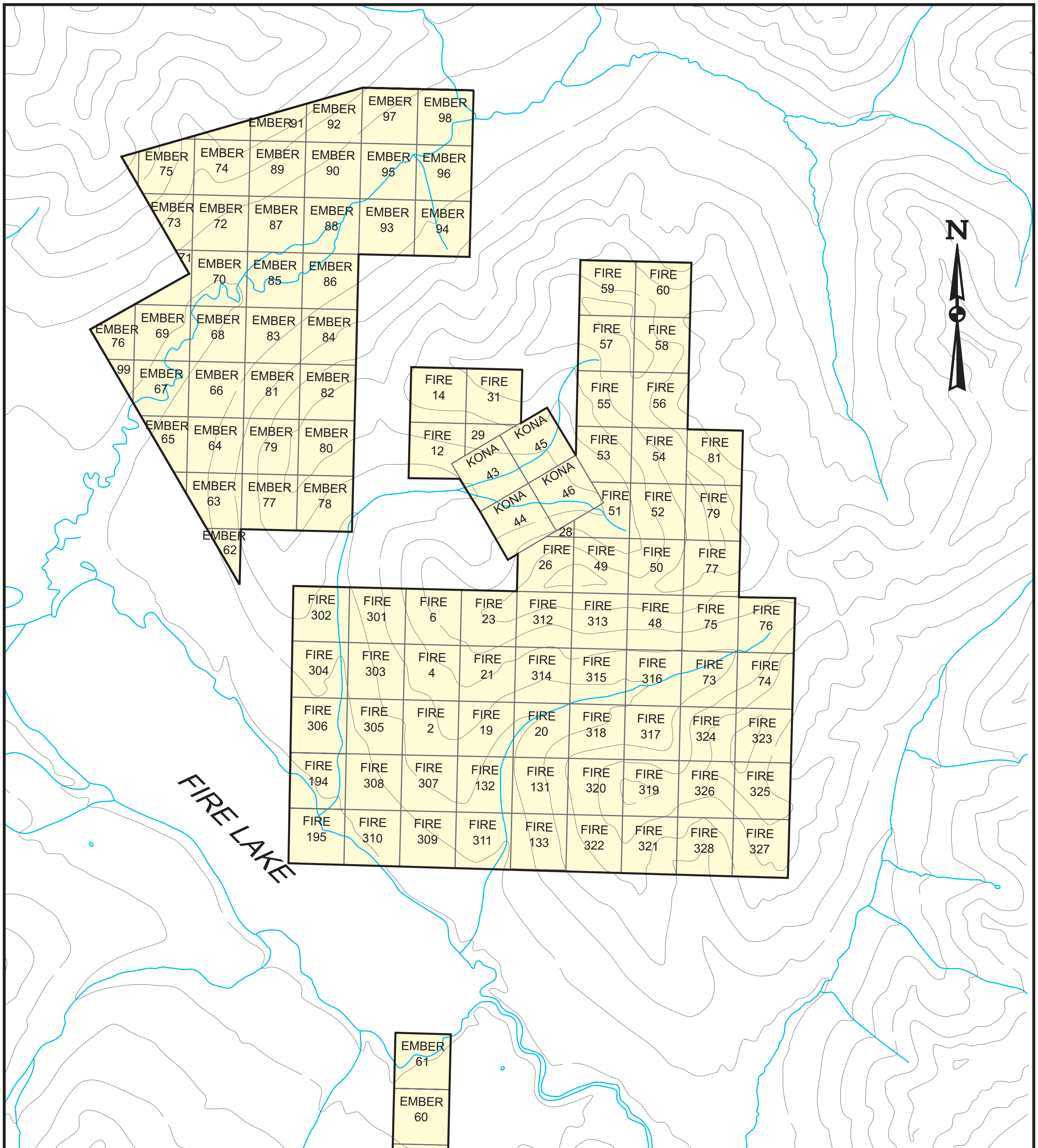
Fire Lake is situated midway along the southeasterly-flowing North River. To the northeast, there are two easterly trending hanging valleys and broad open cirques. The drainage from the northern hanging valley, within which most of the known mineral showings are situated is called 'Kona Creek' and the central valley is drained by 'Outfitters Creek'. The southern drainage which joins North River south of Fire Lake is not named (see Figures 2A and 2B).

The annual mean daily temperature for the eastern Pelly mountains is -5^{o} C ranging from approximately -40^{o} C during the winter months to 25^{o} C during the summer months. Snow cover is minimal, averaging about 60 centimetres by late winter. Permafrost is discontinuous but widespread. Bedrock exposures are generally absent in areas of low to even moderate relief; often limited to stream canyons, ridges and cliffs due to an extensive glacial till cover.

Near Fire Lake a dense spruce forest extends to treeline at an elevation of 1,500 m. To the north, the dense vegetation becomes more open with buckbrush (dwarf birch) and eventually disappears to a caribou moss cover. Kona and Outfitters Creek drainages have sufficient flows of water for diamond drilling purposes until mid-October or later.

Local Resources

The current claim holdings are sufficiently extensive to cover the drill-tested portion of the Kona deposit and its immediate inferred trend but insufficient to cover all of the addition lands required for a viable mining and milling operation should future exploration prove successful. Pacific Ridge should plan to acquire such lands during the early stages of any future exploration work.



(After Claim Data and Maps 105G/1,2,7, 8;
Watson Lake Mining Recorder, 2002)

0 500 1000 1500 2000
metres

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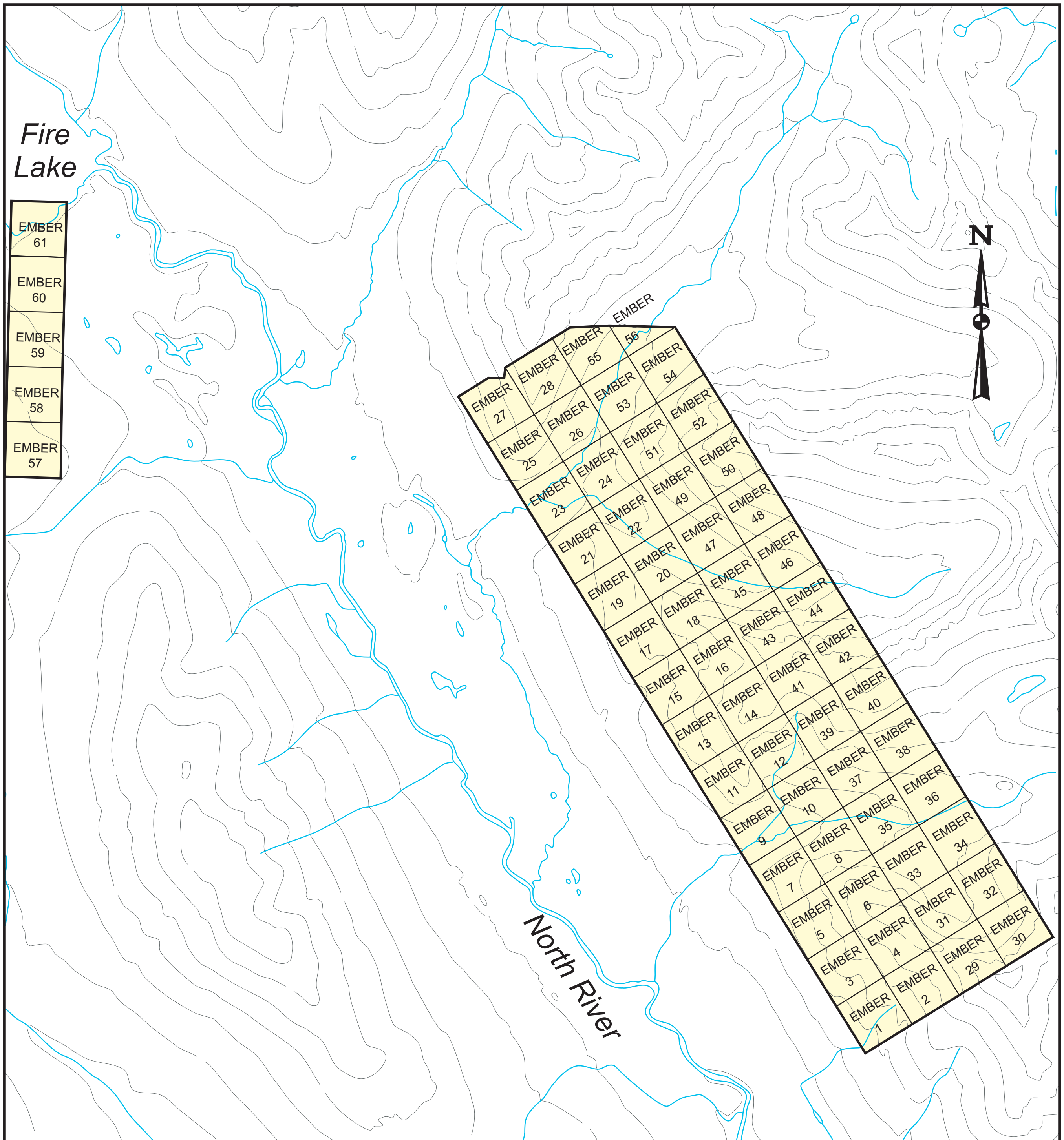
PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

MINERAL CLAIM MAP
North Map-Sheet
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 2A



(After Claim Data and Maps 105G/1,2,7, 8;
Watson Lake Mining Recorder, 2006)

0 500 1000 1500 2000
metres

To Accompany Report By J. D. Blanchflower, P. Geo., 2006

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

MINERAL CLAIM MAP
South Map-Sheet
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 2B

TABLE I

MINERAL CLAIM DATA

Claim Name	NTS	Record Number		Record Date	Expiry Date	Registered Owner
FYRE LAKE PROPERTY (original joint venture claims)						
KONA 43	105G/2	YA	56602	9-Sep-90	9-Sep-06	Columbia Gold Mines Ltd.
KONA 44	105G/2	YA	56603	9-Sep-90	9-Sep-06	Columbia Gold Mines Ltd.
KONA 45	105G/2	YA	56604	9-Sep-90	9-Sep-06	Columbia Gold Mines Ltd.
KONA 46	105G/2	YA	56605	9-Sep-90	9-Sep-06	Columbia Gold Mines Ltd.
FIRE 2	105G/2	YB	33749	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 4	105G/2	YB	33751	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 6	105G/2	YB	33753	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 12	105G/2	YB	33759	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 14	105G/2	YB	33761	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 19	105G/2	YB	33766	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 20	105G/2	YB	33767	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 21	105G/2	YB	33768	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 23	105G/2	YB	33770	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 26	105G/2	YB	33773	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 28	105G/2	YB	33775	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 29	105G/2	YB	33776	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 31	105G/2	YB	33778	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 48	105 G/1	YB	33795	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 49	105 G/1	YB	33796	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 50	105 G/1	YB	33797	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 51	105 G/1	YB	33798	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 52	105 G/1	YB	33799	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 53	105 G/1	YB	33800	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 54	105 G/1	YB	33801	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 55	105 G/1	YB	33802	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 56	105 G/1	YB	33803	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 57	105 G/1	YB	33804	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 58	105 G/1	YB	33805	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 59	105 G/1	YB	33806	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 60	105 G/1	YB	33807	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 73	105 G/1	YB	33820	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 74	105 G/1	YB	33821	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 75	105 G/1	YB	33822	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 76	105 G/1	YB	33823	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.

TABLE I

MINERAL CLAIM DATA

Claim Name	NTS	Record Number		Record Date	Expiry Date	Registered Owner
FIRE 77	105 G/1	YB	33824	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 79	105 G/1	YB	33826	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 81	105 G/1	YB	33828	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 131	105G/2	YB	33878	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 132	105G/2	YB	33879	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FIRE 133	105G/2	YB	33880	31-Dec-90	31-Dec-06	Columbia Gold Mines Ltd.
FYRE LAKE PROPERTY (Pacific Ridge 100% ownership)						
FIRE 194	105 G/2	YB	85203	8-Jul-96	8-Jul-06	Columbia Gold Mines Ltd.
FIRE 195	105 G/2	YB	86834	14-Aug-96	14-Aug-06	Columbia Gold Mines Ltd.
FIRE 301	105 G/2	YB	94275	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 302	105 G/2	YB	94276	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 303	105 G/2	YB	94277	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 304	105 G/2	YB	94278	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 305	105 G/2	YB	94279	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 306	105 G/2	YB	94280	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 307	105 G/2	YC	22651	9-Dec-02	9-Dec-06	Pacific Ridge Exploration Ltd.
FIRE 308	105 G/2	YC	22652	9-Dec-02	9-Dec-06	Pacific Ridge Exploration Ltd.
FIRE 309	105 G/2	YC	22653	9-Dec-02	9-Dec-06	Pacific Ridge Exploration Ltd.
FIRE 310	105 G/2	YC	22654	9-Dec-02	9-Dec-06	Pacific Ridge Exploration Ltd.
FIRE 311	105 G/2	YC	22655	9-Dec-02	9-Dec-06	Pacific Ridge Exploration Ltd.
FIRE 312	105 G/2	YB	94281	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 313	105 G/2	YB	94282	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 314	105 G/2	YB	94283	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 315	105 G/2	YB	94284	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 316	105 G/2	YB	94285	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 317	105 G/2	YB	94286	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 318	105 G/2	YB	94287	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 319	105 G/2	YB	94288	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 320	105 G/2	YB	94289	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 321	105 G/2	YB	94290	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 322	105 G/2	YB	94291	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 323	105 G/2	YB	94292	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 324	105 G/2	YB	94293	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 325	105 G/2	YB	94294	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.

TABLE I

MINERAL CLAIM DATA

Claim Name	NTS	Record Number		Record Date	Expiry Date	Registered Owner
FIRE 326	105 G/2	YB	94295	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 327	105 G/2	YB	94296	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
FIRE 328	105 G/2	YB	94297	12-Nov-02	12-Nov-06	Pacific Ridge Exploration Ltd.
EMBER 1	105 G/1	YB	88808	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 2	105 G/1	YB	88809	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 3	105 G/1	YB	88810	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 4	105 G/1	YB	88811	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 5	105 G/1	YB	88812	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 6	105 G/1	YB	88813	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 7	105 G/1	YB	88814	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 8	105 G/1	YB	88815	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 9	105 G/1	YB	88816	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 10	105 G/1	YB	88817	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 11	105 G/1	YB	88818	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 12	105 G/1	YB	88819	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 13	105 G/1	YB	88820	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 14	105 G/1	YB	88821	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 15	105 G/1	YB	88822	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 16	105 G/1	YB	88823	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 17	105 G/1	YB	88824	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 18	105 G/1	YB	88825	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 19	105 G/1	YB	88826	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 20	105 G/1	YB	88827	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 21	105 G/1	YB	88828	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 22	105 G/1	YB	88829	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 23	105 G/1	YB	88830	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 24	105 G/1	YB	88831	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 25	105 G/1	YB	88832	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 26	105 G/1	YB	88833	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 27	105 G/1	YB	88834	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 28	105 G/1	YB	88835	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 29	105 G/1	YB	88836	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 30	105 G/1	YB	88837	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 31	105 G/1	YB	88838	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 32	105 G/1	YB	88839	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.

TABLE I

MINERAL CLAIM DATA

Claim Name	NTS	Record Number		Record Date	Expiry Date	Registered Owner
EMBER 33	105 G/1	YB	88840	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 34	105 G/1	YB	88841	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 35	105 G/1	YB	88842	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 36	105 G/1	YB	88843	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 37	105 G/1	YB	88844	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 38	105 G/1	YB	88845	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 39	105 G/1	YB	88846	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 40	105 G/1	YB	88847	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 41	105 G/1	YB	88848	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 42	105 G/1	YB	88849	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 43	105 G/1	YB	88850	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 45	105 G/1	YB	88852	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 46	105 G/1	YB	88853	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 47	105 G/1	YB	88854	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 48	105 G/1	YB	88855	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 49	105 G/1	YB	88856	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 50	105 G/1	YB	88857	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 51	105 G/1	YB	88858	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 52	105 G/1	YB	88859	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 53	105 G/1	YB	88860	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 54	105 G/1	YB	88861	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 55	105 G/1	YB	88862	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 56	105 G/1	YB	88863	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 57	105 G/2	YB	88864	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 58	105 G/2	YB	88865	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 59	105 G/2	YB	88866	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 60	105 G/2	YB	88867	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 61	105 G/2	YB	88868	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 62	105 G/2	YB	88869	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 63	105 G/2	YB	88870	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 64	105 G/2	YB	88871	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 65	105 G/2	YB	88872	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 66	105 G/2	YB	88873	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 67	105 G/2	YB	88874	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 68	105 G/2	YB	88875	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 69	105 G/2	YB	88876	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.

TABLE I

MINERAL CLAIM DATA

Claim Name	NTS	Record Number		Record Date	Expiry Date	Registered Owner
EMBER 70	105 G/2	YB	88877	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 71	105 G/2	YB	88878	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 72	105 G/2	YB	88879	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 73	105 G/2	YB	88880	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 74	105 G/2	YB	88881	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 75	105 G/2	YB	88882	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 76	105 G/2	YB	88883	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 77	105 G/2	YB	88884	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 78	105 G/2	YB	88885	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 79	105 G/2	YB	88886	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 80	105 G/2	YB	88887	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 81	105 G/2	YB	88888	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 82	105 G/2	YB	88889	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 83	105 G/2	YB	88890	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 84	105 G/2	YB	88891	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 85	105 G/2	YB	88892	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 86	105 G/2	YB	88893	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 87	105 G/2	YB	88894	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 88	105 G/2	YB	88895	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 89	105 G/2	YB	88896	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 90	105 G/2	YB	88897	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 91	105 G/2	YB	88898	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 92	105 G/2	YB	88899	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 93	105 G/2	YB	88900	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 94	105 G/2	YB	88901	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 95	105 G/2	YB	88902	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 96	105 G/2	YB	88903	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 97	105 G/2	YB	88904	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 98	105 G/2	YB	88905	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.
EMBER 99	105 G/2	YB	88906	3-Dec-96	3-Dec-06	Columbia Gold Mines Ltd.

Total Number of Mineral Claims (original J.V.)	**40**
Total Number of Mineral Claims (Pacific Ridge 100%)	**129**
TOTAL	**169**

History

The Pelly Mountains were originally mapped by the Geological Survey of Canada in 1958 and 1959 and the results of this work were published as the ‘Finlayson Lake’ map-sheet in 1960 (Wheeler, Green and Roddick, 1960). In September, 1960 prospectors employed by Cassiar Asbestos Corporation discovered a 2.5 by 2.0-metre massive sulphide float boulder on a glacial esker near a guide/outfitter’s cabin at the south end of Fire Lake. Shortly after, prospectors discovered massive pyrite mineralization exposed in Kona Creek; they called this showing the ‘E’ zone. During the fall and winter of 1960 Cassiar Asbestos staked the ‘TOP’ mineral claims covering the southwesterly facing slopes of Fire Lake. In 1961 Cassiar Asbestos explored their claim holdings with prospecting, geological mapping, geophysical surveys (electromagnetics and magnetics), trenching and drilling. The drilling comprised twenty-three shallow packsack drill holes, totalling 224 metres, and twelve AX-core diamond drill holes, totalling 582 metres. Most of their efforts were concentrated on assessing the ‘E’ and ‘K’ mineral showings where they reportedly encountered mineralization with an average grade of 1.0% copper, 0.95% zinc, 4.80 gpT (0.14 opt) silver and 0.72 gpT (0.021 opt) gold (Crawford, 1981).

In December, 1965 Atlas Copper Ltd. (later ‘Atlas Explorations Ltd.’) optioned the ‘DUB’ mineral claims and in June, 1966 additional claims were staked to cover the Fyre Lake mineral showings. An airborne electromagnetic and magnetometer survey was conducted over the Kona Creek cirque and along the eastern slopes of Fire Lake and North River. This survey identified two target areas, called ‘DUB I’ and ‘DUB II’, for ground surveying. A cut survey control grid, totalling 31.5 line-kilometres, was established over the northern ‘DUB II’ area, and electromagnetics (horizontal loop electromagnetic - HLEM) and magnetometer surveying with concurrent soil geochemical sampling for copper, lead and zinc were undertaken. Six diamond drill holes (66-01 to -05A), totalling 593.44 metres, tested and extended the copper-bearing pyritic formation at ‘I’ and ‘K’ mineral showings which had been identified by earlier Cassiar Asbestos drilling. Intercepts of massive sulphide mineralization up to 12.2 metres thick are reported from this drilling (Sadlier-Brown, 1966). Selected drill core from three of the drill holes is stored at the Hugh Bostock Core Library in Whitehorse, Y.T. (Stroshein, 1991).

In 1967 Atlas Explorations explored the southern DUB I target area near the original massive sulphide float boulder discovery site. A 15.5 line-kilometre survey control grid was established, and the grid area was explored with horizontal loop electromagnetics (HLEM) and magnetometer surveying, soil geochemical sampling for copper lead and zinc, and diamond drilling (3 AX-core holes totalling 252.68 metres). The diamond drilling intersected disseminated pyrite and pyrrhotite mineralization but no significant base or precious metal mineralization (Sadlier-Brown, 1967).

Between 1974 and 1977 D. J. Tempelman-Kluit of the Geological Survey of Canada remapped and revised the regional geology of the Pelly Mountains (GSC Open File 486). As a consequence of this work, Amax Potash Limited restaked the Fyre Lake massive sulphide showings in 1976 and conducted a limited geological mapping and rock geochemical sampling program. Due to a lack of funding for the project the claims were allowed to lapse in 1977 (Crawford, 1981).

In late August and early September, 1980 Welcome North Mines Ltd. staked sixty-eight ‘KONA’ mineral claims, covering the Fyre Lake massive sulphide showings, after they discovered disseminated copper mineralization in metamorphosed volcanic rocks approximately 2 kilometres north of the known mineral showings. This work was carried out as part of the Basin Joint Venture with Esperanza Explorations Ltd. (a predecessor corporation to Columbia Gold Mines Ltd.) Unfortunately an early snowfall and other work priorities prevented further exploration that year. The Basin Joint Venture was terminated in 1981.

In 1981 Welcome North extended the soil sampling coverage from the Kona cirque area (16.9 line-km survey grid, 255 soil samples for copper, lead and zinc) and completed a geological study of the mineral showings without additional geological mapping. According to Crawford (1981), the results of the Welcome North exploration work identified mineralization in intermittent outcrops for 2.5 kilometres in a northwesterly direction. The mineralization is dominantly hosted by a cupriferous iron formation facies and varies in character from laminated massive pyrite (± chalcopyrite, sphalerite, quartz), through banded cupriferous iron formation (quartz, magnetite, chlorite, chalcopyrite, and/or sphalerite), to disseminated chalcopyrite and pyrite in greenschists. The mineralization occurs within the Nisutlin Allochthon in a dark green chlorite schist unit approximately 100 metres beneath its contact with an overlying quartz-sericite schist unit.

In 1981 J. Morin of the Geology Section, Department of Indian and Northern Affairs reported on a 1979-80 geological and geochemical study of the mineralization at Fire Lake and several other mineral showings in the district. According to Morin (1981), the Fyre Lake mineralization is characteristic of a ‘Besshi-type’ volcanogenic exhalative model.

Placer Dome Inc. optioned the KONA mineral claims from Welcome North Mines Ltd. on November 30, 1990 and, within the provisions of their agreement, staked the ‘FIRE 1’ to ‘FIRE 184’ mineral claims in December, 1990. Prior to signing their agreement, Placer Dome had contracted Dighem Surveys and Processing Inc. to carry out a helicopter-supported airborne survey of a 36-square kilometre area (308 flight line-kilometres) centred on the Fyre Lake mineral showings within the Kona Creek drainage. Dighem prepared a report with electromagnetics, total field magnetics, calculated vertical gradient, resistivity and Very Low Frequency electromagnetics interpretations and maps (Smith, 1990).

In 1991 Placer Dome conducted a surface exploration program, including: geological mapping, geophysical surveying (Apex MaxMin EM) and soil, silt and rock geochemical sampling, based upon the 1990 airborne geophysical survey results. Two survey control grids were established for the 1991 exploration work. Using the old Atlas Exploration baseline location and orientation (340^o), Placer Dome contractors extended the baseline south-southeasterly from 110000 to 8200 North and cut east-northeasterly and west-southwesterly gridlines (070^o-250^o) covering the southwesterly-facing slopes between Outfitters Creek and the south end of Fire Lake, totalling 31.5 line-kilometres. For the northern survey control grid, Placer Dome contractors re-established portions of the old Atlas Explorations grid and cut five north-northwesterly gridlines over the ‘M’ zone showing, totalling 3.5 line-kilometres. Apex MaxMin EM-II surveying with a 100-metre coil separation and a ground magnetometer survey were conducted over both grid areas. A total of 1,750 B- and C-horizon soil samples and 7 stream sediment samples were collected during the program, and these samples were analyzed for their gold content and a 27-element suite of base metal, pathfinder, trace and mineral forming elements. In addition, 112 rock geochemical samples were collected and analyzed for ten elements including precious and base metals.

The results of the 1991 exploration program by Placer Dome determined that the Fyre Lake mineral showings are volcanogenic massive sulphide deposits hosted by metamorphosed and highly deformed Late Devonian volcanic rocks. Furthermore, the iron formations hosting the mineralization can be traced by airborne and ground geophysics and soil geochemistry for 1 kilometre southeast of the Kona Creek cirque. Placer Dome terminated the property option agreement in 1992, thereby relinquishing all property rights to Welcome North Mines Ltd.

In November 1995 Columbia Gold Mines Ltd. negotiated an agreement with Welcome Opportunities Ltd. (formerly Welcome North Mines Ltd.) to acquire the ‘KONA’ and ‘FIRE’ mineral claims, and in 1996 Columbia Gold Mines Ltd. staked additional mineral claims west and south of the joint venture claim holdings.

According to an ‘Annual Information Form’ filed by Columbia Gold Mines Ltd. (1998, p. 14-19),

> “The 1996 and 1997 exploration programs evaluated three zones of known and inferred iron formation-related mineralization along a 13-kilometre strike length, and diamond drilling discovered volcanogenic copper-cobalt-gold mineralization within one of the zones, known as the Kona deposit.
>
> A comprehensive helicopter-supported two-stage exploration program to evaluate the exploration potential of the Fyre Lake Property was financed and operated by Company during the 1996 field season (i.e. from June 15 to October 9). Total project expenditures for 1996 were \$3,018,907.

This exploration work included:

a) staking 121 Yukon Quartz claims (i.e. ‘FIRE 193’ to FIRE 214’ and ‘EMBER 1’ to ‘EMBER 99’);

b) preparation of topographic plans at scales of 1:1000 and 1:2500;

c) land surveying using Global Positioning System (GPS) and electrodistamat instrumentation to

d) establish topographic control within the property for subsequent linecutting and diamond drilling;

e) establishing three metric survey control grids (totalling 150.80 line-km) over the ‘KONA’, ‘LAKE’ and ‘DUB’ exploration target areas;

f) construction of 36-person field camp and core logging facilities;

g) prospecting within and adjacent to the three survey control grids;

h) detailed geological mapping at a scale of 1:1000 within the Kona and Dub survey control grids;

i) collecting and analyzing 2,256 soil geochemical samples for 31-elements by I.C.P. methods;

j) collecting and analyzing 17 rock geochemical samples (8 samples for gold (F.A./A.A.) plus 31-element I.C.P. analysis and 9 samples for copper, cobalt, zinc, silver and gold assays) ;

k) conducting 142.70 line-kilometres of combined ground magnetics and horizontal loop (MaxMin II) electromagnetics geophysical surveying;

l) completing 72 NQ2- and/or BQTK-core diamond drill holes (totalling 9667.93 m or 31,716 ft);

m) assaying and/or analyses of 1,210 drill core samples, check-assaying of 67 drill core samples, analyses of 1,293 drill core samples for various lithogeochemical studies, analyses of 19 magnetite-rich drill core samples for platinum group elements, whole-rock analyses of 20 drill core samples;

n) preparing petrographic study of 8 thin sections from various host-rock lithologies; and

o) subsequent collation, compilation and documentation of the program results.

In summary, the 1996 soil geochemical sampling results from the Kona grid area delineated a series of copper- and cobalt-in-soil anomalies with elevated zinc-in-soil values along and immediately west (i.e. downslope) of the trend of ground magnetic and Horizontal Loop electromagnetic (MaxMin II) geophysical anomalies from the volcanogenic massive sulphide showings at the headwaters of Kona Creek north-northwesterly to the northern ridge of the cirque. Soil geochemical values appear to decrease northward from the north fork of Kona Creek; possibly reflecting alpine glacial transport, more chert-hosted mineralization, and/or deeper overburden. Other coincident copper-, cobalt-, and/or zinc-in-soil geochemical anomalies occur east and west of the main geochemical-geophysical trend indicating other possible mineralized horizons.

The 1996 ground magnetic and HLEM geophysical surveys within the Kona grid delineated an extensive 1,800-metre long, northwest-southeast trending structure the southeastern portion of which is interpreted to be the subsurface strike extensions of the central Kona zone mineralization. The geophysical signature of the mineralization consists of a magnetic response with the electromagnetic conductor being proximal to the magnetic highs. In the northwestern portion of the surveyed area the electromagnetic conductor is coincident with magnetic highs scattered along the same trend from grid coordinates 112900 N by 14700 E to 112250 N by 15150 E. South of grid line 112250 N, the magnetic response is more intense, indicative of a near-surface source along the axis of the structure. From 111700 N by 15500 E to 111400 N by 15650 E, in the vicinity of the known mineralization, the HLEM conductor shows a strong response coinciding with a sharp increase in the magnetic signature from about 6500 nT on the northeast side of the conductor to over 9000 nT southwest of the conductor.

Detailed diamond drilling undertaken during the 1996 exploration program discovered three horizons of massive to semi-massive sulphide and magnetite mineralization hosting significant copper-cobalt-gold (± zinc, silver values) over a combined thickness of 70 to 80 metres, a continuous strike length of more than 800 metres and widths in excess of 100 metres. This mineralization, called the ‘Kona’ deposit, is open for expansion along both strike and dip directions. The mineralization is dominantly hosted within the upper section of a chlorite-actinolite-quartz schist (i.e. metavolcanic) unit, called the ‘Lower’ horizon, and immediately beneath the chlorite-actinolite-quartz schist and phyllite (i.e. metasedimentary) stratigraphic contact, called the ‘Upper’ horizon. The mineralization remained open for expansion and a 1997 program of additional drilling was recommended.

The 1997 exploration program was designed to expand previously defined mineralization comprising the open-ended Kona deposit and to explore for additional volcanogenic massive sulphide mineralization related to several coincident soil geochemical and geophysical anomalies outlined in the Kona and Lake grid areas during the 1996 field program. Field work was conducted during the period from April 4 to October 9. Total project expenditures for 1997 were $3,648,213. The work included the following:

a) 44 NQ2 and/or BQTK core diamond drill holes totalling 13,598.98 m or 44,616 feet;

b) cut-line grid extensions to the west, east and south of the Kona grid, totalling 53.85 kilometres of cut-line;

c) 58.6 line-kilometres of ground magnetics and 59.275 line-kilometres of horizontal loop (Max-Min II) electromagnetic geophysical surveying;

d) a Global Positioning System (GPS) survey of the Kona, Lake and Dub grids in order to establish a tight control for plotting data collected in 1996 and 1997 and to aid in the control for geological mapping as well as establishing the positions of claim posts;

e) detailed geological mapping at a scale of 1:2,000 within the Kona cirque and 1:5,000 scale for the area surrounding the Kona cirque;

f) a large loop time domain electromagnetic (UTEM-3) test survey of 4 drill holes and 7 surface grid lines;

g) (collecting) 1,009 soil geochemical samples and analyzing 757 samples for 31 elements by ICP methods;

h) (collecting) 181 silt geochemical samples and analyzing for 31 elements by ICP methods;

i) (collecting) 39 rock geochemical samples (and analyzing) 21 samples for copper, cobalt, zinc, gold (F.A./A.A.) plus 31 element ICP analysis, 16 samples for copper, cobalt, zinc, gold assays plus 31 element ICP analysis and 2 samples for lithogeochemical studies;

j) assaying and analysis of 767 drill core samples, check assaying 53 drill core samples, analysis of 969 drill core samples for lithogeochemical studies;

k) initiating a water and baseline environmental study of the claims and surrounding area;

l) surveying of collar locations of all diamond drill holes;

m) prospecting in and immediately around the claim area;

n) staking 4 Yukon Quartz claims (FIRE 55F, 56F, 215F and 216F);

o) constructing an additional four sleeping accommodation tents, (and) doubling the size of the core logging facilities;

p) preparing a photo mosaic and various scale topographic maps of the claim and surrounding area;

q) a preliminary scoping study of the Kona deposit by Kilborn Engineering Pacific Ltd.;

r) a resource study of the Kona deposit prepared by J. D. Blanchflower of Minorex Consulting Ltd. and the Company;

s) preliminary metallurgical testwork of the Kona mineralization; and

t) compiling and interpreting all of the combined 1997 and 1996 data and the subsequent documentation of the results.

In summary, during 1997 the Columbia Gold Mines carried out a helicopter-supported diamond drilling program to evaluate several coincident geophysical and soil geochemical anomalies within the Lake zone and expand mineralization in the Kona zone. Eight widely-spaced (greater than 100 metres apart) NQ2-core diamond drill holes (i.e. 97-72 to 97-79) tested several coincident HLEM, ground magnetometer and soil geochemical anomalies within the Lake zone, east of the southern end of Fyre Lake. A majority of the drill holes identified favourable stratigraphy but did not intersect significant mineralization.

Thirty-six holes (hole numbers 97-88 to 97-115) were completed on the Kona grid and expanded the limits of the Kona deposit to a drill-inferred length of 1,500 metres over an average width of 250 metres. The deposit remains open to the southeast and northwest. As of December 31, 1997, the Company had drilled 115 holes totalling 23,200 metres (76,000 feet) most of which tested 2.0 kilometres of a 3.5 kilometres long geophysical-geochemical target. Exploration has outlined the Kona deposit which remains open for expansion as well as several additional large targets worthy of drill testing.

Two step-out holes, 97-114 and 97-115, were collared 450 metres to the southeast of the Kona cirque and were successful in extending the dimensions of the deposit. Drill hole 97-114 intersected 2.56 metres of banded semi-massive sulphides. This mineralization has been interpreted as belonging to the Upper Horizon of East Kona as it occurs immediately below the contact of the metavolcanics and metasediments. Drill hole 97-115 intersected 16.3 metres of siliceous hosted mineralization belonging to West Kona. The step-out holes were successful in confirming strike extension and thus have provided a large exploration target southeast of the area drilled to date.

A second large magnetic anomaly measuring 3.0 kilometres long by 700 metres wide occurs two kilometres to the west of the Kona deposit. This feature, partially delineated by grid surveys, is underlain by metasediments, the immediate hangingwall to the host stratigraphy of volcanogenic massive sulphide mineralization. Three of eight NQ2 diamond drill holes were completed in the southwestern corner of this exploration target that identified favourable stratigraphy but did not intersect significant mineralization.

Soil geochemical sampling within the Kona-Lake grid areas delineated a series of copper- and cobalt-in-soil anomalies along with elevated zinc-in-soil values. The largest and most intense copper-cobalt anomaly, measuring 1.5 kilometres long by 700 metres wide, trends northwesterly from the drill defined edge of the Kona deposit to the boundary of the survey grid. The size and intensity of the soil anomalies suggest potential for additional mineralization. A trend of discontinuous copper-gold soil anomalies along with a 2.0 kilometre length in the Lake target may represent glacial dispersed mineralization from the nearby host horizon.

Metallurgical testwork and a scoping study were undertaken to confirm that metal grades of mineralization would have potential economic feasibility. Copper-cobalt-gold mineralization from diamond drill hole reject material was sent to Lakefield Research Ltd. to determine potential metal recoveries. Eight flotation tests were completed that suggested a copper-gold concentrate could be produced at a grade of approximately 20% copper and 16 grams gold per tonne with recoveries of 90% for copper and 70% for gold.

The company contracted Kilborn Engineering Pacific Ltd. (“Kilborn”) to carry out a preliminary economic study to determine the economic parameters of developing the Kona deposit into a combined open-pit and underground mine. Kilborn was provided with a conceptual tonnage and grade objective for the deposit. Kilborn prepared capital and operating cost estimates and developed a financial model. Kilborn proposed the mining methods used in the study and metallurgical testwork completed at Lakefield Research Labs in Ontario was utilized to establish a milling circuit and determination of metal recoveries.

Preliminary resource calculations of the Kona deposit were prepared by the Company. Calculations utilizing the sectional block method show the Kona deposit to contain a (drill) indicated resource of 8.2 million tonnes grading 2.08% copper, 0.11% cobalt and 0.73 g/T gold (***Note: This historic resource estimate is NOT compliant with current NI 43-101 standards of resource definition classification and should NOT be relied upon***).

The expenditures incurred by the Company on the Fyre Lake Property to December 31, 1997 totalled $6,669,567.”

The procedures and results of the above referenced historical metallurgical testwork by Lakefield Research are discussed later in this report (see ‘Metallurgical Testing’ section). The preliminary economic study by Kilborn Engineering Pacific Ltd. was a theoretical investigation, unsupported by drilling data. The purpose of this study is documented in the ‘Other Relevant Data and Information’ section of this report but the results of the study are not discussed or interpreted by the writer because of its conceptual nature utilizing out-of-date financial models. The above referenced historical resource estimates were completed by the writer utilizing 1966, 1996 and 1997 diamond drill data. Although the writer did the mineral resource estimates, he was not an Independent Qualified Person at the time since Minorex Consulting Ltd., a company he controls, received the majority of its income at the time from Columbia Gold Mines Ltd. (NI 43-101, Section 1.2), and the classifications of mineral resources were combined at the time (NI 43-101, Section 1.3); prior to the adoption of National Instrument 43-101. ***Thus, the 1997 resource estimates prepared by the writer and Columbia Gold Mines Ltd. do not conform with National Instrument 43-101 and should NOT be relied upon***.

Columbia Gold Mines Ltd. did not proceed with exploration work in 1998 due to insufficient exploration funding, and in 1999 the Company decided to remove all drilling and exploration equipment and supplies from the property until sufficient funds were available to continue exploration (see foregoing 'Reclamation and Permitting' section of this report).

In August 2002 Rock Resources Inc., a Vancouver-based junior mining company, negotiated an option agreement with Pacific Ridge to earn a sixty percent interest in the subject property. Rock Resources retained the writer to prepare a report on the property documenting all exploration work. In addition, the writer was retained to undertake a NI 43-101 compliant mineral resource study estimating the individual indicated and inferred mineral resources of the known Kona deposit. This work was competed by the writer and a 128-page document titled "Report on the Fyre Lake Property", dated August 31, 2002, was delivered to Rock Resources Inc.

According to Roberts (2006), Rock Resources Inc. did not undertake any exploration work on the property; thus, terminating their option agreement with Pacific Ridge. Since then Rock Resources Inc. has undertaken a name change to 'Adroit Resources Inc.' Therefore, no exploration work has been undertaken on the subject property since the completion of the 1997 exploration program (Roberts, 2006), and there is no reported mineral production.

Geological Setting

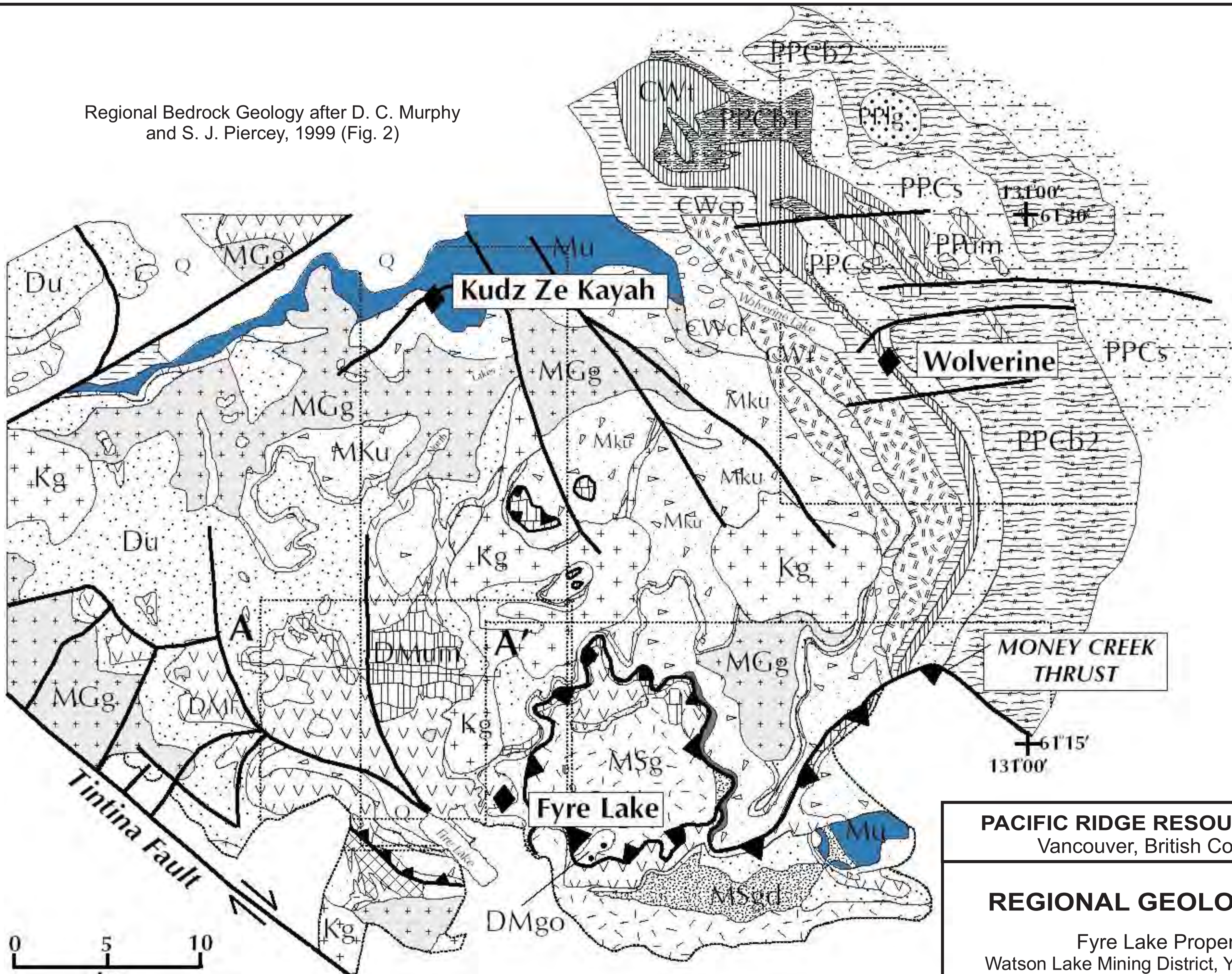
The regional geology of the eastern Pelly Mountains has been the subject of several Geological Survey of Canada mapping projects; published as G.S.C. Map 8-1960 (J. O. Wheeler et al., 1960) and G.S.C. Open File 486 (D. J. Tempelman-Kluit, 1977). More recently, the Yukon Tanana Terrane ("YTT") has been studied by various geologists associated with the Yukon Geology Program. These geologists, such as J. A. Hunt (1999c, 1998a, 1998b, 1997), D. C. Murphy (2001, 1998, 1997a), S. J. Piercey *et al* (2001, 2000, 1999) and J. A. Mortensen *et al* (1985), have mapped and studied the regional geology, evolution and mineral deposits of the Yukon-Tanana Terrane. In 2002 the results of the works by J. A. Hunt were published by the Exploration and Geological Services Division, Yukon Region, Indian and Northern Affairs Canada as "Volcanic-associated massive sulphide (VMS) mineralization in the Yukon-Tanana Terrane and coeval strata of the North American miogeosyncline, in the Yukon and adjacent areas" (Bulletin 12; Hunt, 2002). Much of the following text is based upon the observations and interpretations derived from this bulletin. See Figures 3A, 3B, 4, 5A and 5B of this report for the geological setting of the property (after Hunt, 2002; Murphy and Piercey, 1999).

Regional Geology

The Fyre Lake property is underlain regionally by the Yukon-Tanana Terrane ("YTT") which underlies much of the Yukon, east-central Alaska and parts of British Columbia. This terrane lies between the ancestral North American continental margin to the east and exotic terranes to the west (Hunt, 2002). According to Hunt (2002), "The YTT is made up primarily of poorly exposed, polydeformed, metaigneous and metasedimentary rocks. Pre-mid-Mesozoic rocks (Devonian to Permian) of the YTT consist mainly of pelitic to quartzo-feldspathic metasedimentary schist and gneiss with minor marble, and deformed mafic to felsic metavolcanic and metaplutonic rocks (cf. Mortensen, 1992a). Most units within YTT display a penetrative ductile deformation fabric and have been affected by regional-scale thrust faulting (Mortensen and Jilson, 1985; Mortensen 1992a; Dusel-Bacon et al., 1998)."

In the Finlayson Lake area the YTT is approximately 300 kilometres long by less than 50 kilometres wide. According to Hunt (2002), it is juxtaposed against Proterozoic and Paleozoic miogeosynclinal strata of the ancestral North American continental margin along the Tintina fault zone to the southwest, and along the Finlayson Lake fault zone to the northeast (see Figure 3A). Upon restoration of approximately 425 kilometres of Late Cretaceous right-lateral strike-slip movement on the Tintina Fault the main part of the YTT, which underlies most of west-central Yukon, is contiguous with the Finlayson Lake portion (Roddick, 1967; Tempelman-Kluit, 1976). The Finlayson Lake fault zone has been interpreted by Mortensen and Jilson (1985) to be a transgressive suture, or by Plint and Gordon (1997) to be a thrust fault zone.

Regional Bedrock Geology after D. C. Murphy
and S. J. Piercey, 1999 (Fig. 2)



To Accompany Report By J. D. Blanchflower, P. Geo., 2006

PACIFIC RIDGE RESOURCES LTD. Vancouver, British Columbia	
REGIONAL GEOLOGY MAP Fyre Lake Property Watson Lake Mining District, Yukon Territory	
Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 3A

- LEGEND -

Q — unconsolidated surficial deposits

Legend for Regional Bedrock Geology after D. C. Murphy and S. J. Piercey, 1999 (Fig. 2)

Intrusive rocks

- Kg — weakly foliated biotite-muscovite granite
- Pg — foliated granitic dyke
- PPlg — leucogabbro
- PPum — serpentinized ultramafic rock
- MSg — biotite-hornblende granite and quartz monzonite
- MSgs — strongly foliated unit MSg
- MSgd — biotite-hornblende granodiorite
- MGg — strongly foliated granitic metaplutonic rock
- DMq — quartz-porphyritic granite
- DMd — hornblende-biotite meta-diorite
- DMum — serpentinized ultramafic rock; lesser gabbro and pyroxenite
- DMgo — gabbro

Layered rocks

CAMPBELL RANGE SUCCESSION

- PPCb2 — fragmental, massive and pillowed basalt
- PPCs — carbonaceous argillite, sandstone, and quartz grit; chert, chert-pebble conglomerate in northeast
- PPCb1 — fragmental, massive and pillowed basalt

WOLVERINE SUCCESSION

- CWt — felsic tuff and exhalite
- CWcp — carbonaceous phyllite and quartz sandstone, lesser felsic metavolcanic rocks
- CWf/CWq — felsic metavolcanic rock, locally porphyritic (CWq)
- CWcl — carbonaceous argillite, feldspathic metasandstone and conglomerate

GRASS LAKES SUCCESSION

- Mu — units Mcp, Mq, Mm, and Mcg undifferentiated in Fig. 2
- Mcp — carbonaceous phyllite
- Mq — quartzite
- Mm — biotite-actinolite-chlorite schist
- Mcg — quartz-pebble conglomerate
- MKu — units MK, MKcp and Mkq undifferentiated in Fig. 2
- MK — Kudz Ze Kayah felsic metavolcanic unit: feldspar-muscovite-quartz schist
- MKcp — carbonaceous phyllite and grey quartzite
- MKq — quartz-feldspar-augen schist
- Cl — foliated crinoidal limestone
- DMFr — feldspar-muscovite-quartz schist of volcanic or volcaniclastic protolith
- DMF — Fire Lake mafic metavolcanic unit: biotite-plagioclase-actinolite-chlorite schist, basalt and volcaniclastic sedimentary rocks
- DMFcp — carbonaceous phyllite and quartzite, pale green chert and grey marble
- Dq — biotite-muscovite-feldspar-quartz schist, micaceous quartzite and psammite, quartz-biotite-muscovite metapelitic schist
- Dqc — marble

To Accompany Report By J. D. Blanchflower, P. Geo., 2006

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

LEGEND FOR REGIONAL GEOLOGY MAP

Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 3B

The geology of the Finlayson Lake area (N.T.S. 105G) has been the subject of several mapping studies by Wheeler et al (1960a), Tempelman-Kluit (1977a) and Mortensen and Jilson (1985) at scales of 1:250,000. However, for the purposes of this report the writer will document the results of more recent works by the various geologists of the Yukon Geology Program. According to Hunt (2002),

> " The deformed and metamorphosed volcanic, intrusive and sedimentary rocks in the Finlayson Lake area have been divided by Murphy and Piercey (1999b, c) into three stratigraphic successions, the Grass Lakes, Wolverine and Campbell Range successions (Figs. 3c, 4). The Devonian to Mississippian Grass Lakes succession is stratigraphically lowest and is made up, in part, of a basal unit (units Dq, Dqc, Dm, Df/Unit 1) of non-carbonaceous, metasedimentary rocks and marble. The basal unit is overlain by rocks of the ~ 365-360 Ma (Piercey et al., 2001) Fire Lake metavolcanic unit (unit DMF/Unit 2) made up of mafic metavolcanic and metaintrusive rocks, and lesser felsic metavolcanic rocks, carbonaceous phyllite and quartzite, and marble. The Fire Lake metavolcanic unit is stratigraphically overlain by the ~360 Ma (ibid.) Kudz Ze Kayah felsic metavolcanic unit (units MK, q, cp/Unit 3). This unit consists dominantly of felsic metavolcanic and carbonaceous metasedimentary rocks with lesser mafic metavolcanic and metaintrusive rocks, and felsic metaintrusive rocks. The remainder of the Grass Lakes succession, overlying the Kudz Ze Kayah felsic metavolcanic unit, is made up of feldspar-quartz-pebble conglomerate, quartzite, biotite-chlorite schist and carbonaceous phyllite and quartzite (units Mcg, Mq, Mm, Mcp/Unit 4).
>
> The Carboniferous Wolverine Lake succession (Figs. 3c, 4) unconformably overlies the Grass Lakes succession. The basal unit of the Wolverine Lake succession is made up of quartz-feldspar-pebble metaconglomerate and metasandstone (unit CWcl/Unit 5). Overlying the basal unit is carbonaceous phyllite and quartz metasandstone (unit CW cp). This is, in turn, overlain by a package of ~ 345 Ma (ibid.) rhyolitic metavolcanic rocks made up of flows, volcaniclastic and epiclastic rocks (unit CWf); locally, this unit is made up of feldspar ± quartz porphyritic rhyolitic metaintrusive rocks (unit CW q). The top of the Wolverine Lake succession is made up of rhyolitic metavolcaniclastic and metaepiclastic rocks, rhyolitic metaintrusive rocks and metavolcanic flow rocks, and carbonaceous phyllite (unit CWt/unit 6). Iron formation horizons occur locally within this unit.
>
> The Wolverine Lake succession (Figs. 3c,4) is overlain by the Pennsylvanian to Permian (Plint and Gordon, 1997), and possibly younger (Orchard in Appendix VI-3) Campbell Range succession. The Campbell Range succession is made up of massive to pillowed and brecciated basalt, maroon and green chert, gabbro and diabase separated by sedimentary rocks that include carbonaceous argillite, sandstone and quartz grit, diamictite, varicoloured chert and argillite, chert-pebble conglomerate and limestone (Murphy and Piercey, 1999a, b, c).
>
> The older successions are intruded by (Fig. 4; ibid.) Devonian to Mississippian quartz-porphyritic metagranite (unit DMg), ultramafic rocks (units DMum, Dmgo) and hornblende-biotite metadiorite (unit DMd), the granitic to granodioritic Mississippian Simpson Range Plutonic Suite (units MSgd, gs, g) and the granitic to monzonitic Mississippian Grass Lakes metaplutonic suite (unit MGg). Other intrusive rocks in the Finlayson Lake area include Pennsylvanian to Permian ultramafic rocks (unit Ppum) and leucogabbro (unit PPlg), Permian granitic dykes (Pg) and Cretaceous biotite-muscovite granite (unit Kg).
>
> Correlation of individual rock units between areas is difficult. In any volcanic pile individual lithologies tend to be highly variable with respect to thickness and aerial distribution due to irregular paleodepositional surfaces, proximity to volcanic vents and dynamic to catastrophic local tectonics. In addition, the above successions are deformed and metamorphosed. However, the degree of strain is heterogeneous and locally primary textures are preserved. For example, spherulites/amygdules are preserved in mid-Paleozoic rhyolitic rocks just west of Kudz Ze Kayah, and relict quartz and feldspar phenocrysts are visible in Mississippian intrusive rocks in the Wolverine Lake area.

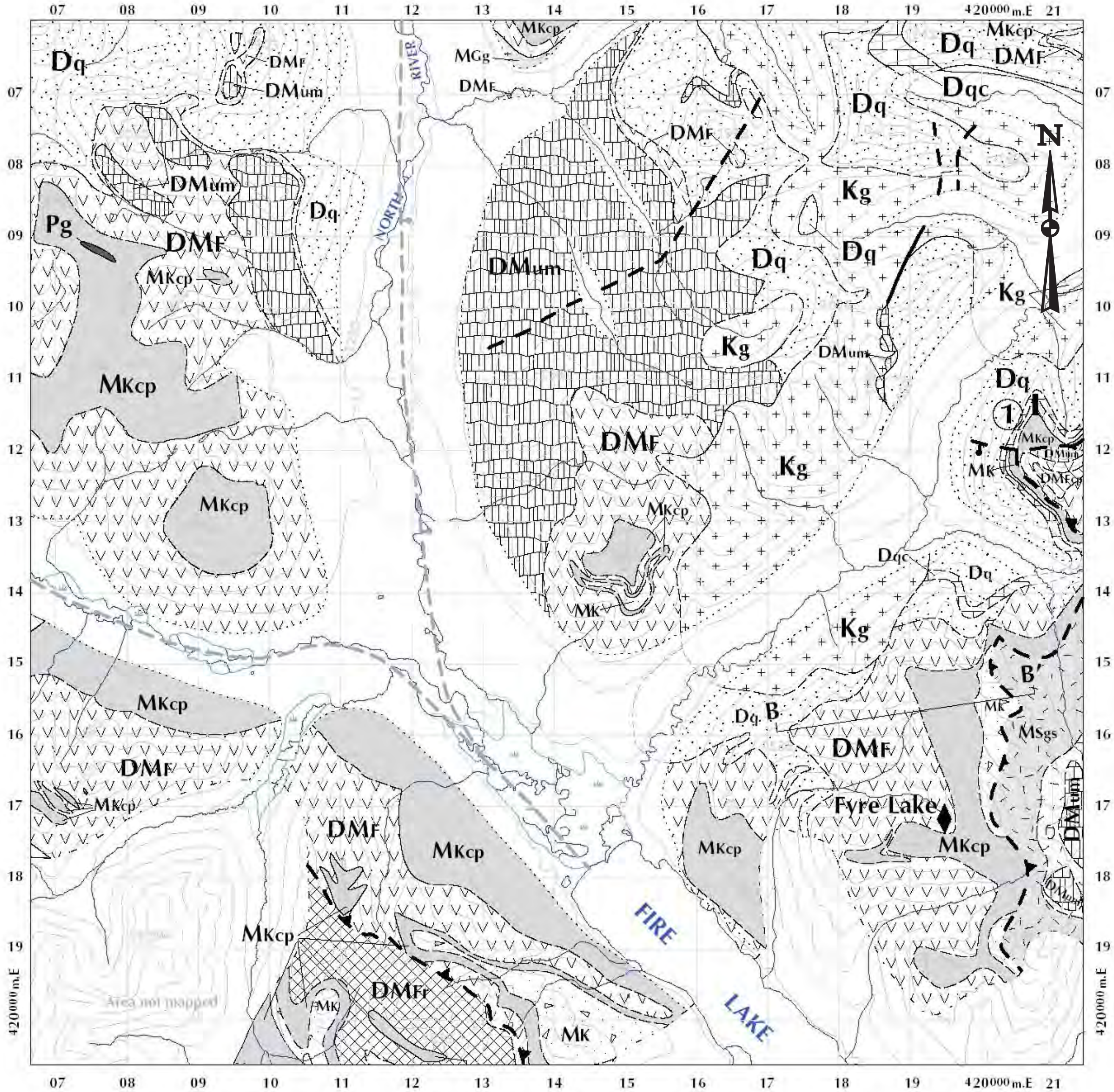
> The above stratigraphy contains at least four horizons that host VMS deposits (Figs 3b, c; synthesized in Murphy and Piercey, 1999a; and partly summarized in Hunt, 1997, 1998a, b, 1999c). The copper-cobalt-gold-bearing Fyre lake deposit is hosted by the Fire Lake mafic metavolcanic unit. It lies close to the contact with overlying carbonaceous phyllite of the Kudz Ze Kayah felsic metavolcanic unit that hosts the zinc-lead-copper and precious metal-rich ABM (Kudz Ze Kayah) deposit. GP4F occurs slightly lower in the stratigraphy than Kudz Ze Kayah (T. Tucker, pers. comm., October, 2000). The zinc-lead-copper and precious-metal-rich Wolverine deposit is hosted by felsic metavolcanic rocks and associated metasedimentary rocks of the Wolverine Lake succession. The copper-cobalt-gold-bearing Ice deposit is hosted by brecciated pillowed mafic volcanic rocks at the northwest end of the CRB (Campbell Range belt). Gossanous pyritic felsic metavolcanic rocks of the basal unit of the Grass Lakes succession may constitute a fifth mineralized horizon (Murphy, 1998)."

The Fire Lake mafic metavolcanic unit (unit DMF/Unit 2) which hosts the Fyre Lake (Kona) deposit, was subdivided by Piercey *et al* (1999) into three lithogeochemical suites (2a, 2b and 2c) based upon trace and major element contents. Suite 2a ranges from subalkalic basalt to andesite and has a boninitic to low-Ti tholeite affinity. Suite 2b has transitional subalkalic basalt/andesite affinities, and suite 2c is made up of basaltic andesite and has chemical affinities intermediate between those of suites 2a and 2b (Hunt, 2002). The Fyre Lake (Kona) deposit is hosted by boninitic rocks of suite 2a (Sebert and Hunt, 1999) which generally occur in forearc and/or back-arc settings (Hunt, 2002).

Two phases of deformation with regional metamorphism are recognized in the region (Mortensen, 1985). The earliest and most pervasive episode affected the Early Paleozoic volcano-sedimentary assemblage and Paleozoic metaplutonic rocks which resulted in a well developed shallow dipping foliation (S_1) accompanied by regional middle greenschist to middle amphibolite metamorphism. The S_1 foliation typically parallels compositional layering and in coarse-grained rocks is accompanied by a stretching lineation. The second phase of deformation involved a locally strong crenulation cleavage accompanied by middle greenschist facies metamorphism. Mortensen (1985) and earlier workers identified six major thrust faults in the region. The faults are cut by mid-Cretaceous intrusions and postdate Late Triassic strata (Stroshein, 1991).

The structural setting within the Fire Lake area is very complex and complicated by syn-depositional normal faulting and post-depositional thrust faulting (Hunt, 2002; Tempelman-Kluit, 1977). Normal faults have been mapped along and perpendicular to the headwaters of Kona Creek. The inferred traces of these faults are reflected by both Kona and Outfitter's drainages. Interpretation of syn-mineralization faults and their re-activation in the Finlayson Lake district were the subject of a paper by Murphy and Piercey (1999). The following text is derived from this paper (Murphy and Piercey, 1999, p. 59-60). See Figure 4 of this report for the local geology (after Murphy and Piercey, 1999, Fig. 3).

> " The characteristics of the host rocks around the Fyre Lake deposit satisfy most of the criteria for the presence of a syn-volcanic fault. Regional mapping has shown that the Fyre Lake deposit is spatially associated with profound changes in the nature and thickness of the host Upper Devonian to Lower Mississippian Fire Lake mafic schist unit (Murphy, 1998; Murphy and Piercey, 1999b, c; unit DMF, Figs. 2 and 3). Four kilometres northeast of the deposit (Location 1, Fig. 3), the Fire Lake unit comprises about 40 m of biotite-actinolite-chlorite schist. Near the deposit, the unit is nearly 800 m thick (section B-B', Fig. 4). At the deposit, unit DMF is at least this thick (the bottom was not intersected in drill holes). It includes 5 to over 200 m of felsic schist of volcanic and volcaniclastic protolith, and siliceous carbonaceous phyllite (quartz-chlorite mica schist of Blanchflower et al., 1997; transition zone of Foreman, 1998; psammitic schist and felsic metavolcanic rocks of Sebert and Hunt, 1999). Massive sulphide mineralization occurs in the upper part of the unit just below the base of the overlying carbonaceous schist, the same carbonaceous schist that overlies the section 4 km to the northeast.



See Figure 3B for Legend

0 2 4
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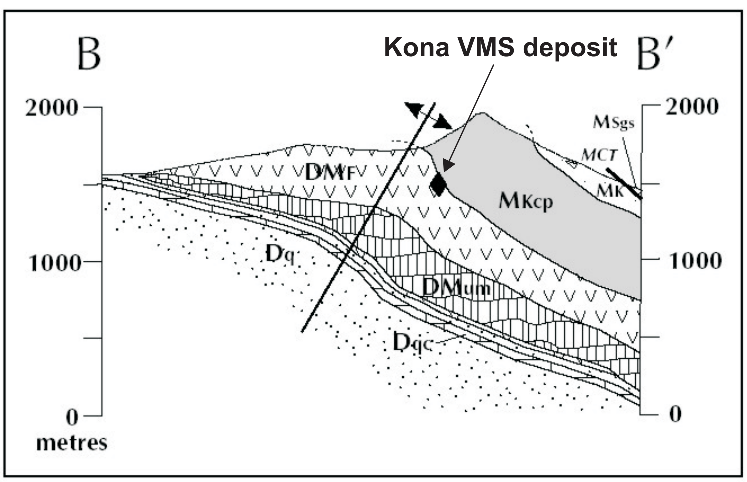


Local Bedrock Geology after D. C. Murphy and S. J. Piercey, 1999 (Fig. 3)

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

LOCAL GEOLOGY MAP

Fyre Lake Property
Watson Lake Mining District, Yukon Territory

To Accompany Report By J. D. Blanchflower, P. Geo., 2006

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 4

The changes in the thickness and nature of the mafic metavolcanic host rocks of the Fire Lake deposit also coincide with a change in the amount of mafic and ultramafic metaplutonic rocks spatially associated with the host schist. No mafic and ultramafic metaplutonic rocks are found in or near the unit 4 km northeast of the deposit (except in the hanging wall of the Money Creek thrust, discussed later). However, 2 km north of the deposit, over 100 m of massively serpentinized ultramafic rock (meta-peridotite), massive coarse-grained amphibolite (meta-pyroxenite) and coarse-grained actinolite-plagioclase-chlorite schist (meta-gabbro) occur at the base of the unit, directly overlying the marble-quartz psammite unit. On the ridge directly north of the deposit, meta-gabbro makes up about 10% of the mafic schist unit.

Mafic and ultramafic meta-plutonic rocks in southeastern Grass Lakes map area (105G/7, Fig. 2, Murphy, 1997), north-northwest along strike of the Fyre Lake deposit, show characteristics and relationships that suggest they are sills that flowed from dykes lying along the trend of thickness changes in unit DMF. These rocks occur primarily in an approximately 600-m-thick sheet lying near the base of unit DMF (section A-A', Figs. 2, 5). The sheet tapers to zero thickness over a 6 km horizontal distance westwardly across the North River valley and over a 4 km horizontal distance eastwardly. In addition, to the east, smaller bodies of ultramafic rock occur at different levels within the muscovite-quartz psammite unit (unit Dq) under the mafic schist. East of the Cretaceous granite in this area, bodies of ultramafic rock occur in the lower part of unit Dq, below the calcareous member Dqc. West of the granite, at approximately the same structural level, ultramafic rocks occur in the part of unit Dq above the calcareous member. These occurrences of ultramafic rock below unit DMF are unusual, observed only in one other place in the area shown in Figure 2, and are interpreted as dykes feeding the stratabound sheet within the overlying mafic schist unit. Furthermore, as two different stratigraphic levels of unit Dq are juxtaposed at the same structural level on either side of the Cretaceous granite, these dykes likely intruded along a fault.

Regionally, the changes identified above occur across a north-northwest-trending corridor that can be traced from the Fyre Lake deposit into east-central Grass Lakes map area and as far north as the large body of Grass Lakes granitic meta-plutonic rock (Fig. 6). Lying directly along this trend, across the Grass Lakes metaplutonic body, is Cominco Ltd.'s Kudz Ze Kayah deposit (Yukon Minfile, 1997, 105G 117). The spatial association of prospects and deposits in unit MK, with the projected trace of the Fyre Lake structure, implies that this feature may have controlled hydrothermal fluid flow during the deposition of unit MK."

Property Geology

The bedrock geology and mineralization of the Fyre Lake property were studied by the writer during his 1996 and 1997 field work. The following text is derived directly from Bulletin 12 (Hunt, 2002, p. 12-20) since it combines the results of the writer's geological work (Blanchflower and Deighton, 1996; Blanchflower, 1997; Blanchflower et al., 1997) with those of other geologists (Hunt and Murphy, 1998; Sebert and Hunt, 1999) and recent results of petrological studies and lithogeochemical analyses by the Yukon Geology Program (Leitch, 1998; Sebert and Hunt, 1998; Piercey *et al*, 1999). See Figures 5A and 5B of this report (after Hunt, 2002) illustrating the bedrock geology of the Kona zone and central portion of the property.

" The Fyre Lake area was mapped at 1:50 000 scale during this study (Fig. 9; Hunt and Murphy, 1998) and in more detail by staff of Columbia Gold Mines Ltd. (now Pacific Ridge Exploration; Blanchflower and Deighton, 1996; Blanchflower, 1997; Blanchflower et al., 1997; Deighton and Foreman, 1997; Foreman, 1998). The property is underlain primarily by the Fire Lake mafic metavolcanic unit (unit DMF/Unit 2: Figs. 3, 4). The majority of this unit, which hosts the Kona deposit, is made up of a dark green, fine-grained chlorite-quartz and chlorite-actinolite-quartz schist and phyllite package. The chloritic schist and phyllite package is underlain by at least 50 m of carbonaceous phyllite, and overlain by a sequence, at least 700 m thick, of fine-grained, finely laminated, well foliated, grey to black carbonaceous phyllite, lesser metasiltstone and metasandstone, and minor limestone (unit MKcp/Unit 3). The under- and overlying carbonaceous phyllites are indistinguishable from one another except by stratigraphic position. Locally, felsic metavolcanic rocks overlie the upper carbonaceous phyllite (unit MK/Unit 3). The dominant foliation is parallel to compositional layering and dips shallowly eastward; lineations plunge shallowly to the southeast, parallel to the trend of mineralization at about 130° (Deighton and Foreman, 1997).

Schistose host rocks to the mineralization are interpreted as a succession of mafic to intermediate flows and tuffs (Fig. 10a) with intercalated volcaniclastic and volcanically derived fine-grained sedimentary rocks (Deighton and Foreman, 1997; Foreman, 1998). The strata are part of a regionally persistent chlorite schist and phyllite unit (unit DMF/Unit 2: Figs. 3, 4), spatially associated with voluminous mafic and ultramafic intrusive rocks (Murphy, 1998; Murphy and Piercey, 1999b). Murphy (1998) interprets the ultramafic rocks as sills, fed by dykes which intruded along a syn-sedimentary fault (not preserved). This fault is inferred to have formed the northeast side of the basin in which the Kona massive sulphide deposit formed.

The underlying carbonaceous phyllite (unit cp on Fig. 9; Fig. 10b) outcrops in Kona creek and was intersected in drilling in the Kona zone in drill hole 97-97 which terminated in metasedimentary rocks beneath mafic schists (Foreman, 1998). It is not clear if the carbonaceous phyllite is structurally juxtaposed or if it represents a separate unit. Its presence as a separate stratigraphic unit would suggest that local faulting likely controlled sedimentation. Such a fault may also have acted as a conduit for mineralizing hydrothermal solutions (Hunt and Murphy, 1998; Murphy, 1998).

Early descriptions of the Fyre Lake property (cf. Stroshein, 1991) show the overlying metasedimentary rocks in thrust fault contact with the underlying mafic metavolcanic rocks, however, recent mapping found no evidence for a thrust fault contact. The contact between the two successions appears to be transitional and is marked by an interval of intercalated quartz-biotite ± chlorite and chlorite ± biotite ± quartz schist 6 to 200 m thick, which thickens to the west (included within unit DMF on Fig. 9; in mixed metasedimentary and metavolcanic rocks on Fig. 12; and as unit INVS in Columbia Gold Mines Ltd. company reports; Foreman, 1998). This interval is described by Foreman (1998) as a zone of interfingering terrigenous sediments and volcanically derived sediments and/or flows.

The affiliation of felsic rocks that overlie the upper carbonaceous phyllite is not clear (Fig. 9, unit MK). They are lithologically similar to felsic metavolcanic rocks in the Kudz Ze Kayah felsic metavolcanic unit (unit Mk/Unit 3). However, they also show similarities to metamorphosed felsic rocks of the Simpson Range Plutonic Suite which is exposed to the east and south of the deposit (units MSgs, MSg on Fig. 9).

Petrography

Specimens of the chlorite schist and phyllite sequence, and felsic rocks overlying the carbonaceous phyllite were examined in thin section by Leitch (1998). Summary results follow. Detailed descriptions are in Appendix VI-3.

Specimens of the chloritic schist and phyllite sequence (Appendix VI-3: specimens JH96-62, JH97-57, 59, 60, 70) are composed of amphibole (0-60%), quartz (10-40%), relict plagioclase (10-30%), chlorite (0-20%), biotite (1-10%), epidote (0-10%), carbonate (0-3%), garnet (0-2%), minor sericite and accessory rutile, apatite, sphene and opaques. Locally, the schists are made up of alternating quartz-plagioclase-rich laminations and amphibole-rich or chlorite-biotite-epidote-garnet-rich laminations. Locally, pale, quartzose boudins 2-20 cm across occur within the schists. Rare orange-red garnet occurs as porphyroblasts up to 4 mm in diameter (Deighton and Foreman, 1997). The specimens examined are classified by Leitch (1998) as mainly metagabbro/diabase and lesser probably intermediate to mafic volcanic rock.

Locally, the chlorite schist and phyllite sequence contains pale layers (Appendix VI-3: specimens JH97-67 and 77) composed primarily of quartz and feldspar that are likely metamorphosed felsic or intermediate volcanic rocks with chlorite-sericite-epidote ± biotite alteration. Locally, the schists are composed primarily of amphibole (likely tremolite-actinolite) and magnesian chlorite (Appendix VI-3: specimen JH97-65) and are interpreted to be metamorphosed ultramafic rocks.

Two specimens (Appendix VI-3: JH96-KZFV, JH97-75) of the felsic rocks were examined by Leitch (1998). Specimen JH97-KZFV is composed of 60-65% (likely albitic) plagioclase, 20% quartz, 10% sericite, 5% chlorite and biotite, and 1-2% opaques (limonite and possibly rutile). Rounded, relict plagioclase phenocrysts, up to 1 mm in diameter, occur in a fine-grained matrix of quartz and plagioclase with minor sericite and chlorite. The protolith appears to have been plagioclase-porphyritic felsic volcanic rock, possibly of dacitic composition. Specimen JH97-75 consists of about 1-mm-thick laminae of alternating quartz-rich, possible feldspar-rich, and sericite-carbonate-rich composition. Locally, larger relict crystals (possibly K-feldspar) are preserved suggesting a possible former porphyritic rock of felsic to intermediate composition.

Metamorphism

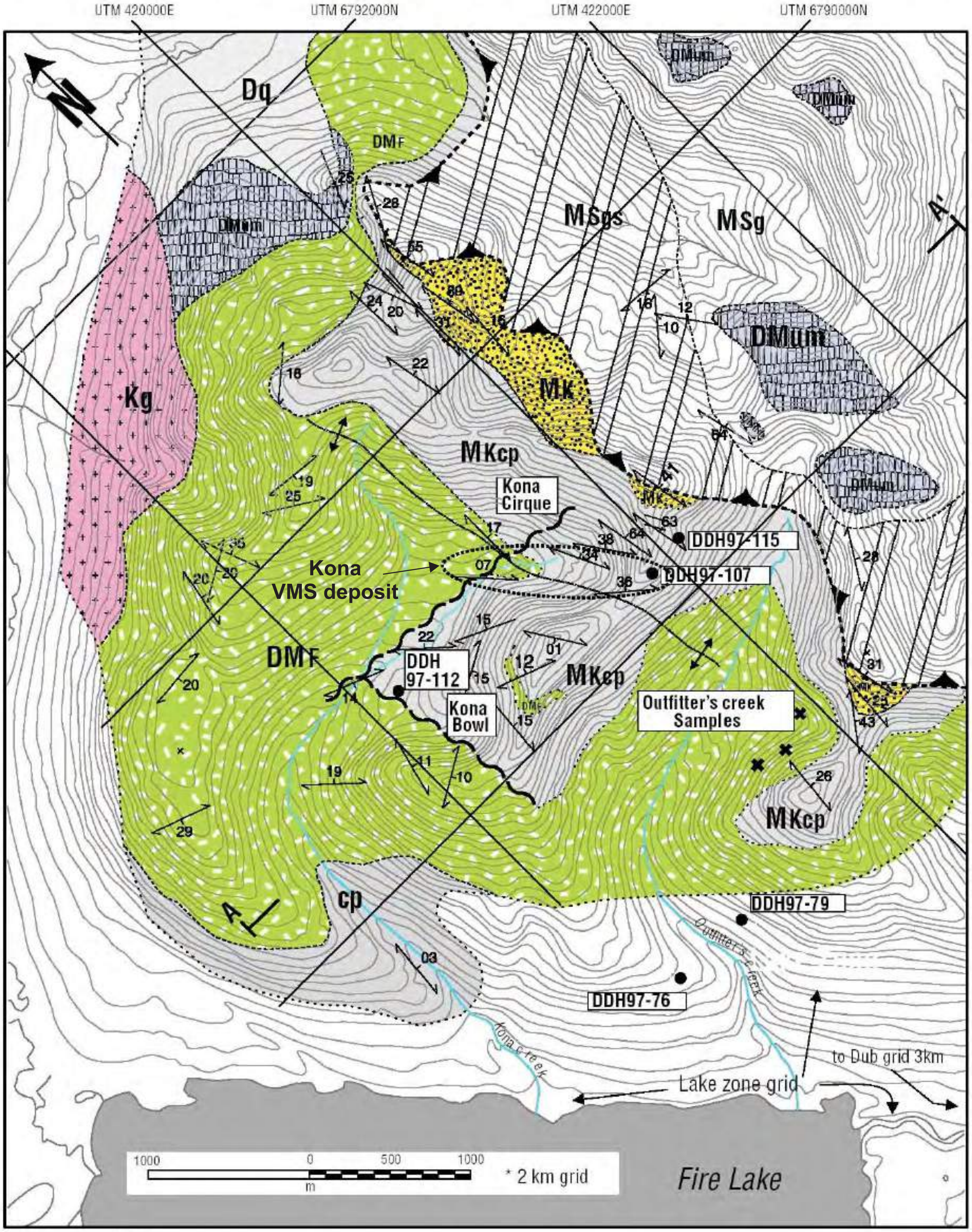
The presence of biotite and garnet indicate that pressure-temperature conditions reached upper greenschist grade in the Fyre Lake area. Garnet porphyroblasts rimmed by chlorite suggest retrograde greenschist metamorphism overprints an earlier higher-grade phase (Sebert, 1997).

Lithogeochemistry

Specimens of mineralization-hosting and non-mineralization-hosting chlorite schist were collected from the Fyre Lake area for lithogeochemical analysis. The data are being analysed as part of an ongoing study of the Fyre Lake deposit (preliminary results are in Sebert and Hunt, 1998) and as part of a regional study of the Finlayson Lake district (preliminary results are in Piercey et al., 1999). Following is a summary of the results to date.

Analysis of chlorite schist from the Fire Lake area indicates there are distinct chemical differences between chlorite schists that host the Kona deposit (Kona cirque specimens) and those that are unmineralized (Lake zone, Outfitter's creek and Kona bowl specimens). In general, the specimens plot in the basalt to andesite fields on an SiO_2 versus Zr/TiO_2 rock classification diagram (Fig. 11a) and are similar to specimens of unit DMF/Unit 2 collected elsewhere in the Finlayson Lake district (see Figs. 6a, b). However, on a V versus Ti tectonic affinity diagram distinct differences are apparent (Fig. 11b). Specimens of chlorite schist that hosts the Kona deposit cluster in the boninite and low Ti-tholeiite field, but non-mineralization-hosting chlorite schist specimens plot in the mid-ocean-ridge basalt (MORB) and back-arc basin basalt (BABB) field. This difference is also seen in rare-earth element (REE) plots for the chlorite schists. The pattern for Kona deposit host chlorite schists is similar to that for boninitic rocks (Figs. 11c, d), but the pattern for non-deposit hosting chlorite schists is similar to that for arc-related tholeiitic rocks. Differences between the chlorite schists can also be seen on other plots, for example MgO versus Zr (Fig. 11e) and Cr versus TiO_2 (Fig. 11f).

Overall, the chemistry of metavolcanic rocks in the Fyre Lake area suggests at least some of them were deposited in an arc-related environment influenced by subduction processes; see Sebert and Hunt (1999) and Sebert et al. (in prep) for more details. However, the tectonic implications of the presence of boninitic rocks is not yet clear. Boninitic rocks usually occur associated with ophiolitic sequences in back-arc basin and forearc settings, for example the Troodos ophiolite in Cyprus and the Izu-Bonin arc in the south Pacific (cf. Crawford, 1989). However, no typical ophiolitic sequence appears to be present in the Fire Lake area, although thick ultramafic rocks are spatially associated with unit DMF/Unit 2 (Figs. 5, 9; Hunt and Murphy, 1998)."



* UTM zone 9, NAD 27

After Hunt, 2002,Fig. 9a (Bulletin 12, p. 18)

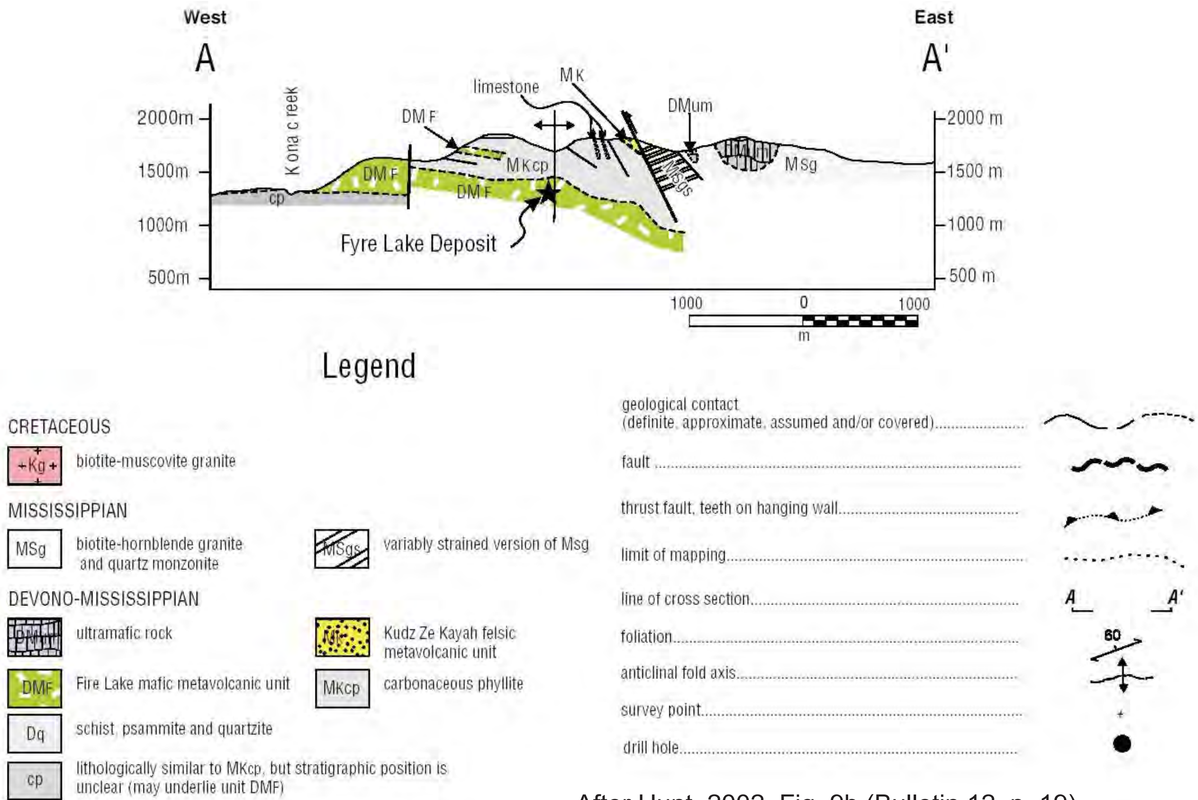
Legend accompanies report as Figure 5B

To Accompany Report By J. D. Blanchflower, P. Geo., 2006

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

PROPERTY GEOLOGY MAP
KONA AREA
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 5A



After Hunt, 2002, Fig. 9b (Bulletin 12, p. 19)

View looking southeasterly along headwaters of Kona Creek (Kona deposit trends 130°)

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

LEGEND for PROPERTY GEOLOGY MAP
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 5B

To Accompany Report By J. D. Blanchflower, P. Geo., 2006

DEPOSIT TYPES

Exploration activity in the Finlayson Lake area since 1993 has led to the delineation of six major deposits, including the Fyre Lake (Kona) deposit on the subject property, the ‘ABM’, ‘GP4F’ and ‘Fault Creek’ deposits on the Kudz Ze Kayah property, the ‘Wolverine’ deposit on the Wolverine property and the ‘Ice’ deposit, plus other numerous base- and precious-metal-bearing mineral showings. The following text contains brief descriptions of the Kudz Ze Kayah, Wolverine and Ice properties to demonstrate the variety, size and tenor of the major volcanogenic massive sulphide deposits in this region. These descriptions are based largely on the observations reported by Hunt (2002).

The Kudz Ze Kayah property (Yukon MINFILE, 2001, 105G 117; 61°28'N, 130°36'W) is located near North Lakes approximately 30 kilometres north of the subject property or about 120 kilometres southeast of the town of Ross River (see Figure 3A). It is now owned entirely by Teck Cominco Limited after Expatriate Resources Ltd. optioned, explored and then dropped their property option between March, 2000 and September, 2001. This property is underlain by a thick, structurally transposed, polydeformed, felsic metavolcanic complex, and lesser mafic metavolcanic and metasedimentary rocks, that have undergone at least mid-greenschist grade metamorphism (Hunt, 2002).

The main mineralization, known as the ‘ABM’ deposit, is roughly tabular and contains several thickened lenses that are collectively up to 22.5 m thick (Hunt, 2002). The mineralization dips moderately to the north near surface and flattens at depth. The deposit extends for about 700 m along strike, about 400 m down dip, and is on average about 18 m thick (Hunt, 2002). It lies stratigraphically above the unit hosting the Fyre Lake deposit within a thick complex of felsic meta- tuffs and sills or flows interlayered with minor mafic sills or flows and metasedimentary rocks, and is overlain by carbonaceous phyllite, mafic metavolcanic rocks, and quartzo-feldspathic conglomerate (Murphy, 1998). Host felsic metavolcanic rocks have a broad alkalic arc affinity (Piercey *et al*, 1999). The strata exhibit isoclinal recumbent folding with bedding generally sub-parallel to schistosity (Hunt, 2002). The deposit has been affected by at least three phases of deformation. Sphalerite, chalcopyrite and galena are the main economic minerals with electrum occurring at the margins of galena and chalcopyrite grains (Hunt, 2002). Gangue includes mixtures of magnetite, barite, pyrrhotite, pyrite and carbonate (Hunt, 2002). Barium and base and precious metal zonation in the deposit, plus the position of proximal chloritic alteration above it, suggest that it has, at least in part, been overturned. This deposit has a reported geological resource of 13,000,000 tonnes of 5.5% Zn, 1% Cu, 1.3% Pb, 125 g/t Ag and 1.2 g/t Au (Schultze, 1996a), including an open pit mineable resource of 11,100,000 tonnes with grades of 5.61% Zn, 0.85% Cu, 1.56 % Pb, 136.9 g/t Ag and 1.33 g/t Au (Hunt, 2002).

The GP4F deposit is located about 6 km southeast of the ABM deposit and appears to be slightly lower in the stratigraphy (Murphy and Piercey, 1999). It is a thin massive sulphide lens hosted by strongly altered tuffs intruded by altered and mineralized quartz-feldspar sills (Hunt, 2002). The mineralization has been traced over a 200 m strike length and 350 m down dip. Reported resources are 1.5 million tonnes of 6.4% Zn, 3.1% Pb, 0.1% Cu, 89.7 g/t Ag and 2.0 g/t Au (Hunt, 2002).

The Fault Creek zone is proximal to the ABM deposit, but is thought to lie at the same stratigraphic level as the GP4F deposit (Hunt, 2002). Reported resources are 50,000 tonnes of 7.1% Zn, 1.0% Pb, 4.7% Cu, 130 g/t Ag, and 2.0 g/t Au (Hunt, 2002).

The Wolverine deposit (Yukon MINFILE, 2001, 105G 072; 61°25'37"N, 130°07'56"W) and the lesser explored Lynx, Fisher, Sable and Puck zones are located near the southeastern end of Wolverine Lake, approximately 30 kilometres northeast of the Fyre Lake property or about 20 kilometres east of Kudz Ze Kayah (see Figure 3A). The property is wholly owned by Yukon Zinc Corporation (formerly Expatriate Resources Ltd.). The deposit is hosted by a thick sequence of Carboniferous rhyolitic metavolcanic rocks and carbonaceous argillite (Tucker et al., 1997). These rocks are interpreted as being part of the Wolverine Lake succession and are stratigraphically higher than the host rocks at Kudz Ze Kayah (Murphy and Piercey, 1999). They are overlain by mafic metavolcanic and associated metasedimentary rocks of the Campbell Range succession while the massive sulphide and/or iron formation rocks are underlain by unaltered to weakly altered porphyritic metaintrusive rocks (Hunt, 2002).

Two thick lenses of stratiform massive sulphide mineralization known as the Wolverine and Lynx zones comprise the Wolverine zone; separated by an area of semi-massive sulphide mineralization and sulphide-stringer mineralization known as the Hump zone (Bradshaw et al., 2001). The deposit dips 35° to 50° northeastward and has been defined over a strike length of 700 m and a down-dip width of 400 m (Hunt, 2002). Polymetallic massive sulphide lenses are made up of pyrite and sphalerite with lesser chalcopyrite, pyrrhotite, galena, silver-rich tetrahedrite-tennantite and arsenopyrite (Tucker et al., 1997; Expatriate Resources Ltd., 1999a; Bradshaw et al., 2001). Ore minerals occur either interstitial to pyrite or as a matrix to disseminated pyrite. Gangue minerals include quartz, calcite, dolomite, ankerite, siderite, chlorite and sericite (Tucker et al., 1997; Bradshaw et al., 2001). Individual sulphide lenses vary in thickness from a maximum of 9.8 m to less than 1 m on the fringes of the deposit, and there are local multiple stacked lenses separated by 4 to 8 m of host rock (Bradshaw et al., 2001). On January 10, 2006 Yukon Zinc Corporation (Sedar; Jan 10, 2006) announced new independent N.I. 43-101 compliant mineral resource estimates for the Wolverine deposit. Measured and indicated mineral resources are now estimated at 4.52 million tonnes grading 12.04% zinc, 351.5 grams per tonne silver, 1.15% copper, 1.68 grams per tonne gold and 1.57% lead. Inferred mineral resources are individually estimated at 1.69 million tonnes grading 12.16% zinc, 385.1 grams per tonne silver, 1.23% copper, 1.71 grams per tonne gold and 1.74% lead.

The Ice property (Yukon MINFILE, 2001, 105G 118; 61°53'N, 131°21'W; NTS 105G/13) is located about 60 km east of Ross River. In 2002, the property was owned entirely by Expatriate Resources Ltd. (now renamed Yukon Zinc Corporation) It is dominantly underlain by variably strained, sub-greenschist to greenschist facies intercalated basalts, ultramafic and plutonic rocks, and ribbon cherts with associated argillite, sandstone and minor limestone, at the northwest end of the Campbell Range belt (CRB), and is the first to be discovered in this belt (Hunt, 2002). Copper-gold-cobalt-bearing massive sulphide mineralization occurs between porphyritic basalt locally interlayered with mudstone and chert and overlying hanging wall massive basalt (Hunt, 2002). The mineralization is made up of an upper massive sulphide horizon and a lower stockwork sulphide zone about 35 m below within brecciated porphyritic basalt (Hunt, 2002). The massive sulphides are dominated by relatively coarse-grained subhedral to euhedral pyrite intergrown with chalcopyrite, minor sphalerite and locally abundant bornite containing digenite in a gangue of milky white quartz and lesser calcite (Hunt, 2002). The deposit is concentrically zoned with a core of thick, high-grade copper mineralization (3.2 to 8.5% Cu over 5.7 to 28.5 m) surrounded by an apron of thinner, lower grade copper mineralization (1.5 to 3% Cu over 1 to 5 m); intersections average 0.5 g/t Au, 15 g/t Ag, 0.3% Zn and 0.08% Co (Hunt, 2002). Cobalt occurs in solid solution with pyrite (Payne, 1996). In 2002, Expatriate Resources Ltd. reported the Ice deposit contains an indicated mineral resource of 4.5 million tonnes grading 1.48% copper, including a near surface, open pittable resource of 3.4 million tonnes grading 1.48% copper of which 2.7 million tonnes are oxidized (Expatriate Resources, 2002).

It is evident from the above text that the recently discovered Fyre Lake, Kudz Ze Kayah, Wolverine and Ice VMS deposits are significant and compare favourably in size and tenor with other Canadian Cu-Zn and Zn-Pb-Cu volcanogenic massive sulphide deposits. The average Canadian Cu-Zn and Zn-Pb-Cu VMS deposits are 5.3 million tonnes grading 1.95% Cu, 4.23% Zn, 0.09% Pb, 19.0 g/t Ag and 0.8 g/t Au and 5.6 million tonnes grading 1.23% Cu, 3.6% Zn, 1.46% Pb, 79.0 g/t Ag and 2.0 g/t Au respectively (Franklin, 1996; Hunt, 2002).

After Columbia Gold Mines discovered the continuous Kona zone massive sulphide horizons in 1996 their geologists began relating the characteristics of the mineralization and its host rocks to various recognized VMS deposit models. It appeared at the time, and now, that the Kona deposit has many characteristics common to the ‘Besshi’-style of volcanogenic massive sulphide deposits. This group of deposits is commonly hosted by subequal amounts of clastic sedimentary rocks and basalt; they typically contain chalcopyrite and pyrite, with lesser sphalerite; and they are stratiform bodies with little or no evidence of stringer zones (Kanehira and Tatsumi, 1970). Their setting is commonly close to a tectonic boundary such as between an ocean floor and an island arc, an ocean floor and a craton, or an ocean floor and continental crust (Franklin *et al*, 1981). The namesake ‘Besshi’ deposits occur in the metamorphosed upper Paleozoic volcanic and sedimentary rocks of the Sambagawa Schist Group in the Shikoku and Iimori districts of Japan (Kanehira, 1970).

According to Franklin *et al* (1981),

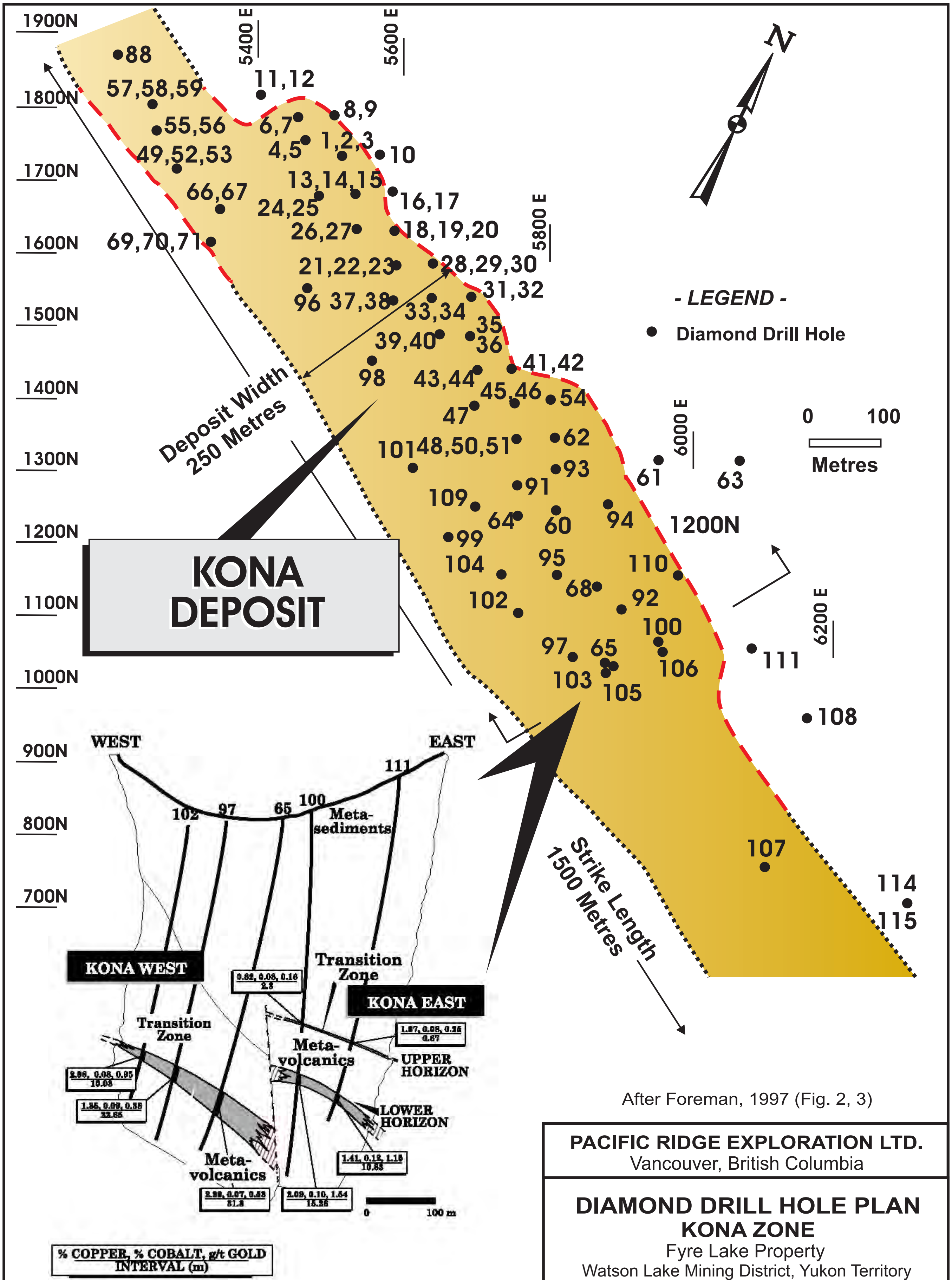
> " The (Besshi) deposits lie close to the abundant volcanic strata (greenschists) and commonly near the transition to more dominantly sedimentary, siliceous strata. Although the deposits vary in detailed stratigraphic position with the Minawa (Besshi district) and Iimori (Iimori district) Formations, the main Besshi orebodies typically lie in a central member of the Minawa Formation. The deposits have a mafic volcanic rock (greenschist) beneath the ore and are immediately overlain by piedmontite-bearing siliceous schist. Magnetite-rich layers (iron-formation) occur within the siliceous schist.
>
> The deposits consist of exceptionally elongate to tabular massive ore zones. The Besshi deposit is 1,800 m in strike length, individual ore beds attain a maximum thickness of only 10 to 20 m (Doi, 1961a). The Motoyasu orebody in the Besshi district is 1,400 m long, 100 to 180 m wide, and 0.6 to 2.5 m thick. The Iimori deposit extends for more than 7,000 m down its plunge and is 250 to 300 m wide and 0.2 to 2.8 m thick (Kanehira, 1970). In most deposits, folding locally increases the thickness of the ore.
>
> Most of the deposits are comprised of two types of ore, massive and banded sulfides. In addition, some deposits have local copper-rich, structurally thickened zones. Kanehira (1970) described the massive ores as containing pyrite, chalcopyrite, sphalerite, and bornite, with minor magnetite, and quartz and calcite as gangue minerals. The banded ore consists of pyrite, with minor chalcopyrite and sphalerite in a gangue of quartz, carbonate, albite, chlorite and minor epidote, amphiboles and tourmaline. Sulfides and gangue are present in subequal amounts. The banded ores and massive ores are mutually transitional. Locally, sulfides have been remobilized into faults and fractures, forming very copper-rich ore containing pyrrhotite (Doi, 1961a), which is otherwise quite rare in this deposit type.
>
> The average copper content of the Besshi ores is 1.40 percent (Sumitomo Metal Mining Company, 1970), with massive ores containing 3 percent copper on average. The total production to 1970 from the Besshi deposits was 28 million metric tons containing 2.46 percent Cu. The ores typically contained 0.3 to 0.9 percent Zn and 20 to 200 ppm Pb (Doi, 1961a), with Zn/Cu ratios of 0.1:0.3; a few deposits have ratios of 0.3:1.0. In the Iimori mine, Kanehira (1970) indicated grades of 1.3 percent Cu for 2.5 million metric tons of ore. In general, the Besshi-type ores contain much higher cobalt contents (1,000 ± 200 ppm) than those of the Kuroko district (Itoh, 1976), and the cobalt is contained almost entirely in pyrite. The ores are all recrystallized, fine-grained aggregates; grain size increases with metamorphic grade."

If, as thought, the Kona deposit has many characteristics of Besshi-style volcanogenic massive sulphide mineralization (see following 'Mineralization' section of this report) then future exploration should be directed at testing its characteristically long trend down plunge; the intervening favourable mineral horizons between the drilled tested near-surface mineralization and that intersected by step-out drill holes FL97-114 and -115, 450 metres to the southeast; a total drill-inferred length of 1,500 metres over an average width of 250 metres. Furthermore, drilling should test northeast and southwest of the, as yet, undelineated edges of the Kona mineralization to investigate whether there are similar buried horizons displaced by pre- and syn-depositional graben-forming fault activity.

MINERALIZATION

When the 1996 exploration program was initiated the only known mineralization on the property was that similar to the massive to semi-massive pyrite-sphalerite boulders found in the headwaters of Kona Creek. Twelve drill holes tested the areal extent of this mineralization with some success but it was not until the drilling of DDH 96-13, -14 and -15 and later DDH 96-21 that the buried massive sulphide horizons to the southeast were discovered. By the end of the 1996 field work Columbia Gold Mines personnel had identified the trend, dip and characteristics of the mineralization. However, it was not until the conclusion of the 1997 exploration season that the relationship of the sulphide- and magnetite-bearing horizons on both sides of an apparent syn-depositional fault structure had been interpreted. The following text is derived directly from Bulletin 12 (Hunt, 2002, p. 20-27) since it incorporates all of the geological observations, including those from the writer's 1996 and 1997 fieldwork (Blanchflower, 1997; Blanchflower *et al*, 1996, 1997) with recent petrographic results (Leitch, 1998).

> " The main mineralization on the Fyre Lake property is known as the Kona deposit. It is located in Kona cirque at the head of Kona creek (Fig. 9). Mineralization within this deposit occurs as two zones of massive to semi-massive sulphide and magnetite mineralization, designated East and West Kona (Figs. 9b, 10c, d, e, 12); the East Kona mineralized zone is divided into upper and lower horizons (Figs. 12 a, b; Blanchflower, 1997; Deighton and Foreman, 1997; Foreman, 1998). The zones are separated by an inferred steeply dipping reverse fault that down-drops the west side about 100 m based on the relative elevation of the metasedimentary-metavolcanic contact (Foreman, 1998). Weakly quartz-chlorite-altered schist occurs in the footwall rocks, however, it is not clear if the chlorite is due to hydrothermal alteration or metamorphism (C. Sebert, pers. comm., 1998).
>
> A mineralized zone, 1500 m long by 250 m wide, encompasses all of the massive and semi-massive Kona deposit mineralization intersected to date (Foreman, 1998). Within this zone there is an open pit target made up of the near-surface portions of the East and West Kona zones (Figs. 12 b, c), and an underground target made up of the high-grade central portions of the East Kona zone and chalcopyrite-rich sections of West Kona zone that consistently have higher cobalt and gold values (Foreman, 1998).
>
> **• East Kona zone - lower horizon**
>
> The East Kona zone lower horizon occurs over a strike length of at least 870 m and is between 100 and 150 m wide (Foreman, 1998). It has been divided into north and south portions separated by an apparent gap in the horizon (Fig. 12b). The northern portion is 3-16 m thick and the southern portion is 2-11 m thick.
>
> Composition of the northern portion varies from bottom to top (Foreman, 1998). The lower part is made up of 65 to 75% massive sulphide with 25 to 35% discontinuous, thin (average 1 m thick) massive magnetite layers. The sulphide mineralization is dominantly made up of layers of fine- to medium-grained pyrite with 3- to 6-m-thick local concentrations of chalcopyrite and pyrrhotite which occur as 2- to 10-cm-thick bands. The upper 0.5 to 1.5 m of the sulphide mineralization is predominantly made up of pyrite with 2 to 6% sphalerite, locally concentrated into 1- to 2-cm-thick bands. The core consists of massive, fine-grained, magnetite-rich layers with about 5% pyrite + chalcopyrite, in a carbonate and/or quartz groundmass. The upper part is predominantly massive, fine- to medium-grained pyrite with 3 to 5% chalcopyrite.
>
> The southern portion of the East Kona zone lower horizon (Fig. 12b) also varies in composition from top to bottom. It is similar in appearance to the northern portion, except that locally the lower sulphide portion contains 0.5- to 3-m-thick layers of banded semi-massive rather than massive) sulphide mineralization, and disseminated to semi-massive banded magnetite, rather than massive magnetite, overlies the sulphide mineralization (Foreman, 1998).



After Foreman, 1997 (Fig. 2, 3)

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

DIAMOND DRILL HOLE PLAN
KONA ZONE
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 6

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Figure 7
Kona Zone Photographs

Figure 7: a) Massive quartz-pyrite boulder (discovery showing) in Kona Creek; b) folded carbonaceous phyllite exposed in Kona Creek (Hunt, 2002); c) and d) massive sulphide mineralization in drill core from the East Kona zone (Hunt, 2002); e) siliceous sulphide mineralization in drill core from the West Kona zone (Hunt, 2002); f) fractured pyrite with chalcopyrite, pyrrhotite and sphalerite, reflected light + plane polarized light, field of view = 1.7 mm (VanRanden, 1997, Leitch, 1998).

Petrography

Specimens of schist and massive sulphide mineralization from the East Kona zone lower horizon were examined by VanRanden (1997) and Leitch (1998). The results, given in detail in Appendix VI-3, are summarized below.

The immediate footwall to the lower horizon is made up of alternating foliae of chlorite and quartz plus minor carbonate, sphene and possibly hydrobiotite, and contains magnetite euhedral up to 3.5 mm across. Away from the lower horizon (Appendix VI-3: specimen DDH FL96-33-76.6 m), the footwall composition changes to fibrous amphibole in a matrix of possible plagioclase and quartz with lesser biotite and minor opaques including euhedral magnetite up to 0.25 mm across.

Lower horizon massive sulphide mineralization (Appendix VI-3: specimens DDH FL96-33-70.8 m, 71.48 m, 72.5 m, 72.9 m and 73.1 m, DDH FL96-34-71.8 m, 72.3 m and 73.45 m) is primarily composed of pyrite, chalcopyrite, sphalerite and lesser pyrrhotite (Fig. 10f), with minor disseminated magnetite occurring in magnetite-rich layers within a gangue of quartz-chlorite and lesser carbonate, sericite and rare amphibole. The lower horizon is locally cut by veinlets of carbonate-quartz-epidote. Green garnet porphyroblasts are also present locally and are partly replaced/pseudomorphed by carbonate-chlorite-sericite; some are sieve-textured due to inclusions of pyrite, and possibly magnetite and sphalerite. Magnetite porphyroblasts, present in only one specimen from the lower horizon, contain exsolution lamellae of hematite (Fig. 10g). In this specimen there are also rare, zoned pyrite crystals. Locally, the sulphides are strongly recrystallized.

The immediate hanging wall to the lower horizon (Appendix VI-3: specimen DDH FL96-33-70.5m) is made up of magnesian chlorite with alternating foliae of ferroan carbonate ± sphene and quartz-(?)plagioclase, similar to the immediate footwall.

Schists separating the upper and lower horizons (Appendix VI-3: specimens DDH FL96-33-47.6 m, 57.87 m and 59.25 m) are foliated, crenulated and locally isoclinally folded. They are made up of amphibole (possibly actinolite) and chlorite, with lesser quartz and feldspar; they locally contain biotite porphyroblasts 2-3 mm in diameter and scattered euhedral magnetite crystals up to 1 mm across. The presence of amphibole, biotite, and possibly magnesian chlorite may indicate an approach to amphibolite facies metamorphism of a mafic rock such as basalt or gabbro/diorite.

• East Kona zone - upper horizon

East Kona zone upper horizon mineralization (Figs. 12a, b) occurs above the lower horizon and is separated from it by approximately 40 to 70 m of chlorite schist. The upper horizon occurs immediately below the contact between overlying metasedimentary (unit MKcp/Unit 3) and underlying metavolcanic (unit DMF/Unit 2) strata. The base of the upper horizon is evident in Kona Creek as boxwork-textured, siliceous grey to white boulders/subcrop (Figs. 13a, b). This horizon has a strike length of at least 630 m, is between 100 and 150 m wide (Foreman, 1998), and has average thicknesses of 8 to 12 m (W. Roberts, pers. comm. 1997; Deighton and Foreman, 1997); the central portion is the thickest part (Foreman, 1998). The upper horizon is fairly consistent throughout and has been divided (Foreman, 1998) into lower, middle and upper layers as below.

The lower layer is an average of 7 m thick (maximum 17 m) and is made up dominantly of metavolcanic rocks and magnetite; the sulphide content is below 10%. Throughout the lower layer finer-grained (< 1 mm) magnetite is concentrated into 1- to 10-mm-thick bands, and occurs within 2- to 20-mm-thick grey siliceous bands. The sulphides in the lower layer occur predominantly as < 1- to 4-mm-long irregular wisps and blebs. The lower layer is overlain by a 3- to 8-m-thick middle layer made up of 1- to 25-cm-thick bands of sulphides and quartz within foliated dark green metavolcanic strata. The middle layer contains 30 to 60% sulphides, dominantly made up of chalcopyrite, with lesser pyrite and pyrrhotite occurring as irregular wisps and blebs. Subhedral to euhedral magnetite porphyroblasts occur throughout the surrounding metavolcanic rocks. The middle layer is overlain by a 1- to 4-m-thick upper layer made up primarily of massive, fine- to medium-grained pyrite, with 2 to 7% very fine-grained chalcopyrite and minor pyrrhotite and sphalerite.

Mineralization of the upper horizon changes to the southeast (down-plunge) where it is dominated by bands of pyrrhotite with 1 to 10% chalcopyrite in chlorite-quartz schist. In addition, the banded magnetite that underlies the upper horizon thickens locally to the southeast to a maximum of 24 m.

Petrography

Specimens of schist and massive sulphide mineralization from the East Kona zone upper horizon were examined by VanRanden (1997) and Leitch (1998; Appendix VI-3). The results are similar to those for the lower horizon and are summarized below.

An examination of specimens from two drill holes (Appendix VI-3: specimens DDH FL96-33-20.15 m and 21.38 m, FL96-34-17.6 m, 24.1 m, 20.85 m and 26.65 m) shows that in general, the three mineralized layers that make up the upper horizon are composed of pyrite, chalcopyrite, sphalerite, and locally, pyrrhotite (Fig 13c), in a gangue of quartz-chlorite with lesser carbonate (locally ferroan) and sericite. Specimen 17.6 m contains a mix of pyrite and marcasite after pyrrhotite with “bird’s eye” textures characteristic of such replacement (Leitch, 1998). Most specimens show evidence of recrystallization, however, possible relict colloform structures appear to be preserved in chalcopyrite in specimen 20.85 m.

Magnetite porphyroblasts and lesser fine-grained, anhedral, disseminated magnetite are present throughout the horizon. Garnet crystalloblasts are present locally (Fig. 13d) within chlorite schist and are partly replaced/pseudomorphed by carbonate-chlorite ± hydrobiotite (Appendix VI-3: specimens DDH FL 96-34-17.6 m, 24.1 m and 26.65 m). The lower layer of the upper horizon has an increased metavolcanic content compared to the rest of the horizon and contains strongly foliated, locally kink-banded, schistose rocks (Appendix VI-3: specimens DDH FL96-33-25.8 m and 29.2 m). These rocks include chlorite schist with porphyroblasts (probably plagioclase) and carbonate pseudomorphs of former mafic crystals, and quartz-chlorite-pyrite-magnetite-minor ferroan carbonate schist with partly carbonate-sericite altered feldspar crystals and possibly relict carbonate and ferriferous biotite-altered mafic crystals. The presence of plagioclase porphyroblasts and mafic crystals suggests that at least some of the chlorite schists had an intermediate volcanic rock precursor.

All sulphide phases in the East Kona zone upper and lower horizons appear to be roughly coeval and most textures likely represent remobilization (Leitch, 1998). However, relict primary textures appear to be preserved locally. For example, there are possible relict colloform, atoll and radiating cockscomb textures in specimens DDH FL96-33-72.5 m, 73.1 m and DDH FL96-33-21.38 m) could represent primary intergrowths of these minerals. Much of the recrystallization of pyrite and pyrrhotite from original fine-grained aggregates (e.g., Appendix VI-3: specimens DDH FL96-33-72.9 m & DDH FL96-34-20.85 m) could have occurred during ongoing hydrothermal activity at the time of sulphide deposition (Leitch, 1981a, b, 1998).

• West Kona zone

West Kona zone mineralization occurs immediately below the metasediment-metavolcanic contact at the same stratigraphic level as East Kona zone upper horizon mineralization, but is separated from it by a reverse fault (Fig. 12a). The West Kona zone has a strike length of at least 1420 m and an inferred width of 75 to 125 m (Foreman, 1998). The thickness of the mineralization varies across this width from about 44 m in the east to less than 1 m at the western margin; the thickness also varies along strike.

West Kona zone mineralization is markedly different from that of the East Kona zone in that it has dominantly siliceous gangue minerals. Greater than 80% of the West Kona zone is made up of siliceous, dominantly fine-grained, disseminated to banded magnetite, with lesser pyrite, chalcopyrite and pyrrhotite mineralization (Fig. 10e). However, it does change laterally to the west to become true massive sulphide mineralization (Foreman, 1998). In the western part of the West Kona zone, the percentage of sulphides within the zone increases to >80% and the mineralization is dominantly made up of fine- to medium-grained pyrite, with lesser fine-grained interstitial chalcopyrite and minor sphalerite, and a noticeable lack of pyrrhotite. The massive sulphides contain 1-10% quartz as blebs. At its western margin the West Kona zone is less than 1 m thick and is composed dominantly of massive pyrrhotite with about 5% blebs and fracture fillings of pyrite and chalcopyrite."

EXPLORATION DRILLING

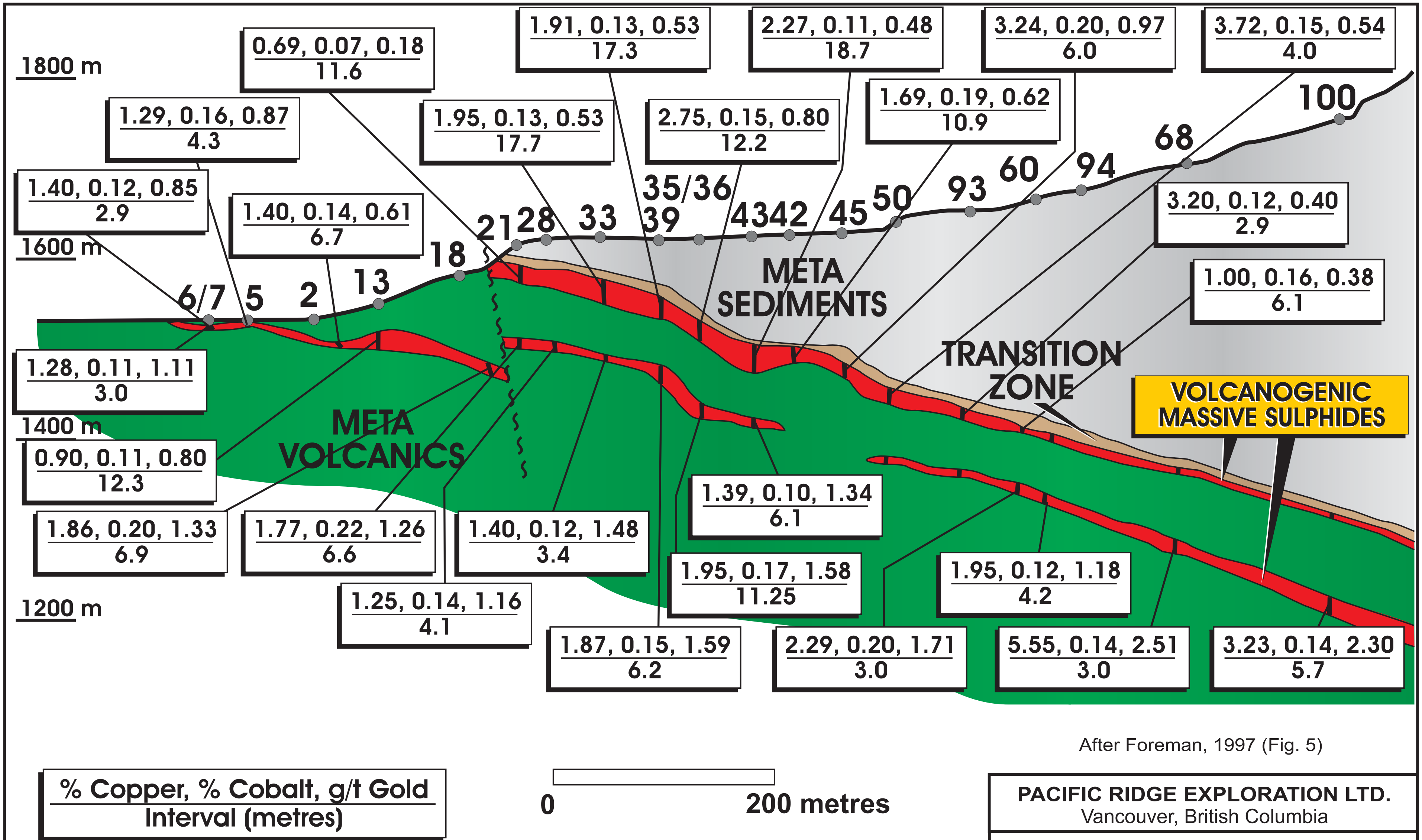
Columbia Gold Mines Ltd. completed 116 holes totalling 23,266.91 m or 76,332 ft. of NQ2- and BQTK-core diamond drilling during the 1996 and 1997 field seasons (Blanchflower, 1997; Deighton and Foreman, 1997). With the nine AX-core diamond drill holes completed by Atlas Explorations Ltd. in 1966 and 1967 (Sadlier-Brown, 1966), the original Fyre Lake property (circa 1997) has been tested by 125 holes totalling 24,113 m or 79,111 ft. of NQ2-, BQTK- and AX-core diamond drilling. This total does not include the 23 shallow packsack drill holes and 12 AX-core diamond drill holes, totalling 582 metres, completed by Cassiar Asbestos Corporation in 1961 because the drill sites and results are poorly documented (see 'History' section of this report).

Most of drill holes that tested the property in 1997 were drilled along 1.5 kilometres of an inferred 3-kilometre long trend. However, only four (i.e. DDH 97-74, -75, -76 and -79) of the eleven drill holes that tested the 'Lake' zone are now covered by currently valid mineral claims. The other seven drill sites were located on mineral claims that were allowed to lapse prior to this report.

J. T. Thomas Diamond Drilling Ltd. of Smithers, British Columbia was contracted in 1996 and 1997 to provide drilling equipment and personnel capable of recovering NQ2- and BQ-size diamond drill core. All of the drilling equipment was designed for a helicopter-supported program. This equipment was initially trucked from Smithers B.C. to the Finlayson Lake airstrip where the drills and support equipment were delivered directly to the first drill sites by helicopter. After the 1996 field work, the drilling equipment and supplies were stored at the campsite where they were readily available for the 1997 exploration program. Upon conclusion of the 1997 drilling, one drill rig and support equipment were again stored at the campsite until they were demobilized during the 1999 reclamation work.

During the 1996 and 1997 exploration programs holes were drilled to intersect the targeted mineralization orthogonally. However, due to structural complexity, drill stem deviation and multiple massive sulphide horizons the drill holes often intersected the reported mineralization at angles varying from 60^o to 90^o to core axis. Thus, any reported drilling intercepts herein are assay interval weighted averages over an apparent length and not true thickness weighted averages. The Kona zone mineralization trends at 130^o and plunges from -0^o to -20^o southeastwardly. Individual horizons dip from -5^o to -35^o northeastward. The individual massive sulphide- and magnetite-bearing horizons comprising the Kona zone have been discussed in the 'Mineralization' section of this report.

See Figure 6 for the locations of the 1996 and 1997 diamond drill sites within the Kona VMS deposit. Figures 8 and 9 are longitudinal sections through the East and West Kona zones illustrating the diamond drilling results for this area. Furthermore, Appendix I of this report contains a summary of the 1966 to 1997 diamond drilling data; including each drill hole with its U.T.M. location, collar elevation, drilled length, azimuth, dip and a summary of any mineralized intercepts with weighted average grades for copper, cobalt, gold and zinc (after Blanchflower, 1997; Pacific Ridge, 2002).



After Foreman, 1997 (Fig. 5)

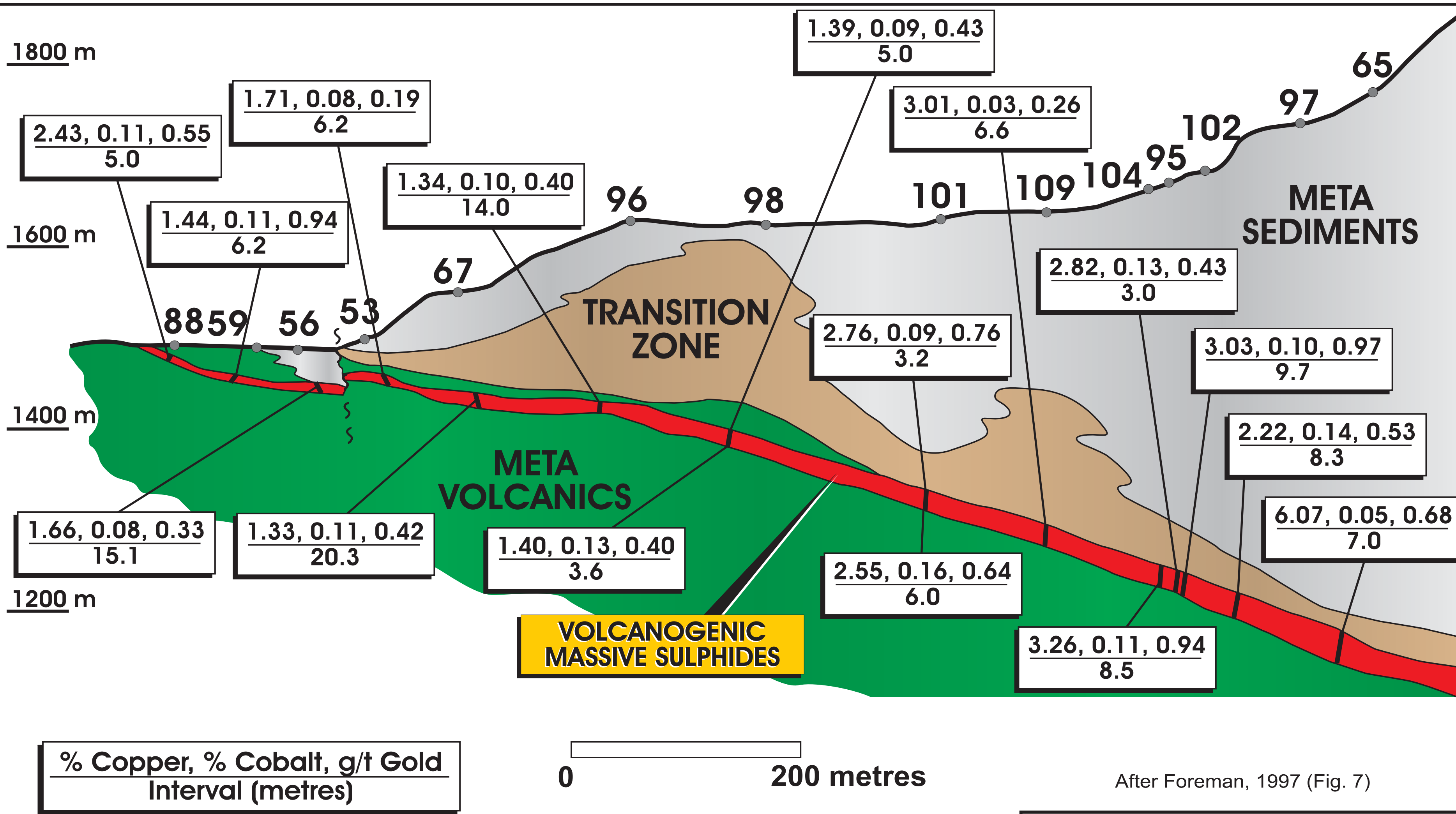
PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

LONGITUDINAL SECTION
EAST KONA ZONE

Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 8

To Accompany Report By J. D. Blanchflower, P. Geo., 2006



After Foreman, 1997 (Fig. 7)

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

LONGITUDINAL SECTION
WEST KONA ZONE
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 9

To Accompany Report By J. D. Blanchflower, P. Geo., 2006

SAMPLING METHODOLOGY

In !996 the writer and Columbia Gold Mines Ltd. established a strict set of sample handling, processing and security procedures prior to the commencement of diamond drilling. These procedures were adhered to during the entire 1996 and 1997 drilling programs.

Sampling Methods and Approach

At the drill rigs the filled wooden drill core boxes were sealed by the drillers and flown directly to the core processing, logging and sampling facilities at the Fyre Lake campsite. There, the footage markers were converted to metric measurements and each box was labelled with a weather-proof aluminium tag inscribed with its respective hole number, box number and drilling length interval. The core was then logged in detail by qualified geologists utilizing a ‘matrix’ coding log form. Geological data was then inputted on site into a computerized database for both documentation and computer-assisted drafting (CAD). Core recovery, rock quality and specific gravity measurements were also logged and recorded. Core recoveries were generally good to excellent (i.e. > 95% recovery by rock volume); except in extremely fractured near-surface rock or wide shear structures. Specific gravity measurements were recorded at 8-metre intervals, except in mineralized zones where measurements were recorded at 1-metre intervals. All of the drill core was photographed prior to splitting and sampling.

Designated sections of mineralization were split in half lengthways using a Longyear manual splitter and sampled between labelled sections. The sampling interval varied for marginally mineralized material but within the mineralized horizons the sampling interval was usually less than or equal to 1 metre long. One-metre samples were collected immediately above and below any massive and/or semi-massive mineralization to detect lithogeochemical signatures for the mineralized horizons. All of the split core samples were labelled, double-bagged, sealed with double wire twists ties and put in wire-sealed fibreglass ‘rice’ bags. The now-triple bagged and sealed samples were then flown directly by fixed-wing aircraft to Watson Lake, Y.T.

At Watson Lake, the project expediter loaded the sealed samples onto a dedicated van and shipped them directly to either the preparation facilities of Min-En Laboratories in Smithers, B.C. or to Chemex Laboratories Ltd. in North Vancouver, B.C. The remaining one-half of the split core was replaced in its original position in its core box and is stored at the core logging and storage facilities at the Fyre Lake campsite on the property. The 1966 drill core was also stored on the property but the old core racks, located approximately 1 kilometre northwest of the Kona Zone, have since rotted and collapsed, and this core is now a large mixed pile.

Lithogeochemical samples of the drill core were commonly collected at 8-metre intervals or less where the country rock lithologies varied but these 15 to 20 cm long samples were not collected from any previously-sampled assay sections. All of the lithogeochemical samples were double-bagged, labelled and shipped, like the above assay samples, to Min-En Laboratories Ltd. in Smithers, B.C. for preparation. Their resultant pulps were then delivered directly under Min-En Laboratories supervision to their laboratories facilities in Vancouver, B.C. for 31-element I.C.P. analyses. The locations of the lithogeochemical samples within the various core boxes were marked with stapled aluminium tags imprinted with “ICP”.

Sample Preparation, Analyses and Security

During the 1996 and 1997 exploration programs Columbia Gold Mines Ltd.

- collected 1,977 mineralized drill core samples for assaying;
- collected 2,262 drill core samples for lithogeochemical studies;
- collected 19 magnetite-rich drill core samples for platinum group element assaying;
- collected 20 drill core samples for whole-rock analyses;
- collected 56 surface rock samples for assaying, 31-element I.C.P. analyses and/or lithogeochemical studies;

- collected 3,265 ‘B’ horizon soil geochemical samples for 31-element I.C.P. analyses;
- collected 181 silt geochemical samples for 31-element I.C.P. analyses, and
- check-assayed 120 mineralized drill core samples (~10% of total mineralized drill core samples).

The mineralized drill core samples were delivered to either Min-En Laboratories in Smithers or to Chemex Laboratories in Vancouver. Initial drill core samples were assayed for copper, lead, zinc, silver and gold but, due to low lead and silver values, later drill core samples were assayed for copper, cobalt, zinc and gold plus 31-element I.C.P. analysis. Geochemical analyses were carried out on drill core samples that the geologist had estimated contained less than 0.50 % copper. If a sample returned a value greater than 10,000 ppm copper, the sample was assayed for its copper content in percent.

In Smithers, Min-En Laboratories’ personnel dried each sample at 65^{o} C before crushing it to minus 1/4 inch. The crushed sample was then reduced to minus 1/8 inch size by a secondary roll crusher. The whole sample was then split on a Jones Riffle to a statistically-representative 300-gram sample pulp. This sample pulp was then pulverized in a ring pulverizer to 95 percent minus 150 mesh, rolled and bagged. All of the sample pulps were then shipped to the Min-En Laboratories facility in Vancouver for assay. The remaining coarse rejects from the Jones Riffle were bagged, catalogued, placed on pallets, covered with plastic and stored in a warehouse in Smithers, B.C.

The drill core samples that were delivered directly to Chemex Laboratories in North Vancouver, B.C. were similarly processed as those samples delivered to Min-En Labs in Smithers. The pulps and coarse rejects that were stored at Chemex Laboratories facilities in North Vancouver, B.C. after the initial assay or analytical procedures were moved to Min-En Laboratories in Vancouver.

At Min-En Laboratories the fire assay procedures for copper, cobalt, zinc, silver and gold utilize a 0.500 to 2.00 gram subsample which is weighed from the sample pulp for analysis. Each batch of 70 assays has a natural standard and a reagent blank included. The samples are digested using a HNO_3 - $KClO_3$ mixture and when the reaction subsides HCl is added before it is placed on a hotplate to digest. After digestion is complete the flasks are cooled, diluted to volume and mixed. The resulting solutions are analyzed on an atomic absorption spectrometer using the appropriate standard sets. The natural standard digested along with this set must be within 2 standard deviations of its known or the whole set is re-assayed. If any of the assays are more than 1 percent copper they are re-assayed at a lower weight, and 10 percent of the submitted samples are assayed in duplicate (Min-En Laboratories, 1995).

The drill core geochemical samples were dried at 65^{o} C, and then they were crushed by a jaw crusher and pulverized by a ceramic-plated pulverizer or ring mill pulverizer. The resultant sample was rolled and sieved to obtain a minus 80-mesh pulp for analysis. A 0.5 gram subsample was digested for 2 hours with an aqua regia mixture and, after cooling, the solution was diluted to standard volume. The resultant solution was then analyzed for its copper, zinc, lead, silver content by atomic absorption methods. The copper, zinc, silver values are quoted as parts per million (“ppm”).

Gold fire assays were undertaken on drill core samples and any ‘blind’ duplicate samples. All gold fire assay procedures at Min-En Laboratories were conducted using one assay ton sample weights. The subsamples were fluxed and a silver inquart was added and mixed. These subsamples were fluxed in batches of 24 assays with a natural standard and a blank. This batch of 26 assays was carried through the whole procedure as described. After cupellation the precious metal beads were transferred into new glassware, dissolved with aqua regia solution, and diluted to volume and mixed. The resulting solutions were analyzed on an atomic absorption spectrometer using a suitable standard set. The natural standard fused along with this set must be within 2 standard deviations of its known or the whole set is re-assayed. Likewise, the blank assay must be less than 0.015 g.p.T. The top 10 percent of all assays per printed page were re-checked and reported along with the standard and blank. Gold values are reported in grams per tonne (“gpT”) with a detection limit of 0.02 gpT.

Other lower copper-content drill core samples were geochemically-analyzed for their gold contents. A 10.0-gram portion from each subsample was weighed and placed into a porcelain crucible, and cindered at 800^{o} C. for 3 hours. All of the subsamples were then transferred to beakers and digested using aqua regia, diluted to volume and mixed. Seventy-five percent of each of the diluted samples was further oxidized, treated and extracted for gold analyses using methyl iso-butyl ketone (“MIBK”). The MIBK solutions were then analyzed by an atomic absorption spectrometer (“AA”) using a suitable standard set and the values of gold (Au) were then reported. Gold values are reported in parts per billion (“ppb”) with a detection limit of 1 ppb.

Nineteen drill core samples were assayed for platinum and palladium at Min-En Laboratories in Vancouver using similar techniques as the above-described gold geochemical analytical procedures.

Chemex Labs Ltd. assayed drill core samples for copper, lead and zinc using similar procedures as Min-En Laboratories Ltd. After processing, the 0.4-gram subsample is digested in a hot nitric-hydrochloric acid mixture, cooled and then transferred to a 100 ml volumetric flask. The final matrix is 25% hydrochloric acid. The solutions are then analyzed on an atomic absorption instrument. Copper, zinc and lead values are reported in percent (“%”) with a universal detection limit of 0.01%.

Silver assays at Chemex Labs Ltd. use a 2-gram subsample which is digested in concentrated nitric acid for one hour. After cooling, hydrochloric acid is added to produce aqua regia and the mixture is digested for an additional hour. The resulting solution is transferred to a volumetric flask, made up to volume and analyzed using atomic absorption techniques with background corrections. Values are reported in grams per tonne.

Chemex Labs Ltd. utilizes standard fire assay techniques for gold assays. A prepared 1 assay ton subsample, weighing 29.166 grams, is fused with a neutral flux inquated with 5 mg of gold-free silver and then cupelled. The silver beads are digested for 1/2 hour in 1 ml diluted 75% nitric acid, then 3 ml of hydrochloric acid is added and digested for 1 hour. The samples are cooled and made to a volume of 10 ml, homogenized and analyzed by atomic absorption spectroscopy. Any samples over 0.4 ounces per ton are re-assayed using a gravimetric finish, and the gravimetrically-determined gold content is substituted into the certificate of analysis. The detection limit is 0.03 gpT gold.

The geochemical procedures for copper, lead, zinc and silver at Chemex Labs Ltd. are similar to those conducted at Min-En Laboratories Ltd. A prepared 1-gram subsample is digested in nitric aqua-regia acids for two hours. The digested solution is cooled and diluted to 25 ml with de-mineralized water. The resulting solution is mixed and the solids allowed to precipitate. The metals are determined using atomic absorption spectroscopy, and background corrections are applied for lead and silver.

Chemex Labs Ltd. uses the following procedure for both gold fire assays and gold atomic absorption analyses. A 10-gram subsample is fused with a neutral lead oxide flux, inquated with 6 mg of gold-free silver, and then cupelled to yield a precious metal bead. These beads are digested for 30 minutes in 0.5 ml concentrated nitric acid, and then 1.5 ml of concentrated hydrochloric acid is added, and the mixture is digested for 1 hour. The solutions are cooled, diluted to a final volume of 5 ml, homogenized and analyzed using atomic absorption spectroscopy.

After the initial assaying work, mineralized drill core samples were selected for trace element geochemical analyses using 31-element induced plasma coupled (“ICP”) techniques. These samples were selected to characterize the geochemistry of the various mineralized horizons. The ICP procedures at Min-En Laboratories Ltd. have been described previously.

Data Verification

Both Min-En Laboratories and Chemex Labs Ltd. carry out standard ‘in-house’ assay verification both on inserted prepared blanks and all ‘over assay limit’ samples. Columbia Gold Mines Ltd. also insisted on re-assaying any notably high grade samples prior to reporting. In addition, approximately ten percent of all mineralized drill core samples were selected by geologists from a range of low, medium and high tenor samples, and the sample pulps were shipped to Acme Analytical Labs Ltd. in Vancouver, B.C. for copper, cobalt, lead, zinc, silver and gold check-assays. As stated, 120 mineralized drill core samples (~10% of total drill core samples) were check-assayed during the two-year period.

The check-assay results usually compared well with the original results from either Min-En Laboratories and Chemex Labs Ltd., within an acceptable range given slightly different assaying procedures. If any of the check-assay results were quite different from the original results then an inquiry was made by the supervising geologist and, if necessary, a cut of the sample pulp was sent to either Min-En Laboratories or Chemex Labs Ltd., whichever one that was not the original assayer, and a second-check assay was done using similar procedures as the original assay. There were procedures in place that should the second check-assay result not be compare favourably with the proceeding results then a new sample was to be split from the original halved core and the assay process was to be repeated. To the writer's recollection, this repeat sampling procedure was never utilized during the drilling programs.

In addition to the above sample verification procedures, as the drill core from each hole arrived at the logging facility a geologist was assigned responsibility for its logging and sampling. While the logging data was being inputted the data was checked and later the geologist re-checked all data including: surface and downhole surveying, geological logging observations, specific gravity measurements, etc. Furthermore, all compiled drilling, geological and assay data was cross-checked again before the data was collated in the program's drilling database.

METALLURGICAL TESTING

Columbia Gold Mines Ltd. initiated preliminary metallurgical testing following the successful 1996 drilling program. In January, 1997 Columbia Gold Mines contracted Westcoast Mineral Testing Inc. to perform preliminary flotation testing on a single composite massive sulphide sample which was prepared by Min-En Laboratories from assay rejects (Hawthorn, 1997) . Five tests were performed to determine the amenability of the composite sample material to the production of a saleable copper concentrate with potential gold and cobalt values. The results indicated that the composite material responded to flotation producing a >25% copper concentrate containing perhaps 4 to 6 g/t gold (Hawthorn, 1997). Furthermore, fine to very fine grinding would be required at some stage in the processing to adequately liberate the chalcopyrite from the pyrite. The I.C.P. analyses of various test products indicated that the composite material was mineralogically quite 'clean' other than 'minor' sphalerite which resulted in the copper concentrate grading about 1.5 to 2.0% zinc (Hawthorn, 1997). Following these initial test results, Columbia Gold Mines decided to proceed with more detailed metallurgical studies following the recommendations of Hawthorn (1997).

Melis Engineering Ltd. and Lakefield Research Limited were contracted in late January, 1997 (Pacific Ridge, 2002) to undertake metallurgical testwork to investigate flotation recovery and deportment of copper, cobalt, zinc, gold and silver in the flotation products. Key flotation parameters for the Fyre Lake mineralization would also be identified. This work was initially conducted on a single blended composite sample prepared from sample rejects and weighing 50 kilograms. The composite sample had a drill calculated head grade of 2.18% copper, 0.145% cobalt, 0.94% zinc, 1.04 g/t gold and 4.11 g/t silver; and an assay head grade of 2.07% copper, 0.16% cobalt, 0.99% zinc, 0.98 g/t gold, 3.7 g/t silver, 26.1% iron and 20.5% sulphur (Melis Engineering, 1997).

Nine flotation scoping tests were completed by Melis Engineering Ltd. and Lakefield Research Limited (1997) during the initial metallurgical testwork. The results of this work indicated target recoveries of 90% for copper and 70% for gold, into a concentrate representing 7.5% of the feed and assaying 21% to 23% copper, 10 g gold/tonne and 0.08% cobalt, would be achieved from a mill feed grade of approximately 2% copper and 1.2 g gold/tonne (Melis Engineering, 1997). Furthermore, cobalt recovery to a pyrite concentrate assaying approximately 0.33% cobalt would be approximately 50% to 75% (Melis Engineering, 1997).

Three scoping leach tests were also conducted by Melis Engineering Ltd. and Lakefield Research Limited on cobalt-bearing pyrite concentrate produced in the ninth flotation test (see above text). This pyrite concentrate assayed 0.34% cobalt, 0.35% copper, 41.6% iron and 0.62 g gold/tonne (Melis Engineering, 1997). Results of this work indicated a 70% cobalt recovery in flotation and 95% recovery from pressure leaching with an overall cobalt recovery of possibly 65 to 70% (Melis Engineering, 1997).

In September, 1997 Lakefield Research Limited conducted mineralogical examinations on one composite head sample (Fyre Lake Test Composite Head) and one gravity concentrate sample (Mozley Concentrate MP128-3) provided by Melis Engineering Ltd. on behalf of Columbia Gold Mines Ltd. The two samples of Fyre Lake Kona deposit mineralization were found to be mainly composed of iron sulphides (pyrite). Minor to trace amounts of sulphides (chalcopyrite, sphalerite, pyrrhotite, chalcocite and covellite) and oxides (magnetite and goethite) were also observed (Lakefield Research, 1997a). According to Lakefield Research (1997a, p. 2),

> " No cobalt minerals were identified during microscopic examination of the two samples. Electron microprobe analysis of pyrite (23 quantitative microprobe analyses) and sphalerite (4 quantitative microprobe analyses) grains detected cobalt in the Fyre Lake Test Composite Head. Cobalt in pyrite ranges from a minimum of 800 ppm (0.08%) to a maximum of 4.9% (one grain) whereas cobalt in sphalerite ranges from a minimum of 470 ppm (0.047%) to a maximum of 0.36%. Pyrite is identified as the main cobalt carrier.
>
> Microscopic gold scans were carried out on the two samples. No visible gold was observed in either sample. Ion microprobe analyses (Secondary Ion Mass Spectrometry – SIMS) confirmed the occurrence of "invisible" gold in all the pyrite grains analysed from the Fyre Lake Test Composite Head. The "invisible" gold in the analyzed pyrite grains occurs more likely as sub-microscopic and/or colloidal gold (< 0.1 um) instead of gold in solid solution.
>
> Copper carriers were identified as chalcopyrite, chalcocite and covellite, and the zinc carrier identified is sphalerite."

The results of the more recent and detailed metallurgical testwork by Melis Engineering Ltd. and Lakefield Research Limited accompany with this report in Appendix II.

During the 1996 and 1997 drilling programs, Columbia Gold Mines geologists measured and recorded 3,828 specific gravity readings from the host rocks and Kona deposit mineralization (see History and Sample Preparation sections). These specific gravity measurements were included in the drilling database and utilized during subsequent mineral resource studies. In summary, the median specific gravities for the six modelled types of Kona mineralization (see Figure 10) are as follows:

Mineralized Kona Deposit Horizon	**3D Solid No.**	**Specific Gravity (g/cc)**
Upper East Kona zone massive sulphide mineralization	100	3.201
Lower East Kona zone massive sulphide mineralization	300	3.668
Upper West Kona zone magnetite (±sulphide) mineralization	400	3.348
Lower West Kona zone massive sulphide mineralization	500	3.447
Lower West Kona zone magnetite (±sulphide) mineralization	600 & 700	2.818

MINERAL RESOURCE ESTIMATES

Qualified Person

The mineral resource estimates contained herein were calculated in August, 2002 by the writer and recently reviewed in January, 2006. The writer was at the time and is presently an independent consulting geologist and president of Minorex Consulting Ltd. He has extensive experience estimating mineral resources, and has completed resource estimates on a variety of mineral deposit types. At the time and now, the writer is a 'Qualified Person', as defined by N.I. 43-101, and did not and does not now hold or expect to hold any interest, directly or indirectly, in the subject property, any adjacent properties, or common stock or options of Pacific Ridge Exploration Ltd.. See 'Statement of Qualifications' section of this report for additional information.

Assumptions, Parameters and Methodology

In August 2002, the writer thoroughly reviewed all drilling, geological and assay data contained in the Fyre Lake database (Pacific Ridge, 2002) which was originally compiled and verified by the writer during his work for Columbia Gold Mines Ltd. in November and December, 1997. This database utilizes the drilling data and assay results reported from the 1966 and 1996-97 exploration programs conducted by Atlas Explorations Ltd. (Sadlier-Brown, 1966) and Columbia Gold Mines Ltd. (Blanchflower, 1997; Deighton and Foreman, 1997) respectively. It contains survey, drilling, geological and assay information from:

- 121 holes totalling 24,113 m or 79,111 ft. of NQ2-, BQTK- and AX-core diamond drilling;
- 427 surface and downhole survey points;
- 1,816 drill core samples with assay results;
- 1,733 geological logging and core recovery observations; and
- 3,828 specific gravity measurements.

No errors in data entry or other information were discovered upon review. Digital topographic data was obtained from Pacific Ridge Exploration (2002). The digital topographic data was specifically surveyed, recorded and compiled on behalf of Columbia Gold Mines Ltd. and supplemented with 1996 and 1997 ground distamat and Global Positioning System ("GPS") surveying information.

Using Gemcom software a solid three-dimensional ("3D") geologic model was first constructed by joining correlatable horizons of similar mineralogy using 3D polylines. These correlations were completed on 25-metre spaced sections with orientations of 070^{o} (True North; 071.54^{o} UTM). Once completed, the sectional polylines were joined together to form a 3D solid body model of that specific mineralization. A total of six distinct 3D solids were created, each representing specific mineralogy.

These solids included:

- Lens 100 (Upper East Kona zone massive sulphide mineralization);
- Lens 300 (Lower East Kona zone massive sulphide mineralization);
- Lens 400 (Upper West Kona zone magnetite (±sulphide) mineralization);
- Lens 500 (Lower West Kona zone massive sulphide mineralization); and
- Lenses 600 and 700 (Lower West Kona zone magnetite (±sulphide) mineralization at the northwestern and southeastern ends of the deposit respectively).

Each lens is hosted by a distinct stratigraphy, is mineralogically distinct, and has a distinct median specific gravity (see 'Metallurgical Testing' section of this report). Once the 3D solid models were completed and verified drill hole assay and specific gravity data were composited to 1-metre intervals inside the respective solids. The 3D geological solids were also utilized to create rock type and percent models for the resource estimate. See Figure 10 of this report illustrating the six 3D solids and their locations with respect to the topography.

A geostatistical study of the assay composites was conducted utilizing variograms generated at various orientations. It was determined that there was insufficient composite data to create individual block models for each lens. However, the grades and specific gravities of individual blocks within a larger, common block model could be readily interpolated using a common anisotropic 'azimuth, dip, azimuth' search ellipsoid with a long-azimuth search distance of 125 metres.

The Kona deposit (approximately 1,500 m long by 250 m wide trending at 130^{o} and plunging 0^{o} to -20^{o} southeastwardly) was block modelled utilizing a rotated (18.46^{o} west of UTM north) model measuring 1,400 m wide by 1,800 m long by 600 m deep (1,600 to 1,000 m AMSL). Individual blocks are 4-metre cubes.

The grades and specific gravities of individual blocks were interpolated using an 'ordinary kriging' method with a 100-metre search radius. A 100-metre search radius was deemed appropriate, given the geostatistical results, since it was a sufficiently conservative distance to interpolate between tested drill sections without projecting grades and specific gravities over unrealistic distances. Furthermore, kriging parameters included a number of restrictions, such as only two assay composites from each hole, to force interpolation along the relatively thin, elongated mineralized horizons.

The estimates of mineral resources for the Kona volcanogenic massive sulphide deposit were conducted in accordance with National Instrument 43-101, and the classification of indicated and inferred resources as defined by the Canadian Institute of Mining and Metallurgy 'CIM Definition Standards on Mineral Resources and Mineral Reserves" (revised November 1, 2004, p.4-5). These classifications state that

> ***"An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes."***

and

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

Once the rock type, percent, specific gravity and copper, cobalt and gold grade block models had been created, indicated and inferred mineral resources were calculated individually using the Gemcom 'Volumetrics' software subroutine. Those resources within a 100-metre search radius with geological and grade continuity were reported as 'Indicated Mineral Resources'. Any mineral resources beyond the 100-metre search radius limits with a lower confidence level of geological and grade continuity were reported as 'Inferred Mineral Resources'.

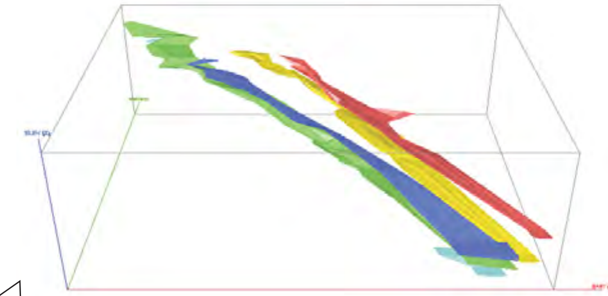
Mineral Resource Estimates

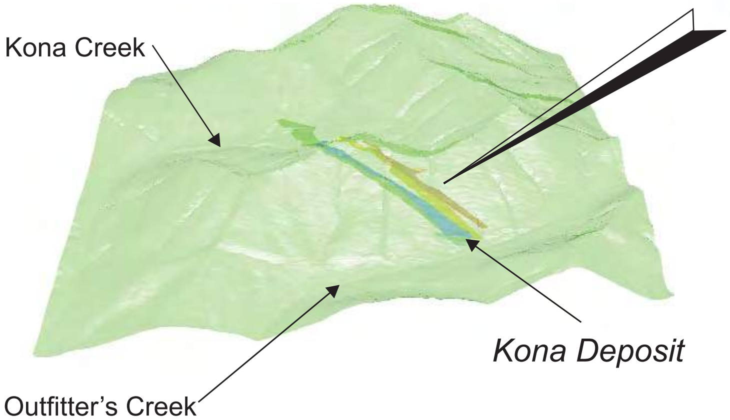
The detailed results of the indicated and inferred mineral resource estimates have been tabulated individually as Table II of this report. In summary, the Kona deposit has indicated and inferred mineral resources as follows (*nb.* all tabulated grades below are rounded to two decimal points).

Indicated Mineral Resources

Copper Cut-Off Grade (% Cu)	**Tonnage (million tonnes)**	**Copper (%)**	**Cobalt (%)**	**Gold (grams/tonne)**
2.00	0.595	2.46	0.11	0.82
1.50	1.639	1.97	0.10	0.70
1.00	**3.571**	**1.57**	**0.10**	**0.61**
0.50	6.415	1.20	0.08	0.50
No Cut-off	7.772	1.05	0.07	0.44

10A: Solid 3D geological model of Kona deposit with transparent topographic surface looking northward.

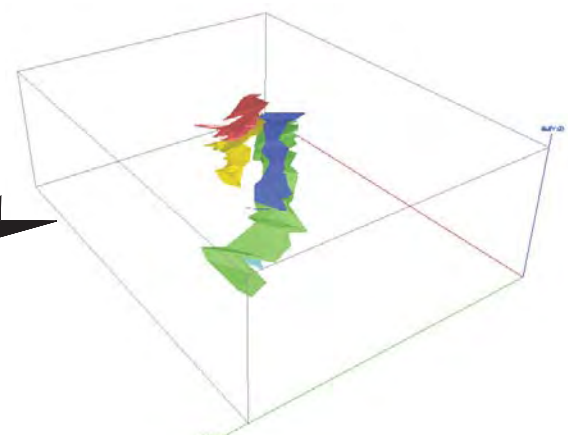


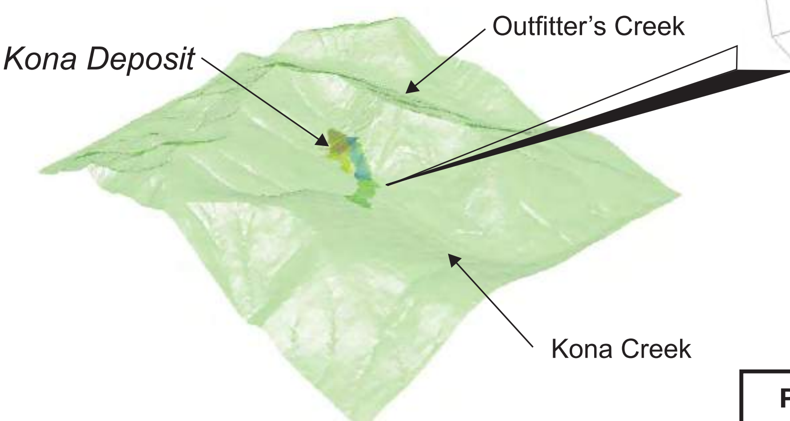


- LEGEND -

- Upper massive sulphide horizon of East Kona zone
- Lower massive sulphide horizon of East Kona zone
- Massive sulphide horizon of West Kona zone
- Upper magnetite-bearing (± sulphide) horizon of West Kona
- Lower magnetite-bearing (± sulphide) horizons of West Kona

10B: Solid 3D geological model of Kona deposit with transparent topographic surface looking southeastward.





To Accompany Report By J. D. Blanchflower, P. Geo., 2006

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

GEOLOGICAL MODELS
KONA DEPOSIT
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 10

Inferred Mineral Resources

Copper Cut-Off Grade (% Cu)	Tonnage (million tonnes)	Copper (%)	Cobalt (%)	Gold (grams/tonne)
2.00	0.418	2.69	0.08	0.61
1.50	2.056	1.87	0.09	0.54
1.00	**5.361**	**1.48**	**0.08**	**0.53**
0.50	9.148	1.18	0.07	0.42
No Cut-off	12.407	0.93	0.06	0.32

Thus, based upon a 1 percent copper cut-off grade the Kona deposit has an indicated mineral resource of 3.571 million tonnes grading 1.57% copper, 0.10% cobalt and 0.61 grams gold per tonne. Its inferred mineral resource, at the same 1 percent copper cut-off grade, is 5.361 million tonnes grading 1.48% copper, 0.08% cobalt and 0.53 grams gold per tonne. Zinc and silver values were not calculated in this mineral resource estimate. **The above mineral resource estimates are NOT mineral reserves and do not have demonstrated economic viability.** See Appendix III of this report for a detailed mineral resource estimate report.

Outstanding Issues

There are currently no known environmental, permitting, legal, title, taxation, socio-economic, or political issues that adversely affect the mineral resources described herein. Pertinent guidelines and regulations governing exploration work in the Yukon Territory are available from the appropriate territorial government agencies. Any further exploration work, especially that requiring surface disturbance (i.e. road building, excavator trenching, etc.), would be subject to the regulations in place at that time.

Preliminary metallurgical testwork results indicate that recoveries of 90% for the copper values and 70% for the gold values could be achieved from a mill feed grade of approximately 2% copper and 1.2 g gold/tonne (Melis Engineering, 1997). Results of this work also indicated a 70% cobalt recovery in flotation and 95% recovery from pressure leaching with an overall cobalt recovery of possibly 65 to 70% (Melis Engineering, 1997).

With further detailed 'fill-in' drilling and trenching, future mineral resource studies should consider estimating the resources for individual mineralized horizons using more discriminating 1- or 2-metre cubic block sizes. Furthermore, such work should separate near-surface and deeply buried mineralization for estimates of possible mineral resources that might be amenable to possible open-pit or underground extraction operations (see Figures 8 and 9 for locations of near-surface mineral resources).

The above mineral resource estimates are NOT mineral reserves and do not have demonstrated economic viability.

OTHER RELEVANT DATA and INFORMATION

In January 1997 Columbia Gold Mines requested Kilborn Engineering Pacific Ltd. ("Kilborn") conduct a preliminary economic study of the 'Fyre Lake project'. Columbia Gold Mines provided Kilborn with basic project parameters and variations on these parameters. The information provided to Kilborn was not supported by drilling data, and it was Kilborn's understanding that the study was a theoretical exercise to be used for strategic planning (Kilborn, 1997). Kilborn investigated a number of variations, including mine life and cobalt recovery, utilizing capital and operating costs based on their 1997 financial model.

The writer will not comment on the results of the 'in-house' Kilborn study since it was a theoretical investigation, unsupported by drilling data, and beyond his scope of expertise.

TABLE II

Estimates of Mineral Resources

Kona Deposit

Incremental Tables

Indicated Mineral Resources

Grade Group (% Cu)	Volume (m3)	Density t/m3	Tonnes (tX1000)	Copper (%)	Co (%)	Gold (g/t)
4.5 - 5.0	0.191	3.520	0.671	4.520	0.101	1.087
4.0 - 4.5	2.349	3.458	8.125	4.229	0.111	0.647
3.5 - 4.0	3.017	3.462	10.447	3.721	0.132	0.870
3.0 - 3.5	16.437	3.394	55.792	3.232	0.119	0.867
2.5 - 3.0	35.423	3.418	121.070	2.719	0.108	0.772
2.0 - 2.5	115.119	3.465	398.894	2.196	0.109	0.823
1.5 - 2.0	305.673	3.416	1044.192	1.697	0.102	0.635
1.0 - 1.5	563.136	3.431	1932.158	1.236	0.090	0.529
0.75 - 1.0	387.217	3.454	1337.401	0.858	0.074	0.426
0.5 - 0.75	433.811	3.473	1506.471	0.619	0.057	0.320
0.01 - 0.5	396.042	3.425	1356.558	0.323	0.030	0.142
Total	2258.415	3.441	7771.780	1.048	0.074	0.440

Inferred Mineral Resources

Grade Group (% Cu)	Volume (m3)	Density t/m3	Tonnes (tX1000)	Copper (%)	Co (%)	Gold (g/t)
> 5.0	2.882	3.447	9.934	6.063	0.062	0.736
4.5 - 5.0	1.166	3.447	4.020	4.796	0.078	0.581
4.0 - 4.5	10.226	3.450	35.283	4.288	0.057	0.595
3.5 - 4.0	4.390	3.452	15.156	3.696	0.071	0.499
3.0 - 3.5	7.086	3.455	24.487	3.237	0.074	0.484
2.5 - 3.0	27.531	3.395	93.477	2.715	0.056	0.670
2.0 - 2.5	68.485	3.439	235.525	2.152	0.089	0.602
1.5 - 2.0	481.987	3.399	1638.034	1.658	0.088	0.521
1.0 - 1.5	956.097	3.456	3304.633	1.231	0.077	0.526
0.75 - 1.0	635.752	3.454	2195.574	0.869	0.063	0.313
0.5 - 0.75	462.688	3.441	1592.036	0.636	0.045	0.204
0.01 - 0.5	948.250	3.437	3258.696	0.223	0.018	0.050
Total	3606.541	3.440	12406.855	0.932	0.056	0.324

TABLE II

Estimates of Mineral Resources

Kona Deposit

Cumulative Tables

Indicated Mineral Resources

Grade Group (% Cu)	Volume (m3)	Density t/m3	Tonnes (tX1000)	Copper (%)	Co (%)	Gold (g/t)
4.5 - 5.0	0.191	3.520	0.671	4.520	0.101	1.087
4.0 - 4.5	2.540	3.463	8.796	4.252	0.110	0.681
3.5 - 4.0	5.557	3.463	19.242	3.964	0.122	0.783
3.0 - 3.5	21.994	3.412	75.035	3.419	0.120	0.846
2.5 - 3.0	57.418	3.415	196.105	2.987	0.112	0.800
2.0 - 2.5	172.537	3.449	594.999	2.457	0.110	0.815
1.5 - 2.0	478.209	3.428	1639.191	1.973	0.105	0.701
1.0 - 1.5	**1041.345**	**3.430**	**3571.349**	**1.574**	**0.097**	**0.608**
0.75 - 1.0	1428.562	3.436	4908.750	1.379	0.091	0.558
0.5 - 0.75	1862.373	3.445	6415.221	1.201	0.083	0.503
0.01 - 0.5	2258.415	3.441	7771.780	1.048	0.074	0.440
Total	2258.415	3.441	7771.780	1.048	0.074	0.440

Inferred Mineral Resources

Grade Group (% Cu)	Volume (m3)	Density t/m3	Tonnes (tX1000)	Copper (%)	Co (%)	Gold (g/t)
> 5.0	2.882	3.447	9.934	6.063	0.062	0.736
4.5 - 5.0	4.048	3.447	13.954	5.698	0.067	0.692
4.0 - 4.5	14.274	3.449	49.237	4.688	0.059	0.622
3.5 - 4.0	18.664	3.450	64.392	4.454	0.062	0.593
3.0 - 3.5	25.751	3.452	88.879	4.119	0.065	0.563
2.5 - 3.0	53.282	3.422	182.357	3.399	0.061	0.618
2.0 - 2.5	121.768	3.432	417.881	2.696	0.077	0.609
1.5 - 2.0	603.754	3.405	2055.915	1.869	0.086	0.539
1.0 - 1.5	**1559.851**	**3.437**	**5360.548**	**1.476**	**0.080**	**0.531**
0.75 - 1.0	2195.603	3.441	7556.123	1.299	0.075	0.468
0.5 - 0.75	2658.291	3.441	9148.159	1.184	0.070	0.422
0.01 - 0.5	3606.541	3.440	12406.855	0.932	0.056	0.324
Total	3606.541	3.440	12406.855	0.932	0.056	0.324

INTERPRETATION and CONCLUSIONS

The Fyre Lake property is a property of merit. Based upon a review of historical exploration results, the potential for discovering additional copper-, cobalt- and gold-bearing volcanogenic massive sulphide mineralization on the subject property is considered very good.

The Kona volcanogenic massive sulphide deposit is sparingly covered by the current mineral claim holdings. It has been tested by 115 holes totalling 22,663 metres or 74,354 feet of diamond drilling. Most of this drilling was conducted along a 1,500-metre section of an apparent 2,100 strike length from near its surface exposure in Kona Creek southeasterly to the ridge between Kona and Outfitter's Creek drainage. The drill-tested mineralization occurs within an area approximately 1,500 m long by 250 m wide, trends at 130^o, and plunges 0^o to -20^o southeastwardly. The deposit is hosted by Devono-Mississippian metavolcanic and metasedimentary rocks of the Grass Lakes succession (Murphy and Piercey, 1999b), part of the Yukon-Tanana Terrane within the Finlayson Lake district.

Two parallel zones of volcanogenic massive sulphide mineralization, East Kona and West Kona, comprise the Kona deposit; separated by an inferred reverse fault. The East Kona zone mineralization is 100 to 150 m wide, and consists of two massive to banded sulphide-bearing horizons (i.e. Upper and Lower East Kona) separated by 40 to 70 metres of chlorite schist.

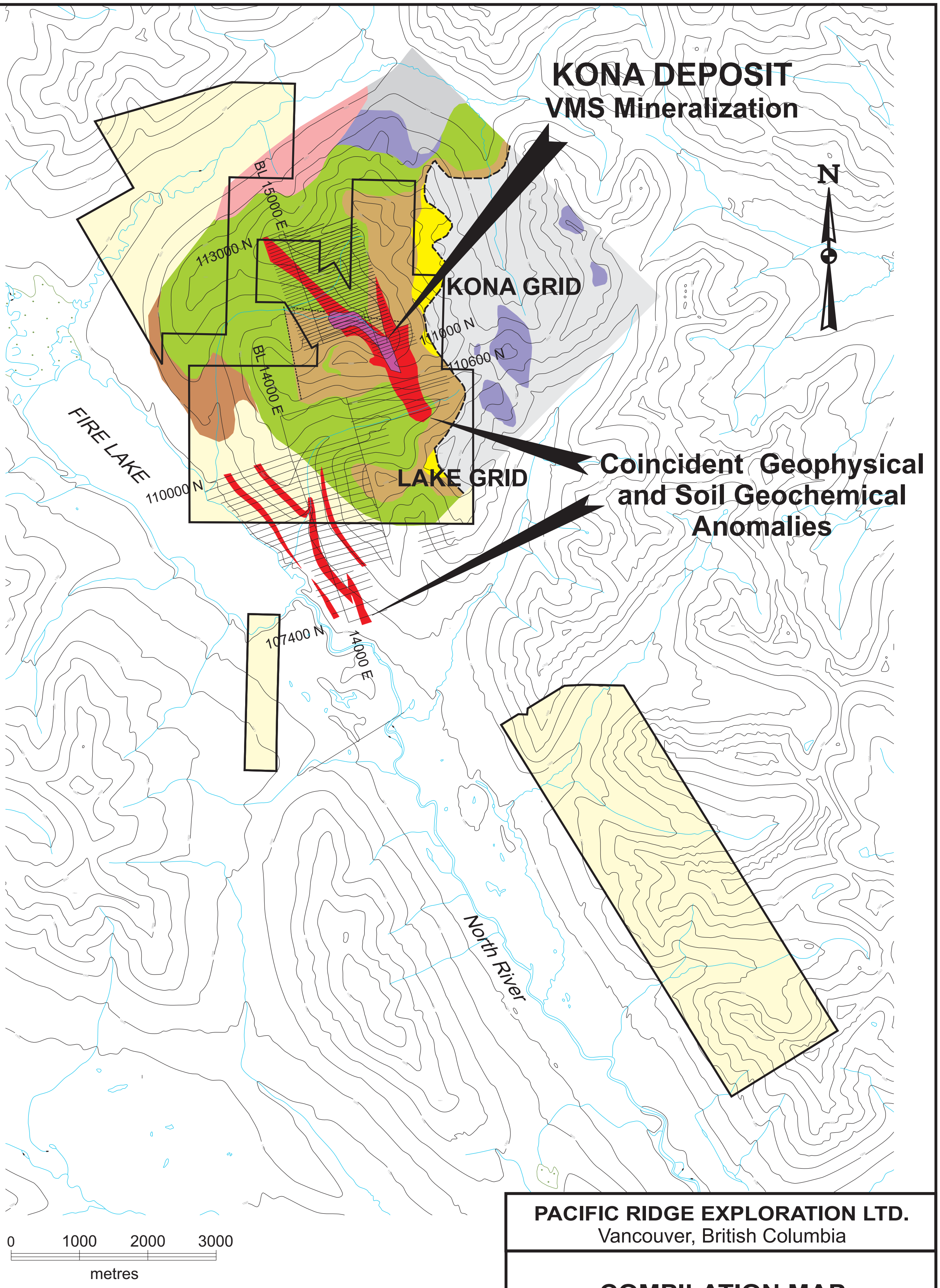
The Lower East Kona horizon has been divided into north and south portions separated by an apparent gap in the horizon. The northern portion is 3 to 16 metres thick and the southern portion is 2 to 11 metres thick. The Upper East Kona horizon averages thicknesses of 8 to 12 metres. These horizons consist mainly of pyrite with lesser pyrrhotite and chalcopyrite, local lenses of massive magnetite, and minor sphalerite.

The West Kona zone is inferred to be 75 to 125 m wide. The thickness of the mineralized horizon varies across this width from about 44 metres in the east to less than 1 metre at the western margin; the thickness also varies along strike. It includes mineralization that changes laterally from magnetite, pyrite and chalcopyrite in a siliceous matrix, through massive pyrite and lesser chalcopyrite, to massive pyrrhotite with minor pyrite and chalcopyrite. This mineralization occurs close to a stratigraphic contact between chlorite schist and overlying carbonaceous phyllite.

The chlorite schists that host the Kona deposit have a distinct boninitic chemical signature that distinguishes them from other similar chlorite schists found elsewhere on the property (Sebert and Hunt, 1999). Furthermore, the deposit has been interpreted by Murphy and Piercey (1999) to be hosted by some of the oldest stratigraphy in the district, and it has many characteristics of Besshi-style volcanogenic massive sulphide mineralization. The Kona volcanogenic massive sulphide mineralization may have been deposited in a back-arc basin, and has since been displaced by syn- and post-depositional normal and reverse faulting prior to regional tectonism and metamorphism.

Geochemical and geophysical anomalies indicate that the mineralized horizons comprising the Kona deposit may continue for at least 600 metres, on a trend of 130^o, to the Outfitter's Creek drainage (see Figures 11 and 12). These same anomalies also suggest that there may be similar volcanogenic massive sulphide mineralization northeast and southwest of the known mineralization hosted by fault-displaced stratigraphy paralleling the Kona deposit trend (see Figures 11 and 13).

Six distinct units of the East and West Kona volcanogenic massive sulphide- and magnetite-bearing mineralization were modelled prior to estimating the mineral resources of the Kona deposit. The results of geostatistical work indicated geological and grade continuity along a 125-metre search radius between drill tested sections of the deposit; however, a more conservative 100-metre search radius was utilized to interpolate the grade and specific gravity (density) block models by ordinary kriging methods.



See Figure 4, 5A and 5B of this report for detailed geology and legend (after Hunt, 2002, Fig. 9a; Bulletin 12, p. 18)

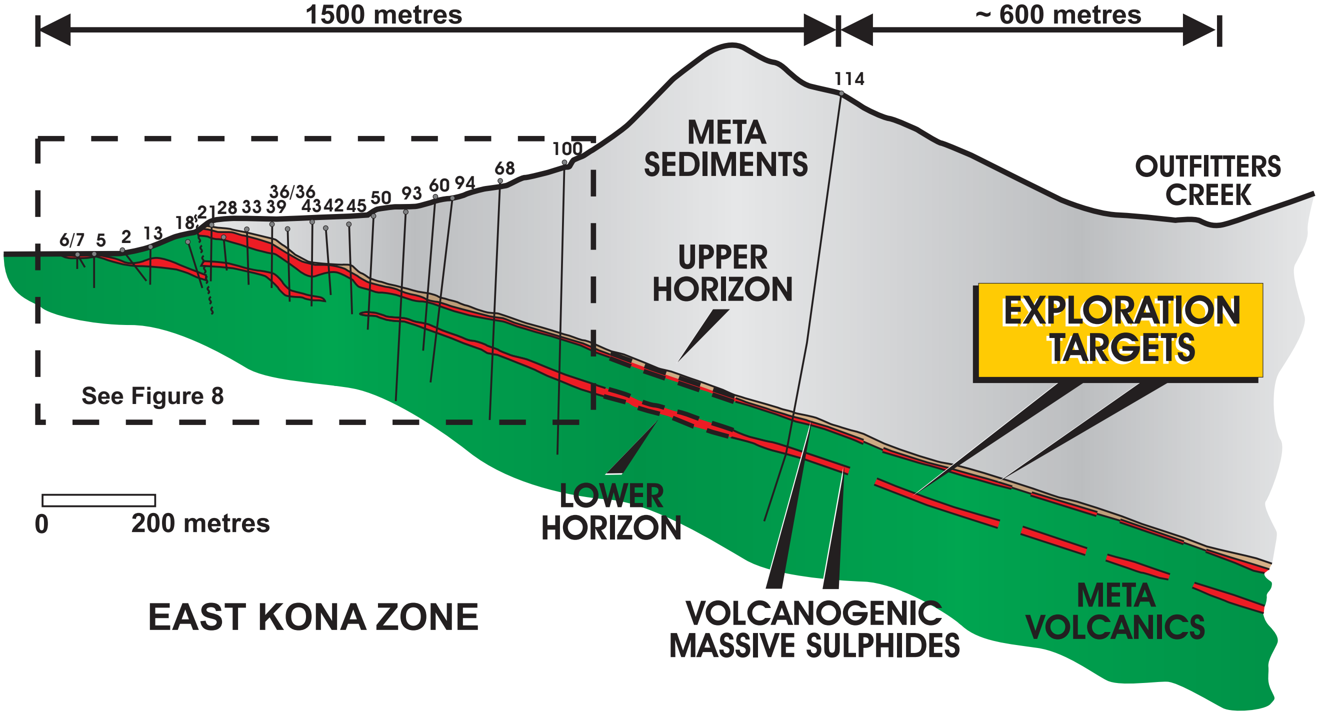
To Accompany Report By J. D. Blanchflower, P. Geo., 2006

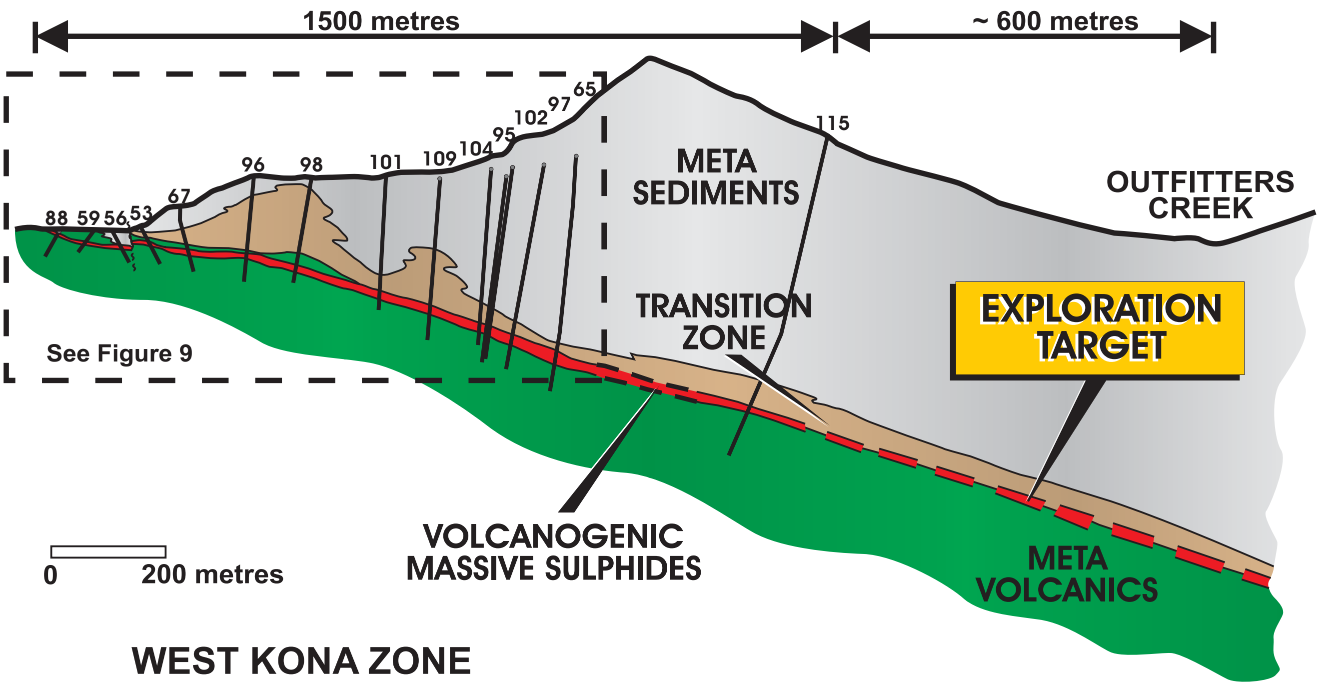
PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

COMPILATION MAP

Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 11





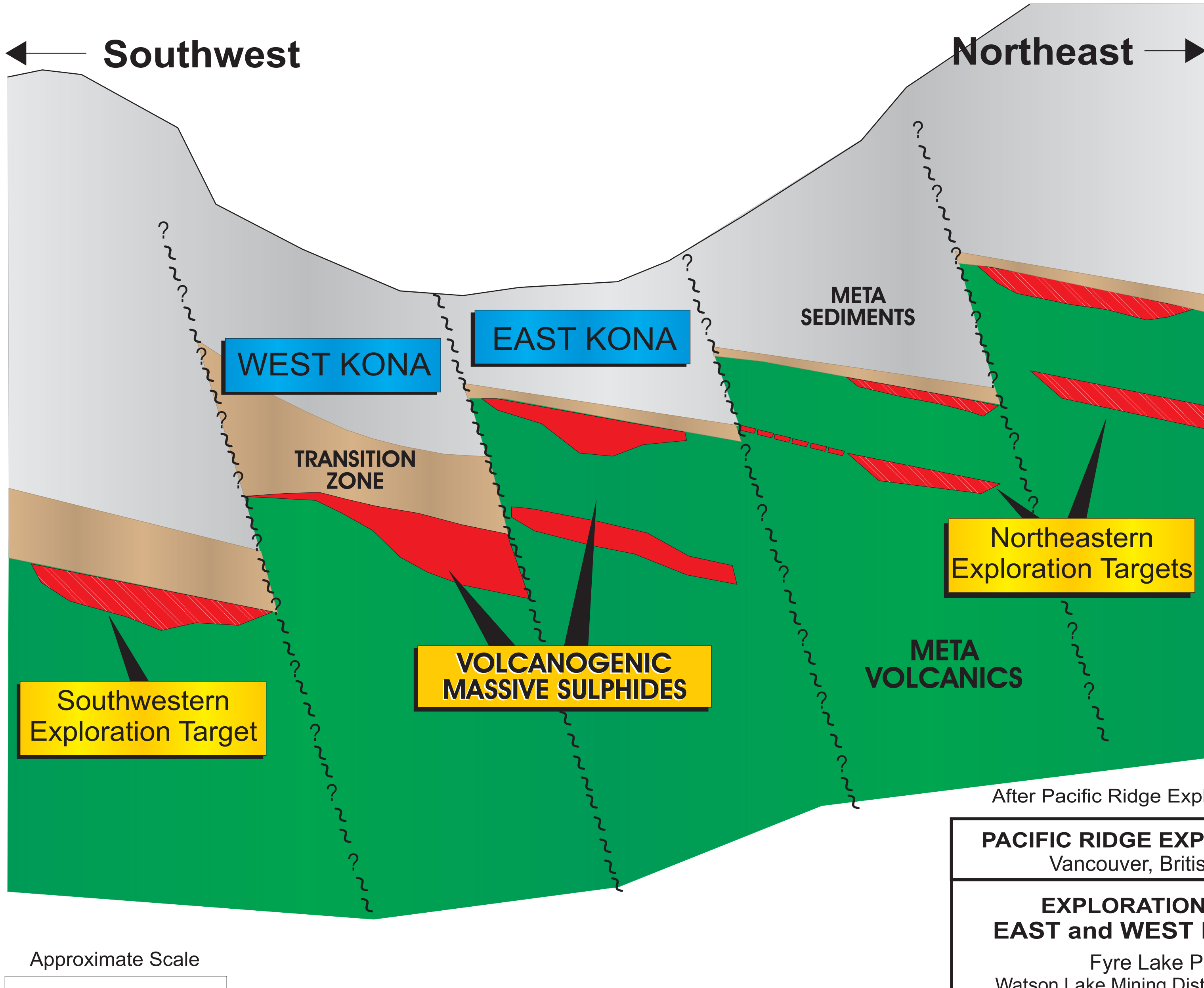
After Foreman, 1997 (Figs. 4, 6)

To Accompany Report By J. D. Blanchflower, P. Geo., 2006

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

LONGITUDINAL PROJECTIONS
EAST and WEST KONA ZONES
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 12



To Accompany Report By J. D. Blanchflower, P. Geo., 2006

After Pacific Ridge Exploration Ltd., 2002

PACIFIC RIDGE EXPLORATION LTD.
Vancouver, British Columbia

EXPLORATION TARGETS
EAST and WEST KONA ZONES
Fyre Lake Property
Watson Lake Mining District, Yukon Territory

Drawn By: JDB	Scale: As Shown
Date: January, 2006	Figure No. 13

Based upon a 1 percent copper cut-off grade the Kona deposit has an indicated mineral resource of 3.571 million tonnes grading 1.57% copper, 0.10% cobalt and 0.61 grams gold per tonne. Its inferred mineral resource, at the same cut-off grade, is 5.361 million tonnes grading 1.48% copper, 0.08% cobalt and 0.53 grams gold per tonne. Zinc and silver grades were not calculated in this mineral resource estimate. However, these mineral resource estimates are not mineral reserves and do not have demonstrated economic viability.

Preliminary metallurgical testwork results indicate that target recoveries of 90% for copper and 70% for gold, into a concentrate representing 7.5% of the feed and assaying 21% to 23% copper, 10 g gold/tonne and 0.08% cobalt, should be achieved from a mill feed grade of approximately 2% copper and 1.2 g gold/tonne (Melis Engineering, 1997). Subsequent scoping leach tests on cobalt-bearing pyrite concentrate indicated a 70% cobalt recovery in flotation and 95% recovery from pressure leaching with an overall cobalt recovery of possibly 65% to 70% (Melis Engineering, 1997). No cobalt minerals were identified during the microscopic examination of two samples, but pyrite was identified as the main cobalt carrier. No visible gold has been observed but it occurs associated with pyrite grains as sub-microscopic and/or colloidal gold (Lakefield, 1997).

Based upon the aforementioned results, it is the writer's opinion that the Fyre Lake property has considerable merit and further exploration work is warranted. Accordingly, the writer has recommended further exploration work to delineate the inferred volcanogenic massive sulphide mineralization along and beside the trend of the Kona deposit, and to continue exploring for similar mineralization elsewhere within the property.

RECOMMENDATIONS

The Fyre Lake property has considerable merit and further exploration work is justified. It is recommended that Pacific Ridge undertake an exploration program to continue drill testing the Kona deposit with some exploratory drilling elsewhere on favourable exploration targets.

The exploration work should include:

1) compiling, correlating and interpreting all available lithogeochemical data for the Kona deposit to identify any possible lithogeochemical trends that could be correlated with thicker or higher grade volcanogenic massive sulphide mineralization for later drill testing;

2) 5,000 metres (~16,400 ft) of NQ2- and/or BQTK-core diamond drilling to continue assessing the Kona deposit and exploratory drilling to investigate any worthy exploration targets elsewhere on the property;

3) additional metallurgical testing to possibly improve cobalt and gold recoveries; and

4) re-estimating the mineral resources of the Kona deposit.

PROPOSED EXPLORATION BUDGET

Item	**Description**	**Estimated Cost ($CDN)**
Analyses	400 lithogeochemical samples @ $15.00/sample	6,000
Assays	400 core samples @ $30.00/sample	12,000
	Check assaying (10%)	1,200
Accommodation	Camp operations – 1,200 man-days @ $30.00/day	36,000
	Hotel/motel during mob/demob	4,000
Consulting -	Geological - Project manager and project geologist	35,000

Item	Description	Estimated Cost ($CDN)
	Metallurgical – additional metallurgical studies	20,000
Drafting	Computer drafting	8,000
Expediting	SatTel rental, telephone, communications	10,000
	Expediting	10,000
Drilling	5,000 m (~16,400 ft.) of diamond drilling @ $27/ft	443,000
	Drill site pad construction and site reclamation	30,000
Equipment	Sperry Sun, generator and radio rentals	10,000
	Other rentals and leases	5,000
	Consumables	10,000
Fuel	Helicopter Jet B fuel	22,000
	Diamond drill and support equipment diesel fuel	20,000
	Camp fuel for heating, generator and propane	5,000
Property Maintenance	Assessment filings	1,000
Salaries/Wages	Geologist, samplers, camp support staff	50,000
Surveying - Control	Drill hole surveying	5,000
Mob/Demo Expenses	Mobilization/demobilization travel expenses	20,000
Transport - Fixed Wing	Camp supplies, drill equipment, samples	20,000
	spring fuel haul (50 trips)	45,000
Transport - Helicopter	Hughes 500D – 200 hours @ $800.00/hr	160,000
Transport - Vehicle	Weekly trucking to and from Smithers, B.C.	10,000
Transport - Freight	Truck support from Watson Lake to staging area	4,800
Total Estimated Expenses for Proposed Work		**$ 1,003,000**

The writer has estimated that the recommended exploration work will cost CAN $1,003,000 excluding G.S.T.

Submitted by,

J. Douglas Blanchflower, P. Geo.
Consulting Geologist

Dated at Aldergrove, British Columbia, Canada this 17th day of January, 2006

STATEMENT OF QUALIFICATIONS

I, J. DOUGLAS BLANCHFLOWER, of Aldergrove, British Columbia, DO HEREBY CERTIFY THAT:

1) I am a Consulting Geologist with a business office at 25856 – 28th Avenue, Aldergrove, British Columbia, V4W 2Z8; and President of Minorex Consulting Ltd.

2) I am a graduate of Economic Geology with a Bachelor of Science, Honours Geology degree from the University of British Columbia in 1971.

3) I am a Registered Professional Geoscientist with the Association of Professional Engineers and Geoscientists of British Columbia (No. 19086), and a Registered Professional Geologist with the Association of Professional Engineers, Geologists and Geophysicists of Alberta (No. M69488).

4) I have practised my profession as a geologist for the past thirty-four years; including:

   Pre-Graduate field experience in Geology, Geochemistry and Geophysics (1966 to 1970);

   Three years as Geologist with the B. C. Ministry of Energy, Mines and Pet. Res. (1970 to 1972);

   Seven years as Exploration Geologist with Canadian Superior Exploration Limited (1972 to 1979);

   Three years as Exploration Geologist with Sulpetro Minerals Limited (1979 to 1982); and

   Twenty-three years as Consulting Geologist and President of Minorex Consulting Ltd. (1982 to present).

5) I own no direct, indirect or contingent interest in the subject claims or any adjacent claims, nor shares in or securities of **PACIFIC RIDGE EXPLORATION LTD.**

6) I supervised the 1996 and 1997 exploration field programs and 1999 reclamation work on the subject property on behalf of Columbia Gold Mines Ltd.

7) I prepared a 'Report on the Fyre Lake Property' for Rock Resources Inc. in August, 2002. During the preparation of said report I estimated the mineral resources of the Kona zone in accordance with N.I. 43-101. These same resources are documented in this report since there has been no exploration activity on the property since 1997. I last visited the property in August, 1999 while supervising the reclamation work.

8) In January 2006 I reviewed the Kona zone mineral resource estimates and confirmed that there were no changes in the herein reported mineral resources as calculated in August, 2002.

9) I prepared this report which summarizes the results of past exploration work and current mineral resource estimates. It contains recommendations for further work on the Fyre Lake property.

10) I consent to the use of this Report on the Fyre Lake property by Pacific Ridge Exploration Ltd. for technical support of other documents to be submitted for regulatory approval.

J. Douglas Blanchflower, P. Geo.
Consulting Geologist

Dated at Aldergrove, British Columbia, Canada this 17th day of January, 2006

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APPENDIX I

Summary of 1996 to 1997 Diamond Drilling Data

APPENDIX I

Summary of 1966 to 1997 Diamond Drilling Data
(After Blanchflower, 1997; Pacific Ridge, 2002)

DDH No.	U.T.M. Easting	Northing	Elev (m)	Lgth (m)	Azim (deg)	Dip (deg)	Mineralized Intercept From (m)	To (m)	Int (m)	Cu (%)	Co (%)	Au (gpT)	Zn (%)
66-001	418636.78	6789422.90	1461.71	187.45	0.0	-90.0	Unreliable assay data						
66-002	418763.46	6789467.41	1479.16	121.01	0.0	-90.0	Unreliable assay data						
66-003	418799.13	6789135.96	1449.90	71.93	0.0	-90.0	Unreliable assay data						
66-004	418595.00	6789122.98	1434.00	125.88	0.0	-90.0	Unreliable assay data						
66-005	418694.62	6789445.16	1474.56	15.85	0.0	-90.0	Unreliable assay data						
66-005A	418694.62	6789445.16	1474.56	71.32	0.0	-90.0	Unreliable assay data						
67-001	418771.00	6785353.00	1180.00	94.49	0.0	-90.0	No Significant Values; Outside current claim holdings						
67-002	418817.09	6785353.51	1185.84	92.66	0.0	-90.0	No Significant Values; Outside current claim holdings						
67-003	418713.69	6785419.85	1175.49	65.53	0.0	-90.0	No Significant Values; Outside current claim holdings						
96-001	419017.03	6789204.72	1477.70	34.14	0.0	-90.0	12.33	16.40	4.07	1.14	0.14	0.48	0.69
96-002	419017.03	6789204.72	1477.63	79.25	160.0	-45.0	36.32	43.00	6.68	1.40	0.14	0.61	0.77
96-003	419017.03	6789204.72	1477.58	91.44	205.0	-45.0	22.30	28.75	6.45	1.02	0.14	0.79	1.09
96-004	418959.42	6789207.19	1472.27	64.01	0.0	-90.0	4.57	12.19	7.62	0.97	0.09	0.47	0.31
							8.50	12.19	3.69	1.60	0.13	0.52	0.20
96-005	418959.42	6789207.19	1472.25	32.00	160.0	-45.0	4.70	9.00	4.30	1.29	0.16	0.87	1.06
							4.70	16.00	11.30	0.90	0.10	0.42	0.46
96-006	418941.15	6789235.89	1472.84	30.48	0.0	-90.0	7.15	10.00	2.85	1.40	0.12	0.85	0.64
96-007	418941.15	6789235.89	1472.77	30.48	160.0	-60.0	7.75	10.70	2.95	1.28	0.11	1.11	1.08
96-008	418987.87	6789253.83	1478.31	70.10	0.0	-90.0	No Significant Values						
96-009	418987.87	6789253.83	1478.26	30.48	160.0	-45.0	No Significant Values						
96-010	419063.44	6789221.70	1481.45	79.86	160.0	-45.0	No Significant Values						
96-011	418882.34	6789248.30	1470.83	45.72	160.0	-45.0	No Significant Values						
96-012	418882.34	6789248.30	1470.93	30.48	0.0	-90.0	No Significant Values						
96-013	419049.42	6789161.62	1482.36	67.06	0.0	-90.0	23.12	35.44	12.32	0.90	0.11	0.80	0.84
							23.12	26.55	3.43	1.18	0.15	1.45	1.31
96-014	419049.56	6789161.25	1482.39	60.96	160.0	-60.0	26.51	33.00	6.49	1.44	0.17	0.88	2.45
							26.51	39.00	12.49	0.76	0.09	0.56	1.32
96-015	419049.73	6789160.84	1482.37	60.96	160.0	-45.0	33.75	43.82	10.07	0.78	0.09	0.70	1.58
96-016	419095.41	6789180.01	1481.72	45.72	0.0	-90.0	No Significant Values						
96-017	419095.54	6789179.49	1481.84	57.91	160.0	-45.0	No Significant Values						
96-018	419114.62	6789131.36	1494.78	91.44	160.0	-45.0	65.07	70.10	5.03	2.24	0.25	1.57	0.55
							63.20	72.55	9.35	1.45	0.15	1.01	0.34
96-019	419114.41	6789131.88	1494.64	91.44	160.0	-70.0	67.00	72.60	5.60	1.12	0.13	0.74	0.80

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Summary of 1966 to 1997 Diamond Drilling Data
(After Blanchflower, 1997; Pacific Ridge, 2002)

DDH No.	U.T.M. Easting	Northing	Elev (m)	Lgth (m)	Azim (deg)	Dip (deg)	Mineralized Intercept From (m)	To (m)	Int (m)	Cu (%)	Co (%)	Au (gpT)	Zn (%)
96-020	419114.30	6789132.09	1494.53	106.68	0.0	-90.0	No Significant Values						
96-021	419132.63	6789085.85	1522.22	100.58	0.0	-90.0	71.70	78.25	6.55	1.77	0.22	1.26	0.73
							11.19	22.80	11.61	0.69	0.07	0.18	0.06
							65.60	78.25	12.65	1.11	0.14	0.89	0.44
96-022	419133.04	6789085.14	1522.15	109.12	160.0	-45.0	12.30	20.00	7.70	0.64	0.09	0.35	0.03
96-023	419132.74	6789085.66	1522.07	91.44	160.0	-70.0	75.80	80.00	4.20	1.40	0.17	1.29	0.41
							12.19	16.80	4.61	0.94	0.09	0.18	0.03
96-024	419002.78	6789143.96	1484.81	79.25	0.0	-90.0	11.00	14.00	3.00	1.11	0.11	0.71	1.25
96-025	419003.00	6789143.18	1485.03	70.10	160.0	-45.0	No Significant Values						
96-026	419066.39	6789115.86	1502.79	76.20	160.0	-70.0	48.82	51.70	2.88	0.62	0.12	0.77	0.08
96-027	419066.61	6789114.92	1502.79	76.20	160.0	-45.0	No Significant Values						
96-028	419176.48	6789105.40	1497.89	82.30	210.0	-45.0	65.80	72.85	7.05	1.02	0.10	0.82	0.28
96-029	419176.74	6789105.80	1497.61	79.25	210.0	-70.0	56.00	59.54	3.54	1.17	0.12	1.00	2.34
							51.80	59.54	7.74	0.82	0.12	0.93	1.40
96-030	419176.82	6789105.96	1497.49	76.20	0.0	-90.0	No Significant Values						
96-031	419243.95	6789078.04	1497.45	91.44	210.0	-45.0	50.00	54.00	4.00	1.31	0.07	0.19	0.14
96-032	419244.46	6789078.72	1497.18	79.25	210.0	-70.0	No Significant Values						
96-033	419194.79	6789061.19	1513.36	100.58	0.0	-90.0	12.19	29.93	17.74	1.95	0.13	0.53	0.28
							16.00	24.00	8.00	2.70	0.19	0.49	0.45
							70.75	74.18	3.43	1.40	0.12	1.48	1.25
96-034	419194.58	6789060.79	1513.46	106.68	210.0	-70.0	13.40	26.00	12.60	2.39	0.17	0.63	0.40
							70.30	76.10	5.80	1.12	0.09	1.11	0.45
96-035	419259.46	6789024.57	1515.02	128.93	210.0	-60.0	53.31	65.50	12.19	2.75	0.15	0.80	0.37
96-036	419259.46	6789024.57	1515.02	137.16	0.0	-90.0	111.05	122.30	11.25	1.95	0.17	1.58	1.68
96-037	419141.22	6789041.18	1534.06	124.97	0.0	-90.0	7.66	12.67	5.01	0.95	0.08	0.49	0.06
96-038	419141.09	6789040.79	1534.15	96.01	210.0	-60.0	8.00	8.80	0.80	0.75	0.12	0.58	0.04
96-039	419217.24	6789015.02	1521.89	112.47	0.0	-90.0	34.06	51.40	17.34	1.91	0.12	0.59	0.12
							34.06	39.50	5.44	3.77	0.12	1.32	0.06
							92.50	97.67	5.17	1.62	0.12	1.40	0.28
96-040	419216.99	6789014.79	1522.20	121.92	225.0	-45.0	37.95	43.70	5.75	1.21	0.11	0.65	0.06
96-041	419326.99	6789001.35	1515.90	167.64	0.0	-90.0	No Significant Values						
96-042	419326.11	6788999.99	1515.50	172.21	210.0	-45.0	93.56	104.50	10.94	1.96	0.19	0.62	0.06
96-043	419285.00	6788984.00	1526.50	152.40	0.0	-90.0	75.83	94.50	18.67	2.27	0.11	0.48	0.41

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Summary of 1966 to 1997 Diamond Drilling Data
(After Blanchflower, 1997; Pacific Ridge, 2002)

DDH No.	U.T.M. Easting	Northing	Elev (m)	Lgth (m)	Azim (deg)	Dip (deg)	Mineralized Intercept From (m)	To (m)	Int (m)	Cu (%)	Co (%)	Au (gpT)	Zn (%)
							75.83	78.50	2.67	3.27	0.12	1.25	0.10
							86.50	93.50	7.00	3.36	0.15	0.53	0.97
							133.01	139.11	6.10	1.39	N/C	1.34	1.42
96-044	419285.00	6788984.00	1526.50	152.10	210.0	-60.0	66.00	75.00	9.00	1.76	0.14	0.48	0.05
96-045	419345.44	6788954.26	1523.06	192.02	210.0	-60.0	99.41	110.50	11.09	2.44	0.16	0.67	0.09
							101.52	107.50	5.98	3.24	0.20	0.97	0.11
							116.39	120.50	4.11	0.97	0.06	0.10	0.02
96-046	419345.68	6788955.02	1522.59	167.64	0.0	-90.0	116.28	117.72	1.44	1.37	0.19	0.58	0.03
96-047	419295.49	6788935.27	1537.51	204.22	210.0	-60.0	No Significant Values						
96-048	419364.86	6788906.55	1539.19	201.17	210.0	-60.0	No Significant Values						
96-049	418800.99	6789119.79	1458.37	91.44	0.0	-90.0	28.50	34.50	6.00	0.95	0.03	0.19	0.07
96-050	419365.22	6788907.21	1538.75	210.31	0.0	-90.0	124.44	138.50	14.06	1.99	0.11	0.46	0.19
							133.50	137.50	4.00	3.72	0.15	0.54	0.43
							181.39	182.99	1.60	1.61	0.18	1.42	1.77
96-051	419365.18	6788906.67	1539.13	198.12	210.0	-75.0	116.07	122.50	6.43	1.89	0.20	0.68	0.05
							116.07	130.50	14.43	1.19	0.11	0.35	0.04
96-052	418798.66	6789118.94	1458.55	73.15	250.0	-45.0	No Significant Values						
96-053	418803.50	6789120.85	1458.82	106.68	70.0	-45.0	43.70	49.93	6.23	1.71	0.08	0.19	0.27
96-054	419392.29	6788974.09	1534.63	274.32	0.0	-90.0	207.18	212.71	5.53	1.72	0.11	1.36	0.61
96-055	418758.36	6789159.47	1448.56	91.13	0.0	-90.0	16.08	24.72	8.64	1.36	0.15	0.38	0.32
96-056	418759.19	6789159.65	1448.61	91.44	70.0	-45.0	39.38	45.86	6.48	3.13	0.15	0.68	0.52
							39.38	42.50	3.12	5.10	0.22	1.10	0.88
96-057	418743.23	6789197.65	1448.35	91.44	0.0	-90.0	24.48	30.70	6.22	1.71	0.09	0.35	0.14
96-058	418744.27	6789197.94	1448.74	91.44	70.0	-45.0	48.72	51.66	2.94	0.36	0.03	0.02	0.04
96-059	418739.52	6789196.21	1447.96	67.06	250.0	-45.0	32.40	38.00	5.60	1.44	0.11	0.54	0.24
96-060	419446.49	6788831.67	1571.15	286.51	0.0	-90.0	184.49	190.58	6.09	1.00	0.16	0.38	0.04
							233.90	238.30	4.40	1.79	0.17	1.35	0.66
96-061	419562.67	6788940.05	1608.86	399.29	0.0	-90.0	No Significant Values						
96-062	419412.55	6788926.94	1540.24	298.69	0.0	-90.0	216.35	229.50	13.15	1.06	0.14	0.93	0.52
96-063	419667.66	6788975.09	1668.63	428.85	0.0	-90.0	No Significant Values						
96-064	419401.85	6788808.12	1577.38	280.42	0.0	-90.0	169.00	172.10	3.10	0.78	0.05	0.05	0.02
96-065	419578.60	6788650.08	1635.86	521.20	0.0	-90.0	428.70	460.00	31.30	2.29	0.07	0.53	0.24
							453.00	460.00	7.00	6.07	0.05	0.68	0.37

APPENDIX I

Summary of 1966 to 1997 Diamond Drilling Data
(After Blanchflower, 1997; Pacific Ridge, 2002)

DDH No.	U.T.M. Easting	Northing	Elev (m)	Lgth (m)	Azim (deg)	Dip (deg)	Mineralized Intercept From (m)	To (m)	Int (m)	Cu (%)	Co (%)	Au (gpT)	Zn (%)
96-066	418877.42	6789081.74	1507.49	152.40	0.0	-90.0	85.25	98.96	13.71	0.71	0.07	0.20	0.08
							111.00	116.00	5.00	0.88	0.04	0.08	0.01
96-067	418877.42	6789081.74	1507.49	152.40	70.0	-65.0	96.08	116.37	20.29	1.33	0.11	0.42	0.29
96-068A	419535.21	6788749.07	1600.76	146.30	0.0	-90.0	Abandoned Due to Drilling Difficulties						
96-068	419535.21	6788749.07	1600.76	445.01	0.0	-90.0	309.00	319.10	10.10	2.66	0.11	1.44	0.88
96-069	418878.96	6789037.88	1539.12	220.98	0.0	-90.0	No Significant Values						
96-070	418878.96	6789037.88	1539.12	185.93	250.0	-70.0	No Significant Values						
96-071	418878.96	6789037.88	1539.12	213.36	70.0	-70.0	132.00	138.34	6.34	1.36	0.13	0.43	0.02
97-072	418651.40	6785676.90	1206.60	152.40	250.0	-60.0	No Significant Values; Outside current claim holdings						
97-073	419054.10	6785816.40	1325.20	150.27	250.0	-52.0	No Significant Values; Outside current claim holdings						
97-074	418832.70	6786164.40	1348.80	182.88	250.0	-50.0	No Significant Values						
97-075	418528.50	6786068.10	1236.20	134.11	250.0	-50.0	No Significant Values						
97-076	418109.60	6786549.50	1276.20	134.11	250.0	-50.0	No Significant Values						
97-077	417686.60	6786374.30	1262.90	140.21	250.0	-50.0	No Significant Values; Outside current claim holdings						
97-078	419242.80	6784957.70	1190.20	150.88	250.0	-50.0	No Significant Values; Outside current claim holdings						
97-079	418758.20	6786351.10	1335.90	152.40	250.0	-50.0	No Significant Values						
97-080	418519.55	6789432.78	1456.79	79.25	70.0	-60.0	No Significant Values						
97-081	418632.57	6789372.21	1459.10	158.50	70.0	-50.0	No Significant Values						
97-082	418663.89	6789276.55	1450.37	104.85	70.0	-50.0	No Significant Values						
97-083	418449.57	6789407.95	1448.16	82.30	70.0	-60.0	No Significant Values						
97-084	418835.67	6789231.58	1466.08	103.63	250.0	-85.0	No Significant Values						
97-085	418457.17	6789521.52	1482.16	109.73	70.0	-85.0	No Significant Values						
97-086	418328.80	6789586.79	1497.64	103.63	70.0	-60.0	No Significant Values						
97-087	418313.33	6789690.49	1529.85	67.06	70.0	-50.0	No Significant Values						
97-088	418675.09	6789228.33	1446.95	117.35	250.0	-50.0	6.10	15.14	9.04	1.54	0.07	0.34	0.02
							10.14	15.14	5.00	2.43	0.11	0.55	0.02
							10.14	12.14	2.00	3.57	0.09	0.73	0.03
97-089	419250.31	6789188.79	1492.56	167.34	250.0	-85.0	No Significant Values						
97-090	419338.18	6789111.08	1525.24	198.12	250.0	-85.0	No Significant Values						
97-091	419387.72	6788850.12	1562.29	368.81	250.0	-85.0	274.70	299.35	24.65	0.81	0.04	0.17	0.09
							278.41	282.70	4.29	1.02	0.05	0.26	0.05
							292.60	297.85	5.25	1.60	0.03	0.20	0.14
							275.70	282.70	7.00	0.88	0.05	0.22	0.09

APPENDIX I

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(After Blanchflower, 1997; Pacific Ridge, 2002)

DDH No.	U.T.M. Easting	Northing	Elev (m)	Lgth (m)	Azim (deg)	Dip (deg)	Mineralized Intercept From (m)	To (m)	Int (m)	Cu (%)	Co (%)	Au (gpT)	Zn (%)
							288.70	297.85	9.15	1.15	0.03	0.18	0.13
97-092	419582.34	6788728.99	1615.27	495.91	250.0	-85.0	269.15	271.15	2.00	1.37	0.04	0.09	0.01
							419.87	422.00	2.13	1.02	0.05	0.18	0.07
97-093	419431.11	6788891.48	1545.06	350.52	250.0	-85.0	170.25	173.16	2.91	3.20	0.12	0.40	0.39
							211.08	213.91	2.83	0.42	0.06	0.83	0.12
							217.73	225.72	7.99	0.49	0.04	0.45	0.18
							221.58	223.57	1.99	0.97	0.06	1.16	0.50
							217.73	223.57	5.84	0.56	0.04	0.54	0.19
97-094	419508.48	6788862.96	1569.80	347.47	250.0	-85.0	218.28	219.95	1.67	2.07	0.13	0.27	0.41
							267.40	279.52	12.12	1.30	0.16	1.28	0.58
							268.40	271.40	3.00	0.99	0.25	1.49	0.09
							274.53	278.70	4.17	1.95	0.12	1.18	0.75
97-095	419464.59	6788739.72	1589.62	416.66	250.0	-85.0	321.67	348.41	26.74	1.36	0.12	0.44	0.23
							321.67	323.93	2.26	2.33	0.09	0.95	0.50
							345.37	348.41	3.04	2.82	0.13	0.43	0.19
							328.35	340.00	11.65	1.30	0.14	0.44	0.13
							328.35	336.00	7.65	1.29	0.16	0.48	0.09
97-096	418998.30	6788991.29	1575.06	225.55	250.0	-88.0	164.39	180.65	16.26	1.24	0.09	0.39	0.17
							164.39	178.39	14.00	1.34	0.10	0.40	0.18
97-097	419513.44	6788673.08	1615.90	486.46	250.0	-85.0	367.15	371.50	4.35	1.45	0.10	0.41	0.09
							367.15	368.50	1.35	2.27	0.11	0.46	0.02
							375.50	389.80	14.30	1.64	0.11	0.43	0.22
							380.50	384.30	3.80	2.75	0.10	0.47	0.22
							385.16	389.80	4.64	1.91	0.15	0.56	0.02
							382.58	389.80	7.22	2.02	0.16	0.52	0.06
							378.40	389.80	11.40	1.91	0.12	0.45	0.15
							381.55	389.80	8.25	2.22	0.14	0.53	0.07
97-098	419122.64	6788958.18	1576.21	234.70	250.0	-75.0	195.46	200.50	5.04	1.39	0.09	0.42	0.40
							205.80	210.77	4.97	1.36	0.11	0.36	0.03
							195.46	210.77	15.31	1.11	0.09	0.31	0.15
							206.70	210.31	3.61	1.40	0.13	0.40	0.03
							200.50	205.80	5.30	0.61	0.07	0.16	0.02
97-099	419323.27	6788767.25	1604.46	419.10	250.0	-85.0	No Significant Values						

APPENDIX I

Summary of 1966 to 1997 Diamond Drilling Data
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DDH No.	U.T.M. Easting	Northing	Elev (m)	Lgth (m)	Azim (deg)	Dip (deg)	Mineralized Intercept From (m)	To (m)	Int (m)	Cu (%)	Co (%)	Au (gpT)	Zn (%)
97-100	419637.53	6788696.56	1637.68	542.85	70.0	-88.0	391.19	406.47	15.28	2.08	0.10	1.54	0.33
							395.60	406.47	10.87	2.48	0.12	1.73	0.29
							395.60	402.76	7.16	2.80	0.14	2.07	0.17
							397.05	402.76	5.71	3.23	0.14	2.30	0.19
							399.86	402.76	2.90	3.86	0.11	2.91	0.22
97-101	419234.67	6788849.95	1588.82	296.88	250.0	-88.0	250.04	269.50	19.46	1.98	0.13	0.54	0.34
							251.00	267.50	16.50	2.15	0.13	0.58	0.39
							255.00	267.50	12.50	2.25	0.13	0.60	0.29
							255.00	258.23	3.23	2.76	0.09	0.76	0.35
							260.50	267.50	7.00	2.36	0.15	0.59	0.33
97-102	419446.74	6788692.16	1609.43	432.82	250.0	-85.0	341.44	351.47	10.03	2.98	0.10	0.95	0.02
							341.44	351.17	9.73	3.03	0.10	0.97	0.02
							342.94	351.17	8.23	3.26	0.10	1.10	0.03
							342.94	347.76	4.82	3.42	0.09	1.24	0.03
97-103	419579.76	6788643.69	1641.45	531.27	175.0	-80.0	No Significant Values						
97-104	419413.70	6788721.40	1603.90	381.00	250.0	-85.0	331.07	339.60	8.53	3.26	0.11	0.94	0.02
97-105	419581.80	6788642.60	1640.10	224.64	175.0	-85.0	No Significant Values						
97-106	419639.10	6788695.40	1639.00	500.48	120.0	-85.0	No Significant Values						
97-107	419883.30	6788459.60	1811.40	805.28	250.0	-85.0	No Significant Values						
97-108	419891.80	6788666.70	1800.20	694.94	250.0	-85.0	No Significant Values						
97-109	419346.10	6788812.40	1582.50	359.66	250.0	-85.0	275.43	290.05	14.62	1.38	0.08	0.62	0.30
							303.41	310.05	6.64	3.01	0.03	0.26	0.10
97-110	419652.00	6788798.80	1637.40	431.91	250.0	-80.0	384.46	390.74	6.28	1.90	0.13	1.45	1.14
97-111	419762.40	6788735.30	1700.70	520.29	250.0	-82.0	482.42	493.25	10.83	1.41	0.12	1.15	0.96
							484.42	493.25	8.83	1.52	0.13	1.22	1.12
97-112	418151.70	6788984.10	1437.50	188.98	250.0	-45.0	No Significant Values						
97-113	418063.50	6788536.30	1612.80	197.51	250.0	-45.0	No Significant Values						
97-114	420095.50	6788472.90	1769.50	831.19	250.0	-85.0	614.17	616.73	2.56	2.17	0.06	0.20	0.03
							615.02	616.73	1.71	2.64	0.06	0.24	0.01
97-115	420094.75	6788472.00	1769.50	825.09	250.0	-80.0	703.60	719.87	16.27	1.28	0.11	0.61	0.21
							711.60	719.87	8.27	1.61	0.11	0.49	0.07

APPENDIX II

1997 Metallurgical Data

Melis Engineering Ltd. and Lakefield Research, 1997

MELIS
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MEMORANDUM

July 18, 1997

MELIS Project No. M-340

TO: Jaakko Levanaho
Kilborn Vancouver

FROM: B.C. Fielder

RE: **Sulphur Concentration in the Pressure Leach Feed**

Sorry for the confusion - we were not talking about the same product.

Unfortunately, we do not have an assay of the sulphur feeding the pressure leach. What we do have is a more detailed assay of the products of that test leach, reproduced below.

Autoclave Leach Results:

Results:

Product	Amount	Assay mg/L, %		Distribution %	
	mL,g	Co	Fe	Co	Fe
Preg	1480	172	1500	95	7
Wash	1500	6.06	129	3	1
Residue	58.5	0.005	46.8	1	92
Cal. Head	75.0	0.36	39.7	100	100
Dir. Head		0.35	42.4		

Additional Assays

Product	Assay(%)			
	$S_{(t)}$	SO_4	S^{2-}	S^o
Residue	7.93	22.7	<0.01	0.83

Regards,
MELIS ENGINEERING LTD.

B C Fielder

B.C. Fielder
Senior Process Engineer

519 45TH STREET WEST, SASKATOON, SK, CANADA S7L 5Z9 (306) 652-4084 FAX: (306) 653-3779 Email: melis@sympatico.ca

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FAX MEMORANDUM

TO:	Mr. W.J. Roberts	DATE:	May 6, 1997
	Columbia Gold Mines Limited	PROJECT NO.:	340
	Vancouver, B.C.	FAX NO.:	(604) 687-4991
cc:	P.N. Dhir	FROM:	L.A. Melis, P.Eng.

This transmission consists of 2 page(s) including this one.

RE: FYRE LAKE PROJECT METALLURGY - COBALT LEACHING TESTS

Three scoping leach tests have been completed on cobalt-bearing pyrite concentrate produced in Flotation Test No. 9 (see Melis Engineering Ltd. fax dated April 18, 1997). This pyrite concentrate assayed 0.34% Co, 0.35% Cu, 41.6% Fe and 0.62 g Au/tonne and, for this particular test, contained 73.6% of the cobalt.

Atmospheric Leach Tests

Two atmospheric leach tests have been completed on unground concentrate. One test was a strong acid leach at 80° C. The second test was an oxidizing mild acid leach, maintaining an emf value of 550 to 600 mV which resulted in a very high hydrogen peroxide consumption. Both tests yielded poor cobalt extractions as noted from the following results:

Fyre Lake Project - Atmospheric Cobalt Leach Tests								
Test No.	Conditions	Results						
		Calc. Head		% Weight Loss	% Leach Extraction		Pregnant Solution	
		% Co	% Fe		Co	Fe	g Co/L	g Fe/L
1	300 g H_2SO_4/L, 90° C, 6 h, 5% Solids(w/w)	0.32	40.6	11.4	1.3	7.9	0.002	1.57
2	30 g H_2SO_4/L, 90° C, 6 h, 5% Solids(w/w), 550 mV, 413 kg H_2O_2/tonne	0.32	40.7	24.2	18.5	22.3	0.028	4.44

Pressure Leach Test

A single pressure leach test was completed on reground (pulverized) concentrate. A 98.9% cobalt extraction was obtained as noted below:

Fyre Lake Project - Pressure Cobalt Leach Test								
Test No.	Conditions	Results						
		Calc. Head		% Weight Loss	% Leach Extraction		Pregnant Solution	
		% Co	% Fe		Co	Fe	g Co/L	g Fe/L
3	225° C, 460 psig P_T, 100 psig Po_2, 2 h, 650 mV	0.36	39.7	22.0	98.9	92.0	0.172	1.5

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The residue from the pressure leach test assayed 7.93% S_T, <0.01% S^{2-}, 0.83% S^0 and 22.7% SO_4, indicating that 10.5% of the sulphur is removed as elemental sulphur (S^0) and the remainder, 89.5%, is precipitated as sulphate compounds.

Remaining Leach Test

One last leach test is being completed to test atmospheric leaching of *reground* concentrate using a strong acid leach and sodium chlorate oxidant. The results of this test are not expected to be much different than the other two atmospheric leach tests but it is being completed to check on atmospheric leaching efficiency using reground concentrate.

Future Leaching Testwork

Leaching work in future test programs will need to confirm that pressure leaching is necessary for cobalt extraction, which is likely, and what will be the minimum pressure leach conditions required for effective, economic leaching of cobalt. Associated with this work will be testing of downstream processes including liquid/solid separation for cobalt solution recovery as well as evaluation of precipitation and solvent extraction for final cobalt recovery.

Cobalt Recovery

Based on 70% cobalt recovery in flotation and 95% recovery from pressure leaching, overall cobalt recovery should be 65% to 70%.

I will forward the results of the last leach test when available. No further work is planned on this at present. It may, however, be worthwhile to investigate cobalt leaching further to better quantify pressure leaching conditions and to evaluate downstream solution recovery of cobalt. Please advise if you feel this work would be warranted at present and we will proceed accordingly.

Yours truly,
MELIS ENGINEERING LTD.

L.A. Melis

Lawrence A. Melis, P.Eng.
President

MELIS
ENGINEERING LTD.

FAX MEMORANDUM

TO:	Mr. W.J. Roberts Columbia Gold Mines Limited Vancouver, B.C.	DATE:	April 18, 1997
		PROJECT NO.:	340
		FAX NO.:	(604) 687-4991
cc:	P.N. Dhir	FROM:	L.A. Melis, P.Eng.

This transmission consists of 7 page(s) including this one.

RE: FYRE LAKE PROJECT METALLURGY - UPDATE OF TESTWORK

HEAD ASSAY

A multi-element analysis was completed on the Fyre Lake test composite. The analysis was as follows:

Element	Unit	Assay	Element	Unit	Assay
Ag	g/tonne	3.7	Mo	ppm	<10
Al	%	1.50	Na	%	0.053
As	%	0.034	Ni	%	0.013
Au	g/tonne	0.98	P	ppm	140
B	ppm	<8	Pb	%	0.004
Ba	ppm	130	S	%	20.5
Be	ppm	<1	Sb	%	0.001
Bi	ppm	<10	Se	ppm	<50
Ca	%	0.34	Si	%	15.8
Cd	ppm	33	Sn	ppm	<20
Co	%	0.16	Sr	ppm	7.5
Cr	ppm	100	Th	%	<0.001
Cu	%	2.07	Ti	ppm	319
Fe	%	26.1	Tl	ppm	<10
Ga	ppm	47	U	ppm	<10
K	%	0.043	V	ppm	104
Li	ppm	8.5	W	%	0.007
Mg	%	1.30	Zn	%	0.99
Mn	%	0.02			

FLOTATION

Test No. 8

Test No. 8 was completed to check on the effect of a finer grind (92.8% minus 200 mesh instead of the 89% minus 200 mesh grind in Test No. 7) and to see if upgrading of the cobalt in the pyrite concentrate would be possible. As can be seen from the grade/recovery curves finer grinding did not improve results over those achieved in Test No. 7. Two stage cleaning of the pyrite concentrate (copper rougher scavenger concentrate) did not improve cobalt grades. The cobalt grade in the pyrite (copper rougher scavenger) concentrate was 0.35% Co and the corresponding grade in the second cleaner concentrate was 0.37% Co.

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Test No. 9

Test No. 9 was completed under similar conditions to Test No. 8 to prepare cobalt-bearing pyrite concentrate for leach scoping tests. A higher concentrate grade was achieved in the test (24.9% Cu), with a corresponding drop in copper recovery (see the copper grade/recovery curve, Figure A-1). For an equivalent copper grade, the gold recovery in Test No. 9 was slightly better than Test No. 8 (see the gold grade/recovery curves, Figures A-2 and A-3).

The copper rougher scavenger (pyrite) concentrate and the copper first cleaner scavenger cleaner tail was combined to provide approximately 650 g of cobalt leach testing material assaying 0.34% Co, 0.35% Cu, 41.6% Fe and 0.62 g Au/tonne.

Zinc Distribution

Copper, gold and cobalt have been the key elements followed in the testwork but a zinc balance was completed on Test No. 9 to measure the deportment of zinc to the various flotation products. The calculated head grade for this test was 0.99% Zn. Since the float was completed as a copper/gold float, the amount of zinc reporting to the copper rougher concentrate was high (88.4%). The final cleaner concentrate, with a 24.9% Cu grade, assayed 7.82% Zn.

Since a typical smelter charge for zinc in copper concentrate is $(U.S.) 3.00/tonne/1% (Pb + Zn) > 4% (Pb + Zn), zinc depression will need to be considered in future testwork. It will also be important to check on potential zinc recoveries if parts of the mineralization grade 1% Zn or higher. In typical copper-zinc concentrators the zinc circuit is not operated when zinc values in the mill feed drop to approximately 1% Zn or less.

Conclusions

From the nine scoping tests completed in this initial part of the Fyre Lake test program, the conditions used in Test No. 7 appear to provide the best results. Future testing will of course need to investigate other reagent schemes to see if any improvements can be made to metallurgical efficiencies.

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Flotation Conditions

Based on work to date, the flotation conditions which appear to be favourable for the Fyre Lake mineralization are as follows:

- grind to approximately 90% minus 200 mesh (K_{80} of ~ 60 μm) in the presence of lime,
- copper rougher flotation at pH 11.0 with Aero 3501 promoter and isopropyl xanthate collector in the last two stages,
- copper rougher scavenger (pyrite) flotation with isopropyl xanthate collector,
- copper rougher concentrate regrind in the presence of lime,
- four-stage cleaning of reground copper rougher concentrate at pH 11.5 with small dosages of Aero 3501 promoter and sodium cyanide in the fourth cleaner, and
- scavenger flotation of the first cleaner tails with a minor dosage of Aero 3501 promoter.

Target Metallurgical Efficiencies

As indicated by the results of Test No. 7, target recoveries of 90% for copper and 70% for gold, into a concentrate representing 7.5% of the feed and assaying 21% to 23% Cu, 10 g Au/tonne and 0.08% Co, should be achievable from a mill feed grade of approximately 2% Cu and 1.2 g Au/tonne. Cobalt recovery to a pyrite concentrate assaying approximately 0.33% Co will be approximately [illegible]% to 75%. Lock-cycle testwork will need to be done in future to determine achievable recoveries and concentrate grades under conditions approaching steady state.

LEACHING TESTS

Leach scoping tests are being completed on the pyrite concentrate (copper rougher scavenger concentrate and copper rougher first cleaner tails scavenger tails) to see what cobalt extractions can be achieved. The following tests are being completed:

- atmospheric leaching using sulphuric acid,
- atmospheric leaching using sulphuric acid and ferric chloride,
- atmospheric leaching using ammonia, and
- pressure leaching using sulphuric acid.

Results will be forwarded to you when they become available.

MELIS
ENGINEERING LTD

Columbia Gold Mines Ltd.
Mr. W.J. Roberts
April 18, 1997

MINERALOGY

As noted in my earlier fax (March 24, 1997), electron microprobe examinations showed that up to 90% of the cobalt in the Fyre Lake mineralization is present as cobaltiferous pyrite. Only a few grains of cobaltite (CoAsS) have been identified.

The mineralogical examination of the Fyre Lake composite will also include an exploratory ion microprobe assessment to check on gold association. I will forward these results when I receive them from Lakefield Research Limited.

The above is a complete update of the current work status. Please let me know if you require a formal report of these initial scoping tests. I can prepare this once the leaching tests are completed and once the mineralogical report is received from Lakefield Research Limited.

Yours truly,
MELIS ENGINEERING LTD.

Lawrence A. Melis, P.Eng.
President

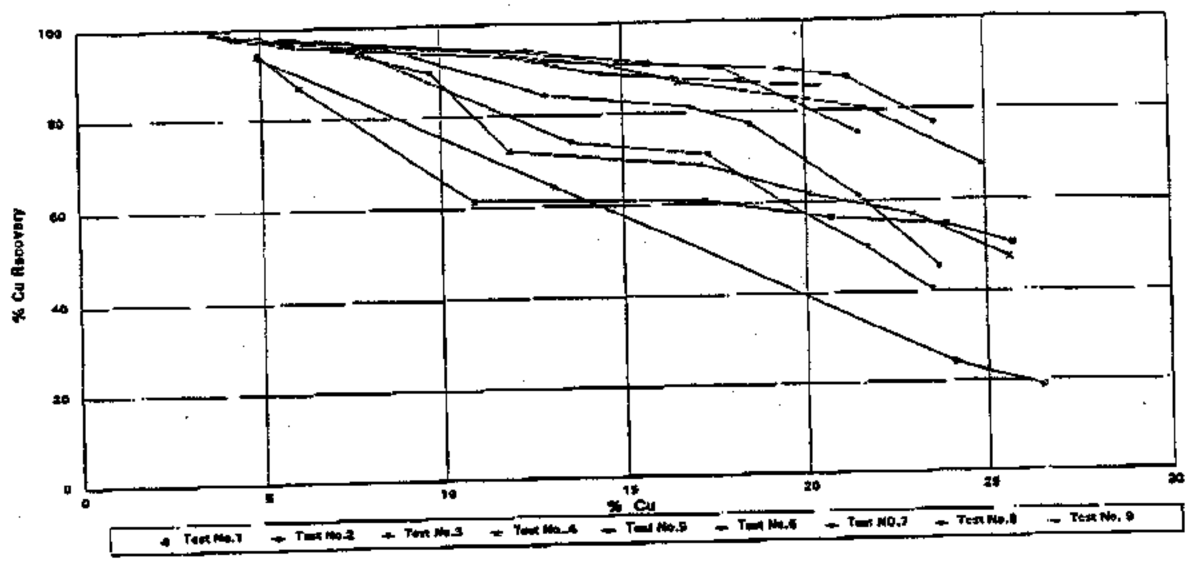
LAM:mb

(f049712.340)

MELIS ENGINEERING LTD.
Project No. 340
April 17, 1997

FIGURE A-1

EYRE LAKE PROJECT
FLOTATION TESTS NO. 1,2,3,4,5,6,7,8 AND 9
COPPER RECOVERY vs % Cu in CONCENTRATE



MELIS ENGINEERING LTD.
Project No. 340
April 17, 1997

FIGURE A-2

EYRE LAKE PROJECT
FLOTATION TESTS NO. 1,2,3,4,5,6,7,8 AND 9
GOLD RECOVERY vs g Au/tonne IN CONCENTRATE

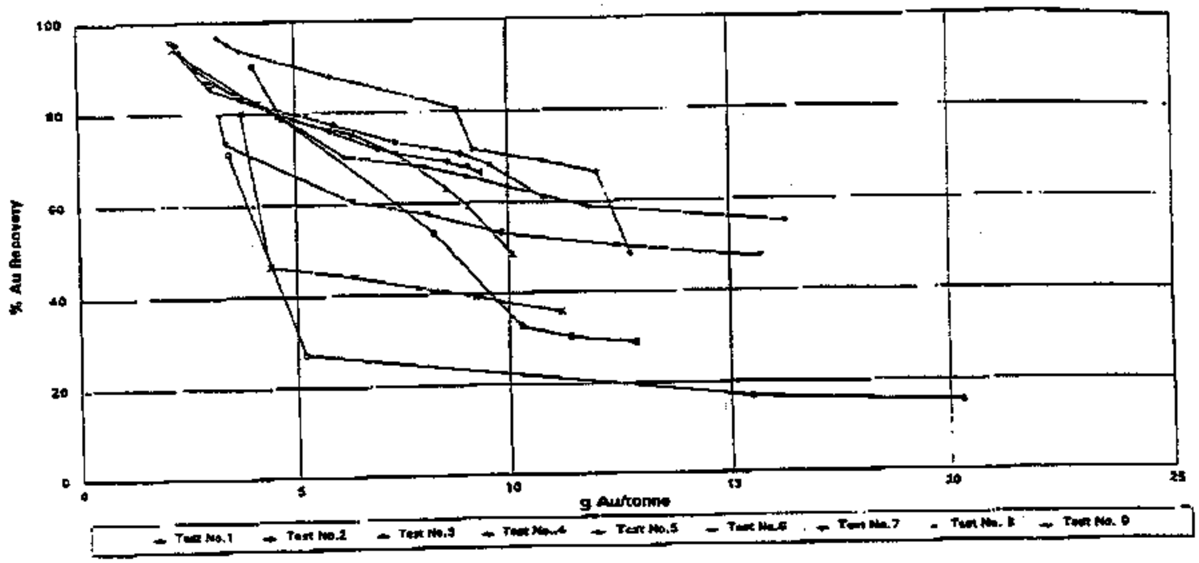
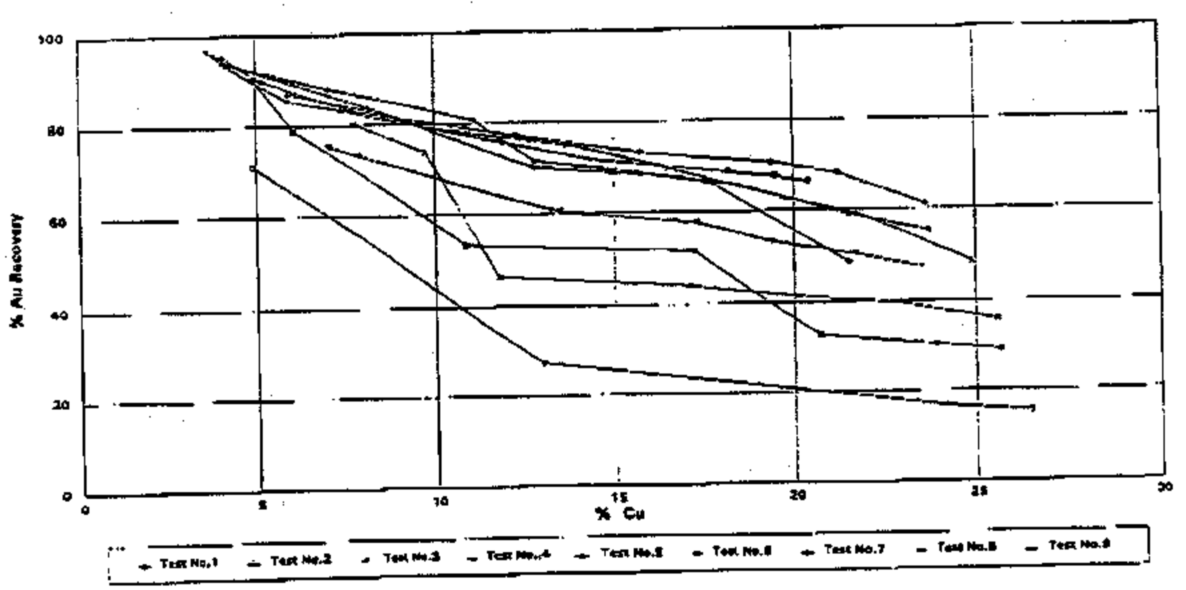


FIGURE A-3

EYRE LAKE PROJECT
FLOTATION TESTS NO. 1,2,3,4,5,6,7,8 AND 9
GOLD RECOVERY vs % Cu IN CONCENTRATE

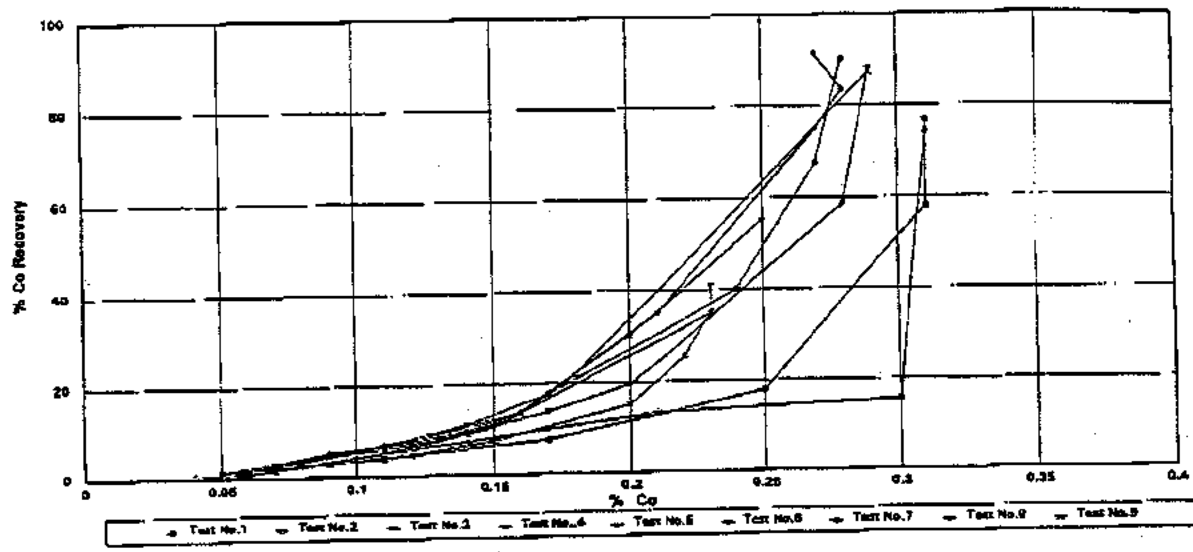


MELIS ENGINEERING LTD.

MELIS ENGINEERING LTD.
Project No. 340
April 17, 1997

FIGURE A-4

EYRE LAKE PROJECT
FLOTATION TESTS NO. 1,2,3,4,5,6,7,8 AND 9
COBALT, RECOVERY vs.% Co IN CONCENTRATE



MELIS ENGINEERING

An Investigation of

The Recovery of Copper, Gold and Cobalt

from a Fyre Lake project sample

submitted by

Columbia Gold Mines Ltd.
per Melis Engineering Ltd.

Progress Report No. 1

Project No. MP-128

NOTE:

This report refers to the samples as received.

The practice of this Company in issuing reports of this nature is to require the recipient not to publish the report or any part thereof without the written consent of Lakefield Research Limited.

LAKEFIELD RESEARCH LIMITED
185 Concession Street, Lakefield, Ontario, K0L 2H0
Tel: (705) 652-2000
Fax: (705) 652-6365

June 23, 1997

Introduction

This report contains the results of flotation testwork conducted on a Fyre Lake project sample, submitted by Columbia Gold Mines Ltd. The objective of the testwork was to evaluate the recovery of a Cu-Au concentrate and a cobalt concentrate. The testwork was directed by Mr. Lawrence Melis, of Melis Engineering Ltd.

Lakefield Research Limited

K.W. Sarbutt

K.W. Sarbutt
Manager - Mineral Processing

Experimental work by: G. Coppaway
Report preparation by: B.J. Scobie

Summary

1. Head Assays

Cu	%	2.07
Co	%	0.16
Zn	%	0.99
Fe	%	26.1
S	%	20.5
Au	g/t	0.98
Ag	g/t	3.70

Al	g/t	15000
As	%	0.034
B	g/t	<8.0
Ba	g/t	130
Be	g/t	<1.0
Bi	g/t	<10.0
Ca	g/t	3400
Cd	g/t	33
Cr	g/t	100
Ga	g/t	47
K	g/t	430
Li	g/t	9
Mg	g/t	13000
Mn	g/t	200
Mo	g/t	<10
Na	g/t	530
Ni	g/t	130
P	g/t	140
Pb	g/t	40
Sb	g/t	0.001
Se	g/t	<50
Si	%	16
Sn	g/t	<20
Sr	g/t	8
Th	%	<0.001
Ti	g/t	319
Tl	g/t	<10
U	%	<0.001
V	g/t	104
W	%	0.007
Zn	g/t	9500

2. Flotation Testwork

Flotation testwork was conducted to evaluate the recovery of Cu-Au and Co concentrates.

Major variables evaluated included fineness of grind and regrind, pH and collector type. A Cu-Au concentrate was recovered at high pH with a selective collector, and then a scavenger concentrate was collected with a xanthate as collector.

The conditions and results of the tests are summarized in Table 1, and Cu grade-recovery curves are shown in Figure 1.

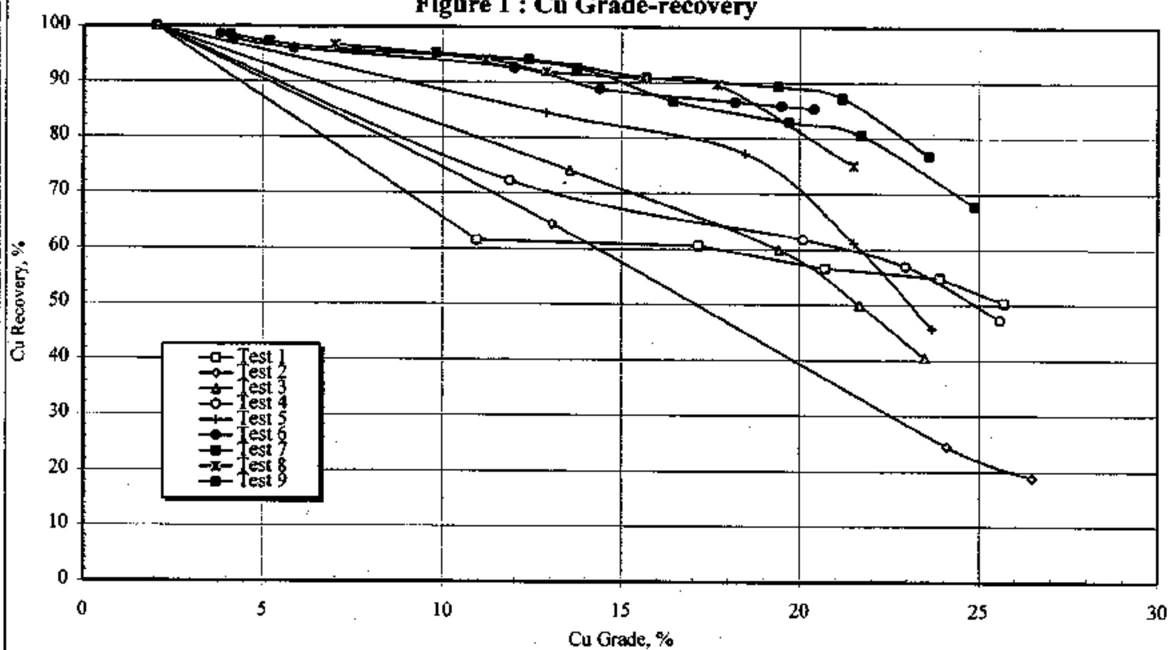
Table 1 : Flotation Conditions and Results

Test	Grind %-200M	Conditions	Product	Weight %	Assays, g/t, % Cu	Au	Co	Fe	Distribution, % Cu	Au	Co	Fe
		Ro pH @ 10.2, 3418 @ 20g/t	Cu Cl Conc	3.99	25.7	12.8	0.05	28.0	50.3	29.1	1.3	4.2
1	80	20g/t SIPX, 8 min	Cu 2nd Cl Conc	4.67	23.9	11.4	0.07	28.1	54.8	30.3	2.0	4.9
		Ro Sc with 10g/t 3418 &	Cu 1st Cl Conc	5.57	20.7	10.3	0.11	29.7	56.6	32.6	3.8	6.2
		10g/t SIPX, 4 min	Cu 1st Cl Conc+Sc Conc	7.20	17.2	12.6	0.17	32.0	60.6	51.6	7.6	8.6
			Cu Ro Conc	11.45	10.9	8.18	0.25	35.3	61.5	53.4	17.9	15.1
		Clnrs @ pH 11.0	Cu Ro Conc+Scav 1	29.41	6.02	4.70	0.31	39.9	86.9	78.9	57.9	43.9
		10g/t NaCN in 3rd Clnr	Cu Ro Conc+Scav1&2	38.98	4.93	4.06	0.31	40.5	94.3	90.3	76.7	59.0
			Cu Ro Sc Tail	61.02	0.19	0.28	0.06	18.0	5.7	9.7	23.3	41.0
			Head (calc)	100.0	2.04	1.75	0.16	26.8				
2	80	Cu Ro pH @ 6.9, R208 @	Cu Cl Conc	1.43	26.5	20.3	0.04	27.5	18.8	15.3	0.4	1.5
		20g/t, 5 min	Cu 1st Cl Conc	2.05	24.1	15.5	0.06	28.3	24.5	16.8	0.8	2.2
		Cu Clnrs @ pH 11.5	Cu Ro Conc	9.97	13.1	5.18	0.21	34.3	64.5	27.3	13.0	12.9
		Py Ro pH @ 6.9	Py Cl Conc	2.33	16.3	25.5	0.16	34.2	18.8	31.4	2.3	3.0
		10g/t 208, 7 min	Py 1st Cl Conc	3.15	13	19.7	0.20	35.9	20.3	32.8	3.8	4.3
		Py Clnr pH @ 11.5	Py Ro Conc	28.57	2.06	2.90	0.34	42.7	29.2	43.8	60.4	45.9
		10g/t 208, 10g/t SIPX in	Cu + Py Cl Conc	3.76	20.2	23.5	0.11	31.7	37.6	46.8	2.7	4.5
		1st Clnr	Py Ro Tail	61.46	0.21	0.89	0.07	17.8	6.4	28.9	26.6	41.2
			Head (calc)	100.0	2.02	1.89	0.16	26.5				
3	80	Pre-gravity conc. of Au	Cu Cl conc	3.48	23.5	15.7	0.07	28.4	40.4	47.5	1.5	3.7
		Cu Ro pH @ 11.2,	Cu 2nd Cl Conc	4.66	21.7	12.4	0.08	28.9	49.9	50.3	2.4	5.0
		20g/t 3501, 5g/t SIPX, 11mi	Cu 1st Cl Conc	6.24	19.4	9.83	0.10	29.9	59.9	53.3	3.9	6.9
		Cu Ro Sc pH @ 10.6	Cu 1st Cl Conc+Sc Conc	8.27	17.3	8.03	0.12	31.2	70.7	57.6	6.2	9.6
		20g/t SIPX, 7 min	Cu Ro Conc	11.06	13.6	6.29	0.15	32.4	74.0	60.4	10.4	13.3
		Cu Clnr pH @ 11.5	Cu Ro Conc+Scav1	24.57	7.77	3.44	0.23	37.4	94.2	73.3	35.1	34.2
			Cu Ro Conc+Scav1 & 2	27.62	6.96	3.14	0.23	37.9	95.0	75.4	40.8	38.9
			Cu Ro Conc+ Sc's+Moz	27.67	na	3.30	na	na	na	79.3	na	na
			Cu Ro Tail	72.33	0.14	0.33	0.13	22.7	5.0	20.7	59.2	61.1
			Head (calc)	100.0	2.02	1.15	0.16	26.9				
4	80	Cu Ro pH @ 11.0, 5100 @	Cu Cl conc	3.77	25.6	11.2	0.07	29.5	47.3	35.8	1.6	4.2
		20g/t, 8 min	Cu 2nd Cl Conc	5.06	23.0	9.20	0.09	29.7	57.0	39.5	2.8	5.7
		Cu Sc pH @ 10.9, 208 @	Cu 1st Cl Conc	6.26	20.1	7.78	0.12	30.4	61.8	41.4	4.5	7.2
		10g/t, 8 min	Cu 1st Cl Conc + Scav	8.21	17.1	6.31	0.15	31.5	68.6	44.0	7.5	9.8
		Cu Clnr pH @ 11.5	Cu Ro Conc	12.41	11.9	4.41	0.20	33.1	72.2	46.5	15.0	15.5
		NaCN in RG, 2nd &	Cu Ro Conc+Scav 1	18.83	9.73	4.62	0.22	35.0	89.8	73.8	25.7	24.9
		3rd Clnr	Cu Ro Conc+Scav 1 & 2	24.96	7.66	3.79	0.23	35.7	93.7	80.2	35.2	33.6
			Cu Ro Tail	75.04	0.17	0.31	0.14	23.4	6.3	19.8	64.8	66.4
			Head (calc)	100.0	2.04	1.18	0.16	26.5				
5	80	Cu Ro pH @ 11.0, 3501 @	Cu Cl conc	3.88	23.7	16.3	0.05	27.4	45.7	55.2	1.3	4.0
		30g/t, SIPX @ 5g/t, 11 min	Cu 2nd Cl Conc	5.72	21.5	11.8	0.07	28.2	61.2	59.0	2.8	6.1
		Cu Sc pH @ 10.5,	Cu 1st Cl Conc	8.40	18.5	8.98	0.10	29.5	77.2	65.8	5.6	9.4
		SIPX 20g/t, 8 min	Cu 1st Cl conc + Scav	9.70	16.8	8.00	0.12	30.6	81.2	67.8	7.7	11.2
		Clnr pH @ 11.5, 3501 in 1st	Cu Ro Conc	13.16	12.9	6.10	0.16	31.9	84.4	70.2	13.7	15.8
		Clnr@ 2.5g/t	Cu Ro Conc+Scav 1	22.48	8.50	4.19	0.20	35.1	95.1	82.4	30.4	29.8
			Cu Ro Conc+Scav 1 & 2	32.88	5.87	3.03	0.25	37.9	96.0	87.1	55.3	47.0
			Cu Ro Tail	67.12	0.12	0.22	0.10	20.9	4.0	12.9	44.7	53.0
			Head (calc)	100.0	2.01	1.14	0.15	26.5				
6	78	Cu Ro pH @ 11.0, 3501 @	Cu Cl conc	8.53	20.4	9.35	0.11	30.0	85.3	66.4	5.9	9.6
		40g/t, SIPX @ 10g/t, 15 min	Cu 3rd Cl Conc	8.98	19.5	9.03	0.12	30.4	85.8	67.6	6.7	10.3
		Cu Sc pH @ 11.0,	Cu 2nd Cl Conc	9.67	18.2	8.53	0.13	31.1	86.5	68.7	8.0	11.3
		SIPX 20g/t, 6 min	Cu 1st Cl Conc	12.56	14.4	6.86	0.17	32.8	88.8	71.8	13.7	15.5
		Clnr pH @ 11.5, 3501 in 1st	Cu 1st Cl conc + Scav	15.77	12.0	5.77	0.20	34.4	92.6	75.8	19.5	20.4
		Clnrs @ 5.0g/t	Cu Ro Conc	33.50	5.85	3.06	0.28	38.3	96.0	85.5	58.5	48.3
		3501 @ 2.5g/t in 2nd & 3rd	Cu Ro Conc+Scav 1	47.60	4.19	2.35	0.29	39.8	97.7	93.3	87.8	71.2
		Clnrs. NaCN in 4th Clnr	Cu Ro Conc+Scav 1 & 2	52.57	3.83	2.18	0.28	39.0	98.7	95.7	94.0	77.0
			Cu Ro Tail	47.43	0.055	0.11	0.02	12.9	1.3	4.3	6.0	23.0
			Head (calc)	100.0	2.04	1.20	0.16	26.6				

Table 1 : Flotation Conditions and Results (cont.)

Test	Grind %-200M	Conditions	Product	Weight %	Assays, g/t, % Cu	Au	Co	Fe	Distribution, % Cu	Au	Co	Fe
7	89	Cu Ro pH @ 11.0, 3501 @	Cu Cl conc	6.72	23.6	10.8	0.07	28.5	76.9	61.1	3.0	7.1
		40g/t, SIPX @ 10g/t, 14 min	Cu 3rd Cl Conc	8.47	21.2	9.56	0.09	29.1	87.2	68.2	5.1	9.2
		Cu Sc pH @ 11.0,	Cu 2nd Cl Conc	9.50	19.4	8.85	0.11	30.2	89.3	70.7	6.8	10.6
		SIPX 20g/t, 4 min	Cu 1st Cl Conc	11.96	15.7	7.27	0.14	31.7	90.9	73.2	11.1	14.1
		Clnr pH @ 11.5, 3501 in 1st	Cu 1st Cl conc + Scav	15.67	12.4	5.86	0.17	33.5	94.1	77.3	17.6	19.5
		Clnrs @ 5.0g/t	Cu Ro Conc	25.99	7.59	3.80	0.24	36.5	95.6	83.1	40.2	35.2
		3501 @ 2.5g/t in 2nd & 3rd	Cu Ro Conc+Scav 1	38.90	5.16	2.75	0.27	38.6	97.2	89.9	67.6	55.8
		Clnrs. NaCN in 4th Clnr	Cu Ro Conc+Scav 1 & 2	49.54	4.10	2.29	0.28	39.5	98.5	95.3	90.2	72.7
			Cu Ro Tail	50.46	0.06	0.11	0.03	14.6	1.5	4.7	9.8	27.3
			Head (calc)	100.0	2.06	1.19	0.16	26.9				
8	93	Cu Ro pH @ 11.0, 3501 @	Cu Cl conc	6.96	21.5	12.7	0.08	28.1	75.0	48.6	3.4	7.4
		45g/t, SIPX @ 10g/t, 14 min	Cu 3rd Cl Conc	10.1	17.7	12.0	0.12	29.9	89.7	66.4	7.3	11.4
		Cu Sc pH @ 11.0,	Cu 2nd Cl Conc	11.61	15.6	10.8	0.14	30.8	90.7	68.9	9.7	13.5
		SIPX 20g/t, 4 min	Cu 1st Cl Conc	14.23	12.9	9.16	0.16	31.7	91.9	71.6	13.8	17.0
		Clnr pH @ 11.5, 3501 in 1st	Cu 1st Cl conc + Scav	16.82	11.2	8.74	0.17	32.7	94.1	80.7	17.8	20.7
		Clnrs @ 5.0g/t	Cu Ro Conc	27.66	6.98	5.79	0.21	34.4	96.7	87.9	35.1	35.8
		3501 @ 2.5g/t in 2nd & 3rd	Cu Ro Conc+Sc Cl Conc	45.11	4.33	3.78	0.27	38.4	97.9	93.6	74.9	65.4
		Clnrs. Scav Clnrs water only	Cu Ro Conc+Ro Sc Conc	53.98	3.66	3.26	0.27	38.2	99.0	96.7	91.5	77.8
			Ro Scav Conc	26.32	0.17	0.61	0.35	42.3	2.3	8.8	56.4	42.0
		and elevated pH.	Cu Ro Tail	46.02	0.043	0.13	0.03	12.8	1.0	3.3	8.5	22.2
			Head (calc)	100.0	2.00	1.82	0.16	26.5				
9	93	Cu Ro pH @ 11.0, 3501 @	Cu Cl conc	5.52	24.9	10.1	0.07	29.1	67.8	48.6	2.4	6.0
		45g/t, SIPX @ 10g/t, 14 min	Cu 3rd Cl Conc	7.51	21.7	9.0	0.09	29.6	80.5	59.2	4.4	8.3
		Cu Sc pH @ 11.0,	Cu 2nd Cl Conc	8.52	19.7	8.5	0.11	30.3	82.8	63.1	5.8	9.6
		SIPX 20g/t, 4 min	Cu 1st Cl Conc	10.68	16.5	7.5	0.14	31.3	86.7	69.7	9.1	12.5
		Clnr pH @ 11.5, 3501 in 1st	Cu 1st Cl Conc + Sc Conc	13.65	13.8	6.3	0.16	32.7	92.5	75.1	13.5	16.7
		Clnrs @ 5.0g/t	Cu Ro Conc	19.70	9.8	4.6	0.18	33.2	95.3	79.2	21.9	24.4
		3501 @ 2.5g/t in 2nd & 3rd	Cu Ro Conc + Ro Sc Conc	48.74	4.1	2.2	0.29	39.1	98.6	94.2	87.2	71.1
		Clnrs. Scav Clnrs water only	Cu 1st Cl Sc Tail + Ro Sc C.	35.10	0.3	0.6	0.34	41.6	6.0	19.1	73.7	54.5
			Head (calc)	100.0	2.03	1.15	0.16	26.8				

Figure 1 : Cu Grade-recovery



Copper concentrate of 20% Cu at 85-90% Cu recovery could be attained. Gold recovery was 60-65% at this grade.

Little concentration of Co could be obtained, and Co and Fe recoveries were similar, indicating that most of the Co was associated with pyrite.

A detailed analysis was conducted on the Cu concentrate from Test 7:

		Test 7 Cu Cl Conc
Cu	%	23.6
Au	g/t	10.8
Co	%	0.07
Fe	%	28.5
Mo	%	0.002
Pt	g/t	0.73
Pd	g/t	0.13
Ag	g/t	30.9
Pb	%	0.007
Zn	%	9.1
Ni	%	0.008
As	%	0.028
Sb	%	0.001
Se	%	0.0085
Bi	%	0.005
Te	%	< 0.0003
Hg	g/t	41
SiO_2	%	0.70
CaO	%	0.14
MgO	%	0.075
Al_2O_3	%	0.16
S	%	37.2
F	%	< 0.01
Cl	g/t	46
Ag	g/t	30.9

Details of Tests

Test No.1 Project No. MP 128 Operator: G.C. Date: Feb.12, 1997

Purpose: First in a series of tests, to evaluate primary flotation of Cu Au from Columbia Gold Mines sample submitted by Melis Engineering.

Procedure: Ore was ground and floated as described below. Products were filtered dried, and submitted for assay.

Feed: 2kg - 10 mesh composite sample

Grind: 30min/2kg in the lab ball mill at 65% solids.

Conditions:

	Reagents added, grams per tonne						Time, minutes			
Stage	Lime	NaCN	3418	SIPX	MIBC		Grind	Cond.	Froth	pH
Primary Grind	1000						30			7.6
Cu Ro	625		10	10	5			2	4	10.2
			10	10				1	4	10.2
Cu Ro Scav 1			10	10	2.5			1	2	10.0
Cu Ro Scav 2									2	10.0
Cu Ro RG	1000						20			11.2
Cu 1st Cl								1	6	11.2
Cu 1st Cl Scav			5		2.5			1	2	11.0
Cu 2nd Cl	35							1	4	11.5
Cu 3rd Cl	80	10						1	3	11.0

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D -1
Speed: r.p.m.	1800	1500

Product	**Weight**			**Assays g/t, %**				**Distribution, %**		
	g	**%**	**Cu**	**Au**	**Co**	**Fe**	**Cu**	**Au**	**Co**	**Fe**
Cu Cl Conc	78.8	3.99	25.7	12.8	0.05	28.0	50.3	29.1	1.3	4.2
Cu 3rd Cl Tail	13.4	0.68	13.4	2.95	0.16	28.9	4.5	1.1	0.7	0.7
Cu 2nd Cl Tail	17.7	0.90	4.07	4.58	0.32	37.9	1.8	2.3	1.8	1.3
Cu 1st Cl Sc Conc	32.1	1.63	5.02	20.4	0.37	40.0	4.0	18.9	3.8	2.4
Cu 1st Cl Sc Tail	83.9	4.25	0.42	0.75	0.38	40.8	0.9	1.8	10.3	6.5
Cu Ro Sc Conc 1	354.3	17.96	2.88	2.49	0.35	42.9	25.4	25.5	40.0	28.8
Cu Ro Sc Conc 2	188.8	9.57	1.59	2.09	0.31	42.3	7.5	11.4	18.9	15.1
Cu Ro Tail	1203.6	61.02	0.19	0.28	0.06	18.0	5.7	9.7	23.3	41.0
Head (calc.)	1972.6	100.00	2.04	1.75	0.16	26.8	100.0	100.0	100.0	100.0
(direct)										

Combined Products									
Cu 2nd Cl Conc	4.67	23.9	11.4	0.07	28.1	54.8	30.3	2.0	4.9
Cu 1st Cl Conc	5.57	20.7	10.3	0.11	29.7	56.6	32.6	3.8	6.2
Cu 1st Cl Conc + Sc Conc	7.20	17.2	12.6	0.17	32.0	60.6	51.6	7.6	8.6
Cu Ro Conc	11.45	10.9	8.18	0.25	35.3	61.5	53.4	17.9	15.1
Cu Ro Conc + Ro Sc 1	29.41	6.02	4.70	0.31	39.9	86.9	78.9	57.9	43.9
Cu Ro Conc + Ro Sc 1 + 2	38.98	4.93	4.06	0.31	40.5	94.3	90.3	76.7	59.0

Test No.2 Project No. MP 128 Operator: G.C. Date: Feb.12, 1997

Purpose: Evaluate Cu Au flotation using a natural pH Cu circuit, followed by a secondary Py flotation circuit.

Procedure: Ore was ground and floated as described below. Products were filtered dried, and submitted for assay.

Feed: 2kg - 10 mesh composite sample

Grind: Conditions: 30min/2kg in the lab ball mill at 65% solids.

	Reagents added, grams per tonne						Time, minutes			
Stage	Lime	NaCN	208	SIPX	MIBC		Grind	Cond.	Froth	pH
Primary Grind							30			6.9
Cu Ro			10		7.5			1	4	6.9
			10					1	1	6.9
Py Ro					5			1	4	6.9
			10					1	3	6.9
Cu Ro RG	1000						15			11.6
Cu 1st Cl								1	4	11.5
Cu 2nd Cl	150				2.5			1	3	11.5
Py Rg	1000	50					30			11.5
Py 1st Cl			10	10				1	3	11.5
Py 2nd Cl								1	3	10.9

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D-1
Speed: r.p.m.	1800	1500

Product	Weight g	Weight %	Assays g/t, % Cu	Au	Co	Fe	Distribution, % Cu	Au	Co	Fe
Cu Cl Conc	28.3	1.43	26.5	20.3	0.04	27.5	18.8	15.3	0.4	1.5
Cu 2nd Cl Tail	12.3	0.62	18.6	4.45	0.11	30.1	5.7	1.5	0.4	0.7
Cu 1st Cl Tail	156.7	7.92	10.2	2.51	0.25	35.8	40.0	10.5	12.2	10.7
Py Cl Conc	46.2	2.33	16.3	25.5	0.16	34.2	18.8	31.4	2.3	3.0
Py 2nd Cl Tail	16.2	0.82	3.73	3.13	0.30	40.6	1.5	1.4	1.5	1.3
Py 1st Cl Tail	502.9	25.41	0.70	0.82	0.36	43.5	8.8	11.0	56.6	41.7
Py Ro Tail	1216.3	61.46	0.21	0.89	0.07	17.8	6.4	28.9	26.6	41.2
Head (calc.)	1978.9	100.00	2.02	1.89	0.16	26.5	####	100.0	100.0	100.0
(direct)										
Combined Products										
Cu 1st Cl Conc		2.05	24.1	15.5	0.06	28.3	24.5	16.8	0.8	2.2
Cu Ro Conc		9.97	13.1	5.18	0.21	34.3	64.5	27.3	13.0	12.9
Py 1st Cl Conc		3.15	13.0	19.7	0.20	35.9	20.3	32.8	3.8	4.3
Py Ro Conc		28.57	2.06	2.90	0.34	42.7	29.2	43.8	60.4	45.9
Cu + Py Cl Conc		3.76	20.2	23.5	0.11	31.7	37.6	46.8	2.7	4.5

Test No.3 Project No. MP 128 Operator: G.C. Date: Feb.12, 1997

Purpose: Evaluate effect of elevated pH on Cu flotation of a post Knelson concentrator tailing.

Procedure: Preconcentration of Au was by using a Knelson concentrator with the water flow set at 3 psi. This concentrate was upgraded on a Mozley Mineral Separator with the water flow at 2.5 gal/min. The combined gravity tailing products were the feed to the flotation circuit. Flotation was carried out as described below.

Feed: 2kg of Knelson concentrator tailings

Grind: Pre-gravity grind of 15 min/2kg at 65% solids in ball mill
Pre-flot grind of 15 min/2kg at 65% solids in ball mill. (80% - 200 mesh)

Conditions:

	Reagents added, grams per tonne						Time, minutes			
Stage	Lime		3501	SIPX	MIBC		Grind	Cond.	Froth	pH
Primary Grind							15			
Gravity testwork										
Flot grind	2000						15			11.2
Cu Ro			10		5			1	4	11.2
			5		2.5				4	
			5	5	5			1	3	11.0
Cu Ro Scav 1				10				1	4	
Cu Ro Scav 2				10				1	3	10.6
Cu Ro RG	1000						20			11.5
Cu 1st Cl								1	4	11.5
Cu 1st Cl Scav			5		2.5			1	3	11.5
Cu 2nd Cl	60							1	3	11.3
Cu 3rd Cl	90							1	3	11.3

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D -1
Speed: r.p.m.	1800	1500

Product	Weight		Assays g/t , %				Distribution, %			
	g	%	Cu	Au	Co	Fe	Cu	Au	Co	Fe
1 Mozley Conc	0.9	0.05		98.6				3.9		
2 Cu Cl Conc	68.6	3.48	23.5	15.7	0.07	28.4	40.4	47.5	1.5	3.7
3 Cu 3rd Cl Tail	23.3	1.18	16.3	2.80	0.12	30.5	9.5	2.9	0.9	1.3
4 Cu 2nd Cl Tail	31.0	1.57	12.8	2.14	0.15	32.8	9.9	2.9	1.5	1.9
5 Cu 1st Cl Sc Conc	40.0	2.03	10.8	2.48	0.18	35.2	10.8	4.4	2.3	2.7
6 Cu 1st Cl Sc Tail	55.0	2.79	2.42	1.14	0.24	35.8	3.3	2.8	4.2	3.7
7 Cu Ro Sc Conc 1	266.2	13.51	3.03	1.10	0.29	41.5	20.2	12.9	24.6	20.9
8 Cu Ro Sc Conc 2	60.2	3.06	0.50	0.77	0.30	41.6	0.8	2.0	5.8	4.7
9 Cu Ro Tail	1425.2	72.33	0.14	0.33	0.13	22.7	5.0	20.7	59.2	61.1
Head (calc.)	1970.4	100.00	2.02	1.15	0.16	26.9	100.0	100.0	100.0	100.0
(direct)										
Combined Products										
Cu 2nd Cl Conc		4.66	21.7	12.4	0.08	28.9	49.9	50.3	2.4	5.0
Cu 1st Cl Conc		6.24	19.4	9.83	0.10	29.9	59.9	53.3	3.9	6.9
Cu 1st Cl Conc + Sc Conc		8.27	17.3	8.03	0.12	31.2	70.7	57.6	6.2	9.6
Cu Ro Conc		11.06	13.6	6.29	0.15	32.4	74.0	60.4	10.4	13.3
Cu Ro Conc + Ro Sc1		24.57	7.77	3.44	0.23	37.4	94.2	73.3	35.1	34.2
Cu Ro Conc + Ro Sc1 + 2		27.62	6.96	3.14	0.23	37.9	95.0	75.4	40.8	38.9
Cu Ro Conc + Ro Sc1 + 2 + Moz		27.67		3.30				79.3		

Company **Lakefield Research** **LR-0**

Size Distribution Analysis

Product: Cu Ro. Scav. Tail **Test No:** 3

Microns	Mesh	Weight Grams	% Weight Ind.	% Weight Cum.	% Weight Passing
75	200	36.0	20.2	20.2	79.8
-75	-200	141.8	79.8	100.0	-
	Total	177.8	100.0	-	-

Test No.4 | Project No. MP 128 | Operator: G.C. | Date: Feb.12, 1997

Purpose: Same as test 3, but omit preconcentration of flotation feed and use 5100 as primary collector.

Procedure: Ore was ground and floated as described below. Products were filtered dried, and submitted for assay.

Feed: 2kg - 10 mesh composite sample

Grind: 30min/2kg in the lab ball mill at 65% solids.

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			
	Lime	NaCN	5100	208	MIBC		Grind	Cond.	Froth	pH
Primary Grind	2000						30			9.1
Cu Ro	590		10		5			1	4	11.0
			10					1	4	11.0
Cu Ro Scav 1				10	2.5			1	4	10.9
Cu Ro Scav 2				10				1	4	10.8
Cu Ro RG	1000	25					15			11.5
Cu 1st Cl								1	4	11.4
Cu 1st Cl Scav	100			5	2.5			1	3	11.4
Cu 2nd Cl	50	5						1	3	11.4
Cu 3rd Cl	60	5						1	3	11.5

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D -1
Speed: r.p.m.	1800	1500

Product	Weight		Assays g/t, %				Distribution, %			
	g	%	Cu	Au	Co	Fe	Cu	Au	Co	Fe
1 Cu Cl Conc	74.7	3.77	25.6	11.2	0.07	29.5	47.3	35.8	1.6	4.2
2 Cu 3rd Cl Tail	25.6	1.29	15.3	3.36	0.15	30.1	9.7	3.7	1.2	1.5
3 Cu 2nd Cl Tail	23.9	1.21	8.13	1.83	0.23	33.5	4.8	1.9	1.7	1.5
4 Cu 1st Cl Sc Conc	38.6	1.95	7.21	1.59	0.25	35.1	6.9	2.6	3.0	2.6
5 Cu 1st Cl Sc Tail	83.3	4.20	1.75	0.69	0.29	36.3	3.6	2.5	7.5	5.8
6 Cu Ro Sc Conc 1	127.3	6.42	5.57	5.02	0.27	38.5	17.5	27.4	10.7	9.3
7 Cu Ro Sc Conc 2	121.6	6.13	1.32	1.23	0.25	37.8	4.0	6.4	9.5	8.8
8 Cu Ro Tail	1488.1	75.04	0.17	0.31	0.14	23.4	6.3	19.8	64.8	66.4
Head (calc.)	1983.1	100.00	2.04	1.18	0.16	26.5	100.0	100.0	100.0	100.0
(direct)										
Cu 2nd Cl Conc		5.06	23.0	9.20	0.09	29.7	57.0	39.5	2.8	5.7
Cu 1st Cl Conc		6.26	20.1	7.78	0.12	30.4	61.8	41.4	4.5	7.2
Cu 1st Cl Conc + Sc Conc		8.21	17.1	6.31	0.15	31.5	68.6	44.0	7.5	9.8
Cu Ro Conc		12.41	11.9	4.41	0.20	33.1	72.2	46.5	15.0	15.5
Cu Ro Conc + Ro Sc 1		18.83	9.73	4.62	0.22	35.0	89.8	73.8	25.7	24.9
Cu Ro Conc + Ro Sc 1 + 2		24.96	7.66	3.79	0.23	35.7	93.7	80.2	35.2	33.6

Test No.5 | Project No. MP 128 | Operator: G.C. | Date: Feb.27, 1997

Purpose: Repeat Test 3, but omit gravity circuit, and slightly increase collector in Ro and 1st Clnr to gain and keep the Ro recovery through the cleaner circuit.

Procedure: Ore was ground and floated as described below. Products were filtered dried, and submitted for assay.

Feed: 2kg - 10 mesh composite sample

Grind: 30min/2kg in the lab ball mill at 65% solids.

Conditions:

		Reagents added, grams per tonne						Time, minutes			
Stage	Lime		3501	SIPX		MIBC		Grind	Cond.	Froth	pH
Primary Grind	2000							30			11.3
Cu Ro			10			5			1	4	11.2
			10			5			1	4	11.1
			10	5					1	3	
Cu Ro Scav 1				10		2.5			1	4	10.6
Cu Ro Scav 2				10		2.5			1	4	10.4
Cu Ro RG	1000							20			11.5
Cu 1st Cl			2.5			2.5			1	4	11.5
Cu 1st Cl Scav			5			2.5			1	4	11.5
Cu 2nd Cl	130								1	3	11.5
Cu 3rd Cl	150								1	3	11.5

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D -1
Speed: r.p.m.	1800	1500

Product	Weight		Assays g/t , %				Distribution, %			
	g	%	Cu	Au	Co	Fe	Cu	Au	Co	Fe
1 Cu Cl Conc	76.9	3.88	23.7	16.3	0.05	27.4	45.7	55.2	1.3	4.0
2 Cu 3rd Cl Tail	36.5	1.84	16.9	2.34	0.12	30.0	15.5	3.8	1.5	2.1
3 Cu 2nd Cl Tail	53.2	2.68	12.0	2.94	0.16	32.2	16.0	6.9	2.9	3.3
4 Cu 1st Cl Sc Conc	25.8	1.30	6.13	1.70	0.24	37.9	4.0	1.9	2.1	1.9
5 Cu 1st Cl Sc Tail	68.8	3.47	1.91	0.79	0.26	35.3	3.3	2.4	6.0	4.6
6 Cu Ro Sc Conc 1	184.8	9.31	2.29	1.50	0.27	39.6	10.6	12.2	16.7	13.9
7 Cu Ro Sc Conc 2	206.5	10.41	0.18	0.52	0.36	44.0	0.9	4.7	24.9	17.3
8 Cu Ro Tail	1331.7	67.12	0.12	0.22	0.10	20.9	4.0	12.9	44.7	53.0
Head (calc.)	1984.2	100.00	2.01	1.14	0.15	26.5	100.0	100.0	100.0	100.0
(direct)										
Cu 2nd Cl Conc		5.72	21.5	11.8	0.07	28.2	61.2	59.0	2.8	6.1
Cu 1st Cl Conc		8.40	18.5	8.98	0.10	29.5	77.2	65.8	5.6	9.4
Cu 1st Cl Conc + Sc Conc		9.70	16.8	8.00	0.12	30.6	81.2	67.8	7.7	11.2
Cu Ro Conc		13.16	12.9	6.10	0.16	31.9	84.4	70.2	13.7	15.8
Cu Ro Conc + Ro Sc 1		22.48	8.50	4.19	0.20	35.1	95.1	82.4	30.4	29.8
Cu Ro Conc + Ro Sc 1 + 2		32.88	5.87	3.03	0.25	37.9	96.0	87.1	55.3	47.0

Test No.6 | Project No. MP 128 | Operator: G.C. | Date: Mar.13, 1997

Purpose: Repeat of test 5, but an additional Ro stage with collector, to target 20% weight recovery and collector additions throughout the cleaners to maintain the recovery.

Procedure: Ore was ground and floated as described below. Products were filtered dried, and submitted for assay.

Feed: 2kg - 10 mesh composite sample

Grind: 30min/2kg in the lab ball mill at 65% solids.

Conditions:

		Reagents added, grams per tonne						Time, minutes			
Stage	Lime	NaCN	3501	SIPX		MIBC		Grind	Cond.	Froth	pH
Primary Grind	2000							30			11.1
Cu Ro			10			5			1	4	11.1
			10			5			1	4	11.0
			10	5					1	4	
			10	5					1	3	11.0
Cu Ro Scav 1				10		2.5			1	3	10.8
Cu Ro Scav 2				10		2.5			1	3	10.7
Cu Ro RG	1500							30			11.5
Cu 1st Cl			5						1	4	11.5
Cu 1st Cl Scav			5			2.5			1	3	11.5
Cu 2nd Cl			2.5						1	4	11.5
Cu 3rd Cl	150		2.5						1	4	11.3
Cu 4th Cl		10				2.5			1	3	11.2

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D-1
Speed: r.p.m.	1800	1500

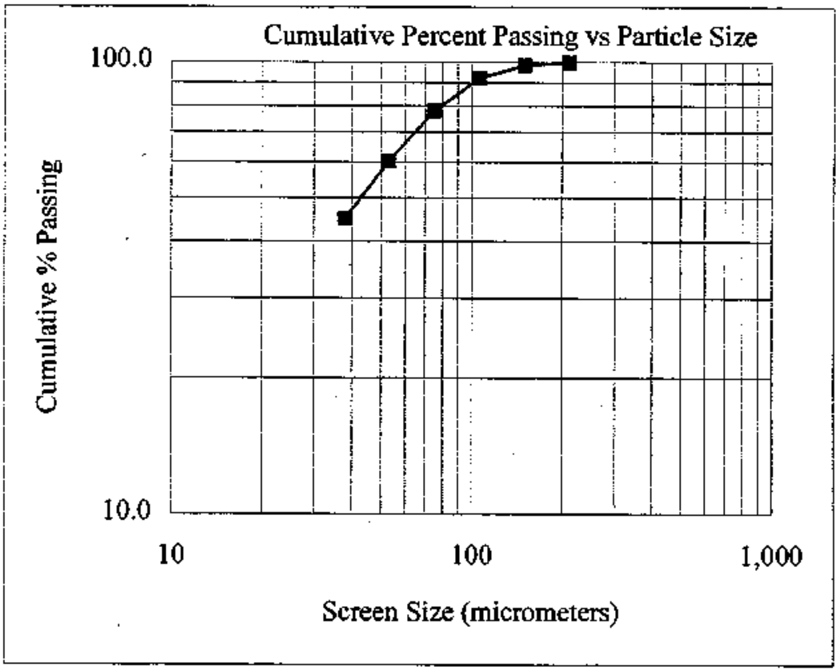
Product	**Weight**			**Assays g/t, %**				**Distribution, %**		
	g	**%**	**Cu**	**Au**	**Co**	**Fe**	**Cu**	**Au**	**Co**	**Fe**
1 Cu Cl Conc	168.2	8.53	20.4	9.35	0.11	30.0	85.3	66.4	5.9	9.6
2 Cu 4th Cl Tail	9.0	0.46	2.37	3.12	0.28	38.6	0.5	1.2	0.8	0.7
3 Cu 3rd Cl Tail	13.5	0.68	2.03	1.96	0.31	39.3	0.7	1.1	1.3	1.0
4 Cu 2nd Cl Tail	57.0	2.89	1.63	1.26	0.31	38.7	2.3	3.0	5.6	4.2
5 Cu 1st Cl Sc Conc	63.3	3.21	2.42	1.50	0.29	40.8	3.8	4.0	5.9	4.9
6 Cu 1st Cl Sc Tail	349.8	17.73	0.39	0.66	0.35	41.8	3.4	9.8	39.0	27.8
7 Cu Ro Sc Conc 1	278.1	14.10	0.25	0.66	0.33	43.4	1.7	7.8	29.2	23.0
8 Cu Ro Sc Conc 2	98.1	4.97	0.42	0.57	0.20	30.9	1.0	2.4	6.3	5.8
9 Cu Ro Tail	935.6	47.43	0.055	0.11	0.02	12.9	1.3	4.3	6.0	23.0
Head (calc.)	1972.6	100.00	2.04	1.20	0.16	26.6	100.0	100.0	100.0	100.0
(direct)										
Combined Products										
Cu 3rd Cl Conc		8.98	19.5	9.03	0.12	30.4	85.8	67.6	6.7	10.3
Cu 2nd Cl Conc		9.67	18.2	8.53	0.13	31.1	86.5	68.7	8.0	11.3
Cu 1st Cl Conc		12.56	14.4	6.86	0.17	32.8	88.8	71.8	13.7	15.5
Cu 1st Cl Conc + Sc Conc		15.77	12.0	5.77	0.20	34.4	92.6	75.8	19.5	20.4
Cu Ro Conc		33.50	5.85	3.06	0.28	38.3	96.0	85.5	58.5	48.3
Cu Ro Conc + Ro Sc 1		47.60	4.19	2.35	0.29	39.8	97.7	93.3	87.8	71.2
Cu Ro Conc + Ro Scavs		52.57	3.83	2.18	0.28	39.0	98.7	95.7	94.0	77.0

Company **Lakefield Research** **MP 128**

Size Distribution Analysis

Sample: Comb Prod **Test No.:** 6

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.2	0.1	0.1	99.9
100	150	2.5	1.6	1.7	98.3
150	106	9.8	6.1	7.8	92.2
200	75	22.9	14.3	22.1	77.9
270	53	28.2	17.6	39.8	60.3
400	38	24.5	15.3	55.1	44.9
Pan	-38	71.9	44.9	100.0	0.0
Total	-	**160.0**	**100.0**	-	-
K80	**80**				

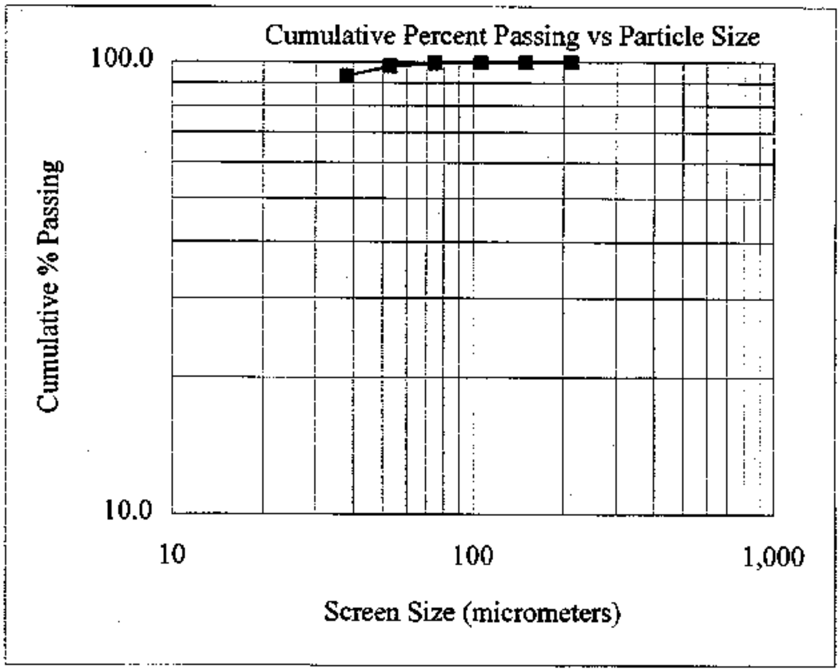


Company **Lakefield Research** **MP 128**

Size Distribution Analysis

Sample: Comb Cl Prod **Test No.:** 6

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.0	0.0	0.0	100.0
100	150	0.0	0.0	0.0	100.0
150	106	0.1	0.1	0.1	99.9
200	75	0.6	0.4	0.4	99.6
270	53	2.6	1.6	2.1	97.9
400	38	7.7	4.8	6.9	93.1
Pan	-38	149.0	93.1	100.0	0.0
Total	-	**160.0**	**100.0**	-	-
K80	**#DIV/0!**				



Test No.7 Project No. MP 128 Operator: G.C. Date: Mar.13, 1997

Purpose: Repeat of test 6 with a finer primary grind.

Procedure: Ore was ground and floated as described below. Products were filtered dried, and submitted for assay.

Feed: 2kg - 10 mesh composite sample

Grind: 40min/2kg in the lab ball mill at 65% solids.

Conditions:

Stage	Lime	Reagents added, grams per tonne						Time, minutes			pH
		NaCN	3501	SIPX		MIBC		Grind	Cond.	Froth	
Primary Grind	2500							40			10.2
Cu Ro	620		10			5			1	4	11.0
	400		10			5			1	4	11.0
			10	5					1	3	
			10	5					1	3	11.0
Cu Ro Scav 1				10		2.5			1	2	11.0
Cu Ro Scav 2				10		2.5			1	2	11.0
Cu Ro RG	1000							20			11.5
Cu 1st Cl			5						1	4	11.5
Cu 1st Cl Scav			5			2.5			1	3	11.5
Cu 2nd Cl			2.5						1	4	11.5
Cu 3rd Cl			2.5						1	4	11.1
Cu 4th Cl		10							1	3	11.1

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D-1
Speed: r.p.m.	1800	1500

Product	Weight		Assays g/t, %				Distribution, %			
	g	%	Cu	Au	Co	Fe	Cu	Au	Co	Fe
Cu Cl Conc	131.8	6.72	23.6	10.8	0.07	28.5	76.9	61.1	3.0	7.1
Cu 4th Cl Tail	34.3	1.75	12.2	4.79	0.18	31.5	10.3	7.1	2.0	2.0
Cu 3rd Cl Tail	20.1	1.03	4.27	2.98	0.27	39.0	2.1	2.6	1.8	1.5
Cu 2nd Cl Tail	48.2	2.46	1.32	1.17	0.27	37.5	1.6	2.4	4.3	3.4
Cu 1st Cl Sc Conc	72.8	3.71	1.79	1.33	0.27	39.5	3.2	4.2	6.5	5.4
Cu 1st Cl Sc Tail	202.3	10.32	0.29	0.67	0.34	41.0	1.4	5.8	22.6	15.7
Cu Ro Sc Conc 1	253.0	12.91	0.27	0.62	0.33	42.9	1.7	6.7	27.4	20.5
Cu Ro Sc Conc 2	208.7	10.65	0.24	0.61	0.33	42.8	1.2	5.5	22.6	16.9
Cu Ro Tail	989.1	50.46	0.062	0.11	0.03	14.6	1.5	4.7	9.8	27.3
Head (calc.)	1960.3	100.00	2.06	1.19	0.16	26.9	100.0	100.0	100.0	100.0
(direct)										

Combined Products

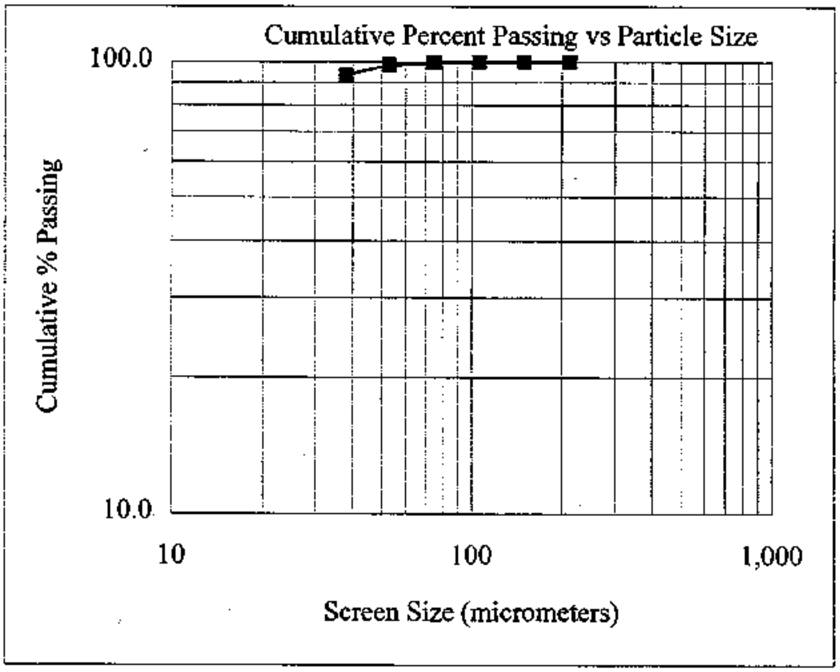
Cu 3rd Cl Conc		8.47	21.2	9.56	0.09	29.1	87.2	68.2	5.1	9.2
Cu 2nd Cl Conc		9.50	19.4	8.85	0.11	30.2	89.3	70.7	6.8	10.6
Cu 1st Cl Conc		11.96	15.7	7.27	0.14	31.7	90.9	73.2	11.1	14.1
Cu 1st Cl Conc + Sc Conc		15.67	12.4	5.86	0.17	33.5	94.1	77.3	17.6	19.5
Cu Ro Conc		25.99	7.59	3.80	0.24	36.5	95.6	83.1	40.2	35.2
Cu Ro Conc + Ro Sc 1		38.90	5.16	2.75	0.27	38.6	97.2	89.9	67.6	55.8
Cu Ro Conc + Ro Scavs		49.54	4.10	2.29	0.28	39.5	98.5	95.3	90.2	72.7

Company **Lakefield Research** **MP 128**

Size Distribution Analysis

Sample: Comb Cl Prod **Test No.:** 7

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.0	0.0	0.0	100.0
100	150	0.0	0.0	0.0	100.0
150	106	0.2	0.1	0.1	99.9
200	75	0.5	0.3	0.4	99.6
270	53	1.7	1.1	1.5	98.5
400	38	8.2	5.1	6.6	93.4
Pan	-38	149.4	93.4	100.0	0.0
Total	-	**160.0**	**100.0**	-	-
K80	**#DIV/0!**				

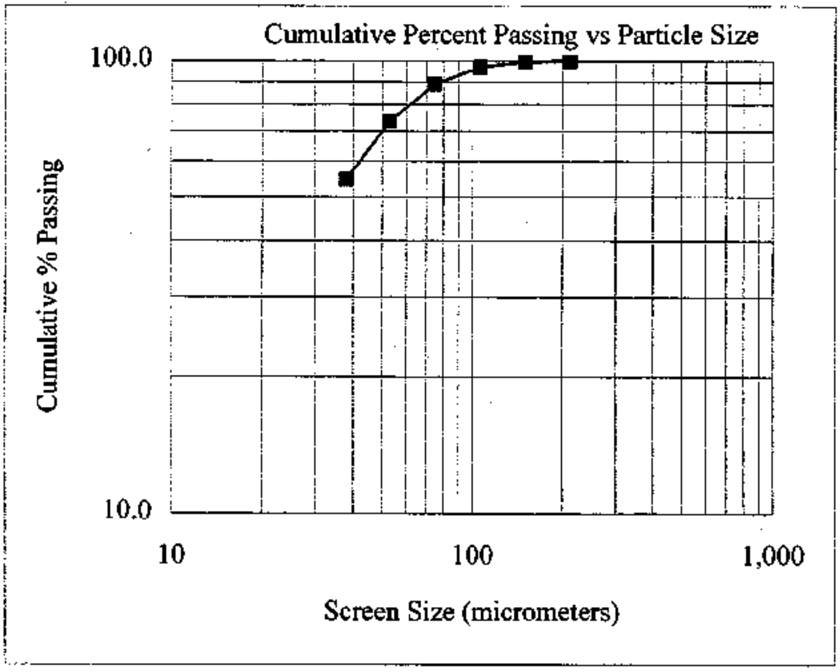


Company **Lakefield Research** **MP 128**

Size Distribution Analysis

Sample: Comb Prod **Test No.:** 7

Size		Weight	% Retained		% Passing
Mesh	μm	grams	Individual	Cumulative	Cumulative
65	212	0.0	0.0	0.0	100.0
100	150	0.7	0.4	0.4	99.6
150	106	4.3	2.7	3.1	96.9
200	75	12.9	8.1	11.2	88.8
270	53	24.4	15.3	26.4	73.6
400	38	30.4	19.0	45.4	54.6
Pan	-38	87.3	54.6	100.0	0.0
Total	-	**160.0**	**100.0**	-	-
K80	**61**				



Test No.8 Project No. MP 128 Operator: G.C. Date: Mar.27, 1997

Purpose: Repeat of test 7 with a finer primary grind, and to cleaner upgrade a Ro Scav Conc. of the gangue Py.

Procedure: Ore was ground and floated as described below. Products were filtered dried, and submitted for assay.

Feed: 2kg - 10 mesh composite sample
Grind: 50min/2kg in the lab ball mill at 65% solids.(93% - 200 mesh)
Conditions:

	Reagents added, grams per tonne							Time, minutes			
Stage	Lime	NaCN	3501	SIPX		MIBC		Grind	Cond.	Froth	pH
Primary Grind	2500							50			9.7
Cu Ro	360		15			10			1	4	11.0
	230		10			5			1	4	11.0
	115		10	5					1	3	
			10	5		2.5			1	3	11.0
Cu Ro Scav 1	240			10		2.5			1	2	11.0
Cu Ro Scav 2				10		2.5			1	2	11.0
Cu Ro RG	1000							20			11.5
Cu 1st Cl			5						1	4	11.5
Cu 1st Cl Scav			5			5			1	3	11.5
Cu 2nd Cl			2.5						1	4	11.5
Cu 3rd Cl			2.5						1	4	11.1
Cu 4th Cl									1	3	11.1
Ro Scav 1st Cl	75					2.5			1	3	11.0
Ro Scav 2nd Cl	95					2.5			1	3	11.0

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D -1
Speed: r.p.m.	1800	1500

Product	Weight		Assays g/t , %				Distribution, %			
	g	%	Cu	Au	Co	Fe	Cu	Au	Co	Fe
1 Cu Cl Conc	137.9	6.96	21.5	12.7	0.08	28.1	75.0	48.6	3.4	7.4
2 Cu 4th Cl Tail	62.2	3.14	9.34	10.3	0.20	33.9	14.7	17.8	3.9	4.0
3 Cu 3rd Cl Tail	29.9	1.51	1.38	3.10	0.26	36.7	1.0	2.6	2.4	2.1
4 Cu 2nd Cl Tail	51.8	2.62	0.85	1.85	0.25	35.9	1.1	2.7	4.0	3.5
5 Cu 1st Cl Sc Conc	51.3	2.59	1.73	6.43	0.25	38.2	2.2	9.2	4.0	3.7
6 Cu 1st Cl Sc Tail	214.7	10.84	0.48	1.21	0.26	36.9	2.6	7.2	17.4	15.1
7 Cu Ro Sc Cl Conc	345.6	17.45	0.13	0.59	0.37	44.9	1.1	5.7	39.8	29.5
8 Cu Ro Sc 2nd Cl Tail	80.9	4.08	0.26	0.68	0.34	40.6	0.5	1.5	8.6	6.3
9 Cu Ro Sc 1st Cl Tail	94.9	4.79	0.26	0.60	0.27	34.2	0.6	1.6	8.0	6.2
10 Cu Ro Scav Tail	911.5	46.02	0.043	0.13	0.03	12.8	1.0	3.3	8.5	22.2
Head (calc.)	1980.7	100.00	2.00	1.82	0.16	26.5	100.0	100.0	100.0	100.0
(direct)										

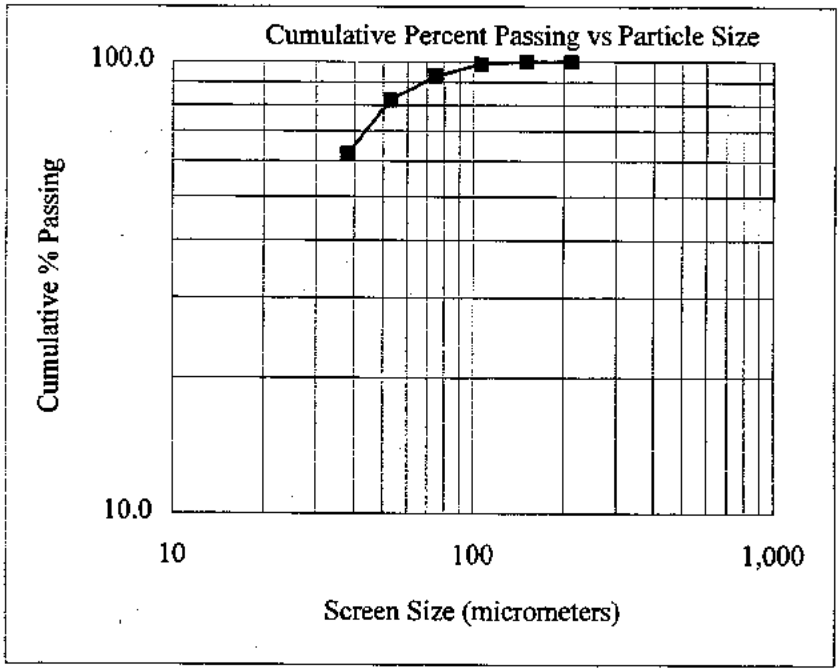
Combined Products

	%	Cu	Au	Co	Fe	Cu	Au	Co	Fe
Cu 3rd Cl Conc	10.10	17.7	12.0	0.12	29.9	89.7	66.4	7.3	11.4
Cu 2nd Cl Conc	11.61	15.6	10.8	0.14	30.8	90.7	68.9	9.7	13.5
Cu 1st Cl Conc	14.23	12.9	9.16	0.16	31.7	91.9	71.6	13.8	17.0
Cu 1st Cl Conc + Sc Conc	16.82	11.2	8.74	0.17	32.7	94.1	80.7	17.8	20.7
Cu Ro Conc	27.66	6.98	5.79	0.21	34.4	96.7	87.9	35.1	35.8
Cu Ro Conc + Ro Sc 2nd Cl Conc	45.11	4.33	3.78	0.27	38.4	97.9	93.6	74.9	65.4
Cu Ro Conc + Ro Sc 1st Cl Conc	49.19	3.99	3.52	0.28	38.6	98.4	95.1	83.5	71.6
Cu Ro Conc + Ro Sc Conc	53.98	3.66	3.26	0.27	38.2	99.0	96.7	91.5	77.8
Ro Sc 1st Cl Conc	21.53	0.15	0.61	0.36	44.1	1.7	7.2	48.4	35.8
Ro Sc Conc	26.32	0.17	0.61	0.35	42.3	2.3	8.8	56.4	42.0

Company	Lakefield Research Size Distribution Analysis	MP 128

Sample: Comb Prod **Test No.:** 8

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.0	0.0	0.0	100.0
100	150	0.1	0.1	0.1	99.9
150	106	1.2	1.2	1.3	98.7
200	75	5.9	5.9	7.2	92.8
270	53	10.6	10.6	17.8	82.2
400	38	20.0	20.0	37.8	62.2
Pan	-38	62.2	62.2	100.0	0.0
Total	-	**100.0**	**100.0**	-	-
K80	**52**				



Test No.9 Project No. MP 128 Operator: G.C. Test9 Date: April 7, 1997

Purpose: Repeat of test 8 to improve recovery in the 4th Cu cleaner.

Procedure: Ore was ground and floated as described below. Products were filtered dried, and submitted for assay.

Feed: 2kg - 10 mesh composite sample
Grind: 50min/2kg in the lab ball mill at 65% solids.(93% - 200 mesh)
Conditions:

	Reagents added, grams per tonne							Time, minutes			
Stage	Lime	NaCN	3501	SIPX		MIBC		Grind	Cond.	Froth	pH
Primary Grind	2500							50			9.8
Cu Ro	550		15			10			1	4	11.0
	305		10			7.5			1	4	11.0
	145		10	5					1	3	11.0
			10	5		2.5			1	3	11.0
Cu Ro Scav	250			10		5			1	2	11.0
				10		2.5			1	2.5	11.0
Cu Ro RG (BM	1000							15			11.5
Cu 1st Cl			5						1	4	11.5
Cu 1st Cl Scav			5			5			1	3	11.5
Cu 2nd Cl			2.5						1	4	11.5
Cu 3rd Cl	90		2.5						1	4	11.1
Cu 4th Cl		10	5			2.5			1	3	11.1
Ro Scav 1st Cl	75					2.5			1	3	11.0
Ro Scav 2nd Cl	95					2.5			1	3	11.0

Stage	Rougher	Cl
Flotation Cell	1000 g D-1	500g D -1
Speed: r.p.m.	1800	1500

Product	Weight		Assays g/t , %					Distribution, %				
	g	%	Cu	Au	Co	Fe	Zn	Cu	Au	Co	Fe	Zn
1 Cu Cl Conc	108.6	5.52	24.9	10.1	0.07	29.1	7.82	67.8	48.6	2.4	6.0	43.7
2 Cu 4th Cl Tail	39.2	1.99	12.9	6.06	0.16	30.8	11.0	12.7	10.5	2.0	2.3	22.2
3 Cu 3rd Cl Tail	19.7	1.00	4.73	4.50	0.23	36.2	2.67	2.3	3.9	1.4	1.4	2.7
4 Cu 2nd Cl Tail	42.5	2.16	3.60	3.49	0.24	35.2	2.42	3.8	6.6	3.2	2.8	5.3
5 Cu 1st Cl Sc Conc	58.4	2.97	4.02	2.11	0.24	37.8	3.94	5.9	5.5	4.5	4.2	11.8
6 Cu 1st Cl Sc Tail	119.0	6.05	0.92	0.78	0.22	34.4	0.44	2.7	4.1	8.3	7.8	2.7
7 Cu Ro Sc Conc	571.3	29.05	0.23	0.59	0.36	43.1	0.22	3.3	14.9	65.3	46.7	6.5
8 Cu Ro Scav Tail	1008.1	51.26	0.056	0.13	0.04	15.1	0.10	1.4	5.8	12.8	28.9	5.2
Head (calc.)	1966.8	100.00	2.03	1.15	0.16	26.8	0.99	100.0	100.0	100.0	100.0	100.0
(direct)												

Combined Products

Cu 3rd Cl Conc		7.51	21.7	9.03	0.09	29.6	8.66	80.5	59.2	4.4	8.3	65.8
Cu 2nd Cl Conc		8.52	19.7	8.50	0.11	30.3	7.96	82.8	63.1	5.8	9.6	68.5
Cu 1st Cl Conc		10.68	16.5	7.48	0.14	31.3	6.84	86.7	69.7	9.1	12.5	73.8
Cu 1st Cl Conc + Sc Conc		13.65	13.8	6.31	0.16	32.7	6.21	92.5	75.1	13.5	16.7	85.7
Cu Ro Conc		19.70	9.81	4.61	0.18	33.2	4.44	95.3	79.2	21.9	24.4	88.4
Cu Ro Conc + Ro Sc Conc		48.74	4.10	2.22	0.29	39.1	1.92	98.6	94.2	87.2	71.1	94.8
* Cu 1st Cl Sc Tail + Ro Sc Conc		35.10	0.35	0.62	0.34	41.6	0.26	6.0	19.1	73.7	54.5	9.2

* Feed for future Co leaching

LAKEFIELD RESEARCH LIMITED

Mineralogical Services

MINERALOGICAL EXAMINATION OF TWO SAMPLES FROM THE FYRE LAKE DEPOSIT, YUKON TERRITORY

submitted by
MELIS ENGINEERING LTD.

for
COLUMBIA GOLD MINES LTD.

Project managed by: Giovanni Di Prisco

Submission date: September 17, 1997

Project No: 8901-058 - FEB3410

Lakefield Research Limited - 185 Concession St., Postal Bag 4300, Lakefield Ontario, K0L 2H0, CANADA
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Fyre Lake Ore Sample - Mineralogical Report

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1.0 SUMMARY

Mineralogical examinations of one composite head sample and one gravity concentrate sample were carried out for Melis Engineering Ltd. on behalf of Columbia Gold Mines Ltd..

Two samples from the Fyre Lake deposit (Fyre Lake Test Composite Head and Mozley Concentrate MP128-3) were submitted for mineralogical examination. The main scope of this investigation was to determine the nature and mode of occurrence of cobalt-, gold-, copper-, and zinc-bearing minerals.

A summary of the mineralogical data is presented in summary form in Appendix A. The samples were found to be mainly composed of iron sulphides (pyrite). Minor to trace amounts of sulphides (chalcopyrite, sphalerite, pyrrhotite, chalcocite, and covellite) and oxides (magnetite and goethite) were also observed.

No cobalt minerals were identified during microscopic examination of the two samples. Electron microprobe analyses of pyrite (23 quantitative microprobe analyses) and sphalerite (4 quantitative microprobe analyses) grains detected cobalt in the Fyre Lake Test Composite Head. Cobalt in pyrite ranges from a minimum of 800 ppm (0.08 %) to a maximum of 4.9 % (one grain) whereas cobalt in sphalerite ranges from a minimum of 470 ppm (0.047 %) to a maximum of 0.36 %. Pyrite is identified as the main cobalt carrier.

Microscopic gold scans were carried out on the two samples. No visible gold was observed in either sample. Ion microprobe analyses (Secondary Ion Mass Spectrometry - SIMS) confirmed the occurrence of "invisible" gold in all the pyrite grains analyzed from the Fyre Lake Test Composite Head. The "invisible" gold in the analyzed pyrite grains occurs more likely as sub-microscopic and / or colloidal gold (< 0.1 μm) instead of gold in solid solution.

Copper carriers were identified as chalcopyrite, chalcocite, and covellite, and the zinc carrier identified is sphalerite.

LAKEFIELD RESEARCH LIMITED
September 17, 1997

Amanda Evans, H.B.Sc.
Mineralogist

Giovanni Di Prisco, Ph.D.
Senior Mineralogist

2.0 MINERALOGICAL OBSERVATIONS

2.1 METHODOLOGY

One composite head sample and one gravity concentrate sample from the Fyre Lake deposit (Table 1) were submitted for mineralogical examination. The main scope of this study was to determine the nature and mode of occurrence of cobalt- and gold-bearing minerals, as well as the nature of the other economic minerals.

Table 1. List of Submitted Samples

Fyre Lake Project	
Sample	**Polished Section No.**
Fyre Lake Test Composite Head	***PS6503***
Fyre Lake Test Composite Head	***PS6504***
Fyre Lake Test Composite Head	***PS6505***
Fyre Lake Test Composite Head	***PS6506***
Mozley Concentrate MP128-3	-

Polished sections were examined with reflected light microscopy in order to identify cobalt-, gold-, and other base metal-bearing minerals (copper and zinc). Gold scans were carried out on the entire surface of the polished sections using reflected light microscopy at magnifications of 200X and 500X. A series of sulphide grains were selected for microprobe analysis to identify cobalt-bearing minerals, and ion probe analyses of pyrite grains were carried out to assess the occurrence of "invisible" gold. Photomicrographs for the Fyre Lake Test Composite Head sample are presented in Appendix B.

The entire Mozley Concentrate MP128-3 sample was scanned for gold using a binocular microscope at magnifications ranging from 240X to 600X.

A list of the examined samples are presented in Table 2 and a list of the mineral abbreviations used in this report is presented in Table 3.

Table 2. List of Examined Samples and Assay Values

Fyre Lake Project							
Sample	**Assay %, g/t**						
	Cu	**Co**	**Zn**	**Au**	**Ag**	**Fe**	**S**
Fyre Lake Test Composite Head	2.07	0.16	0.99	0.98	3.7	26.1	20.5
Mozley Concentrate MP128-3*	-	-	-	-	-	-	-

* no assays available

Table 3. List of Mineral Abbreviations

Abbreviation	Mineral
au	gold
cc	chalcocite
cp	chalcopyrite
cv	covellite
gn	transparent gangue
goe	goethite
mt	magnetite
po	pyrrhotite
py	pyrite
sp	sphalerite

2.2 DETAILS OF MICROSCOPE EXAMINATIONS

2.2.1 Fyre Lake Test Composite Head *(PS6503, PS6504, PS6505, and PS6506)*

The Fyre Lake Test Composite Head sample is mainly composed of opaque minerals (90 to 95 %). Opaque minerals are mainly composed of pyrite aggregates intergrown with minor to subordinate amounts of chalcopyrite and sphalerite. Minor amounts of pyrrhotite, magnetite, goethite, and transparent gangue minerals, and a rare intergrowth of chalcocite and covellite were also observed.

No cobalt minerals were identified during microscopic examination, however, pyrite is known to contain minor amounts of cobalt. Electron microprobe analyses of more than thirty pyrite grains detected cobalt concentrations ranging from of 800 ppm (0.08 %) to 4.9 % (one grain). Pyrite is identified as a cobalt carrier.

Microscopic gold scans were carried out and no visible gold was observed. Ion microprobe analyses (Secondary Ion Mass Spectrometry - SIMS) confirmed the occurrence of "invisible" gold in all the pyrite grains analyzed from the Fyre Lake Test Composite Head.

Copper carriers were identified as chalcopyrite, chalcocite, and covellite, and the zinc carrier identified is sphalerite.

2.2.2 Mozley Concentrate MP128-3

The Mozley Concentrate MP128-3 was examined in order to determine the nature and mode of occurrence of gold. A gold scan was carried out and no gold was observed.

The Mozley Concentrate MP128-3 sample is mainly composed of very fine-grained pyrite (> 98 %). Trace amounts of magnetite and quartz were also observed. Due to the very fine-grained nature of the grains observed in Mozley Concentrate MP128-3, no photomicrographs of Mozley Concentrate MP128-3 were taken.

2.3 ELECTRON MICROPROBE INVESTIGATION

No cobalt minerals were identified by microscopic examination, however, sulphides, in particular pyrite, are known to contain cobalt. Sulphide grains were microprobed in an attempt to identify the most likely cobalt-bearing minerals present in the Fyre Lake Test Composite Head Sample. A series of microprobe analyses of sulphide grains were carried out at the Duncan Derry Microprobe Laboratory of the University of Toronto.

More than thirty pyrite grains were selected for microprobe analyses. Qualitative Energy Dispersive System (EDS) analyses and quantitative Wave Dispersive System (WDS) analyses, with a detection limit of 400 ppm for cobalt, were carried out. Selected quantitative analyses (WDS) are presented in Table 4.

Table 4. Selected Electron Microprobe Analyses of Pyrite Grains

	Fyre Lake Test Composite Head							
Element	**Selected Pyrite Grains**							
(weight %)	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**
Fe	46.873	45.782	46.299	46.385	46.454	45.788	46.522	45.665
Co	**0.085**	**0.748**	**0.105**	**0.129**	**0.153**	**1.119**	**0.241**	**0.536**
S	53.802	53.320	53.528	53.304	53.289	53.219	52.878	53.382
Total	100.760	99.851	99.932	99.818	99.895	100.126	99.641	99.583
	9	**10**	**11**	**12**	**13**	**14**	**15**	**16**
Fe	41.932	45.583	46.459	46.131	46.550	45.253	46.488	46.884
Co	**4.923**	**0.572**	**0.370**	**0.140**	**0.152**	**1.316**	**0.192**	**0.202**
S	53.597	52.935	53.389	53.362	53.631	53.239	53.535	53.583
Total	100.452	99.359	100.217	99.634	100.333	99.809	100.215	100.669
	17	**18**	**19**	**20**	**21**	**22**	**23**	
Fe	46.059	46.237	46.001	46.721	46.515	46.667	45.815	
Co	**0.088**	**0.163**	**0.902**	**0.091**	**0.471**	**0.278**	**0.411**	
S	52.396	53.616	53.374	56.653	53.570	53.302	53.653	
Total	98.543	100.017	100.278	100.465	100.556	100.246	99.879	

All the qualitative and quantitative analyses of the pyrite grains detected cobalt in pyrite. Qualitative analyses indicate that cobalt concentrations in pyrite grains range from 800 ppm (0.08 %) to 4.9 % (one grain). The average amount of cobalt in pyrite is 0.4 %.

A series of microprobe analyses for cobalt were also carried out on selected chalcopyrite and sphalerite grains. In all the chalcopyrite grains analyzed, cobalt was not detected (400 ppm detection limit). The amount of copper in the chalcopyrite grains ranges from 33.5 % to 34.9 %. Semi-quantitative analyses (WDS) of selected sphalerite grains are presented in Table 5.

Table 5. Electron Microprobe Analyses of Selected Sphalerite Grains

Fyre Lake Test Composite Head				
Element	**Selected Sphalerite Grains**			
(weight %)	**1**	**2**	**3**	**4**
Fe	8.232	9.212	9.113	9.537
Cu	0.278	0.015	0.143	0.052
S	33.502	34.170	33.805	33.533
Zn	57.733	56.927	57.106	56.837
Co	**0.245**	**0.162**	**0.358**	**0.047**
Total	99.989	100.486	100.532	100.006

Semi-quantitative analyses indicate that cobalt in sphalerite ranges from 470 ppm (0.047 %) to 0.36 %. The average amount of cobalt in sphalerite is 0.2 %. The amount of iron in sphalerite ranges from 8.2 % iron in a grain with 57.7% zinc to 3.6 % iron in a grain with 61.5 % zinc.

2.4 ION MICROPROBE INVESTIGATION

Gold scans were carried out and no visible gold was identified. Follow-up analyses were carried out with the ion probe (Secondary Ion Mass Spectrometry - SIMS) at the Surface Science Laboratory at the University of Western Ontario. Individual pyrite grains were analyzed with an ion probe (Secondary Ion Mass Spectrometry - SIMS) in order to assess the concentration of "invisible" gold. Ion probe technology allows for the detection of very low concentrations (down to 0.001 ppm). The samples were analyzed using a Camera IMS-3f Ion Microprobe (SIMS). All measurements were made with a cesium primary beam of about 500 nA at 14 KeV.

Table 5. Ion Microprobe Analyses of Pyrite Grains

Fyre Lake Test Composite Head			
Sample 1		Sample 2	
Grain	**Gold Content (Au ppm)**	**Grain**	**Gold Content (Au ppm)**
1	5	1	0.026
2	0.260	2	3*
			1*
			0.065*
		3	195*
			0.130
		4	0.195
			0.098
		5	2

* gold not evenly distributed in pyrite lattice

A few pyrite grains were probed and confirmed the occurrence of "invisible" gold. Gold concentrations varied widely, ranging from a low of 0.026 ppm to a high of 195 ppm. The "invisible" gold in the analyzed pyrite grains occurs more likely as sub-microscopic and / or colloidal gold (< 0.1 μm) instead of gold in solid solution.

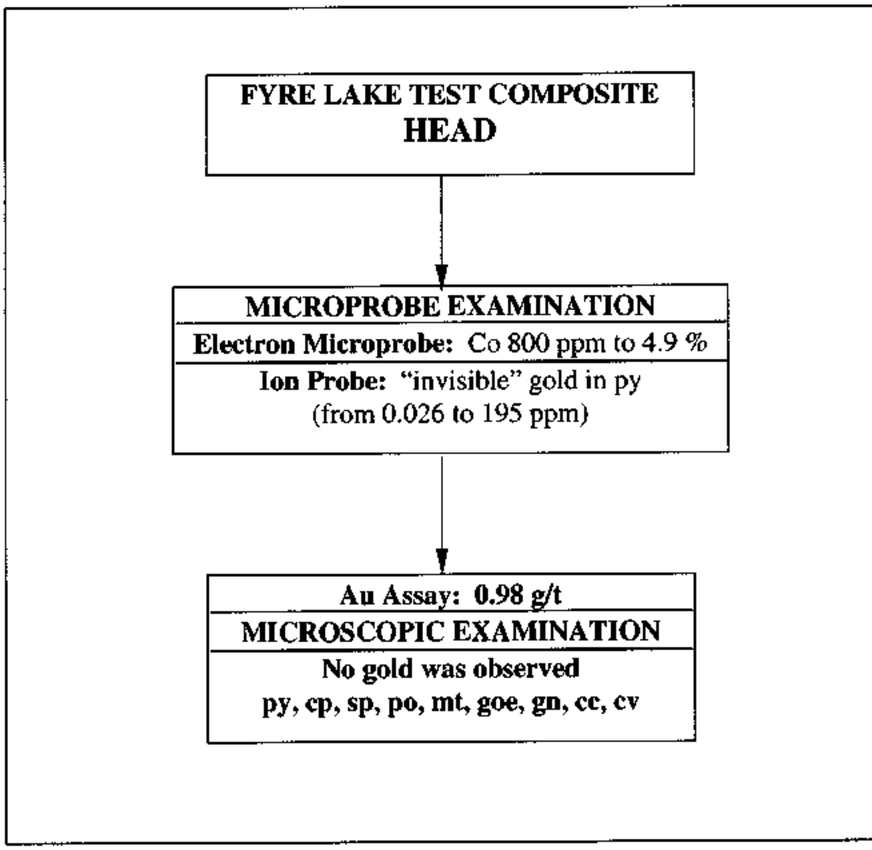
3.0 CONCLUSIONS

The mineral composition of the two submitted samples is similar. The bulk of the samples (> 90 %) is composed of opaque minerals. The opaque minerals are mainly composed of aggregates of pyrite, intergrown with minor amounts of chalcopyrite and sphalerite. Minor amounts of pyrrhotite, magnetite, goethite, and transparent gangue minerals, and a rare intergrowth of chalcocite and covellite were also observed.

Microscopic examination did not reveal any cobalt sulphide minerals. Follow-up studies included electron microprobe analyses of sulphide grains to identify cobalt. Cobalt (up to 4.9 weight %) was detected in pyrite grains with electron microprobe analyses and minor amounts of cobalt were also detected in sphalerite grains (up to 0.36 %). In the submitted sample (Fyre Lake Test Composite Head), pyrite was found to be the main cobalt-bearing mineral. Ion probe analysis (Secondary Ion Mass Spectrometry - SIMS) was able to confirm that some of the gold occurs as sub-microscopic "invisible" gold in pyrite.

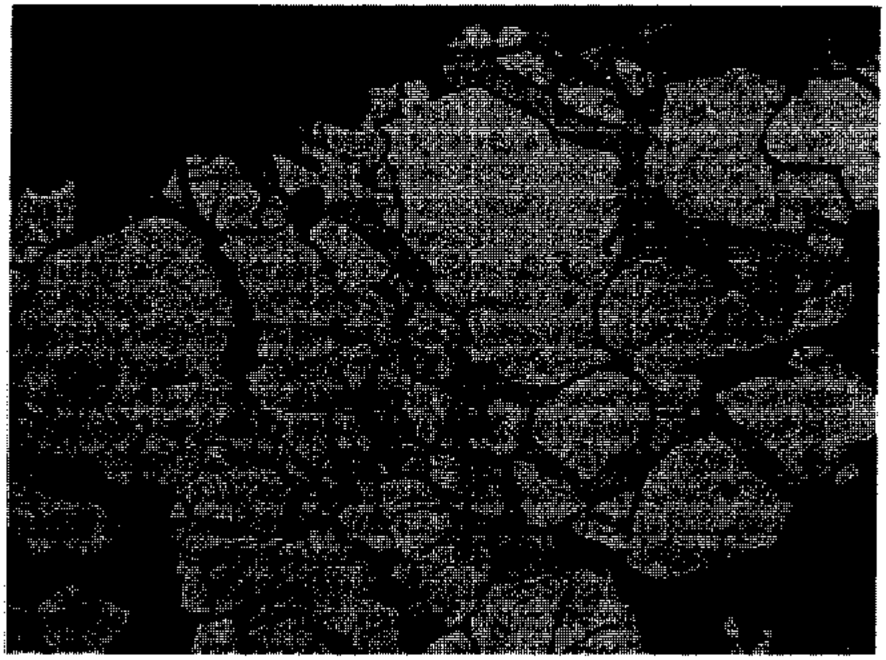
APPENDIX A

Summary of Assay and Mineralogical Data from the Fyre Lake Project



APPENDIX B

PHOTOMICROGRAPHS - FYRE LAKE TEST COMPOSITE HEAD



100 µm

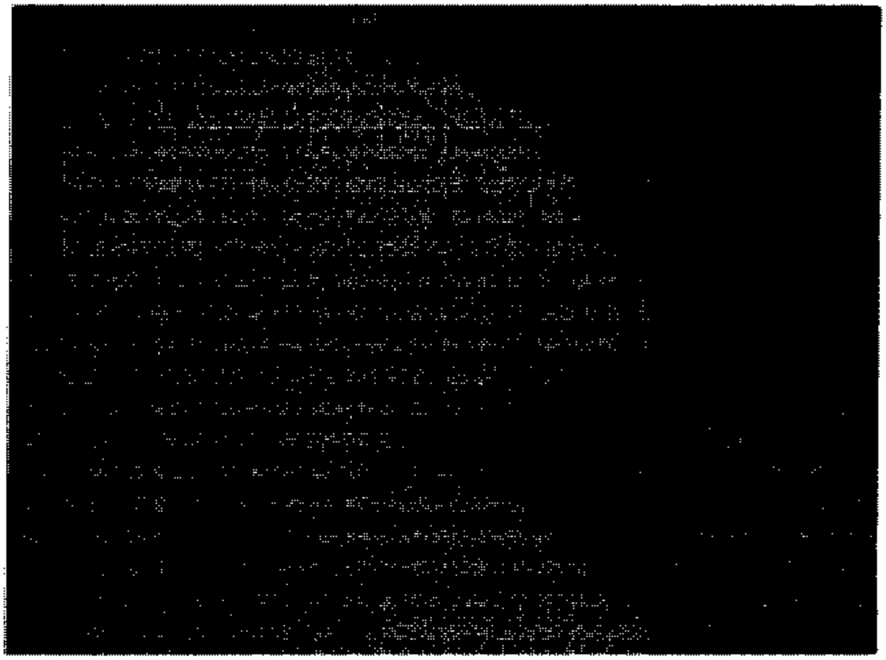
Photo 1. Fyre Lake Deposit
PS6506
Mag 100X Reflected Light

Aggregate of medium- to coarse-grained subhedral pyrite (whitish) with interstitial chalcopyrite (yellow) and sphalerite (dark grey). Surrounding dark patches are epoxy. Pinkish tarnish on pyrite may be the result of relatively high amount of cobalt in pyrite grains.

50 µm

Photo 2. Fyre Lake Deposit
PS6506
Mag 200X Reflected Light

Coarse-grained subhedral pyrite (whitish) and coarse-grained interstitial sphalerite (dark grey) with minor interstitial chalcopyrite (yellow). Surrounding dark patches are epoxy. Pinkish tarnish on pyrite may be the result of relatively high amounts of cobalt in pyrite grains.



100 µm

Photo 3. Fyre Lake Deposit
PS6506
Mag 100X Reflected Light

Aggregate of coarse-grained pyrite (whitish) with interstitial chalcopyrite (yellow). Surrounding dark patches are epoxy. Pinkish tarnish on pyrite may be the result of relatively high amounts of cobalt in pyrite grains.

50 µm

Photo 4. Fyre Lake Deposit
PS6506
Mag 200X

Aggregate of medium-grained pyrite (whitish) surrounded by coarse-grained sphalerite (dark grey) with inclusions of chalcopyrite (yellow). Dark patches are epoxy. Pinkish tarnish on pyrite may be the result of relatively high amounts of cobalt in pyrite grains.

APPENDIX III

Kona Zone Resource Report

2002 Mineral Resource Estimate

Gemcom Reserves Report Detailed Information Page

Included Report Contents:

Obtain rock type from block model
Obtain grades from block model
Use surfaces
Obtain densities from block model
Create ASCII report file
Include report details
Use clipping polygon
Clipping Polygon : D:\PMDBKZ\zone.ABP

Needle Information:

Needle Orientation : Block Model Levels
Needle Integration Level: Level 1 - 1 needle per cell
Needle Pattern Type : Regular Gauss

Reserves Report Profile Information

Reserves Reporting Profile: CLASS2

Group Number	Rock Group	Grade Group
1	2000	KZ_RES
2	3000	KZ_RES

Supression of Zero Value Categories: YES

Rock Group Profiles Information

Rock Group Name	Active Rock Code
2000	2000
3000	3000

Reserves Report Rock Code Information

Code	Ore ?	Density	Block Model Folder	Block Model Code
2000	YES	2.7000	Standard	2000
3000	YES	2.7000	Standard	3000

NOTE: Only solids with ORE rock types will access block models for grades/density.
Non-ORE solids use the values assigned to the individual solid.

Default Rock Code: 3000

Grade Group Profiles Information

Grade Group Name : KZ_RES

Control Element 1

Description	CU From	To
5.0-100	5.0000	100.0000
4.5-5.0	4.5000	5.0000
4.0-4.5	4.0000	4.5000
3.5-4.0	3.5000	4.0000
3.0-3.5	3.0000	3.5000
2.5-3.0	2.5000	3.0000
2.0-2.5	2.0000	2.5000
1.5-2.0	1.5000	2.0000
1.0-1.5	1.0000	1.5000
0.75-1.0	0.7500	1.0000
0.5-0.75	0.5000	0.7500
0.01-0.5	0.0100	0.5000

Default Grade:

Grade Element	Value
CU	0.0000
CO	0.0000
AU	0.0000
ZN	0.0000
CLASS	0.0000

Reserves Report Active Surface Information

Number	Name 1	Name 2	Name 3	Category	Above Material	Below Material
1	KZ_TOPO	TOPO	REVISED	SURFACE TYPE		
2	Resource	Bottom	1000m	Excavation	<none>	<none>

Reserves Report Active Surface Information

Top Surface: KZ_TOPO TOPO REVISED
Bottom Surface: Resource Bottom 1000m

Reserves Report Block Model Information

The following block models were accessed:

Block Folder: Standard

- Class (Rock Model)
- Density (Density Model)
- Percent (Percent Model)
- Cu % Grade (Grade 1 Model)
- Co % Grade (Grade 2 Model)
- Au gpT Grade (Grade 3 Model)
- Rock Type (Grade 5 Model)

Report 1 of 2: Incremental

Totals for ROCKGROUP 2000

GRADEGROUP	VOLUME M**3	DENSITY T per M**3	TONNAGE T	CU Grade	CO Grade	AU Grade
4.5-5.0	0.191	3.520	0.671	4.520	0.101	1.087
4.0-4.5	2.349	3.458	8.125	4.229	0.111	0.647
3.5-4.0	3.017	3.462	10.447	3.721	0.132	0.870
3.0-3.5	16.437	3.394	55.792	3.232	0.119	0.867
2.5-3.0	35.423	3.418	121.070	2.719	0.108	0.772
2.0-2.5	115.119	3.465	398.894	2.196	0.109	0.823
1.5-2.0	305.673	3.416	1044.192	1.697	0.102	0.635
1.0-1.5	563.136	3.431	1932.158	1.236	0.090	0.529
0.75-1.0	387.217	3.454	1337.401	0.858	0.074	0.426
0.5-0.75	433.811	3.473	1506.471	0.619	0.057	0.320
0.01-0.5	396.042	3.425	1356.558	0.323	0.030	0.142
Total	2258.415	3.441	7771.780	1.048	0.074	0.440

Totals for ROCKGROUP 3000

GRADEGROUP	VOLUME M**3	DENSITY T per M**3	TONNAGE T	CU Grade	CO Grade	AU Grade
5.0-100	2.882	3.447	9.934	6.063	0.062	0.736
4.5-5.0	1.166	3.447	4.020	4.796	0.078	0.581
4.0-4.5	10.226	3.450	35.283	4.288	0.057	0.595
3.5-4.0	4.390	3.452	15.156	3.696	0.071	0.499
3.0-3.5	7.086	3.455	24.487	3.237	0.074	0.484
2.5-3.0	27.531	3.395	93.477	2.715	0.056	0.670
2.0-2.5	68.485	3.439	235.525	2.152	0.089	0.602
1.5-2.0	481.987	3.399	1638.034	1.658	0.088	0.521
1.0-1.5	956.097	3.456	3304.633	1.231	0.077	0.526
0.75-1.0	635.752	3.454	2195.574	0.869	0.063	0.313
0.5-0.75	462.688	3.441	1592.036	0.636	0.045	0.204
0.01-0.5	948.250	3.437	3258.696	0.223	0.018	0.050
Total	3606.541	3.440	12406.855	0.932	0.056	0.324

Report 2 of 2: Cumulative

Totals for ROCKGROUP 2000

GRADEGROUP	VOLUME M**3	DENSITY T per M**3	TONNAGE T	CU Grade	CO Grade	AU Grade
4.5-5.0	0.191	3.520	0.671	4.520	0.101	1.087
4.0-4.5	2.540	3.463	8.796	4.252	0.110	0.681
3.5-4.0	5.557	3.463	19.242	3.964	0.122	0.783
3.0-3.5	21.994	3.412	75.035	3.419	0.120	0.846
2.5-3.0	57.418	3.415	196.105	2.987	0.112	0.800
2.0-2.5	172.537	3.449	594.999	2.457	0.110	0.815
1.5-2.0	478.209	3.428	1639.191	1.973	0.105	0.701
1.0-1.5	1041.345	3.430	3571.349	1.574	0.097	0.608
0.75-1.0	1428.562	3.436	4908.750	1.379	0.091	0.558
0.5-0.75	1862.373	3.445	6415.221	1.201	0.083	0.503
0.01-0.5	2258.415	3.441	7771.780	1.048	0.074	0.440
Total	2258.415	3.441	7771.780	1.048	0.074	0.440

Totals for ROCKGROUP 3000

GRADEGROUP	VOLUME M**3	DENSITY T per M**3	TONNAGE T	CU Grade	CO Grade	AU Grade
5.0-100	2.882	3.447	9.934	6.063	0.062	0.736
4.5-5.0	4.048	3.447	13.954	5.698	0.067	0.692
4.0-4.5	14.274	3.449	49.237	4.688	0.059	0.622
3.5-4.0	18.664	3.450	64.392	4.454	0.062	0.593
3.0-3.5	25.751	3.452	88.879	4.119	0.065	0.563
2.5-3.0	53.282	3.422	182.357	3.399	0.061	0.618
2.0-2.5	121.768	3.432	417.881	2.696	0.077	0.609
1.5-2.0	603.754	3.405	2055.915	1.869	0.086	0.539
1.0-1.5	1559.851	3.437	5360.548	1.476	0.080	0.531
0.75-1.0	2195.603	3.441	7556.123	1.299	0.075	0.468
0.5-0.75	2658.291	3.441	9148.159	1.184	0.070	0.422
0.01-0.5	3606.541	3.440	12406.855	0.932	0.056	0.324
Total	3606.541	3.440	12406.855	0.932	0.056	0.324

NOTE: Total Needle Volume = 6514249.311