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REPORT ON THE METALLURGICAL TESTING OF THE LUMBY MUSCOVITE - GRAPHITE - GOLD DEPOSIT OF THE QUINTO MINING CORPORATION, VERNON MINING DIVISION, BRITISH COLUMBIA, CANADA

LOCATED

NEAR LUMBY, B.C.

50 DEGREES 15.9 MINUTES NORTH LATITUDE
118 DEGREES 56.3 MINUTES WEST LONGITUDE
N.T.S. MAP AREA 82L/7W, ~~82L/6E~~

CONSISTING OF 19 CLAIMS (251 units)
(B.S. 1-2, 4-5, P.S., P.S. 2-4, P.S. 7,
QUIN, M.M. 1-5, LUM 1-4)

FOR

THE QUINTO MINING CORPORATION
606-626 WEST PENDER STREET
VANCOUVER, BRITISH COLUMBIA
V6B 1V9

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VANCOUVER, B.C.

BY

A.D. Drummond, Ph.D., P.Eng.

D.A. Howard, M.Sc., P.Eng.

September 15, 1993

23029

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SUMMARY

The Quinto Mining Corporation owns 251 mineral claim units covering Saddle Mountain and the area north of the town of Lumby, B.C. in the Vernon Mining Division of British Columbia. Road access is excellent to and on the property. Local elevation of Saddle Mountain is from 500 metres (1,640 ft. asl.) to about 900 metres (about 3,000 ft. asl.). Geomorphologically the area has rounded hills, sparse outcrop and is covered with pine, cedar, shrubs and mountain grasses. Coordinates are 50 degrees 15.9 minute north latitude and 118 degrees 56.3 minutes west longitude. N.T.S. area is 82L/6E and 7W.

The mineral claim group covers Upper Triassic aged sedimentary and metamorphic rocks and has been intruded by Jurassic and Cretaceous intrusions. The Saddle Mountain portion of The Quinto Mining Corporation property has been the site of most of the exploration work. The southern part of Saddle Mountain is underlain by a hornblende diorite of Cretaceous age which has intruded a Upper Triassic sequence of argillaceous - micaceous schist rocks which near the hornblende diorite intrusion dip at about 45 degrees to the south toward the intrusion. It is within the tilted argillaceous horizon that the Plateau Shear Zone has formed and which is the site of the herein described muscovite-graphite-gold mineralization.

The Plateau Shear Zone has been traced across Saddle Mountain in an east-west direction for about 1,000 metres (3,280 ft.) by drilling. The muscovite, graphite and gold mineralization within the shear zone has been confirmed down dip in excess of 150 metres (about 500 ft.) and is still open. The down dip length from the ridge of Saddle Mountain to the valley floor is 500 metres (1,640 ft.).

Gold mineralization occurs as very fine grained native gold within pyrite grains and along hairline fractures in the abundant fine grained to massive pyrite.

Previous investigations had suggested that the Plateau Shear Zone contained a very high percentage of graphite because of its shiny black appearance. After extensive metallurgical testing, microscope study,and research into the analytical methods used to analyze for graphite it was found that the flotation product only contained 5 to 6 percent graphite, therefore it is probable that the original rock contained somewhat less. The graphite (identified with a petrographic microscope - 1000 power) is ultra fine grained (0.1-0.3 by 2-5 microns) and occurs as interleaved grains between fine grained muscovite/sericite grains. The reason that the high percentage of muscovite floated was that the graphite adheres to some of the muscovite thus making it readily floatable.

Recent metallurgical investigations by Process Research

Associates Ltd. has shown that three products can be extracted from the Plateau Shear Zone material. Mine run rock can be ground to 70% -200 mesh and floated in sequence to produce the three products. These are a very fine grained muscovite-graphite mix which has been termed "Schillerite No. 1", a pyrite-gold concentrate from which gold can be recovered and a very fine grained muscovite product which has been termed "Schillerite No. 2". The most recent metallurgical testing has indicated that variations can be made which would enhance the suggested flowsheet by possibly changing a flotation stage for a hydrocyclone stage. Metallurgical testing will continue as will mine planning in preparation for bulk sampling. At this stage, the "Schillerite No. 1 and No. 2" products have been sent to end users for acceptance testing.

INTRODUCTION

The firm of D.D.H. Geomanagement Ltd., 422-470 Granville Street, Vancouver, B.C., V6C 1V5 has been requested by The Quinto Mining Corporation, 606-626 West Pender Street, Vancouver, B.C., V6B 1V9 to supervise, coordinate and report on recent metallurgical investigations into the recovery of very fine grained muscovite/graphite, muscovite and gold. Previous investigations by the writers (Drummond and Howard, March 9, 1993) established that the above products could be recovered from the Lumby deposit with simple metallurgical procedures. The present metallurgical work is being conducted by Process Research Associates Ltd. under the direction of Dr. Bernhard Klein Ph.D.

The purpose of the present metallurgical testing is to establish a mineral dressing flow sheet that can be used to produce the graphite-muscovite and muscovite products and also recover the gold contained in the pyrite fraction.

To accomplish the present assignment, all available data both public and private has been reviewed, the writers have been involved in further research and in the day to day management supervision of the metallurgical testing. The property has been visited and sampled by the writers.

In anticipation of future pilot tests the existing mill on the property is being cleaned up and repaired.

The present report contains a number of sections verbatim from the writer's earlier assessment report (Drummond and Howard, March 9, 1993) and is included for purposes of clarity.

LOCATION, ACCESS AND INFRASTRUCTURE

The Lumby gold-graphite deposit (Figures 1 and 2) of The Quinto Mining Corporation is located in south central British

FIGURE 1

THE QUINTO MINING CORPORATION

LOCATION MAP

LUMBY PROJECT

Scale: as shown

D.D.H. GEOMANAGEMENT LTD.

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9/15/93

A. D. DRUMMOND

Sept 15/93

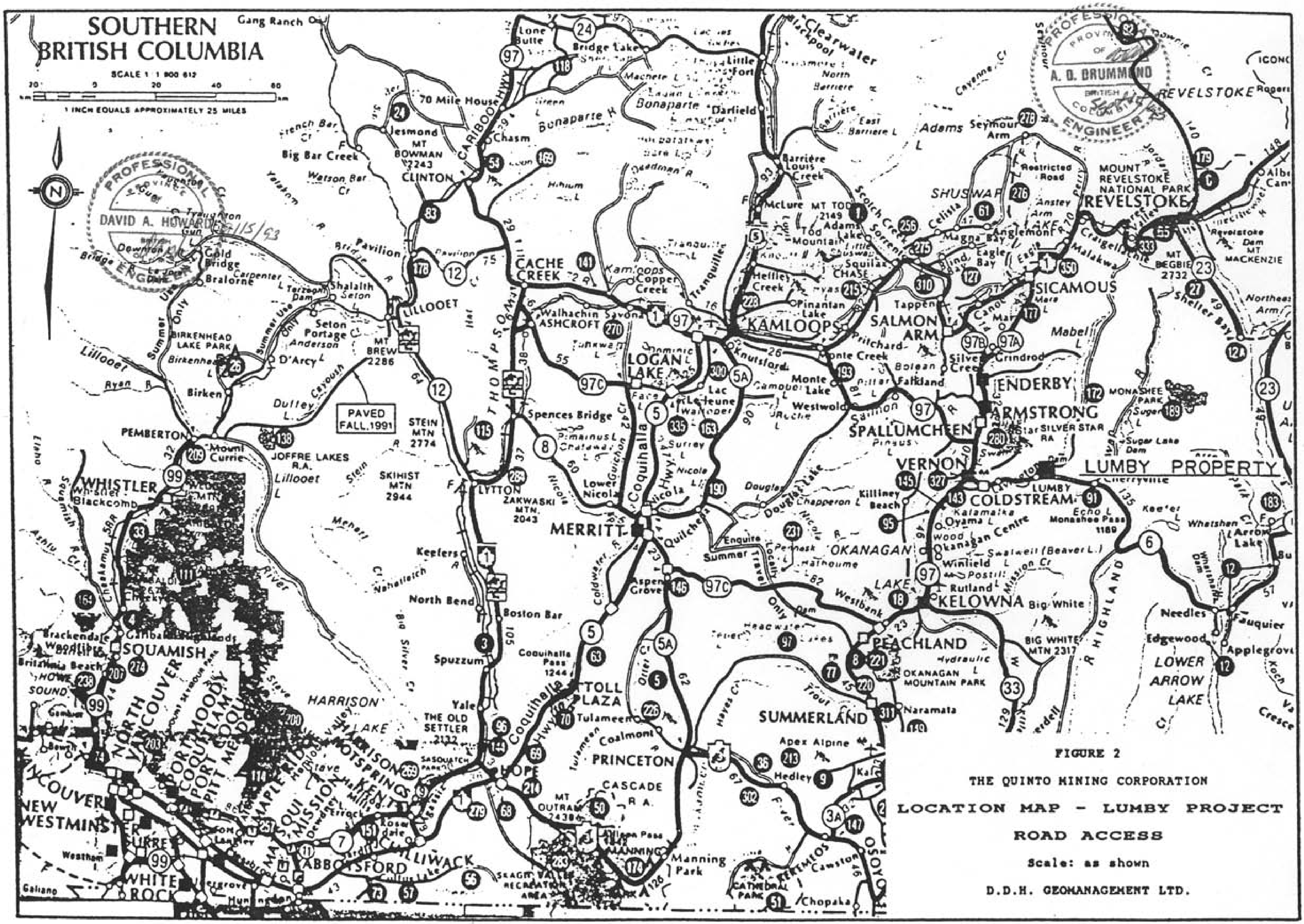


FIGURE 2

THE QUINTO MINING CORPORATION

LOCATION MAP - LUMBY PROJECT

ROAD ACCESS

Scale: as shown

D.D.H. GEOMANAGEMENT LTD.

Columbia just outside the city, but in part within the village limits of Lumby, B.C. and is traversed by B.C. Highway No. 6.

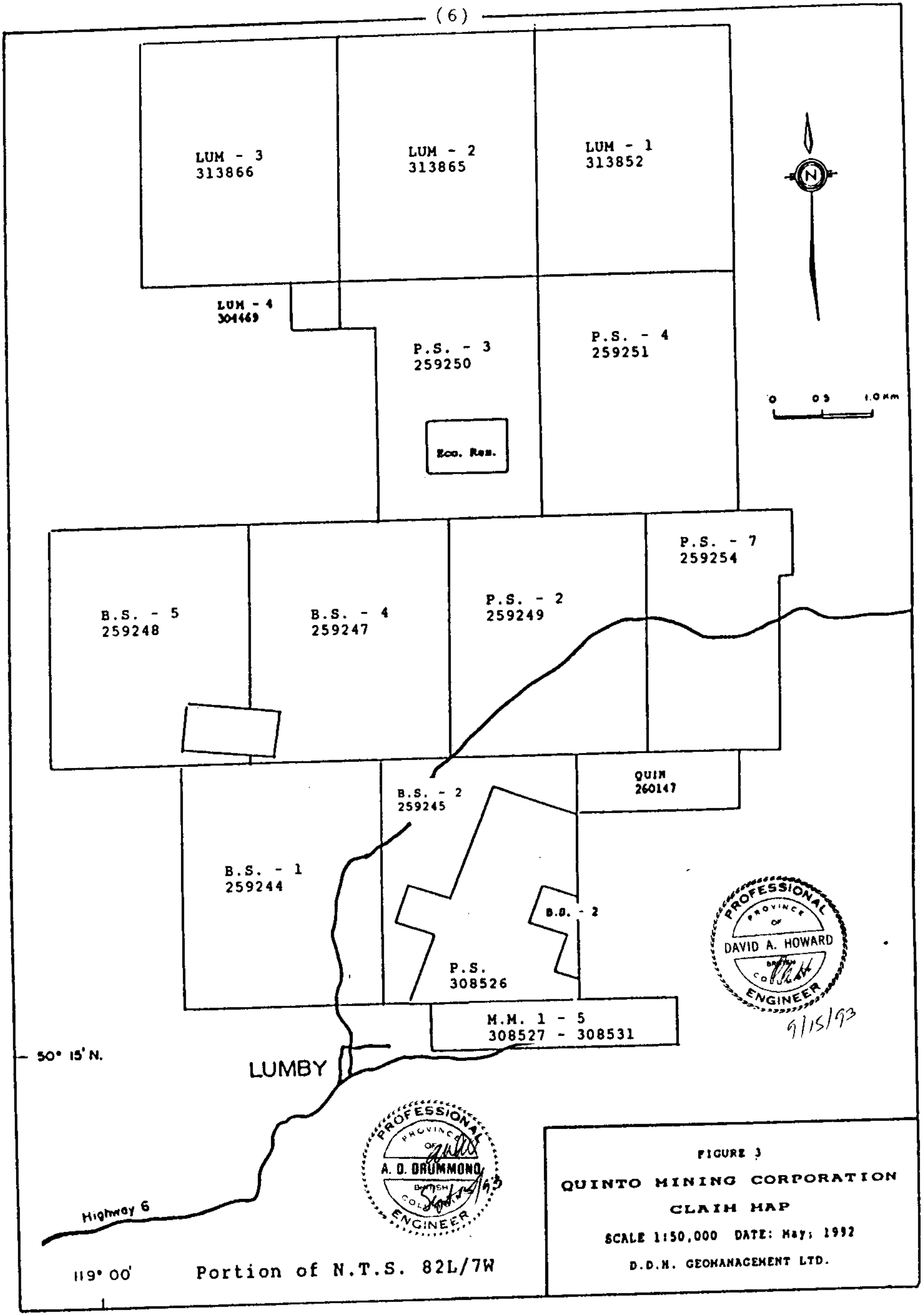
Access to the property is via a network of narrow two and/or four wheel drive old logging roads. A good gravel road is in place to the underground workings and the 150 ton mill.

The town of Lumby is a service centre for local logging and agriculture located 22 kilometres east of Vernon, B.C. The town contains all necessary facilities in terms of lodging, restaurants and fuel to support any exploration program. A 3 phase power line extends from Lumby to the above mill and could be activated upon request.

PROPERTY AND TITLE

The Lumby property comprises 251 units contained in 19 modified grid claims (See Figure 3). The claims all lie within the Vernon Mining Division on N.T.S. map sheets 82L/7W and 82L/6E. The claims are listed below:

CLAIM NAME	RECORD NO.	UNITS	MIN. TEN. NO.	EXPIRY DATE
B.S. - 1	2002	20	259244	9/24/95
B.S. - 2	2003	20	259245	9/24/95
B.S. - 4	2005	20	259247	9/24/93
B.S. - 5	2006	20	259248	9/24/93
P.S.	-	20	308526	4/01/96
P.S. - 2	2007	20	259249	9/24/93
P.S. - 3	2008	20	259250	9/24/93
P.S. - 4	2009	20	259251	9/24/93
P.S. - 7	2012	15	259254	9/24/94
QUIN	3536	10	260147	5/07/96
M.M. - 1	-	1	308527	3/31/95
M.M. - 2	-	1	308528	3/31/95
M.M. - 3	-	1	308529	3/31/95
M.M. - 4	-	1	308530	3/31/95
M.M. - 5	-	1	308531	3/31/95
LUM - 1	-	20	313852	9/26/93
LUM - 2	-	20	313865	9/28/93
LUM - 3	-	20	313866	9/28/93
LUM - 4	-	1	304469	9/27/93
	TOTAL	251		



DAVID A. HOWARD
9/15/93

A. D. DRUMMOND
Sept ../93

FIGURE 3

QUINTO MINING CORPORATION

CLAIM MAP

SCALE 1:50,000 DATE: May, 1992

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HISTORY

Historical development of mining activity on Saddle Mountain adjacent to Lumby, B.C. is summarized from Kuran (1986), Lebel (1987) and Bradley (1990).

Mineralization at the south end of Saddle Mountain adjacent to Lumby, B.C. was noted in the early 1900's by a local teacher whose prospect workings have been named the Teacher Showing.

After a hiatus of some 50 years, Chaput Logging Company exposed silver-lead-zinc-copper veins during logging operations on the west side of Saddle Mountain in the 1960's at which time the showing was staked (since named the Mine Showing). In 1968, F.K. Explorations Ltd. acquired the claims covering this showing, started underground development and constructed a 50 tonne per day flotation mill. During the period 1968 - 1970, some 1500 tonnes of concentrate were shipped to the smelter at Trail, B.C. Work was terminated.

In 1971, Alberta Gypsum Ltd. acquired the property and mill, undertook underground and surface exploration in an attempt to establish mineable reserves. Work was terminated in 1973. Coast Interior Ventures Ltd. acquired the property in 1974 and operated sporadically in the period 1974 to 1979. The mill was expanded to 150 tons capacity in 1980. Operations were terminated and the plant closed in 1981.

In 1983, The Quinto Mining Corporation purchased the Chaput (Lumby) property and increased the property size. Geochemical and geophysical surveys outlined coincident geochemical and V.L.F.-E.M. anomalies near the top of Saddle Mountain. A trenching program followed which exposed the Plateau Shear Zone. Sampling of the shear zone produced encouraging gold and silver results from brecciated quartz veins in a graphitic host. During 1985, an initial reverse circulation drill program of 10 holes was completed and followed by 1,396 metres of diamond drilling in 13 holes. In 1986, the Saddle Mountain portion of the property was geologically mapped, additional ground V.L.F and magnetometer surveys conducted and 2,700 metres of NQ diamond drilling completed along the Plateau Shear Zone. During 1987, 32 reverse circulation and 7 diamond drill holes for a combined total of 3,030 metres was completed. Additional ground V.L.F.-E.M., magnetometer and geochemical surveys were conducted. An initial metallurgical test was completed by Lakefield Research. In early 1988, Kilborn Engineering constructed a computer generated model of the Plateau Shear Zone based on 21 vertical sections.

The Quinto Mining Corporation entered into a joint venture agreement with Golden Seville Resources Ltd. in the spring of 1988. Purpose of this agreement was to test a proprietary column leach process concept. During the period July to November, 1988, Sancold Resources Contractors Inc. completed 186 metres (610 feet) of 3.05 x 3.66 metre (10 x 12 feet) exploration drift in the hanging wall of the Plateau Shear Zone and two cross-cuts totalling 105 metres (344 feet) of 2.74 x 3.05 metres (9 x 10 feet). A preliminary feasibility study was completed by Bechtel Canada, Inc. in December 1988. At this point, Golden Seville Resources Ltd. could not fulfil its financial commitments and the column leach testing project was terminated. The property was inactive during 1989. During 1990, M. Bradley (1990) geologically mapped and sampled the Plateau Shear Zone workings. The Quinto Mining Corporation initiated this work having noted not only the variable gold content (nugget effect) but also the presence of abundant graphite. Bradley recommended bulk sampling to evaluate the gold and graphite content over the entire width of the Plateau Shear Zone. To date, the Bradley (1990) recommendations have not been implemented. Prior to March 9, 1993 a few preliminary metallurgical tests were completed (See Drummond and Howard, March 9, 1993).

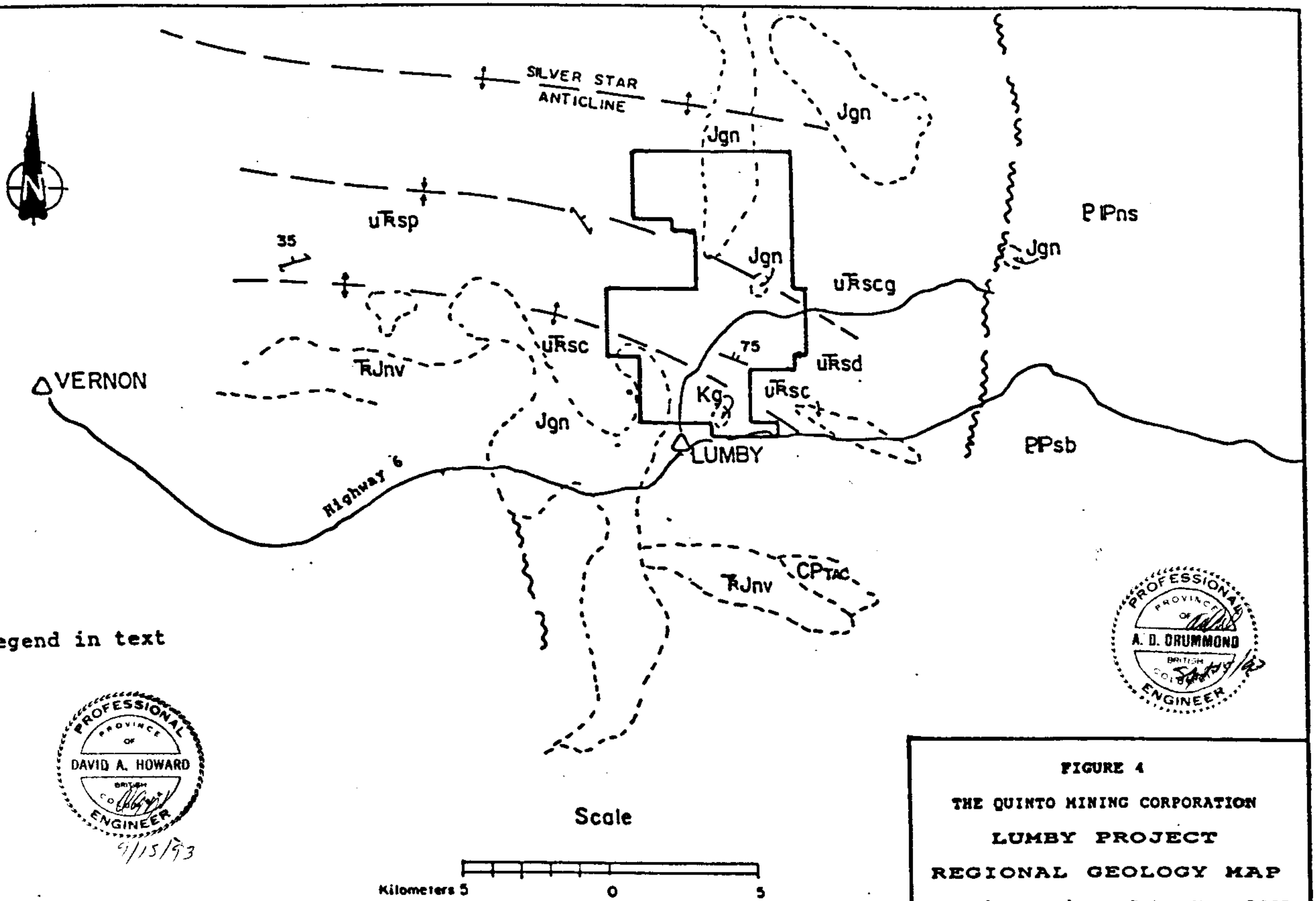
During the period to September 15, 1993, the main thrust has been metallurgical investigation (Process Research Associates Ltd.) with analytical work by Dr. Jim McLeod, Cominco Research Laboratory and by Acme Analytical Laboratories Ltd.

REGIONAL GEOLOGY

Regional geology of the Lumby area is taken from Okulitch,(1979) and is shown on Figure 4. Modifications have been made to the geology map base for purposes of clarity and to reduce the number of rock units, some of which have no bearing on the presence or absence of mineralization.

Earlier geologic mapping by Jones (1959) of the Vernon map area shows the Lumby property mainly underlain by undefined rocks of the Shuswap Terrane-Monashee Group (Archean or later) and bracketed by two parallel northwest trending major faults.

Subsequent work by Okulitch,(1979) totally redefines this earlier interpretation. Jones (1959) considered the Shuswap Metamorphic Complex to be either an extension of the Archean and Proterozoic Canadian Shield (Okulitch, 1979). Okulitch (1979) has found that along the western margins of the Shuswap Metamorphic



Legend in text

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
DAVID A. HOWARD

9/15/93

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
A. D. DRUMMOND

FIGURE 4

THE QUINTO MINING CORPORATION

LUMBY PROJECT

REGIONAL GEOLOGY MAP

Scale: as shown Date: May, 1992

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Modified from Okulitch, 1978 - Open File 637

LEGEND FOR FIGURE 4
(Modified from Okulitch, 1979)

MESOZOIC

Cretaceous

Kg Granite, Granodiorite, Lesser Quartz Monzonite and Quartz Diorite.

Jurassic

Jgn Massive and foliated, syntectonic Pegmatite, Aplite, Leucocratic Granite, and Quartz Monzonite bordering and within Shuswap Metamorphic Complex and Okanagan Plutonic and Metamorphic Complex; Silver Star Intrusions: (May include Orthogneiss of Paleozoic and Proterozoic ages).

INTRUSIVE CONTACT

Triassic and Jurassic
Nicola Group (Possibly includes Slocan Group)

TRJnv
Andesite and Basalt flow rocks, porphyritic augite Andesite, Breccia, Tuff, Agglomerate, Greenstone, chloritic Phyllite, minor Argillite, Limestone, Sericite Schist.

Slocan Group
Sicamous Formation

uTRsc
Sericitic, graphitic and argillaceous Limestone, calcareous Phyllite, Argillite.

uTRsp
Shale, Argillite, massive Siltstone, Phyllite, Tuff and calcareous Pelite, minor Conglomerate, Limestone.

uTRscg
Conglomerate.

Proterozoic and Paleozoic (may include Archean)

Shuswap Metamorphic Complex

PIPns
Undivided granitoid Gneiss, Paragneiss, Schist, minor Quartzite, Marble, Amphibolite.

PIPsb
Quartz mica Schist, commonly garnet and sillimanite bearing.

Complex, i.e., the Lumby area, units of probable Mesozoic and Paleozoic age can be traced into the Complex. Where this appears to be the case the units have been extracted from the Complex and correlated with those of known stratigraphic affinity (Okulitch, 1979).

Okulitch (1979) has mapped the Lumby area as being mainly underlain by shale, argillite, massive siltstone, conglomerate, phyllite, tuff, minor andalusite, staurolite and kyanite schist, limestone, greenstone and chloritic phyllite of the Slocan Assemblage. Okulitch (1979) could not locate the faulted boundary between the Slocan and Sicamous Formations and therefore has included everything in the Slocan Formation. The faulted block of Jones (1959) is not present. Okulitch (1979) shows the area covered by the Lumby property as being centred on a parallel westerly trending anticline-syncline structure that is parallel to the well defined Silver Star Anticline whose axis is just north of the property (See Figure 4).

PROPERTY GEOLOGY

Property geology for this report will be restricted to the Saddle Mountain portion of the property. Kuran (1986) mapped a hornblende diorite at the southern end of Saddle Mountain to the north of which occurs variably metamorphosed sedimentary rock and/or fine grained volcanic rocks. A sequence of argillaceous rocks lies interbedded with fine grained micaceous schists, the later being metamorphosed sedimentary rocks in the opinion of the writer rather than metamorphosed acid volcanic tuffs as suggested by Kuran (1986).

Of particular importance is the presence of the micaceous graphite-gold bearing Plateau Shear Zone. The Plateau Shear Zone has been traced for 2300 metres (7546 feet) by a Dighem III E.M. survey and confirmed for over a 1000 metres (3280 feet) by drilling This shear zone strikes east-west with a dip of 45 degrees to the south toward the hornblende diorite intrusive and is the site of contorted quartz-carbonate (siderite)-pyrite veins bounded by massive foliated micaceous graphitic (mylonitic ?) schist containing pods of massive pyrite and scattered disseminated and sheared pyrite. The shear limits are bounded by harder, coarsely jointed, fine grained micaceous schist. The entire sequence appears to have been of a sedimentary nature originally. Through the mechanism of regional metamorphism and tilting during intrusion of the diorite, the less competent argillaceous beds became a plane of weakness along which movement occurred. This movement allowed the introduction of not only mineralizing solutions but also heat the result of which produced the quartz-pyrite-gold veins, the pyritic gold-bearing pods and the formation of graphite. A continuum of structural readjustments along this plane of weakness produced the contorted and foliated Plateau Shear Zone that is observed today. Kuran (1985) geology map is reproduced herein as Figure 5.

GRAPHITE INVESTIGATION

Previous investigations by the writer and A.D. Drummond, Ph.D., P.Eng. lead to the conclusion that the Plateau Shear Zone contained a very high percentage of graphite due to the dense black sheen exhibited by the material in the zone. The black material behaved like graphite including its ability to mark paper. Subsequent assaying by the classical "Leco" method indicated that the rock only contained a trace of graphite. This inconsistency was finally addressed by changing the assay procedure. The "classical leco method" involves pre-heating the sample to 600 degrees centigrade to remove any organic carbon and/or carbonate carbon prior to placing it in the Leco Analyzer. The new method involved using a nitric acid wash to remove any organics, sulphur, calcium carbonate and soluble iron followed by a hydrofluoric acid wash to remove the silica. The sample was then analyzed by the "leco" method without the pre-heat step to determine carbon content. A comparison of the two methods is shown in Table 1 below:

TABLE 1

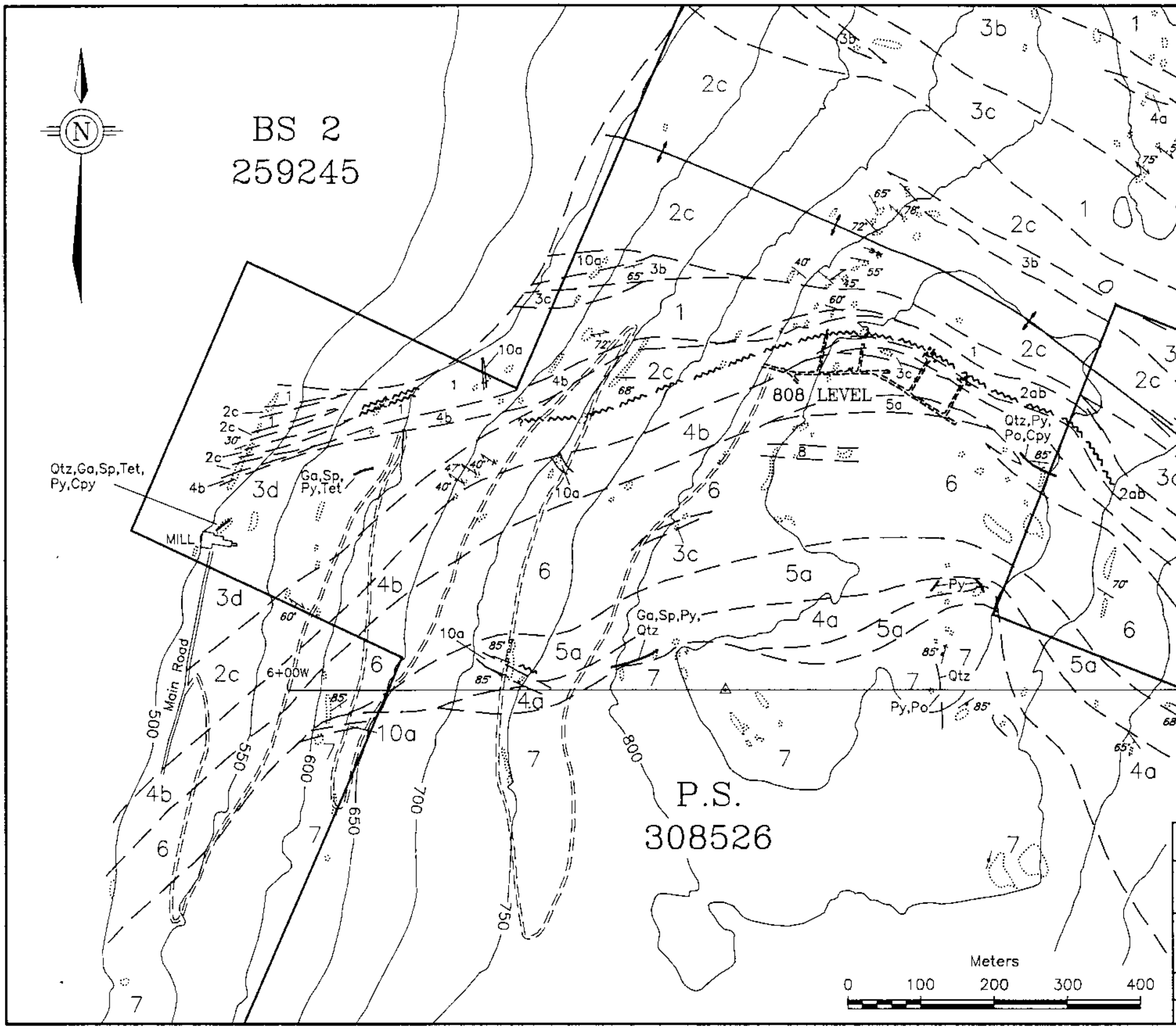
A comparison of carbon assay methods

Sample No. 5015B

Method 1	Method 2
Preheat oxidation/leco	Wet chemical oxidation/leco
Graphite carbon percent	Graphite carbon percent
0.23	5.35

Sample 5015B is a flotation product produced by Westcoast Minerals Testings Inc. from a composite sample of fresh graphitic material from the underground workings.

The above discrepancy in carbon content between the two assay methods is explained by the fact that the oxidation rate (i.e. burning) is related to surface area of the mineral/element particle. In the present case it is now known that the graphite in the Plateau Zone samples is extremely fine grained. Under 1000



NOTE: LEGEND ON FOLLOWING PAGE.

DAVID A. HOWARD

THE QUINTO MINING CORP.

LUMBY PROJECT
SADDLE MOUNTAIN

GEOLOGY MAP

Modified from Kuran (1987)

D.D.H. GEOMANAGEMENT LTD.

DATE: JULY, 1994	REVISED:	SCALE: AS SHOWN	FIGURE: 5

LEGEND FOR KURAN (1987) GEOLOGY MAP - FIGURE 5

TABLE 2

Table of Formations

10		Mafic Dykes
	10a	Hornblende Gabbro; medium to coarse crystalline, granular, dark green.
	10b	Hornblendite; very fine grain, dark green to black.
		-- Intrusive Contact --
9		Biotite Granite
		Very fine to fine crystalline, 5-15% fine pyrite, light grey.
		-- Intrusive Contact --
8		Hornblende Feldspar Porphyry Dyke
		Light grey, fine crystalline, hornblende lathes.
		-- Intrusive Contact --
7		Polyphase Hornblende Diorite
		Undifferentiated very fine chilled to coarse crystalline, hornblende or feldspar porphyritic, weakly foliated, variably pyritic.
		-- Intrusive Contact --
6		Argillite
		Well bedded to laminated, black, interbeds of fine grey felsic ash, variably calcareous.
5		Biotite Siltstone
	5a	*Fine grained, moderately to poorly sorted,* well bedded, minor argillite, beds of argillite rip-up clasts.
	5b	Turbeditic cycles, fine siltstone to coarse pebbles, graded into unit 4b.
4		Intermediate Volcanic Tuffs
	4a	Ash tuff; fine to medium grain, drab green, massive.
	4b	Lithic tuff; poorly sorted, turbeditic cyclic members, polymictic matrix supported.
	4c	Argillaceous ash tuff; fine grained, finely bedded.
3		Acid Volcanic Tuffs
	3a	Ash Tuff; fine grain, rare lapilli, ligh grey, locally sericitic rich and crenulated, *minor disseminated pyrite.*
	3b	Lapilli-ash tuffs; ash to medium lapilli size, dacite to andi-dacite composition, variably calcareous.
	3c	Crystal tuff; fine sericitic ash matrix, 15% 0.5-2 mm subrounded *feldspar* or quartz crystals.
	3d	Spotted-banded tuff; calcareous, greenish grey, knots of brown biotite and pyrrhotite.
2		Argillite / Shear Zone
	2a	Argillite; moderately to intensely sheared, graphitic gouge, pyritized, minor quartz veining.
	2b	Quartz veins; sugary to bullish, variably pyritic, up to 60% pyrite, minor pyrrhotite, trace galena and sphalerite.
	2c	Argillite; poorly sheared, 2-3% 1 cm buckshot pyrite cubes.
1		Acid Volcanic Tuffs
		Crystal tuff; dark grey fine felsic matrix, 10-20% 0.5-1 cm euhedral feldspar crystals, minor subrounded quartz eyes.

magnification it was determined that most of the graphite grains were in the 3-5 by 0.1-0.3 micron range which translates into a maximum surface area for each grain. Therefore in the thermal oxidation/leco method, the pre-heating to 600 degrees centigrade of the sample not only drove off the organic and carbonate carbon but also most of the fine grained graphite carbon prior to putting it in the Leco Analyzer. The thermal oxidation of very fined grained graphite was confirmed by submitting two chemically pure graphite samples for analysis using the "Classical Leco" method. The two samples (KS-75 and No. 763) were very fine grained (approx. 325 mesh) and were guaranteed to contain 99.9 and 96 percent carbon respectively. As can be seen in the following Table 2 the assay results using the "Classical Leco" method does not even approximate the correct value.

TABLE 2

Classical Leco assays of graphite standards

Sample KS-75 (99.9% graphite)			Sample No. 763 (96% carbon)		
L.O.I.	TOT/C	GRA/C	L.O.I.	TOT/C	GRA/C
91.5%	100.49%	38.16%	91.6%	89.39%	7.37%

L.O.I. = Loss on ignition, TOT/C = Total carbon, GRA/C = Graphite

PREVIOUS METALLURGICAL TESTING

The previous metallurgical testing by Westcoast Mineral Testing Inc. (Drummond and Howard, March 9, 1993) was originally initiated to define the possibility of recovering graphite in economic quantities as a by product of the gold mineralization. The results of the initial investigation showed that graphite was present but in a form that would probably not be recoverable as a separate product (See "GRAPHITE INVESTIGATION" above).

In the previous investigation approximately 10 kilograms of graphitic underground material was dry crushed to 90 percent passing 20 mesh by the Placer Research Centre. Westcoast Mineral Testing Inc. combined this material with higher percentage pyritic material to approximate "mine run" rock. A summary of the Westcoast Mineral Testing Inc. test is as follows (From Hawthorn, 1/3/93):

SAMPLE COMPOSITION

The test sample consisted of 73 percent graphitic material assaying 0.024 O.P.T. gold and 27 percent pyritic material assaying 0.60 O.P.T. gold for a combined head assay of 0.180 O.P.T. gold.

PROCEDURE

Grind 1000 gm / 5 min. / 67% solids
Pan: K = +200:1
Float: Staged rougher to completion
Wet/dry screen rougher tailings to 325 mesh

METALLURGICAL CALCULATIONS

			Assay			Distribution%
Product	Wt%	Au oz./t	LOI%	Fe%	S%	Au
Pan conc.	1.1	2.28				9.1
F-1 RC	15.2	0.136	11.3	5.5	2.3	7.6
F-2 RC	20.9	0.980	25.1	34.4	35.6	74.6
F	36.1					82.2
Tails	62.8	0.038				8.7
Feed(calc)	100.0	0.275				100.0
	Assay	0.180				

RC = Rougher Concentrate

Note when comparing the results of the latest test above it again shows a very significant increase in gold content in comparison to the head assay.

The F-1 product although looking like pure graphite only contained 5.35 percent graphite carbon. Visually under high magnification (60x) fine grained pyrite (4-5 percent) was visible which explains the above iron and sulphur assays. A whole rock analysis of the F-1 product confirmed that one or more silicate minerals constituted approximately 90 percent of the product. The percentages of silica, aluminum and potassium were consistent with a muscovite/sericite composition as indicated by the X.R.D work.

Preliminary petrographic analysis suggested that the graphitic portion of the host rock is mainly composed of very fine grained muscovite/sericite. The reason that it appears so readily floatable is the ultra fine grained nature of the interleaved graphite which makes the muscovite grains behave like graphite. Muscovite is not normally floatable with low concentration of DF 250 (Dow Froth 250). It should be noted that in excess of 15 percent by weight of the original sample was a flotation product.

PRESENT METALLURGICAL TESTING

Process Research Associates Ltd. under the direction of Dr. Bern Klein with assistance from the authors has outlined the following Flow Sheet, Figure 6. (See Lumby Graphite/Sericite/Gold Property: Metallurgical Testing, dated July 20, 1993 in Appendix A).

The rock from the Plateau Shear Zone is ground to 70% -200 mesh. At a pulp density of 15% with Dowfroth 250 as a conditioner, the rock is floated to produce a black very fine grained product composed mostly of fine muscovite (estimated at about 80% plus) with attached very fine graphite (estimated at about 10%). This product is then leached with HCl to remove any entrained calcite (carbonate mineral) and then leached with HNO3 to remove any entrained pyrite. The final product is then dried at low temperature to insure that the fine graphite does not oxidize. The product is black, platy and has a grain size of 50% < 400 mesh. This product is termed "Schillerite No. 1" and comprises about 15% of the original head material.

The tailing material from the first float then becomes the feed for the second float step. After changing the pH to slightly basic, a pyrite collector is added and pyrite is floated. This pyrite concentrate comprises about 15% of the original head material and contains above 90% of the gold in the original material. At this stage in the development of the flow sheet, this material is set aside as the immediate goal is to produce the industrial mineral products for market testing.

The tailing material from the pyrite float then becomes the feed for the third flotation step which is carbonate flotation. The reason for this step is that in order to float muscovite the pH must be reduced to 3 and conditioned at pH 3 for 24 hours before the amine collectors will work for muscovite. Consequently, removal of carbonate by flotation reduces the amount of acid required, reduced the time required for conditioning or stabilization at pH 3 and eliminates the chemical production of gypsum in the product. This innovation was not incorporated into the flow sheet at the time of the July 20, 1993 report by Dr. Klein.

The tailing material from the carbonate flotation then becomes the feed for the fourth float step. After the change in pH to 3 as previously mentioned above, the muscovite collector amine is added and a muscovite product is floated. This product comprises about 20 to 30% of the original head material and is composed of a very fine grained platy muscovite mineral which has a grain size

FIGURE 6

FLOWSHEET DEVELOPED FOR SCHILLERITE NO.1 AND NO. 2 PRODUCTS

(This FLOWSHEET is subject to change with further research)

HEAD MATERIAL

Mine run from Plateau Shear Zone crushed to 70% -200 mesh.
.
STEP ONE
.
FLOTATION (Dowfroth 250 only)
.
.....F-1 concentrate + HCl leach/wash
.   + HNO3 leach/wash/air dry
.   to produce **SCHILLERITE NO. 1 PRODUCT**
.
. Tailing
Feed .
STEP TWO
.
FLOTATION (adjust pH, add collector)
.
.....Pyrite-gold concentrate
.   (product is reserved for later treatment)
.
. Tailing
Feed .
STEP THREE-A
. Condition for 24 hours to stable pH 3
. (to remove carbonate)

OR

STEP THREE-B
. CARBONATE FLOTATION (to remove carbonate)
.
STEP FOUR-A
.
FLOTATION (From 3-A at pH 3)
.
.....Mica concentrate + H20 wash + filter + dry
.   to produce **SCHILLERITE NO. 2 PRODUCT**

OR

STEP FOUR-B
.
HYDROCYCLONE (From 3-B tailing)
.   (to effect a sand/slime size fraction separation)
.....Mica concentrate + filter + dry
.   to produce **SCHILLERITE NO. 2 PRODUCT**
.
.
Final tailing

in the order of 50% -400 mesh. During the mica flotation, it was noted that the fine grained nature of the micaceous product may be separated from the remaining rock gangue by the use of hydrocyclones as the settling rate of the mica product was very much slower than the settling rate for the rock gangue grains. This observation will lead to investigations of these settling rates and the use of hydrocyclones instead of the fourth flotation step outlined in this paragraph. This may increase the yield of this micaceous product which is termed "Schillerite No. 2". Any residual entrained graphite is removed from the product by heating during the final drying stage.

The final tailing material which comprises about 40 to 50% of the original rock, has a high quartz content, some feldspar and some micaceous material.

The primary thrust of this new metallurgical test work has been to develop a practical and workable flow sheet to produce the Schillerite No. 1 and Schillerite No. 2 products on a bench scale. When these products have been market tested and requests for additional product for further testing have been received, the flow sheet developed to date will guide the type of on-site milling facility required for testing a larger bulk sample.

CONCLUSIONS AND RECOMMENDATIONS

The post March 1993 research program on The Quinto Mining Corporation Lumby property has concentrated on the development of a flow sheet that could be use to recover the pyrite/gold, graphite/muscovite and muscovite products that constitute at least 50 percent of the rock mass in the Plateau Shear Zone.

The favourable metallurgical results to date have produced a graphite/muscovite product that contains approximately 88 percent muscovite and 12 percent graphite. This product is termed "Schillerite No. 1" comprises about 15 percent of the original head material.

After removal of the pyrite and carbonate from the tails of the above product, a relatively pure very fine grained muscovite product is floated with amine as a collector. The muscovite product is termed "Schillerite No. 2" and constitutes approximately 20-30 percent of the original head material.

Samples of the Schillerite No. 1 and No. 2 products are being evaluated by potential end users so after their evaluation some modifications may have to be made to the flow sheet to accommodate individual requirements.

Based on the successes to date in recovering the Schillerite Products it is recommended that the following investigations be instituted.

(1) Conduct further mineralogical studies to better define the character and composition of the fine grained muscovite in the zone.

(2) Continue with the metallurgical testing to optimise the recovery of the muscovite - muscovite/graphite product with special attention given to producing a product with ASTM specifications.

(3) Proceed with the planned underground development to test mining methods and provide material for bulk testing.

(4) Proceed with the rehabilitation of the on site mill so it will be ready to process bulk material from the planned underground development.

PERSONNEL TIME DISTRIBUTION

(B.S 1-5, P.S., P.S. 2-4, 7, Quin, M.M. 1-5, Lum 1-4)

A.D. Drummond, Ph.D., P.Eng. (Geological Engineer)

Office	Research and consulting services Period April 1, 1993 - Sept. 15, 1993	5.5 months

D.A. Howard, M.Sc., P.Eng. (Geological Engineer)

	Research and consulting services Period April 1, 1993 - Sept. 15, 1993	5.5 months

Bernhard Klein, Ph.D. (Senior Process Metallurgist)
Time included in Process Research Associates Ltd. invoice

E.M. Pelto P.Eng. (Consultant)
Time included in M.D. Mine Design invoice

Jim McLeod P.Eng. (Consultant)
Time included in Cominco Exploration Invoices

David Jenkins (Consultant)
Time included in Ainsworth-Jenkins invoice

Jim MacNeil (Mill Wright, Supervisor)	3.0 months
Arron MacNeil (Mill Wright)	4.0 months
Tony Zagar (Mill Wright)	4.0 months
Randy Walters (Labour)	2.0 months
Glen Jensen (Labour)	2.0 months
Lou Manning P.Eng. (Consultant)	1.0 day
David Jenkins P.Eng. (Consultant)	2.0 hours

COST STATEMENT

PERSONNEL

D.D.H. Geomanagement Ltd. A.D. Drummond, Ph.D., P.Eng. D.A. Howard, M.Sc., P.Eng.	$10,813.63
Jim MacNeil	2,323.41
Arron MacNeil	6,388.36
Tony Zagar	6,495.06
Randy Walters	1,054.85
Glen Jensen	1,381.07
Labour sub-total	$28,456.38

EXPENSES AND DISBURSEMENTS

Mill Overhaul (Includes replacement parts and equipment)	$25,000.00
Process Research Associates Ltd.	32,593.27
Cominco Exploration	521.63
M.D. Mine Design	6,140.73
Ainsworth-Jenkins	104.33
L.J. Manning and Associates	615.25
Westcoast Mineral testing Inc.	1,918.45
Acme Analytical Laboratories Ltd.	28.89
4 x 4 truck use at $1000/mo.	4,000.00
Accommadation and food for period	4,000.00
Western Reproductions Group	77.38
Expenses sub-total	$74,999.93
Total amount expended	$103,456.31

Respectfully submitted,

A. D. DRUMMOND

Sept 15/93

A.D. Drummond P Eng

A.D. Drummond, Ph.D., P.Eng.

DAVID A. HOWARD

9/15/93

D,A. Howard, M.Sc., P.Eng.

CERTIFICATION

I, Arthur Darryl Drummond of the City of Vancouver, Province of British Columbia, hereby certify as follows:

1. I am a geological engineer residing at 20-5760 Hampton Place, Vancouver, B.C., V6T 2G1 and employed by D.D.H. Geomanagement Ltd., with an office at 422 - 470 Granville St., Vancouver, B.C., V6C 1V5.

2. I am a registered Professional Engineer of the Province of British Columbia, certificate number 5778. I graduated from the University of British Columbia in 1959 with a B.A.Sc. in geological engineering, and in 1961 with a M.A.Sc. in geological engineering. I graduated from the University of California at Berkeley in 1966 with a Ph.D. in geology.

3. I have practised my profession continuously for 30 years primarily with the Placer Development Group of Companies at Craigmont, Endako and Gibraltar mines, and in mineral exploration in Canada, United States of America, Chile, Argentina, Mexico and the Philippines.

4. I am a co-author of this report which is based on personal supervision of the described work program and from data contained in the files of D.D.H. Geomanagement Ltd., private reports and government publications.

5. I hold shares of The Quinto Mining Corporation.

6. This report may be utilized for development of the property provided that no portion may be used out of context in such a manner as to convey a meaning which differs from that set out in the whole.

7. Consent is hereby given to The Quinto Mining Corporation to use or reproduce this report or any part of it for the purposes of development of the property, or related to the raising of funds.

Dated at Vancouver, B.C. this 15th day of September, 1993.

A. D. DRUMMOND

A.D. Drummond, Ph.D., P.Eng.
D.D.H. GEOMANAGEMENT LTD.

CERTIFICATION

I, David A. Howard, of the City of Vancouver, Province of British Columbia, hereby certify as follows:

1. I am a geologist residing at 9040 Glenallan Gate, Richmond, B.C., with an office at 422-470 Granville Street, Vancouver, B.C.

2. I am a registered Professional Engineer of the Province of British Columbia. I graduated from Montana State University in 1964 and from the University of Washington in 1967.

3. I have practised my profession continuously since June, 1966.

4. I am a co-author of this report which is based on personal supervision of the described work program and from data contained in the files of D.D.H. Geomanagement Ltd., private reports and government publications.

5. I hold shares of The Quinto Mining Corporation.

6. This report may be utilized for development of the property provided that no portion may be used out of context in such a manner as to convey a meaning which differs from that set out in the whole.

7. Consent is hereby given to The Quinto Mining Corporation to use or reproduce this report or any part of it for the purposes of development of the property, or related to the raising of funds.

Dated at Vancouver, B.C. this 15th day of September, 1993.

DAVID A. HOWARD

David A. Howard, M.Sc., P.Eng.

REFERENCES

Bradley, M. (1990) Report On Chip/Channel Sampling And Geological Mapping Of The 140East And 190East Cross-cuts, 808 m Level Underground, Chaput 5 Claim: a private report for The Quinto Mining Corporation by Mike Bradley and Associates, dated October 1, 1990.

Drummond, A.D. and Howard, D.A. (1993) Report Of The Exploration Potential Of The Muscovite-Graphite-Gold Deposit Of The Quinto Mining Corporation, Vernon Mining Division, British Columbia, Canada: a private report by D.D.H. Geomanagement Ltd., dated March 9, 1993.

Jones, A.G. (1959) Vernon Map-Area, British Columbia: Geol. Surv. Can., Memoir 296, pp.182.

Klein, B. (1993) Lumby Graphite/Sericite/Gold Property: Metallurgical Testing: a private report by Process Research Associates Ltd., dated July 20, 1993.

Kuran, D.L. (1986) Report On The Lumby Project: an internal report for The Quinto Mining Corporation, dated February 3, 1986.

Kuran, D.L. (1987) Report On The Lumby Project, Lumby, B.C., Vernon Mining Division, Phase III Exploration: a private report for The Quinto Mining Corporation, co-signed by Alfred R. Allen, P.Eng., dated June 22, 1987.

LeBel, J.L. (1987) Report On The Lumby Project, Vernon M.D., British Columbia: a private report for Greyhawk Resources Ltd., by LeBel Geophysics, dated August 10, 1987.

Okulitch, A.V. (1979) Thompson-Shuswap-Okanagan Map-Area, British Columbia: Geol. Surv. Can., Open File 637, Scale 1:250,000.

APPENDIX A

METALLURGICAL TEST RESULTS

LUMBY GRAPHITE/SERICITE/GOLD PROPERTY:

METALLURGICAL TESTING

For:

THE QUINTO MINING CORPORATION

Attention:

Mr. Paul Schiller, President

Prepared by:

PROCESS RESEARCH ASSOCIATES LTD.
9145 Shaughnessy Street
Vancouver, B.C.
V6P 6R9

Bernhard Klein
Bernhard Klein, Ph.D.
Senior Metallurgist

Date: July 20, 1993

1. Summary

Metallurgical testwork was performed on a composite sample from the Lumby property near Vernon, B.C. Several samples were blended to prepare the 728 kg composite which was then jaw and cone crushed. Representative splits were then subjected to flotation and leach tests to produce graphite/sericite, sericite and gold bearing pyrite products.

Flotation tests were performed on samples ground to approximately 70% -200 mesh. A low pulp density (15% solids by weight) was used to reduce the potential for the entrainment of slimes in the froth product.

The naturally hydrophobic graphite and associated sericite was floated first with product yields ranging from 9.6% to 15.3%. Subsequent leaching with HCl to dissolve carbonates and HNO_3 to dissolve sulphides reduced the product yields to 8.3% to 11.7%. X-ray fluorescence (XRF) whole rock analyses of the products revealed that they contained greater than 56% SiO_2 as compared to the stoichiometric content for sericite of approximately 45%. The addition of a dispersant and performing a cleaning stage of flotation decreased the SiO_2 content slightly to 52.4%. The inability to reduce the silica(quartz) content further indicates that it does not occur as liberated grains but rather it must be physically attached to the graphite/sericite grains.

Pyrite flotation produced a concentrate yielding 11.2% of the feed which contained 90.2% of the sulphur and 79.1% of the gold. Lower recoveries were achieved (sulphur 76.6% and gold 60.4%) when floating a sample that had been pulverized dry (as compared to wet grinding). The carbon in leach (CIL) cyanidation gold extraction from the concentrate (52.4% -400 mesh) was 81.7%. Pulverizing the concentrate (from 52.4% to 84.5% -400 mesh) increased extraction to 88.8%.

Following pyrite flotation, the remaining sericite was floated using an amine collector (Armac T) while depressing the quartz by maintaining the pH at 3.0 using H_2SO_4. Prior to the flotation, the pH was maintained at pH 3.0 for 24 hours requiring 100.6 kg H_2SO_4 per tonne of feed to dissolve the calcium carbonate. Floating the carbonate minerals prior to sericite flotation would reduce the acid consumption.

The sericite concentrate SiO_2 content was approximately 39% indicating good quartz rejection from flotation. The product also contained 7.1% gypsum which precipitated during H_2SO_4 leaching of the calcium carbonate. The gypsum content was reduced to 3.2% by leaching it with water for 24 hours.

2. Introduction

Ore samples from the Lumby property (located near Vernon, B.C.) were blended to produce a composite sample. The samples were not identified but were from various locations on the property. Testwork was performed on representative splits from this sample with the objective of developing a process to produce a graphite/sericite product, a sericite product and gold bearing pyrite concentrate.

The graphite/sericite and sericite products are to be used as fillers for plastics. Since these products are to be tested as potential "new" replacements for fillers that are presently being used, details of the product specifications were not known. Flotation and leach tests were performed to attempt to minimize the levels of contaminants (eg. calcite, quartz, pyrite and other mineral matter) in the products.

Once a satisfactory process has been established, it is intended to use the process to produce products for evaluation by plastic manufacturers. This report summarizes the results of the testwork that has been completed to date.

3. Procedures & Results

Several bags and buckets of material were received from Quinto Mining's Lumby property. The material was dried, weighed and blended by coning and quartering. The sample was then jaw and cone crushed to reduce the particle top size from approximately 3 inches to -6 mesh. The total dry weight of the sample was 728 kg. Several representative splits were riffled from the sample for subsequent testwork.

Four sets of flotation/leach tests were performed on the splits. The procedures and results of the tests are described below.

3.1 Test F1

As a first attempt to produce a graphite/sericite product, a flotation test was performed on a representative sample. The test procedure and the metallurgical balance have been tabulated and are presented in the Appendix.

A 1 kg sample was ground to 72.8% -200 mesh and then subjected to flotation. Considerable amounts of very fine (clay) particles were observed in the ground product. To reduce the potential for entrainment of these fines into the froth product, a low pulp density of 15% solids by weight was used for the flotation test. To float the naturally hydrophobic graphite, DF 250 (frother) was added for the first stage of flotation. Much of the sericite was expected to float with the graphite since previous mineralogical analyses found that sericite lamellae were attached to graphite flakes (Dave Howard verbal communication).

Frother was added as required until the froth appeared barren. The flotation kinetics were quite slow with the flotation lasting 30 minutes and yielding 15.3% of the feed weight. The graphite product appeared much darker than the tails indicating concentration of the graphite.

Leach tests were performed on the graphite/sericite product to attempt to dissolve carbonate and sulphide minerals. A 50 g sample was leached for four hours in 100 mL of 10% HCl to dissolve the carbonate minerals leaving 41.1 g of solids. The residue was then leached with 30 g of concentrated HNO_3 per 100 grams of sample to dissolve the sulphide minerals (twice the dosage required to dissolve the sulphur present based on 2.63 kg HNO_3 per kg pyrite). To determine if all of the sulphide had dissolved, the residue should be analyzed for sulphide sulphur. The final residue weight was 38.1 g which is 76.2% of the initial product weight and corresponds to an overall yield of 11.7%.

The composition of the product was determined from by XRF whole rock analysis. The analysis indicated that the product contained 72% SiO_2. Based on the stoichiometric composition of sericite, the SiO_2 content of a pure product should be approximately 45%.

The high SiO_2 content may be explained partially by the entrainment of fine siliceous particles in the flotation froth. Due to the slow flotation kinetics, the amount of fines that become entrained in the froth can be substantial. In this case, increasing the flotation kinetics should reduce the silica content. It may be possible to improve the flotability of the graphite and thereby the flotation kinetics by conditioning with kerosene prior to flotation.

Previous mineralogical analyses (J. McCleod, Cominco Exploration) found that very fine (micron size) SiO_2 grains were trapped between the sericite lamellae and graphite flakes. It would be impractical to try to separate this form of SiO_2 from the product.

Following the graphite flotation, potassium amyl xanthate (PAX) was added to the pulp to float the pyrite along with the associated gold. All products were assayed for gold and sulphur.

The pyrite concentrate contained 48.1% of the gold and only 10.0% of the sulphur in a product yielding 4.7% of total feed weight. While some upgrading occurred, the recovery of sulphur was quite low.

3.2 Test F2

Following test F1, two additional flotation tests (F2 & F3) were performed. The primary objective of these tests was to decrease the SiO_2 content of the graphite/sericite product. The procedures and metallurgical balances have been tabulated and are presented in the Appendix.

For test F2, a 1 kg sample was pulverized dry using a plate pulverizer. The results from this test are to be compared to results from F3 in which wet grinding was used (see section 3.3). The particle size range was 66% -200 mesh which is slightly finer than the grind used in test F1 (73% -200 mesh).

Prior to floating the graphite, the pulp pH was adjusted to 10.5 with lime to depress the pyrite. The floated graphite/sericite product was acid leached to remove carbonate and sulphide minerals using the same procedure as described for test F1.

The graphite/sericite product yield was 10.8% which is less than the yield produced from test F1 (15.3%). Leaching with HCl and HNO_3 reduced the product yield to 76.4% of the product weight which corresponds to an overall yield of 8.25%. From the XRF whole rock analysis, the SiO_2 content was 56.7% which is a significantly lower than the 72% recorded from test F1 but still exceeds the stoichiometric content for sericite of approximately 45%.

To determine the amount of SiO_2 that is present as fines, the leached product was screened at 400 mesh and the fine fraction was submitted for XRF whole rock analysis. The -400 mesh fraction contained 97.9% of sample, so it is not surprising that there was no significant change in its composition (SiO_2 content equal to 56.4%).

Following the graphite flotation, the pH was adjusted to 7.5 with H_2SO_4 for pyrite flotation. A rougher and a scavenger stage of flotation were performed with PAX added at each stage.

The pyrite flotation resulted in a sulphur recovery of 76.6% and a gold recovery of 60.4%. The improvements in the flotation results can be explained by the higher collector dosage used (350 g/t PAX versus 25 g/t PAX in F1). Since the gold recovery and sulphur recovery are different, it is evident that either

i. not all of the gold is associated with the pyrite, or that
ii. different forms of pyrite mineralization have different gold contents.

3.3 Test F3

For test F3, a 1 kg sample was wet ground to 71.3% -200 mesh. The graphite and pyrite flotation procedures were similar to those used for test F2. In addition, cyanidation tests were performed on the pyrite concentrate and the flotation tailings product was subjected to sericite flotation.

The graphite/sericite product yield was 11.2% which is similar to the yield from test F2. Acid leaching reduced the graphite/sericite product yield to 9.3% (82.6% of flotation product remained in residue). The SiO_2 content of the leach residue was 56.0% which is almost equal to the SiO_2 content of the F2 product.

The -400 mesh fraction of the leach residue was analyzed to determine if the SiO_2 is concentrated in the fines. As was found from test F2, almost all of the material was finer than 400 mesh (96.7%) and therefore it was not surprising that the SiO_2 contents were equal.

Pyrite flotation produced a concentrate with a sulphur content of 26.1% containing 90.2% of the sulphur and 79.1% of the gold. These results are significantly better than those obtained from test F2 in which dry pulverizing was used.

Cyanidation tests were performed on the sulphide concentrate with two different grinds. The first test was performed on the concentrate as is (52.4% -400 mesh) and the second test was performed on the concentrate following pulverizing (84.5% -400 mesh). A carbon in leach procedure was used to avoid the risk of preg robbing which can occur during the cyanidation of gold ores containing graphite. Details of the procedures and the results are presented in the Appendix.

The gold extraction for the pulverized sample was 88.8% which is significantly higher than the 81.7% extracted from the sample that was not pulverized. The improvement in the extractions can be explained by the presence of gold that is completely occluded within the mineral grains. Finer grinding exposes this gold so that it can be leached by cyanidation.

The flotation tailings were subjected to sericite flotation. To selectively float sericite from other silicate minerals using amine collectors, it is necessary to maintain the pH in the 3.0 to 4.0 range. Due to the high carbonate mineral content of the tailings material, it is difficult to maintain the pH in this range. Therefore, prior to sericite flotation, the carbonate minerals were acid leached for 24 hours with H_2SO_4 at pH 3.0. The consumption of H_2SO_4 was 100.6 kg/t (100 mL of 12 N H_2SO_4 for a sample weighing 676 g). To reduce the acid consumption, the carbonate minerals could be floated prior to floating the sericite.

Two stages of flotation (rougher and scavenger) were performed with each product being analyzed separately. The product yields were 6.4% and 15.9% of the sericite flotation feed, respectively. The SiO_2 contents of the two products were 39.9 and 38.1% respectively which are well within acceptable limits for sericite.

The CaO content of the rougher concentrate was 5.0% and for the scavenger product was 9.5%. The high CaO contents can be explained by the precipitation of gypsum during leaching of calcium carbonate with H_2SO_4. In fact, the precipitation of gypsum resulted in a net weight gain in the sample.

The rougher concentrate sulphate content was determined to be 3.98% which is equivalent to 7.1% gypsum. To determine if the gypsum could be redissolved, a 25 g sample was added to 1 litre of distilled water and the mixture was agitated for 24 hours. (Based on the solubility of gypsum of 2.23 g/L and the gypsum content of the sample it was determined that 0.8 L of distilled water should be sufficient to dissolve all of the gypsum present in a 25 g sample). The solution was sampled at 1, 2, 4, 8 & 24 hours and analyzed for sulphate. The results were plotted and are presented in Figure 1 which shows that the sulphate stopped dissolving after approximately 8 hours. The sulphate level of the residue was reduced to 1.81% which is equivalent to a gypsum content of 3.2%.

3.4 Test F4

A fourth flotation test was performed with the objective of reducing the level of SiO_2 in the graphite/sericite product. For this test, sodium silicate was added to the grind (69.4% -200 mesh) to ensure dispersion of fine silicate mineral particles. Following a rougher stage of flotation, the product was refloated in a cleaner stage to attempt to reject any SiO_2 remaining. The cleaner concentrate was subjected to HCl and HNO_3 leaches. The leach residue and the cleaner flotation tailings were submitted for XRF whole rock analysis (see Appendix for procedures and results).

The rougher flotation product yield was 9.6% which was reduced to 3.7% by the cleaner flotation. The cleaner concentrate had a SiO_2 content of 52.4% which is only slightly lower than levels achieved from tests F2 and F3. The cleaner flotation tailings had an SiO_2 content of 48.6%. The lower SiO_2 content of the tailings can be explained by the presence of non-silica contaminants (carbonate and sulphide mineral particles) that effectively dilute the SiO_2 content. Evidently, the addition of a dispersant and the incorporation of a cleaning stage of flotation did not significantly reduce the level of SiO_2 in the product.

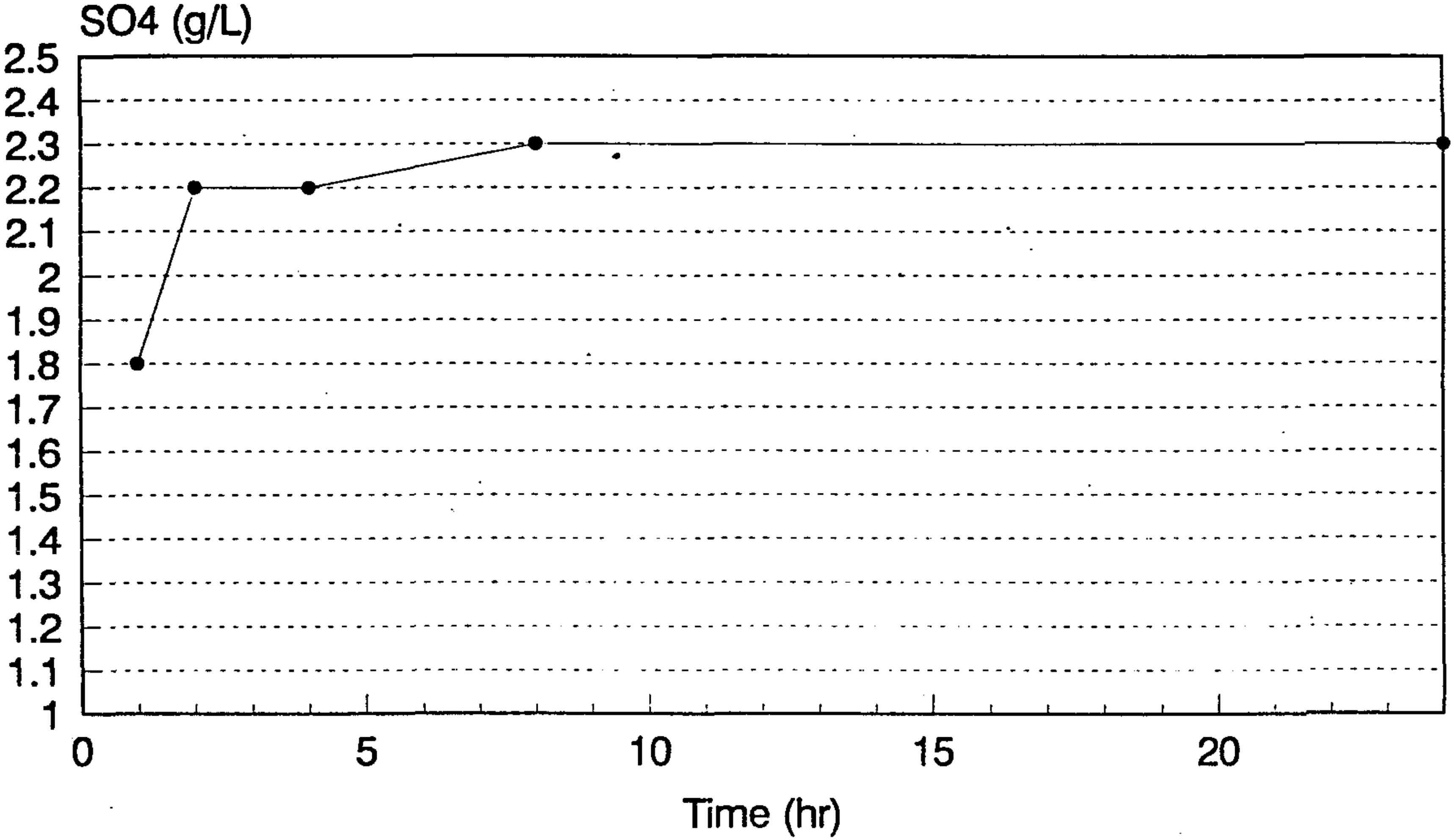


Figure 1. Dissolved sulphate concentration as a function of mixing time for a suspension containing 25 g of the F3 sericite product in 1 L of distilled water.

APPENDIX

TEST F1 RESULTS

TESTWORK PROCEDURE

Test No: 93-016 F1 **Date:** 10-May-93

Purpose: Initial bench flotation scoping test

STAGE	TIME (Minutes)	ADDITIONS	
		g/tonne	REAGENT
Grind	14		
Graphite/Sericite Flotation			1 kg flotation @ 15% solids by weight
Rougher float	30	625	DF250 (stage addition) pH=8.1 Final pH=8.0
Pyrite Flotation			
Condition	5	25	Potassium Amyl Xanthate
Rougher float	11		

SIZE DISTRIBUTION

SAMPLE NO-93-016 F1 head

Size Fraction (Tyler mesh)	Individual Percentage Retained	Cumulative Percentage Passing
65	0.1	99.9
100	2.6	97.3
150	9.8	87.4
200	14.6	72.8
325	17.8	55.1
400	3.4	51.7
Undersize	51.7	

MATERIAL BALANCE

Project no : 93-016
Test no : F1

Date : May 10, 1993

Sample description : Composite ore ground to 72.8% -200 mesh

Products	Weight		Assays			% Distribution		
			Au	Au	S(T)	Au	Au	S(T)
	(g)	(%)	(oz/ton)	(g/t)	(%)			
Graphite/Sericite Ro Conc	152.6	15.3	0.01	0.38	3.42	5.4	5.4	22.9
Pyrite Ro Conc	47.5	4.7	0.31	10.76	4.81	48.1	48.1	10.0
Final Tails	799.7	80.0	0.02	0.62	1.91	46.5	46.5	67.1
Calculated head	999.8	100.0	0.03	1.08	2.28	100.0	100.0	100.0
Assay head			0.03	0.99	3.52			

LEACH TEST REPORT

Objective: To dissolve carbonate and sulphide minerals from the F1 graphite/sericite concentrate using HCl and HNO_3, respectively.

Procedure:

HCl Leach

Sample: Test F1 graphite/sericite product
Vessel: Beaker with magnetic stirrer
Weight: 50.0 g
Solution: 100 mL of 10% HCl solution
Leach Time: 4 hours
Temperature: Ambient

HNO_3 Leach

Sample: Dried F1 HCl leach residue
Vessel: Beaker with magnetic stirrer
Weight: 41.1 g
Solution: 100 mL distilled water 30 g HNO_3 per 100 g solids
Leach Time: 2.5 hours
Temperature: Ambient

Table of Weights:

Product	Weight (g)	Weight (%)
Feed	50.0	100.0
HCL Leach Residue	41.1	82.2
HNO_3 Leach Residue	38.1	76.2

XRF - ANALYSIS

Sample: Residue from Test F1 HCl & HNO_3 leach of graphite/sericite product.

Parameter	**Level**
CaO (%)	0.09
K_2O (%)	3.45
P_2O_5 (%)	0.01
SiO_2 (%)	72.04
Al_2O_3 (%)	14.83
MgO (%)	0.72
Na_2O (%)	1.63
Fe_2O_3 (%)	2.44
TiO_2 (%)	0.62
MnO (%)	0.01
LOI (%)	4.40
Total (%)	99.79
Ba (ppm)	1041

TEST F2 RESULTS

TESTWORK PROCEDURE

Test No: 93-016 F2 **Date:** 12-May-93

Purpose: Initial bench flotation scoping test on dry ground materials

STAGE	TIME (Minutes)	ADDITIONS	
		g/tonne	REAGENT
Grind (pulverized)			
Graphite/Sericite Flotation			1 kg flotation @ 15% solids by weight
Condition	5	1287	Lime pH = 8.6 to pH = 10.7
Rougher float	30	625	DF250 (stage addition)
			Final pH = 8.9
Pyrite Flotation			
pH adjustment		1854	Conc H2SO4 pH = 6.9
Rougher float	10	100	Potassium Amyl Xanthate
Scavenger float	30	250	Potassium Amyl Xanthate
			Final pH = 7.9

SIZE DISTRIBUTION

SAMPLE NO-93-016 F2 head

Size Fraction (Tyler mesh)	Individual Percentage Retained	Cumulative Percentage Passing
65	9.8	90.2
100	8.9	81.3
150	8.5	72.8
200	6.7	66.1
325	9.0	57.1
400	2.2	54.9
Undersize	54.9	

MATERIAL BALANCE

Project no : 93-016
Test no : F2

Date : May 10, 1993

Sample description : Composite ore pulverized to 60.1% -200 mesh

Products	Weight		Assays			% Distribution		
	(g)	(%)	Au (oz/ton)	Au (g/t)	S(T) %	Au	Au	S(T)
Graphite/Sericite Ro Conc	105.9	10.8	0.09	3.08	2.26	32.6	32.6	6.7
Pyrite Ro Conc	184.1	18.9	0.10	3.29	14.90	60.4	60.4	76.6
Final Tails	686.2	70.3	0.00	0.10	0.87	7.0	7.0	16.7
Calculated head	976.2	100.0	0.03	1.03	3.67	100.0	100.0	100.0
Assay head			0.03	0.99	3.52			

LEACH TEST REPORT

Objective: To dissolve carbonate and sulphide minerals from the F2 graphite/sericite concentrate using HCl and HNO_3, respectively.

Procedure:

HCl Leach

Sample: Test F2 graphite/sericite product
Vessel: Beaker with magnetic stirrer
Weight: 50.0 g
Solution: 100 mL of 10% HCl solution
Leach Time: 4 hours
Temperature: Ambient

HNO_3 Leach

Sample: Dried F2 HCl leach residue
Vessel: Beaker with magnetic stirrer
Weight: 41.1 g
Solution: 100 mL distilled water 30 g HNO_3 per 100 g solids
Leach Time: 2.5 hours
Temperature: Ambient

Table of Weights:

Product	Weight (g)	Weight (%)
Feed	50.0	100.0
HCL Leach Residue	41.8	83.6
HNO_3 Leach Residue	38.2	76.4

XRF - ANALYSIS

Sample: Residue from Test F2 HCl & HNO_3 leach of graphite/sericite product.

Parameter	**Level**
CaO (%)	0.09
K_2O (%)	5.17
P_2O_5 (%)	0.02
SiO_2 (%)	56.71
Al_2O_3 (%)	20.73
MgO (%)	1.32
Na_2O (%)	1.41
Fe_2O_3 (%)	1.65
TiO_2 (%)	1.04
MnO (%)	0.01
LOI (%)	11.59
Total (%)	99.74
Ba (ppm)	1535

XRF - ANALYSIS

Sample: -400 mesh fraction of residue from Test F2 HCl & HNO_3 leach of graphite/sericite product.

Parameter	Level
CaO (%)	0.11
K_2O (%)	5.17
P_2O_5 (%)	0.02
SiO_2 (%)	56.41
Al_2O_3 (%)	20.85
MgO (%)	1.29
Na_2O (%)	1.23
Fe_2O_3 (%)	1.66
TiO_2 (%)	1.04
MnO (%)	0.02
LOI (%)	11.86
Total (%)	99.67
Ba (ppm)	1498

TEST F3 RESULTS

TESTWORK PROCEDURE

Test No: 93-016 F3 **Date:** 13-May-93

Purpose: Initial bench flotation scoping test on wet ground materials

STAGE	TIME (Minutes)	ADDITIONS g/tonne	ADDITIONS REAGENT
Grind (pulverized)			
Graphite/Sericite Flotation			1 kg flotation @ 15% solids by weight
Condition	5	1716	Lime pH = 8.1 to pH = 10.7
Rougher float	30	625	DF250 (stage addition)
			Final pH = 9.0
Pyrite Flotation			
pH adjustment		1854	Conc H_2SO_4 pH = 6.9
Rougher float	7	100	Potassium Amyl Xanthate
Scavenger float	1	50	Potassium Amyl Xanthate
			Final pH = 7.8

SIZE DISTRIBUTION

SAMPLE NO-93-016 F3 head

Size Fraction (Tyler mesh)	Individual Percentage Retained	Cumulative Percentage Passing
65	0.2	99.8
100	2.9	96.9
150	10.1	86.8
200	15.5	71.3
325	17.2	54.1
400	3.4	50.7
Undersize	50.7	

MATERIAL BALANCE

Project no : 93-016
Test no : F3

Date : May 12, 1993

Sample description : Composite ore ground to 71.3% -200 mesh

Products	Weight		Assays			% Distribution		
			Au	Au	S(T)	Au	Au	S(T)
	(g)	(%)	(oz/ton)	(g/t)	(%)			
Graphite/Sericite Ro Conc	112.1	11.2	0.03	1.06	1.85	12.5	12.5	6.4
Pyrite Ro Conc	111.9	11.2	0.20	6.71	26.10	79.1	79.1	90.2
Final Tails	776.0	77.6	0.00	0.10	0.14	8.4	8.4	3.4
Calculated head	1000.0	100.0	0.03	0.95	3.24	100.0	100.0	100.0
Assay head			0.03	0.99	3.52			

LEACH TEST REPORT

Objective: To dissolve carbonate and sulphide minerals from the F3 graphite/sericite concentrate using HCl and HNO_3, respectively.

Procedure:

<u>HCl Leach</u>

Sample:	Test F3 graphite/sericite product
Vessel:	Beaker with magnetic stirrer
Weight:	50.0 g
Solution:	100 mL of 10% HCl solution
Leach Time:	4 hours
Temperature:	Ambient

<u>HNO_3 Leach</u>

Sample:	Dried F3 HCl leach residue
Vessel:	Beaker with magnetic stirrer
Weight:	41.1 g
Solution:	100 mL distilled water 30 g HNO_3 per 100 g solids
Leach Time:	2.5 hours
Temperature:	Ambient

Table of Weights:

Product	Weight (g)	Weight (%)
Feed	50.0	100.0
HCL Leach Residue	43.6	87.2
HNO_3 Leach Residue	41.3	82.6

XRF - ANALYSIS

Sample: Residue from Test F3 HCl & HNO_3 leach of graphite/sericite product.

Parameter	**Level**
CaO (%)	0.08
K_2O (%)	5.57
P_2O_5 (%)	0.02
SiO_2 (%)	55.96
Al_2O_3 (%)	22.21
MgO (%)	1.22
Na_2O (%)	1.40
Fe_2O_3 (%)	1.33
TiO_2 (%)	1.21
MnO (%)	0.02
LOI (%)	10.45
Total (%)	99.47
Ba (ppm)	1617

XRF - ANALYSIS

Sample: -400 mesh fraction of residue from Test F3 HCl & HNO_3 leach of graphite/sericite product.

Parameter	Level
CaO (%)	0.13
K_2O (%)	5.57
P_2O_5 (%)	0.02
SiO_2 (%)	56.02
Al_2O_3 (%)	22.20
MgO (%)	1.13
Na_2O (%)	1.32
Fe_2O_3 (%)	1.34
TiO_2 (%)	1.20
MnO (%)	0.02
LOI (%)	10.68
Total (%)	99.63
Ba (ppm)	1560

CYANIDATION TEST REPORT

Project no : 93-016
Test no : CF3C2a
Date : 20-May-93

SAMPLE DESCRIPTION
F3 Pyrite Concentrates

INITIAL CONDITIONS

Feed solids weight :	48.15 g	NaCN :	5 g/L
Water weight :	112.4 g	pH :	10.5
% solids :	30.0 %	Carbon :	1.84 g

TEST PROGRESS

Time (hr)	NaCN [g/L]	NaCN (g)	Ca(OH)2 [g/L]	Ca(OH)2 (mL)	Initial pH	Final pH	dO2 [mg/L]
0.0		0.56		3.3	5.4	10.9	
24.0	2.60		0.00			10.8	

Reagent consumption

NaCN = 5.6 kg/t
Lime Ca(OH)2 = 9.7 kg/t

Reducing power

RP = 200 mL of N/10 KMnO4 per litre solution

CYANIDATION TEST REPORT

Project no : 93-016
Test no : CF3C2a

Date : 20-May-93

TEST DATA

	Time (hr)	Volume (mL)	Weight (g)	Au (mg/L)	Ag (mg/L)	Au (g/t)	Ag (g/t)
Solution	24	124.63		0.04	0.05		
Carbon			1.84				
Residue			47.39			1.16	2.74

DISTRIBUTION

	Total Au (mg)	Total Ag (mg)	Au (%)	Ag (%)
Solution	0.00	0.01	1.7	3.7
Carbon	0.24	0.03	80.0	18.5
Residue	0.06	0.13	18.3	77.7
Total	0.30	0.17	100.0	100.0

HEAD ASSAY

	Au (g/t)	Ag (g/t)	Au (oz/ton)	Ag (oz/ton)
Measured	6.71		0.20	
Calculated	6.36	3.53	0.19	0.10

CYANIDATION TEST REPORT

Project no : 93-016 **Date :** 20-May-93
Test no : CF3C2b

SAMPLE DESCRIPTION
F3 Pyrite Concentrates (Pulverized)

INITIAL CONDITIONS

Feed solids weight :	46.8 g	NaCN :	5 g/L
Water weight :	106 g	pH :	10.5
% solids :	30.6 %	Carbon :	1.74 g

TEST PROGRESS

Time (hr)	NaCN [g/L]	NaCN (g)	Ca(OH)2 [g/L]	Ca(OH)2 (mL)	Initial pH	Final pH	dO2 [mg/L]
0.0		0.53		3.0	5.7	10.8	
24.0	4.30		0.00			10.7	

Reagent consumption

NaCN = 1.6 kg/t
Lime Ca(OH)2 = 9.2 kg/t

Reducing power

RP = 150 mL of N/10 KMnO4 per litre solution

CYANIDATION TEST REPORT

Project no : 93-016
Test no : CF3C2b

Date : 20-May-93

TEST DATA

	Time (hr)	Volume (mL)	Weight (g)	Au (mg/L)	Ag (mg/L)	Au (g/t)	Ag (g/t)
Solution	24	120.43		0.02	0.05		
Carbon			1.74				
Residue			46.02			0.75	1.92

DISTRIBUTION

	Total Au (mg)	Total Ag (mg)	Au (%)	Ag (%)
Solution	0.00	0.01	0.8	4.8
Carbon	0.27	0.03	88.0	24.1
Residue	0.03	0.09	11.2	71.0
Total	0.31	0.12	100.0	100.0

HEAD ASSAY

	Au (g/t)	Ag (g/t)	Au (oz/ton)	Ag (oz/ton)
Measured	6.71		0.20	
Calculated	6.67	2.70	0.19	0.08

TESTWORK PROCEDURE

Test No: 93-016 F3B

Date: 19-May-93

Purpose; Flotation of Sericite from acid leached F3 tailings

Sample I.D. : F3 Rougher Flotation Tails

STAGE	TIME (Minutes)	ADDITIONS	
		g/tonne	REAGENT
Acid leach	1290	100601	Conc. H2SO4 pH = 7.0 to pH = 3.0
Flotation			
Condition	15		@ 40% solids
		750	Armac T
		183	Conc. H2SO4 pH = 3.5 to pH = 3.0
Rougher float	5		@ 15% solids
Condition	5	1500	Armac T
Scavenger float	5		

LEACH TEST REPORT

Objective: To dissolve carbonate minerals from the F3 flotation tailings product using H_2SO_4 which is necessary prior to floating the sericite.

Procedure:

Sample:	Test F3 flotation tailings
Vessel:	Overhead stirred tank
Weight:	676 g
Solution:	1 L H_2O with 12 N H_2SO_4 to maintain pH at 3.0
Leach Time:	24 hours
Temperature:	Ambient

Acid Consumption:

Acid:	12 N H_2SO_4
Volume:	110 mL
Consumption:	100.6 kg/t
Temperature:	Ambient

Table of Weights:

Product	Weight (g)	Weight (%)
F3 Flotation Tails	676	100
H_2SO_4 Leach Residue	716	106

MATERIAL BALANCE

Project no : 93-016
Test no : F3b

Date : May 19, 1993

Sample description : H2SO4 leached F3 Rougher Tails Residué

Products	Weight		Assays	% Distribution
	(g)	(%)	SO4 (%)	
Rougher Conc.	46.1	6.4	3.98	
Scavenger Conc.	114.2	15.9		
Final Tails	555.9	77.6		
Insol. Rougher Conc.			1.81	
Calculated head	716.2	100.0		
Assay head				

XRF - ANALYSIS

Sample: Sericite flotation products from flotation test F3b.

Parameter	Rougher	Scavenger
CaO (%)	5.04	9.54
K_2O (%)	5.74	4.14
P_2O_5 (%)	0.11	0.13
SiO_2 (%)	39.93	38.14
Al_2O_3 (%)	22.51	17.27
MgO (%)	3.06	2.57
Na_2O (%)	0.71	1.02
Fe_2O_3 (%)	4.34	4.14
TiO_2 (%)	0.28	0.31
MnO (%)	0.11	0.09
LOI (%)	17.95	21.06
Total (%)	99.78	98.44
Ba (ppm)	1559	1184

TEST F4 RESULTS

TEST WORK PROCEDURE

Test No: 93-016 F4 **Date:** 04-Jun-93

Purpose: To reduce the amount of quartz in the graphite concentrate by addition of sodium silicate to the primary grind and the rougher flotation stage.

STAGE	TIME (Minutes)	ADDITIONS	
		g/tonne	REAGENT
Grind (2kg)	14	1000	Na2SiO3
Condition (1Kg) at 15% solids	5	930 250	Lime pH = 8.6 to pH = 10.5 Na2SiO3
Rougher float	25	625	DF 250 (stage addition)
Cleaner float	15	0	DF 250

SIZE DISTRIBUTION

Project no : 93-016
Sample id : F4 Head

Date : June 4, 1993

Size Fraction (Tyler mesh)	Individual Percentage Retained	Cumulative Percentage Passing
65	0.2	99.8
100	3,4	96.3
150	11.6	84.8
200	15.4	69.4
325	16.6	52.7
400	3.6	49.1
Undersize	49.1	

MATERIAL BALANCE

Project no : 93-016
Test no : F4

Date : June 14, 1993

Sample description : Ground to 69.4% passing 200.mesh

Products	**Weight**		**Assays**	**% Distribution**
	(g)	**(%)**		
Cleaner Conc	37.7	3.7		
Cleaner Tails	59.6	5.9		
Total Rougher Conc	**97.3**	**9.6**		
Final Tails	916.8	90.4		
Calculated head	1014.1	100.0		
Assay head				

LEACH TEST REPORT

Objective: To dissolve carbonate and sulphide minerals from the F4 graphite/sericite concentrate using HCl and HNO_3, respectively.

Procedure:

HCl Leach

Sample: Test F4 graphite/sericite product
Vessel: Beaker with magnetic stirrer
Weight: 50.0 g
Solution: 100 mL of 10% HCl solution
Leach Time: 4 hours
Temperature: Ambient

HNO_3 Leach

Sample: Dried F4 HCl leach residue
Vessel: Beaker with magnetic stirrer
Weight: 41.1 g
Solution: 100 mL distilled water 30 g HNO_3 per 100 g solids
Leach Time: 2.5 hours
Temperature: Ambient

Table of Weights:

Product	Weight (g)	Weight (%)
Feed	50.0	100.0
HCL Leach Residue	41.1	82.2
HNO_3 Leach Residue	38.1	76.2

XRF - ANALYSIS

Sample: Residue from Test F4 HCl & HNO_3 leach of graphite/sericite product.

Parameter	Level
CaO (%)	0.09
K_2O (%)	5.11
P_2O_5 (%)	0.03
SiO_2 (%)	52.36
Al_2O_3 (%)	21.00
MgO (%)	1.40
Na_2O (%)	1.23
Fe_2O_3 (%)	1.82
TiO_2 (%)	1.69
MnO (%)	0.01
LOI (%)	14.33
Total (%)	99.07
Ba (ppm)	1438

XRF - ANALYSIS

Sample: Graphite/sericite cleaner flotation tails from test F4.

Parameter	Level
CaO (%)	2.78
K_2O (%)	5.26
P_2O_5 (%)	0.29
SiO_2 (%)	48.57
Al_2O_3 (%)	21.82
MgO (%)	2.94
Na_2O (%)	1.14
Fe_2O_3 (%)	5.23
TiO_2 (%)	0.62
MnO (%)	0.06
LOI (%)	10.84
Total (%)	99.55
Ba (ppm)	1466