TEESHIN RESOURCES LTD.

DOME MOUNTAIN PROJECT

M.P.D. CONSULTANTS INC.

March 1988
(Amended)

1987 SUMMARY REPORT OF THE DOME MOUNTAIN PROJECT

DOME MOUNTAIN

OMINECA MINING DIVISION

BRITISH COLUMBIA

NTS 93L/10E, 15E

Latitude 54° 44.5' North
Longitude 126° 37.0' West

January - December 1987

For

TEESHIN RESOURCES LTD.

581 Argus Road

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K. H. Schippers

March 1988

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1.0 GENERAL

1.1 INTRODUCTION

In May 1987, Teeshin Resources Ltd. commenced an underground exploration program on the Boulder Zone location on its Dome Mountain Property. Preliminary feasibility studies indicated that the Boulder combined with the Argillite Zone was economically viable at todays gold prices. These studies were based on diamond drill information. To bring the property to a final production decision a comparison of actual and diamond drill ore reserves had to be made both for continuity and grade. Bulk metallurgical testing was necessary and ground conditions needed to be examined for mining purposes.

The 1370 level on the Boulder Zone was chosen for a number of reasons.

1. The 1370 elevation occurred where drill hole intersections were at their greatest widths and best gold grades. These intersections proved critical in the initial diamond drill ore reserves.

2. The Boulder Zone represented approximately 70% of the total mineable tons.

3. The Boulder Zone was more assessable then the Argillite Zone.

The following report summarizes the 1370 level underground exploration program undertaken between May and September 1987 on the Boulder Zone.

1.2 LOCATION AND ACCESS

The Dome Mountain Property is located approximately 35 km due east of Smithers, British Columbia. Smither, with a population of about 5,000 residents, is a government centre and natural resources oriented community with principal industries of wood products and mining. Good school, hospital, housing, shipping and recreational facilities are available to complement a stable workforce. The town is situated on the main CNR rail line and Highway 16, connecting Prince George and Prince Rupert. PWA serves the community with daily flights to Vancouver.

The property is presently accessed via the Babine Lake Road for 35 km, then 18 km south on the Chapman Lake Road to the property access road. The total distance from Smithers to the site is 57 km and takes about 1 hour and 15 minutes driving time.

1.3 TOPOGRAPHY

The Boulder Zone deposit lies on the east flank of Dome Mountain, below the tree line at a mean elevation of about 1,400 m above sea level. The topography falls away to the east with a 65 m drop in elevation along the strike length of the ore zone.

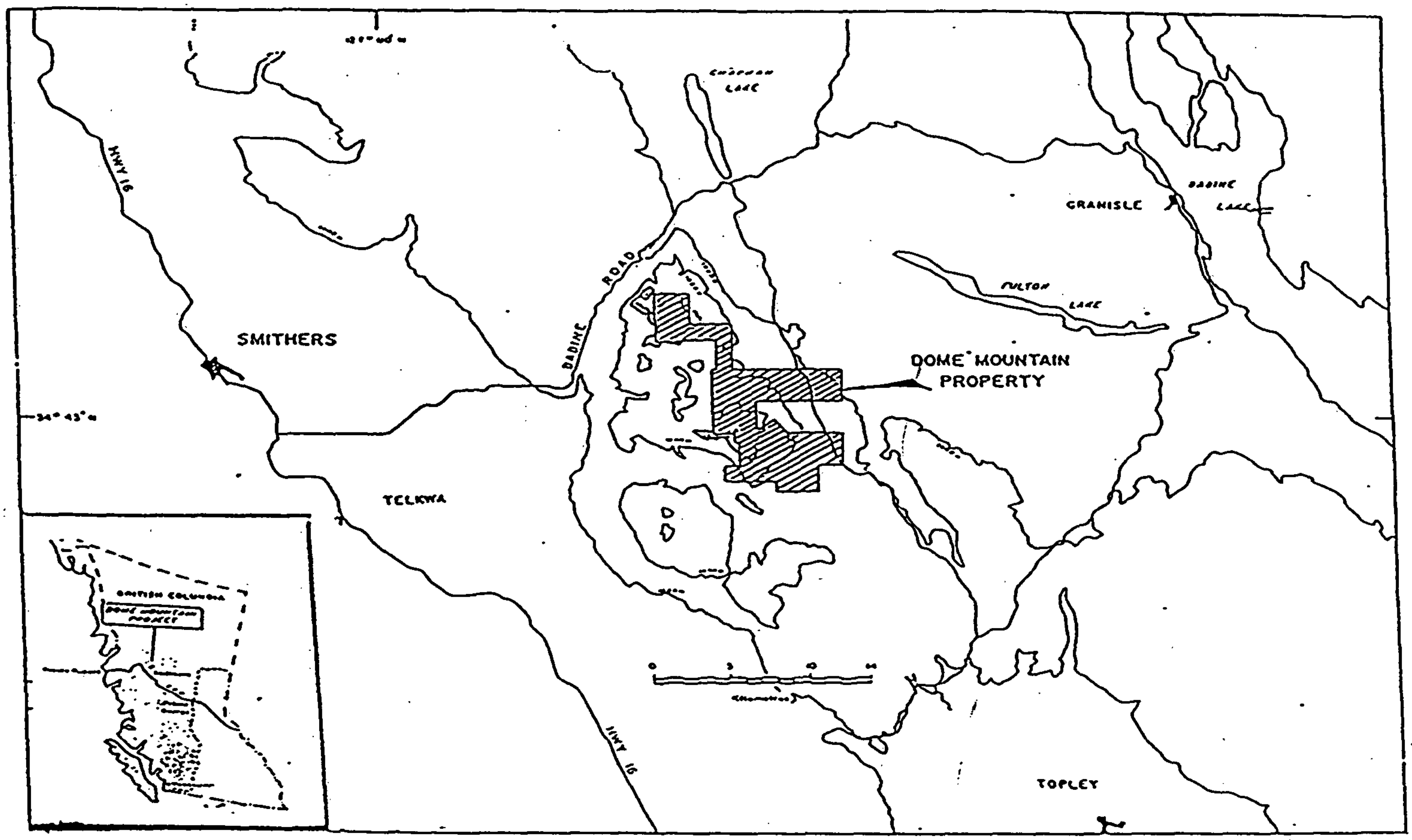
The area is drained by Boulder Creek, which flows parallel to the mineralized zone about 50 m. to the north and about 5 m. to 10 m. lower in elevation. Vegetation in the area consists mostly of balsam, fir and spruce. Overburden covering the ore zone is mostly sandy clays up to 3 m. in depth at the west end and increasing to 25 m. on the steeper slope to the east.

1.4 CLIMATE

The Smithers area receives approximately 102 cm. precipitation annually, with 2 m. to 3 m. of snow. Accumulated snow loads of up to 2 m. can be expected. The area is generally snow free from June to mid October, with temperatures ranging from a low of -40^{0} C. in July/August.

1.5 PROPERTY DESCRIPTION

The property covers an area of 5,354.7 hectares (13225 acres) made up of 65 claims containing 237 units.



LOCATION MAP FIG. 1

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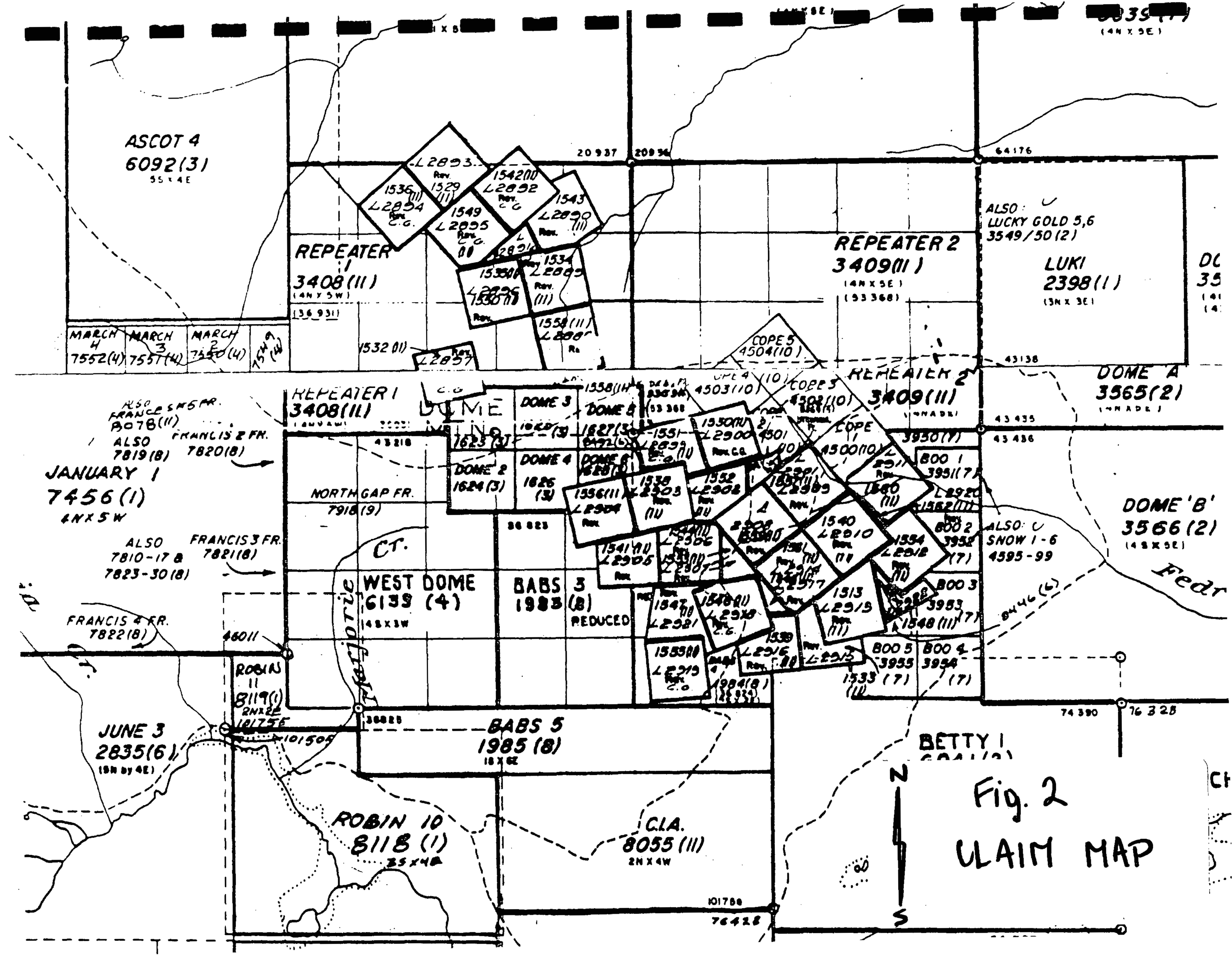


Fig. 2
CLAIM MAP

1.6 DOME MOUNTAIN CLAIM INVENTORY

Claim Name	Claim Type	No. of Units	Area in Hectares
L'Orsa Option			
Byron 1	MG	14	350.0
Byron 2	MG	12	300.0
Emily	TP	1	20.9
Harold	TP	1	20.9
Tony	MG	16	400.0
		44	1091.8
L'Orsa et al. Option			
Betty 1	MG	20	500.0
Boo Fraction	FR	1	10.5
Boo 1	TP	1	20.9
Boo 2	TP	1	20.9
Boo 3	TP	1	20.9
Boo 4	TP	1	20.9
Boo 5	TP	1	20.9
Cope 1	TP	1	20.9
Cope 2	TP	1	20.9
Cope 3	TP	1	20.9
Cope 4	TP	1	20.9
Cope 5	TP	1	20.9
No. 2	RC	1	20.9
No. 3	RC	1	20.9
No. 6	RC	1	20.9
Whistler	RC	1	20.9
		35	803.1
Reako Property Option			
Bert I	MG	20	500.0
Bert II	MG	20	500.0
Dome B	MG	20	500.0
Mat 1	MG	20	500.0
Repeater 1	MG	20	500.0
		100	2500.0
McIntyre Mines Option			
Bertha Fraction	RC	1	5.7
Elk	RC	1	12.5
Gem	RC	1	20.8
New York	RC	1	19.0
Pioneer	RC	1	20.5
Porcupine	RC	1	16.8
Trail	RC	1	20.9
		7	116.2

Warren Option			
Dome 1	TP	1	20.9
Dome 2	TP	1	20.9
Dome 3	TP	1	20.9
Dome 4	TP	1	20.9
Dome 5	TP	1	20.9
Dome 6	TP	1	20.9
Hawk	RC	1	20.9
No. 1	RC	1	20.9
No. 4	RC	1	20.9
Snowdrop	RC	1	20.9
Wallace	RC	1	20.8
Wallace Fraction	RC	1	0.2
		12	230.0
Silver Standard Option			
Babs 3	MG	8	150.0
Babs 4	MG	8	100.0
Babs 5	MG	6	100.0
Dome	RC	1	20.9
Eagle	RC	1	20.9
Eagle Fraction	RC	1	5.0
Freda	RC	1	19.9
Grizzly	RC	1	18.8
Hercules	RC	1	20.9
Josie	RC	1	20.9
No. 5	RC	1	20.3
Ptarmigan	RC	1	20.9
Raven	RC	1	17.8
Telkwa	RC	1	12.7
Tom Fraction	RC	1	7.5
Trail Fraction	RC	1	16.2
Triangle Fraction	RC	1	5.0
Vancouver	RC	1	15.1
Victoria Fraction	RC	1	2.6
Whistler Fraction	RC	1	18.2
		39	613.6
GRAND TOTAL	65	237	5354.7

Dome Mountain Claims Groups - Forks Group

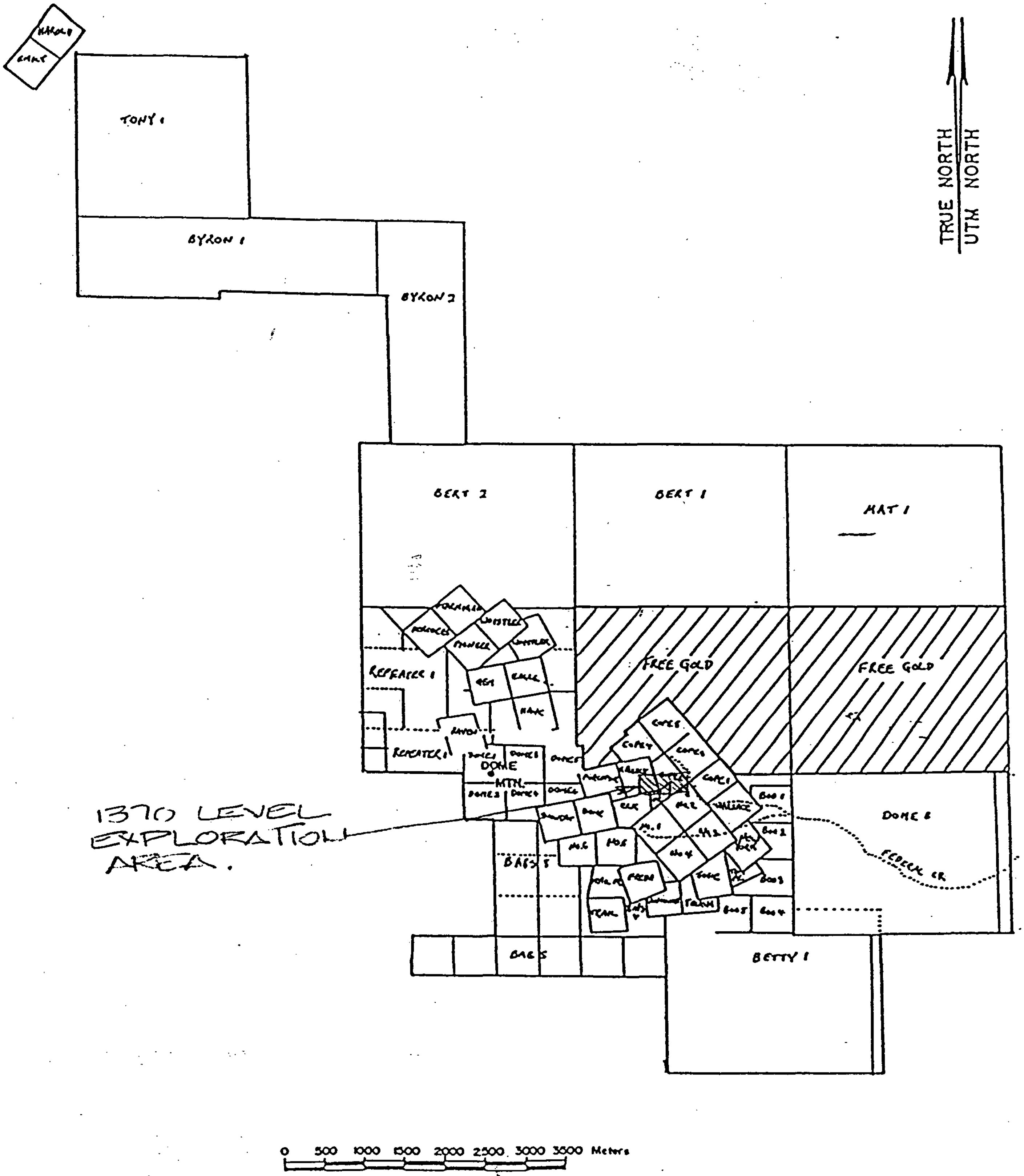
Claim Name		# of Units	Record Number	Mo. of Record	Area (hectares)
Raven	L2897	1	1532	Nov.	17.80
Snowdrop	L2904	1	1556	Nov.	20.90
No. 6	L2905	1	1541	Nov.	20.90
No. 2	L2909	1	1557	Nov.	20.90
No. 3	L2910	1	1540	Nov.	20.90
Wallace	L2911	1	1560	Nov.	20.80
New York	L2912	1	1554	Nov.	19.00
Josie	L2913	1	1531	Nov.	20.90
Telkwa	L2915	1	1533	Nov.	12.70
Vancouver	L2916	1	1539	Nov.	15.10
Victoria Fr.	L2917	1	1545	Nov.	2.60
Freda	L2918	1	1546	Nov.	19.90
Trail	L2919	1	1555	Nov.	20.90
Wallace Fr.	L2920	1	1562	Nov.	0.20
Trail Fr.	L2921	1	1547	Nov.	16.20
Tom Fr.	L2922	1	1548	Nov.	7.50
Dome 1		1	1623	Mar.	20.90
Dome 2		1	1624	Mar.	20.90
Dome 3		1	1625	Mar.	20.90
Dome 4		1	1626	Mar.	20.90
Dome 6		1	1628	Mar.	20.90
Babs 3		8	1983	Aug.	200.00
Babs 4		8	1984	Aug.	200.00
Babs 5		6	1985	Aug.	150.00
Dome B		20	3566	Feb.	500.00
Boo Fr.		1	3950	Jul.	10.50
Boo 1		1	3951	Jul.	20.90
Boo 2		1	3952	Jul.	20.90
Boo 3		1	3953	Jul.	20.90
Boo 4		1	3954	Jul.	20.90
Boo 5		1	3955	Jul.	20.90
Cope 1		1	4500	Oct.	20.90
Cope 3		1	4502	Oct.	20.90
Cope 4		1	4503	Oct.	20.90
Cope 5		1	4504	Oct.	20.90
Betty 1		20	6041	Feb.	500.00
		93			2110.30

Dome Mountain Claims Groups - Dome North Group

Claim Name		# of Units	Record Number	Mo. of Record	Area (hectares)
Hawk	L2888	1	1558	Nov.	20.90
Eagle	L2889	1	1534	Nov.	20.90
Whistler Fr.	L2890	1	1543	Nov.	18.20
Eagle Fr.	L2891	1	1535	Nov.	5.00
Whistler	L2892	1	1542	Nov.	20.90
Ptarmigan	L2893	1	1529	Nov.	20.90
Hercules	L2894	1	1536	Nov.	20.90
Pioneer	L2895	1	1549	Nov.	20.50
Gem	L2896	1	1550	Nov.	20.80
Porcupine	L2899	1	1551	Nov.	16.80
Grizzly	L2900	1	1530	Nov.	18.80
Triangle Fr.	L2901	1	1537	Nov.	5.00
Elk	L2902	1	1552	Nov.	12.50
Dome	L2903	1	1538	Nov.	20.90
No. 5	L2906	1	1544	Nov.	20.30
Bertha Fr.	L2907	1	1553	Nov.	5.70
No. 1	L2908	1	1559	Nov.	20.90
No. 4	L2914	1	1561?	Nov.	20.90
Dome 5		1	1627	Mar.	20.90
Repeater 1		20	3408	Nov.	500.00
Mat 1		20	3839	Jul.	500.00
Cope 2		1	4501	Oct.	20.90
Bert I		20	4831	Oct.	500.00
Bert II		20	4832	Oct.	500.00
		100			2352.60

DOME MOUNTAIN

FIG. 3 - INDEX MAP



Scale: 1:50,000

1.7 CLAIMS COVERED BY UNDERGROUND DEVELOPMENT

The 1370 Level Portal was collared on the Cope 2 and the actual underground workings covered the Cope 2 and Grizzly claims, North Dome Grouping.

1.8 PROPERTY HISTORY

The Dome Mountain area first attracted prospectors as early as 1914 when narrow, high grade quartz-sulphide veins were discovered on the mountain. The area staked and actively explored until 1924. The most intensive exploration activity occurred during 1923 and 1924 when the Dome Mountain Mining Company completed underground development work at the Ptarmigan, Jane-Chisholm, Cabin and Forks areas. The underground results were however, disappointing when compared to the surface results and work was discontinued. A detailed account of the early exploration activity on Dome Mountain has been compiled by Myers (1984a).

The Dome Mountain area was explored intermittently until 1984 when Noranda Exploration acquired an option on claims comprising most of the known showings. Linecutting and B horizon soil sampling was undertaken in 1984 (Myers, 1984b) and was followed in 1985 with reconnaissance and detailed geological mapping, trenching and sampling of historical showings and geochemical anomalies, and diamond drilling at the Hawk, Hoopes, Cabin and Forks-9800 areas (Myers 1985, Myers 1986a, Myers 1986b). VLF-EM and magnetometer surveys were also undertaken (Myers, 1986a). During 1985, the Boulder vein and 9800 showing were discovered by trenching of geochemical soil anomalies.

With the discovery of the Boulder vein in 1985 and the Argillite Zone in 1986, the exploration emphasis shifted towards delineating these two zones. The remainder of the property received little attention during 1985-1986 except for three drill holes completed at the Forks showing by Canadian-United Minerals, Inc. late in 1985.

During 1987, a limited, shallow depth exploration program was completed by MPD Consultants Inc. on the Hawk, Cabin-Federal and Jane-Chisholm zones. The drilling confirmed strike and depth extensions of the mineralization at various zones but failed to locate any reserves of current economic potential. Seven drill holes were also completed in the Forks-9800 area.

Drifting along the Boulder commenced in May of 1987 with details of the program outlined in this report.

2.0 GEOLOGY

2.1 REGIONAL AND PROPERTY GEOLOGY

Dome Mountain is situated within the northwest-southeast trending Babine Range located in west-central British Columbia. The Babine Range is a "host of folded and faulted Jurassic and Cretaceous volcanic and sedimentary rocks bounded to the west and east by grabens containing late Cretaceous and younger rocks", (MacIntyre, 1987). The regional geology is described by Tipper and Richards (1986). The following excerpts from MacIntyre (1985) summarized the regional geology and stratigraphy specific to the Dome Mountain area.

Regional Geological Setting

The Dome Mountain area is underlain by subaerial to submarine volcanic, volcaniclastic and sedimentary rocks of the Hazelton Group. The Hazelton Group is an island-arc assemblage that was deposited in the northwest trending Hazelton Trough between Early Jurassic and Middle Jurassic time. Tipper and Richards (1986) divide the Hazelton Group into three major formations in the Smithers map-area (93L). These are the Late Sinemurian to Early Pliensbachian Telkwa Formation, the Early Pliensbachian to Middle Toarcian Nilkitkwa Formation, and the Middle Toarcian to Lower Callovian Smithers Formation.

The Telkwa Formation, which is comprised of subaerial and submarine pyroclastic and flow rocks with lesser intercalated sedimentary rocks, is the thickest and most extensive formation of the Hazelton Group. The mixed subaerial to submarine Babine Shelf facies of the Telkwa Formation, which separates the subaerial Howson facies to the west and the submarine Kotsine facies to the east, underlies the Babine Range (Tipper and Richards, 1976).

The Nilkitkwa Formation conformably to disconformably overlies the Telkwa Formation. West of Dome Mountain it is comprised of predominantly Toarcian red pyroclastic rocks; to the east it includes Early Pleinsbachian to Middle Toarcian marine sedimentary rocks with intercalated rhyolite to basalt flows.

In the Babine Range, the Smithers Formation disconformably overlies the Nilkitkwa Formation; it is predominantly Bajocian in age. It is comprised of fossiliferous sandstone and siltstone with lesser intercalated felsic tuff.

Dome Mountain Geology

The core of Dome Mountain is underlain by a large southwest-verging, southeast-plunging anticlinal structure that has been cut by northeast and northwest-trending high angle faults (Fig. 14a). The oldest rocks are well exposed on the crest of the mountain and a good stratigraphic column (Fig. 14b) has been established on the basis of this section. Seven major map units are recognized. Going up section these are: (1) fragmental volcanic unit (+1000 metres?); (2) red volcaniclastic-green flow unit (150-200 metres); (3) volcanic wacke-conglomerate-felsic tuff unit (20-50 metres); (4) rusty argillite or shale unit (50-100 metres); (5) dark gray siltstone unit (250-300 metres); (6) thin-bedded limestone-siltstone-wacke unit (50-100 metres); and (7) greenish gray massive volcaniclastic unit (+500 metres). The ages of these units and their correlations with Hazelton Group formations are not well established.

Several small plugs or dykes of diabase or diorite intrude the Hazelton Group on Dome Mountain; a stock of quartz porphyry or quartz monzonite is exposed near the Freegold showing.

2.2 PRELIMINARY STRATIGRAPHIC COLUMN DOME MOUNTAIN

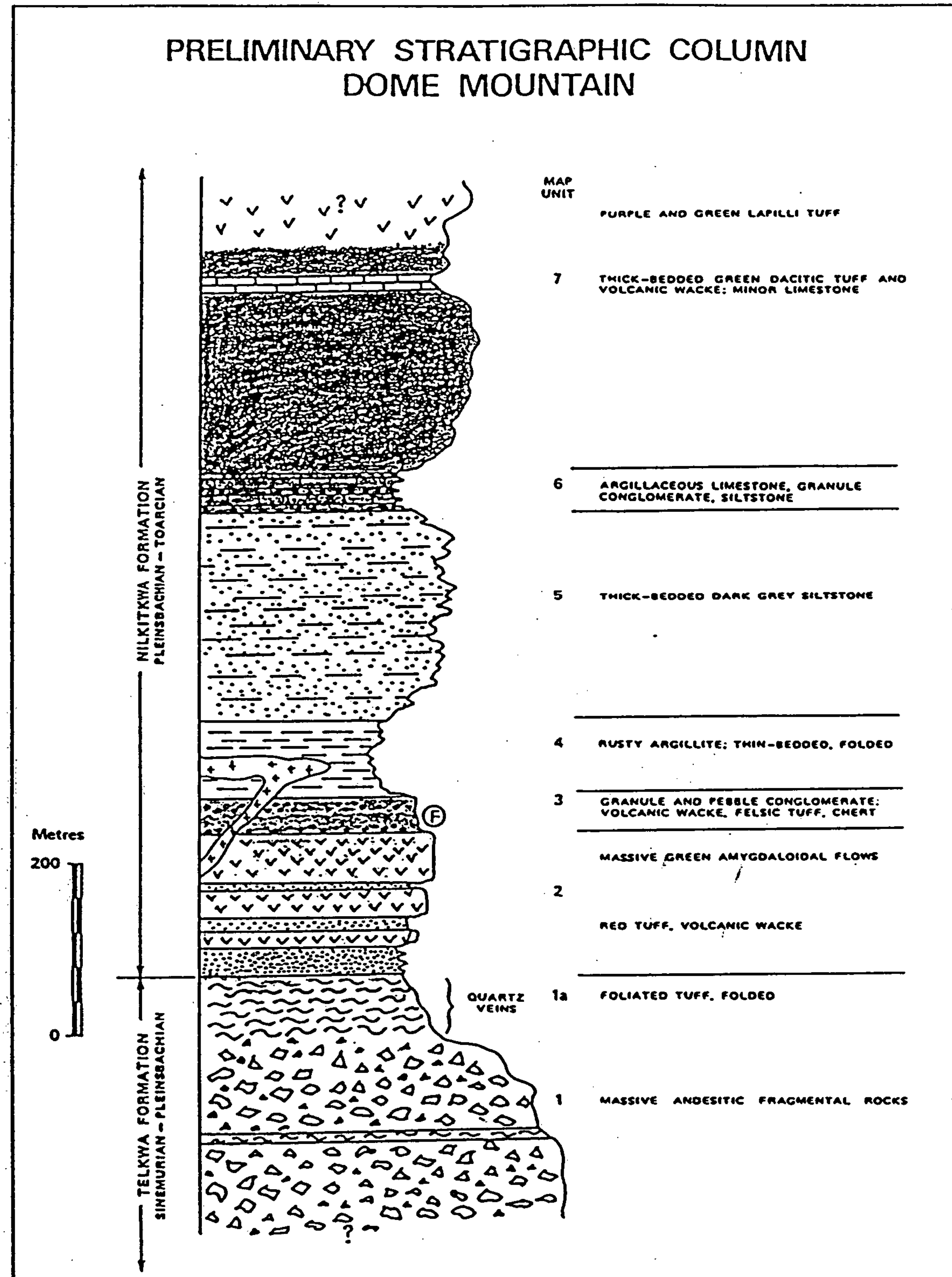


Figure 4: Preliminary stratigraphic column, Dome Mountain gold camp.

(From McIntyre, 1985)

2.3 BOULDER-FEDERAL SHEAR ZONE

The Boulder-Federal Shear Zone lies central to the property, striking east-west for a known distance of 1000 meters. From east to west it includes four mineralized zones: the Argillite Zone, Boulder Zone, Cabin Vein and Federal Vein. At the eastern end, the structure turns in a southeastern direction and veins appear to splay out into several smaller quartz stringers marking the eastern extent. On the western end, the full extent has not yet been determined. To date, this structure is the largest and most explored; it has the best economic potential disclosed on the property. Although the structure occurs for more than 1000 metres, economic reserves have only been established in eastern end in the Argillite and Boulder Zones.

Although base metal and precious metals have only low values along much of the structure, pyrite and alteration minerals are present for its entire length. The widest portion of the shear zone occurs just west of the Cabin Vein, while the area falling between the Cabin and the western end of the Boulder Zone is quite narrow (50 cm.). The structure dips from 45 to 60 degrees south on the east and is sub-vertical dipping north on the west. Although the quartz veins hosted within the shear zone are characteristically erratic in orientation and continuity, the Boulder-Federal Shear Zone maintains uniformity in strike direction. To date, no other structure having a similar east-west orientation has been located on the property. The various zones in the structure differ somewhat in character, grade and economic potential.

2.4 BOULDER ZONE

The Boulder Zone was discovered by Noranda Exploration in late 1985 through trenching. It occurs along the eastern end of the Boulder-Federal Shear Zone with an east-west strike and dips from 45 to 65 degrees south. The Boulder Zone appears to plunge south-east, with the main zone having a south-westerly rake. The gold-bearing quartz and quartz carbonate vein is segmented as quartz veins, lenticular pods and areas of brecciation. The main shear generally marks the contact between the altered quartz-chlorite hanging wall and the unaltered maroon volcanic andesite and lapilli tuff footwall. Above the shear occur one or more quartz veins enveloped within a bleached zone of intense alteration, averaging seven meters in width. The erratic nature of the quartz veining is clearly exposed in the underground drifting. The width and orientation of a particular vein can change very abruptly by one or more of the following structural controls: folding of the vein; offset or truncation of a vein by post-ore faulting or shearing; thickening of the vein by a series of smaller thrust shears within the vein; or, areas of brecciation where the vein has been fragmented and then re-cemented with quartz.

The Boulder Zone is in fact a bleached zone within one or more highly variable ore-bearing quartz veins or lenses are hosted. Within the boundaries of the bleached zone, the quartz vein may split into two or more shoots or occur as overlapping pods or lenses divided by a section of barren, but highly altered, bleached rock. Associated with the vein in the hanging wall only are smaller quartz vein stringers. Some of these stringers contain economic mineralization and others contain only pyrite mineralization. In areas where thickening or stacking has taken place within the vein, widths up to 5 metres can be achieved with no dilution of grade. In areas where veins widened through folding, brecciation or flooding, grades are often diluted. In the delineation of the Boulder Zone, it is better to outline the bleached zone rather than to attempt to correlate each vein structure.

Alteration features surrounding the Boulder Zone are relatively clear. From surface to the bleached zone on the hanging wall, rock alteration ranges from weak to moderate. The andesite, lapilli tuff and agglomerate units vary in colour from maroon to dark green to light green depending on the degree of alteration. Alteration is a product of extraction and replacement, where the iron content has been leached out and replaced by chlorite, sericite and silica. The bleached zone is characterized by a buff to lime green to white colour and often appears foliated. All evidence of volcanic fragments has been obscured. Disseminated cubic pyrite crystals are commonly present in this unit. On the footwall side of the vein, the andesite, lapilli tuff and agglomerate units are relatively unaltered. Only small quartz-carbonate and carbonate stringers related to the mineralizing hydrothermal activity occur in the maroon to brick red footwall rock. Small 10 to 30 centimetre envelopes of chlorite-silica alteration flooding are visible along some of the stringers.

The Boulder Vein hosts a base and precious metal assemblage which generally occurs in the following order of abundance: pyrite, sphalerite, galena, chalcopyrite, silver and gold. Base metal sulphide minerals often occur as segregated bands of pyrite, sphalerite and galena. Chalcopyrite is most frequently intermixed with the pyrite mineralization. Pyrite can occur from thin banding to massive pods and has a granular to massive texture. If a vein hosts pyrite in cubic or sub-cubic crystal form, other sulfide minerals and precious metals are not associated. Cubic pyrite is also the only sulfide which has flooded out into the wall rock. It is felt that the cubic pyrite was deposited during a separate mineralizing phase.

Gold grades vary from as high as several ounces to lesser amounts, but cannot be seen as visible gold. Microscopically, gold occurs as tiny granules which are loosely bonded along the outer boundaries of the pyrite grains and within minute fractures within the pyrite sulfides. For this reason, liberation of gold in metallurgical testing has resulted in an anticipated high rate of mill recovery. Little is known about the accompanying silver, which grades several ounces. Silver is felt to be directly associated with the presence of galena and/or sphalerite and probably occurs as tetrahedrite. Due to low anticipated recoveries of the silver, it will have limited economic contribution.

3.0 1370 LEVEL UNDERGROUND EXPLORATION PROGRAM

The drifting and raising program was contracted to Vicore Mining, from Vancouver in late March. By early May, the contractor arrived on site and constructed a twenty-man camp and mobilized equipment to the portal-site. The portal construction commenced on May 20th and was timbered and ready by June 2nd.

3.1 DRIFTING AND RAISING

The total length of drifts and crosscuts on the 1370 metre level is 361 metres. Drift measurements are 3 metres wide and 2.4 metres high. Safety bays are on 23 metre centres. Overall advance was 6.9 metres/per day based on two 8 hour shifts per day six days a week. The stability of the hanging wall, footwall rock and the ore zone is very satisfactory. Little reinforcement "back" has been required, and only in two small areas have rock bolts and wire mesh been required to prevent slabbing. Thus far, our underground experience has shown that rock stability and excessive dilation within stopes should not be a problem to the future mining operation.

Raise 1 is 37.5 metres long and reaches surface. Raise 2 is located at the western limit of that 1370 level and does not reach surface. From the raise mining program, it is confirmed the average dip of the orebody is 42 degrees. Raise 1 is completely within the ore zone but weakens in sulfide content over the last 15 metres. Raise 2 follows the ore but loses it in places because of low dip undulations within the ore zone.

3.2 SAMPLING PROCEDURES

Two methods of sampling were done, muck sampling and chip sampling. Muck samples in places were highly diluted because of the drift width in comparison to the vein exposure. The muck samples were taken at three points during the mucking cycle. Two ten pound bags were taken after the second bucket. (3.5 yard loader), two after the seventh or eighth and finally two after the twelfth or thirteenth bucket. Total sample weight averaged 60 lbs. These samples were dried and twelve analyses performed, four for each of the three (two bag) samples. Chip samples were taken across the back and checked against face and wall samples where taken.

The drift was completely mapped geologically and sample results plotted as shown on Plates 2 through 6.

Gold assays from chip samples taken across the vein correlated with diamond drill grades. In no case did drill information vary to a point where it would affect grade or original tonnages on the Boulder Zone.

Muck samples were low in places due to narrow veins and dilution of up to 70%.

3.3 ROCK MECHANICS

The ore in the Boulder Zone occurs as quartz veins within a shear zone. This zone is broken but generally competent with irregular clean fractures. Ore exposures will be stable without support for spans up to 5 metres in most areas. Some stable unsupported exposures up to 3 to 4 metres wide have already been made in the exploration development. A number of continuous gouge-filled structures exist usually striking along the ore. These sometimes form the immediate footwall of the ore and usually are a barrier to mineralization in the footwall. Although structurally weak, they should not significantly affect stability in a cut and fill method. They would form the footwall if exposed and should not affect dilution. The footwall and hanging wall strata are generally very competent and strong presenting no problems for access development or ore passes. No exposure has been developed in the Argillite Zone.

4.0 BOULDER ZONE ORE RESERVES

4.1 ORE RESERVE SUMMARY

Ore reserves have been identified in quartz-carbonate veins in a major shear zone. The Boulder Zone strikes approximately east-west with a strike length of approximately 350 m. (1150 ft.), with vein widths ranging from 0.5 m. to 3.7 m. (1.6 to 12.1 ft.) and an average dip of 45-50^{o} south.

Diluted probable mineable ore reserves at 20% dilution are estimated at 228,179 short tons at .337 o.p.t. Au and 1.973 o.p.t. Ag.

Boulder Zone

Tons	Undiluted Au	(O.P.T.) Ag	Tons Diluted @ 20%	Au	(O.P.T.) Ag
190.149	0.404	2.367	228,179	0.337	1.973
Tonnes	gms/t	gms/t	tonnes	gms/t	gms/t
172.549	13.85	81.15	207,059	11.55	67.65

Exploration to date on the Boulder Zone is as follows:

1) 51 diamond drill holes on the Boulder Zone along a strike length of 350 m. (1150 ft.);

2) an underground exploration drift on the 1370-level of the Boulder Zone for 361 m. (1185 ft.) including crosscuts, intersecting the ore-body about one-third from the top;

3) two raises totalling 67.5 m. (221.5 ft.).

A longitudinal projection of the ore blocks by section of the mineralized zone is shown on plate: 7.

4.2 METHODOLOGY

Ore reserves have been calculated using conventional cross-section methods. The key assumptions are:

- areas of influence of each drill-hole were extended half-way to the next drill-hole;
- sections were extended half-way to the next section;
- the areas were drawn on each cross-section by contouring the zone then measured;
- a minimum mining width of 1.5 meters was assumed in drawing the areas.
- grades were calculated across drawn widths at intercepts;
- in the 1790 section, the grade from the drift was used to over-ride the grade from drill-hole 86-40;
- in the 1880 section, the grade from the drift (65g/tonne over 2.5 metres) was cut to 34.3 g./tonne; this grade was then averaged with the grade from drill-hole 86-15 (16.9 g./tonne over 2.24 metres) to give a new pierce point.
- specific gravity of 2.785 (11.5 cubic feet per shot ton)
- dilution of 30% was assumed over all tonnage.

4.3 ORE RESERVE DETAILS

TABLE 1 ORE RESERVE DETAILS BOULDER ZONE

HOLE NO.	SECTION	GOLD (g/tonne)	SILVER (g/tonne)	GRADE AREA (sq. m)	INTERVAL (m)	VOLUME (cu. m)	TONNES
T-86-37	1650	5.73	69.6	27.5	20.0	550.0	1531.8
T-86-38	1650	3.60	48.4	46.9	20.0	937.5	2610.9
T-86-33	1690	11.63	93.3	31.5	25.0	787.5	2193.2
T-86-31	1720	4.80	43.6	94.0	30.0	2820.0	7853.7
T-86-22	1750	10.36	89.2	20.0	25.0	500.0	1392.5
T-86-23	1750	10.12	56.3	66.3	25.0	1656.3	4612.7
T-86-20	1770	6.17	49.7	27.0	15.0	405.0	1127.9
RAISE	1780	7.89	48.7	36.0	10.0	360.0	1002.6
T-86-18	1790	40.06	153.7	36.8	15.0	551.3	1535.2
T-86-19	1790	18.56	134.1	51.3	15.0	768.8	2141.0
T-86-40/DRIFT	1790	6.86	12.3	23.8	15.0	356.3	992.2
T-86-40 BOT	1790	5.45	72.0	16.3	15.0	243.8	678.8
T-86-16	1810	29.19	210.9	50.0	15.0	750.0	2088.8
T-86-17	1810	20.92	28.5	56.0	15.0	840.0	2339.4
DRIFT	1820	8.92	27.8	112.5	15.0	1687.5	4699.7
—TB-87-2	1820	7.20	93.3	173.8	15.0	2606.3	7258.4
AP2/T86-4	1840	9.26	36.0	45.0	15.0	675.0	1879.9
T-86-15/DRIFT	1840	25.59	207.5	67.5	15.0	1012.5	2819.8
—TB-87-3	1840	3.43	16.8	56.9	15.0	853.1	2376.0
T-86-41 TOP	1840	3.77	16.8	126.9	15.0	1903.1	5300.2
T-86-41 BOT	1840	17.80	43.2	90.0	15.0	1350.0	3759.8
T-86-5	1850	91.96	172.2	100.0	10.0	1000.0	2785.0
T-86-6	1850	8.88	50.8	152.8	10.0	1527.5	4254.1
—TB-87-1	1850	19.21	80.9	72.0	10.0	720.0	2005.2
DRIFT-RAISE	1860	31.45	179.7	255.5	15.0	3832.5	10673.5
—TB-87-1	1860	19.21	80.9	119.3	15.0	1788.8	4981.7
T-86-7 TOP	1880	9.74	36.0	45.4	20.0	908.0	2528.8
T-86-8 TOP	1880	48.02	56.9	77.5	20.0	1550.0	4316.8
T-86-9	1880	12.90	50.8	67.5	20.0	1350.0	3759.8
T-86-7 BOT	1880	6.04	46.3	77.5	20.0	1550.0	4316.8
T-86-8 BOT	1880	21.03	145.1	33.0	20.0	660.0	1838.1
T-86-11	1900	16.22	46.3	30.0	20.0	600.0	1671.0
T-86-13	1920	19.14	87.5	49.9	15.0	748.1	2083.5
T-86-14	1920	23.91	154.7	134.0	15.0	2010.0	5597.9
T-86-13	1930	19.14	87.5	45.0	20.0	900.0	2506.5
T-86-14	1930	23.91	154.7	72.0	20.0	1440.0	4010.4
T-86-48	1930	13.41	81.3	115.5	20.0	2310.0	6433.4
T-84-24	1960	4.87	9.9	45.0	20.0	900.0	2506.5
T-86-25	1960	5.15	37.0	130.0	20.0	2600.0	7241.0
T-86-24	1970	4.87	9.9	57.0	20.0	1140.0	3174.9
T-85-25	1970	5.15	37.0	52.0	20.0	1040.0	2896.4
T-86-29	2000	28.09	72.6	25.5	25.0	637.5	1775.4
T-86-30	2000	3.50	14.4	20.3	25.0	506.3	1409.9
T-86-51	2000	22.60	194.5	29.3	25.0	731.3	2036.5
*							
TOTAL		16.40	81.0			52063.6	144997.2

4.4 DRILL HOLE CALCULATIONS

TABLE 2

BOULDER ZONE

SECTION	HOLE #	FROM - TO m	INTERCEPT m	GRADE (opt) Ag	Au
1650	86-37	15.40 - 16.40	1.0	3.28	0.226
		16.40 - 17.40	1.0	0.86	0.109
			2.0	2.03	0.167
	86-38	17.85 - 19.85	2.0	1.41	0.105
1690	86-33	21.00 - 22.00	1.00	4.08	0.498
		22.00 - 22.50	0.50	0.01	0.021
			1.50	2.72	0.337
1720	86-31	28.20 - 29.20	1.00	0.99	0.133
		29.20 - 30.20	1.00	0.11	0.012
		30.20 - 31.20	1.00	2.42	0.301
		31.20 - 32.20	1.00	1.56	0.115
			4.00	1.27	0.140
1750	86-22	13.83 - 14.83	1.00	1.73	0.512
		14.83 - 15.83	1.00	3.47	0.092
			2.00	2.60	0.302
	86-23	22.63 - 23.63	1.00	2.66	0.564
		23.63 - 24.63	1.00	0.63	0.085
		24.63 - 25.63	1.00	0.67	1.093
		25.63 - 26.62	0.99	3.39	0.583
		26.62 - 27.62	1.00	0.84	0.152
			4.99	1.64	0.295
1770	86-20	17.15 - 17.68	0.53	0.09	0.018
		17.68 - 18.58	0.90	2.35	0.288
		18.58 - 18.65	0.07	0.10	0.020
			1.50	1.45	0.180
1780	RAISE			1.42	0.230
1790	86-18	25.04 - 26.04	1.00	5.94	1.320
		26.04 - 26.50	0.46	1.68	0.938
		26.50 - 26.54	0.04	-	-
			1.50	4.48	1.168
1790	86-19	34.78 - 35.78	1.00	4.30	0.632
		35.78 - 36.78	1.00	6.93	0.740
		36.78 - 37.78	1.00	0.49	0.201
			3.00	3.91	0.541

DRILL HOLE CALCULATIONS (cont.)

SECTION	HOLE #	FROM - TO m	INTERCEPT m	GRADE (opt) Ag	Au
	86-40	68.10 - 68.60	0.50	0.05	0.010
		68.60 - 69.60	1.00	3.13	0.233
			1.50	2.10	0.159
1790	DRIFT			0.36	0.200
1810	86-16	24.03 - 25.03	1.00	3.64	1.020
		25.03 - 26.03	1.00	4.49	0.944
		26.03 - 26.60	0.57	9.87	0.390
			2.57	6.15	0.851
	86-17	38.59 - 39.59	1.00	1.15	0.875
		39.59 - 39.95	0.36	0.27	0.110
		39.95 - 40.09	0.14	-	-
			1.50	0.83	0.610
1820	DRIFT			0.81	0.260
	87-2		4.00	2.72	0.210
1830	86-15	50.00 - 51.00	1.00	6.43	0.350
		51.00 - 52.00	1.00	3.68	0.178
		52.00 - 52.24	0.24	0.78	0.145
			2.24	4.60	0.492
	DRIFT			9.48	1.900
	Cut to:			7.50	1.000
	Cut Average:			6.05	0.492
	86-41	111.50 - 112.50	1.00	0.23	0.288
		112.50 - 113.50	1.00	0.89	0.267
		113.50 - 114.50	1.00	0.61	0.870
		114.50 - 115.50	1.00	3.29	0.650
			4.00	1.26	0.519
1850	86-5	22.39 - 23.39	1.00	2.68	0.482
		23.40 - 24.40	1.00	2.55	0.469
		24.40 - 25.40	1.00	12.59	9.680
		25.40 - 26.40	1.00	2.24	0.091
			4.00	5.02	2.681
	86-41	105.5 - 106.5	1.00	0.41	0.349
		106.5 - 107.5	1.00	0.61	0.55
		107.5 - 108.5	1.00	0.29	0.216
		108.5 - 109.5	1.00	-	0.012
		109.5 - 110.5	1.00	-	0.013
		110.5 - 111.5	1.00	-	0.028
			6.00	0.49	0.112

DRILL HOLE CALCULATIONS (cont.)

SECTION	HOLE #	FROM - TO m	INTERCEPT m	GRADE (opt) Ag	Au
1840	DRIFT			2.36	0.36
	86-6	42.00 - 43.00	1.00	2.56	0.267
		43.00 - 44.00	1.00	1.14	0.184
		44.00 - 45.00	1.00	0.43	0.103
		45.00 - 46.00	1.00	0.23	0.028
		46.00 - 47.00	1.00	0.22	0.031
		47.00 - 48.56	1.56	3.27	0.698
			6.56	1.48	0.259
	RAISE			8.12	1.480
1860	DRIFT			2.36	0.360
	87-1	82.00 - 82.50	0.50		1.010
		82.50 - 83.00	0.50		0.200
		83.00 - 83.60	0.60		0.130
			2.01	2.36	0.560
1880	86-7 Top	23.78 - 24.85	1.07	1.43	0.298
		24.85 - 25.28	0.43	0.10	0.008
			1.50	1.05	0.284
	Bottom	28.20 - 29.20	1.00	3.52	0.239
		29.20 - 30.20	1.00	0.16	0.022
		30.20 - 30.02	0.42	0.40	0.129
		30.02 - 32.12	1.50	0.96	0.249
			3.92	1.35	0.176
	86-8 Top	38.11 - 38.95	0.84	3.88	6.820
		38.95 - 40.71	1.76	-	-
		40.71 - 41.71	1.00	0.67	0.232
		41.71 - 42.71	1.00	2.99	0.452
		42.71 - 43.24	0.53	3.05	1.450
			5.13	1.66	1.400
	Bottom	52.50 - 53.50	1.00	3.23	0.331
		53.50 - 54.50	1.00	5.22	0.896
			2.00	4.23	0.613
1880	86-9	79.84 - 80.84	1.00	2.21	0.456
		80.84 - 81.36	0.52	1.22	0.513
		81.36 - 81.76	0.40	-	-
			1.92	1.48	0.376
1900	86-11	22.33 - 23.47	1.14	1.77	0.622
		23.47 - 23.83	0.36	0.01	0.003
			1.50	1.35	0.473

DRILL HOLE CALCULATIONS (cont.)

SECTION	HOLE #	FROM - TO m	INTERCEPT m	GRADE (opt) Ag	Au
1920	86-13	28.43 - 28.83	0.40	0.27	0.051
		28.83 - 29.93	1.10	3.69	0.812
			1.50	2.55	0.558
	86-14	44.92 - 45.92	1.00	6.37	0.756
		45.92 - 46.92	1.00	5.91	1.162
		46.92 - 48.01	1.09	1.26	0.213
			3.09	4.43	0.697
	86-48	116.98 - 117.98	1.00	3.89	0.668
		117.98 - 119.03	1.05	0.84	0.114
			2.05	2.37	0.391
1960	86-24	53.73 - 53.99	0.26	1.02	0.550
		53.99 - 55.20	0.21	0.05	0.005
		55.20 - 56.14	0.94	0.16	0.074
		56.14 - 56.33	0.19	0.01	0.001
			1.60	0.29	0.142
	86-25	60.50 - 61.50	1.00	1.03	0.181
		61.50 - 62.50	1.00	2.63	0.139
		62.50 - 63.50	1.00	1.03	0.096
		63.50 - 64.50	1.00	0.13	0.013
		64.50 - 65.50	1.00	1.01	0.304
		65.50 - 66.50	1.00	0.29	0.074
			6.00	1.08	1.150
2000	86-29	35.96 - 37.00	1.04	3.11	1.180
		37.00 - 37.46	0.46	0.05	0.002
			1.50	2.17	0.819
	86-30	34.47 - 35.05	0.58	0.05	0.015
		35.05 - 35.97	0.92	0.65	0.162
			1.50	0.42	0.102
2000	86-51	65.46 - 66.35	1.09	7.79	0.906
		66.55 - 67.55	0.41	0.03	0.004
			1.50	5.67	0.659

APPENDIX 1

A REPORT ON TESTWORK CARRIED OUT ON SAMPLES RECEIVED FROM THE DOME MOUNTAIN PROPERTY, B.C.

OROCON INC.

G.C. Dickson
General Manager

M. Ahmed
Chemist

BOTTLE LEACH CYANIDATION

Initial cyanidation tests were conducted on the samples under the following conditions, 0.5g/L NaCN, 50% solids, 0.5g/L CaO. The samples were ground in a ball mill for varying times to produce grinds of 55%, 70% and 85% minus 200 mesh, prior to cyanidation. The cyanidation was carried out in bottles on rolls for 48 hours.

The results are shown below:

Test No.	Sample No.	Time Hr.	Grind %-200 mesh	Reagent Kg / NaCN	Cnsmd tonne CaO	Gold Extr. %	Calc Head oz/ston	Residue Gold oz/ston
1	A	48	55	1.43	1.13	96.8	0.458	0.0145
2	A	48	55	1.38	1.20	96.8	0.454	0.0145
3	A	48	70	1.44	1.13	98.2	0.450	0.0080
4	A	48	70	1.45	1.15	98.5	0.445	0.0065
5	A	48	85	1.47	1.20	99.2	0.437	0.0035
6	A	48	85	1.46	1.17	99.1	0.436	0.0040
7	F	48	55	2.27	1.47	31.9	0.247	0.1680
8	F	48	55	2.29	1.47	29.8	0.304	0.2140
9	F	48	70	2.32	1.47	26.5	0.295	0.2170
10	F	48	70	2.28	1.49	29.3	0.322	0.2280
11	F	48	85	2.27	1.65	28.8	0.305	0.2170
12	F	48	85	2.27	1.67	28.9	0.305	0.2170
13	E	48	55	1.97	1.07	ND	ND	ND
14	E	48	55	1.98	1.10	ND	ND	ND
15	E	48	70	2.03	1.10	ND	ND	ND
16	E	48	70	2.01	1.10	ND	ND	ND
17	E	48	85	2.02	1.11	ND	ND	ND
18	E	48	85	2.01	1.12	ND	ND	ND
34	A&E	48	85	0.87	0.73	37.4	0.412	0.2580
35	HW	48	85	1.53	1.56	80.4	0.056	0.0110
36	OA	48	85	1.85	1.53	96.9	0.550	0.0170
37	HW&OA	24	85	1.26	0.64	95.5	0.532	0.0240

Due to the graphitic nature of sample E it was decided that a pregnant robbing test should be carried out (see Test-21). The test showed that the graphitic material was capable of reducing gold levels in pregnant solution by 90% in one hour and by 99% in 20 hours.

BOTTLE LEACH C.I.L.

A mixture of the sample A and the graphitic material were leached under C.I.L. conditions at 90% - 200 mesh. The solution conditions were the same as in the previous cyanidation tests.

Test No.	Sample No.	Time Hr.	Grind %-200 mesh	Reagent Kg / NaCN	Cnsmd tonne CaO	Gold Extr. %	Calc Head oz/ston	Residue Gold oz/ston
19	A&E	47	90	2.37	0.63	42.4	0.314	0.1810

FLOTATION

Flotation test were carried out on samples A and F at three different grinds:

55% - 200 mesh
70% - 200 mesh
85% - 200 mesh

Test No.	Sample No.	% -200 mesh	Rougher Conc. Wt %	Au %	Au oz/ston	Rougher Tails Wt %	Au %	Au oz/ston
22	A	55	13.2	67.8	2.23	86.8	32.2	0.160
23	A	70	17.2	80.8	1.94	82.8	19.2	0.096
24	A	85	20.7	58.2	1.18	79.3	41.8	0.220
25	F	55	8.6	44.5	1.71	91.4	55.5	0.200
26	F	70	17.7	72.4	1.35	82.3	27.6	0.110
27	F	85	7.9	56.5	2.29	92.1	43.5	0.150

A further three tests were carried out on sample F after conditioning in copper sulphate:

Test No.	Sample No.	% -200 mesh	Rougher Conc. Wt %	Au %	Au oz/ston	Rougher Tails Wt %	Au %	Au oz/ston
28	F	55	14.4	50.9	1.41	85.6	49.1	0.230
29	F	70	13.5	44.0	1.31	86.5	56.0	0.260
30	F	85	14.1	43.6	1.22	85.9	56.4	0.26

A further floatation test was carried out on a mixture of sample A and 20% sample E to see if the effects of the graphitic material could be reduced. The rougher tails underwent cyanidation.

Test No.	Sample No.	% -200 mesh	Rougher Conc. Wt %	Au %	Au oz/ston	Rougher Tails Wt %	Au %	Au oz/ston
31	A&E	70	8.3	33.2	1.54	91.7	66.8	0.28

The results of the bottle leach are shown below:

Test No.	Sample No.	Time Hr.	Grind %-200 mesh	Reagent Kg / NaCN	Cnsmd tonne CaO	Gold Extr. %	Calc Head oz/ston	Residue Gold oz/ston
31	A&E	48	70	0.94	0.63	15.7	0.296	0.249

SPECIFIC GRAVITY

The specific gravity of the ore in sample A was determined to be 2.71.

SAMPLE PREPARATION

On August 28th, 1987 four 45 gallon drums of material were delivered to Orocon's laboratories.

Three samples were obtained from these drums:

SAMPLE A
SAMPLE F
SAMPLE E

On November 2nd, 1987 a further ten samples of drill core were delivered to Orocon's laboratories.

From these samples two composite were made:

SAMPLE HW
SAMPLE OA

FIRE ASSAY

Sample	Au oz/ston
A	0.4024
B	0.3436
E	0.0066
HW	0.0600
OA	0.5232

OTHER ASSAYS

QUANTITATIVE ANALYSIS

	Sample A	Sample E	Sample F
Antimony ppm	110	30	130
Arsenic ppm	140	570	3600
Copper ppm	3400	120	700
Iron %	5.05	5.20	8.70
Lead ppm	2400	600	2400
Mercury ppm	0.56	0.30	0.44
Sulphur %	4.95	2.18	7.21
Zinc ppm	2700	1600	16000

SEMIQUANTITATIVE ANALYSIS

	Sample A	Sample E	Sample F
Aluminum %	3.23	5.52	2.25
Barium ppm	520	500	220
Beryllium ppm	<1	<1	<1
Bismuth ppm	<10	<10	<10
Calcium %	1.24	4.46	0.55
Cadmium ppm	520	260	2000
Cerium ppm	<10	<10	<10
Chromium ppm	40	75	30
Cobalt ppm	14	17	16
Germanium ppm	<20	<20	<20
Niobium ppm	<10	50	<10
Gallium ppm	<10	<10	<10
Lanthanum ppm	<5	<5	<5
Lithium ppm	<50	<50	<50
Manganese ppm	850	6600	11200
Magnesium ppm	4300	8300	3600
Molybdenum ppm	10	17	7
Nd + Sm ppm	38	<10	170
Nickel ppm	<10	90	<10
Phosphorus ppm	1000	850	400

SEMIQUANTITATIVE ANALYSIS (CON'T)

	Sample A	Sample E	Sample F
Silver ppm	150	33	240
Silicon %	33.2	23.5	26.3
Sodium ppm	1700	7200	1100
Strontium ppm	59	179	33
Tantalum ppm	<100	<100	<100
Titanium ppm	1720	3000	980
Tin ppm	<20	<20	<20
Thorium ppm	<20	<20	<20
Tungsten ppm	<20	<20	2200
Uranium ppm	0.9	<0.2	<0.2
Vanadium ppm	75	190	50
Yttrium ppm	9	27	6
Zirconium ppm	31	64	18

INVENTORY

Sample	Wt/kg
A	225
F	22
E	7
HW	3.40

CONFIRMATORY TESTWORK

Confirmatory testwork was carried out on samples A & E by ORF (Ontario Research Foundation). Results are shown below.

BOTTLE LEACH CYANIDATION

Test No.	Sample No.	Time Hr.	Grind %-200 mesh	Reagent Kg / NaCN	Cnsmd tonne CaO	Gold Extr. %	Calc Head oz/ston	Residue Gold oz/ston
1	A	48	74.9	2.40	2.37	93.9	0.546	0.031
2	A&E	48	74.9	2.57	3.40			
3	F	48	74.9	1.60	3.48	27.3	0.220	0.160
4	E	48	74.9	0.42	1.29			0.001

FLOTATION

SAMPLE A

PRODUCT	Wt %	Au oz/ston	Au %
Cleaner Conc.	11.6	3.010	92.9
Cleaner Tailings	5.3	0.110	1.6
Rougher Tailings	83.1	0.025	5.5
Head (calc)	100.0	0.380	100.0

SAMPLE A&E

PRODUCT	Wt %	Au oz/ston	Au %
Cleaner Conc.	10.4	2.740	95.7
Cleaner Tailings	7.7	0.006	0.2
Rougher Tailings	81.9	0.015	4.1
Head (calc)	100.0	0.300	100.0

DETAILS OF TESTS

Test No. 1

Purpose: To cyanide Sample A at a grind of 55% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 55% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.4 - 11.5

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.20	-	-
1	0.30	0.29	0.01	0.20	-	-
24	0.30	0.11	0.19	0.20	-	-
32	0.55	0.27	0.28	0.40	-	-
48	0.55	0.12	0.43	0.45	0.11	0.34

Reagent Consumption (kg/tonne of dry feed) NaCN: 1.43

CaO: 1.13

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.0145	3.2
Solution	300ml	0.4440	96.8
Head (calc)	-	0.4580	100.0

DETAILS OF TESTS

Test No. 2

Purpose: To cyanide Sample A at a grind of 55% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 55% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.4 - 11.5

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.20	-	-
1	0.30	0.29	0.01	0.20	-	-
24	0.30	0.11	0.19	0.20	-	-
32	0.55	0.27	0.28	0.40	-	-
48	0.55	0.14	0.41	0.45	0.09	0.36

Reagent Consumption (kg/tonne of dry feed)	NaCN:	1.38
	CaO:	1.20

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.0145	1.2
Solution	300ml	0.4390	96.8
Head (calc)	-	0.4540	100.0

DETAILS OF TESTS

Test No. 3

Purpose: To cyanide Sample A at a grind of 70% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 70% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.3 - 11.4

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.20	-	-
1	0.30	0.29	0.01	0.20	-	-
24	0.30	0.10	0.20	0.20	-	-
32	0.55	0.26	0.29	0.40	-	-
48	0.55	0.12	0.43	0.45	0.11	0.34

Reagent Consumption (kg/tonne of dry feed) NaCN: 1.44

CaO: 1.13

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.008	1.8
Solution	300ml	0.442	98.2
Head (calc)	-	0.450	100.0

DETAILS OF TESTS

Test No. 4

Purpose: To cyanide Sample A at a grind of 70% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 70% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.3 - 11.4

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.20	-	-
1	0.30	0.29	0.01	0.20	-	-
24	0.30	0.09	0.21	0.20	-	-
32	0.55	0.27	0.28	0.40	-	-
48	0.55	0.11	0.44	0.45	0.11	0.34

Reagent Consumption (kg/tonne of dry feed) NaCN: 1.45

CaO: 1.15

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.0065	1.5
Solution	300ml	0.4380	98.5
Head (calc)	-	0.4450	100.0

DETAILS OF TESTS

Test No. 5

Purpose: To cyanide Sample A at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 85% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.1 - 11.4

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.20	-	-
1	0.30	0.29	0.01	0.20	-	-
24	0.30	0.10	0.20	0.20	-	-
32	0.55	0.26	0.29	0.40	-	-
48	0.55	0.11	0.44	0.45	0.09	0.36

Reagent Consumption (kg/tonne of dry feed) NaCN: 1.47

CaO: 1.20

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.0035	0.8
Solution	300ml	0.4330	99.2
Head (calc)	-	0.4370	100.0

DETAILS OF TESTS

Test No. 6

Purpose: To cyanide Sample A at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carries out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 85% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.1 - 11.4

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.20	-	-
1	0.30	0.29	0.01	0.20	-	-
24	0.30	0.09	0.21	0.20	-	-
32	0.55	0.26	0.29	0.40	-	-
48	0.55	0.11	0.44	0.45	0.10	0.35

Reagent Consumption (kg/tonne of dry feed) NaCN: 1.46

CaO: 1.17

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.0040	0.9
Solution	300ml	0.4320	99.1
Head (calc)	-	0.4360	100.0

DETAILS OF TESTS

Test No. 7

Purpose: To cyanide Sample F at a grind of 55% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 55% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.2 - 11.1

Reagent Balance:

Time Hr	Sodium Cyanide Added g	Sodium Cyanide Remaining g	Sodium Cyanide Consumed g	Lime Added g	Lime Remaining g	Lime Consumed g
0	0.30	-	-	0.20	-	-
1	0.30	0.14	0.16	0.20	-	-
24	0.55	0.08	0.47	0.25	-	-
32	0.80	0.21	0.59	0.45	-	-
48	0.80	0.12	0.68	0.55	0.11	0.44

Reagent Consumption (kg/tonne of dry feed)	NaCN:	2.27
	CaO:	1.47

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.168	68.1
Solution	300ml	0.079	31.9
Head (calc)	-	0.247	100.0

DETAILS OF TESTS

Test No. 8

Purpose: To cyanide Sample F at a grind of 55% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 55% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.0 - 11.0

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.20	-	-
1	0.30	0.13	0.17	0.20	-	-
24	0.55	0.08	0.47	0.25	-	-
32	0.80	0.21	0.59	0.45	-	-
48	0.80	0.11	0.69	0.55	0.11	0.44

Reagent Consumption (kg/tonne of dry feed) NaCN: 2.29

CaO: 1.47

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.2135	70.2
Solution	300ml	0.0910	29.8
Head (calc)	-	0.3040	100.0

DETAILS OF TESTS

Test No. 9

Purpose: To cyanide Sample F at a grind of 70% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 70% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.1 - 10.9

Reagent Balance:

Time Hr	Sodium Cyanide Added g	Sodium Cyanide Remaining g	Sodium Cyanide Consumed g	Lime Added g	Lime Remaining g	Lime Consumed g
0	0.3	-	-	0.20	-	-
1	0.3	0.09	0.21	0.20	-	-
24	0.6	0.11	0.49	0.25	-	-
32	0.8	0.24	0.56	0.45	-	-
48	0.8	0.11	0.69	0.55	0.11	0.44

Reagent Consumption (kg/tonne of dry feed) NaCN: 2.32

CaO: 1.47

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.217	73.5
Solution	300ml	0.078	26.5
Head (calc)	-	0.295	100.0

DETAILS OF TESTS

Test No. 10

Purpose: To cyanide Sample F at a grind of 70% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 70% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.1 - 10.9

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.3	-	-	0.20	-	-
1	0.3	0.10	0.20	0.20	-	-
24	0.6	0.11	0.49	0.25	-	-
32	0.8	0.24	0.56	0.45	-	-
48	0.8	0.12	0.68	0.55	0.10	0.45

Reagent Consumption (kg/tonne of dry feed) NaCN: 2.28

CaO: 1.49

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.228	70.7
Solution	300ml	0.094	29.3
Head (calc)	-	0.322	100.0

DETAILS OF TESTS

Test No. 11

Purpose: To cyanide Sample F at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 85% minus 200 mesh.

Solution Volume: 300 ml   Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.5 - 10.6

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.2	-	-
1	0.30	0.07	0.01	0.2	-	-
24	0.65	0.21	0.19	0.3	-	-
32	0.80	0.24	0.28	0.5	-	-
48	0.80	0.12	0.43	0.6	0.11	0.49

Reagent Consumption (kg/tonne of dry feed)   NaCN: 2.27

CaO: 1.65

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.217	71.2
Solution	300ml	0.088	28.8
Head (calc)	-	0.305	100.0

DETAILS OF TESTS

Test No. 12

Purpose: To cyanide Sample F at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 85% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.5 - 10.6

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.2	-	-
1	0.30	0.06	0.24	0.2	-	-
24	0.65	0.19	0.46	0.3	-	-
32	0.80	0.24	0.56	0.5	-	-
48	0.80	0.12	0.68	0.6	0.10	0.50

Reagent Consumption (kg/tonne of dry feed)		
	NaCN:	2.27
	CaO:	1.67

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	0.217	71.1
Solution	300ml	0.088	28.9
Head (calc)	-	0.305	100.0

DETAILS OF TESTS

Test No. 13

Purpose: To cyanide Sample E at a grind of 55% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 55% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.4 - 11.4

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.3	-	-	0.20	-	-
1	0.3	0.14	0.16	0.20	-	-
24	0.5	0.11	0.39	0.20	-	-
32	0.7	0.24	0.46	0.40	-	-
48	0.7	0.11	0.59	0.45	0.13	0.32

Reagent Consumption (kg/tonne of dry feed) NaCN: 1.97

CaO: 1.07

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	ND	ND
Solution	300ml	ND	ND
Head (calc)	-	ND	ND

DETAILS OF TESTS

Test No. 14

Purpose: To cyanide Sample E at a grind of 55% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 55% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.5 - 11.3

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.3	-	-	0.20	-	-
1	0.3	0.14	0.16	0.20	-	-
24	0.5	0.09	0.41	0.20	-	-
32	0.7	0.23	0.47	0.40	-	-
48	0.7	0.11	0.59	0.45	0.12	0.33

Reagent Consumption (kg/tonne of dry feed)	NaCN:	1.98
	CaO:	1.10

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	ND	ND
Solution	300ml	ND	ND
Head (calc)	-	ND	ND

DETAILS OF TESTS

Test No. 15

Purpose: To cyanide Sample E at a grind of 70% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 70% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.4 - 11.3

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.3	-	-	0.20	-	-
1	0.3	0.13	0.17	0.20	-	-
24	0.5	0.10	0.40	0.20	-	-
32	0.7	0.24	0.40	0.40	-	-
48	0.7	0.09	0.61	0.45	0.12	0.33

Reagent Consumption (kg/tonne of dry feed) NaCN: 2.03

CaO: 1.10

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	ND	ND
Solution	300ml	ND	ND
Head (calc)	-	ND	ND

DETAILS OF TESTS

Test No. 16

Purpose: To cyanide Sample E at a grind of 70% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 70% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.4 - 11.2

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.3	-	-	0.20	-	-
1	0.3	0.14	0.16	0.20	-	-
24	0.5	0.09	0.41	0.20	-	-
32	0.7	0.23	0.47	0.40	-	-
48	0.7	0.10	0.60	0.45	0.12	0.33

Reagent Consumption (kg/tonne of dry feed) NaCN: 2.01

CaO: 1.10

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	ND	ND
Solution	300ml	ND	ND
Head (calc)	-	ND	ND

DETAILS OF TESTS

Test No. 17

Purpose: To cyanide Sample E at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 85% minus 200 mesh.

Solution Volume: 300 ml Pulp Density 50% solids.

Solution composition: 0.5 g/L NaCN, 0.5g/L CaO

pH Range: 9.3 - 11.3

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.3	-	-	0.20	-	-
1	0.3	0.12	0.18	0.20	-	-
24	0.5	0.09	0.41	0.20	-	-
32	0.7	0.24	0.46	0.40	-	-
48	0.7	0.09	0.61	0.45	0.12	0.33

Reagent Consumption (kg/tonne of dry feed) NaCN: 2.02

CaO: 1.11

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300g	ND	ND
Solution	300ml	ND	ND
Head (calc)	-	ND	ND

DETAILS OF TESTS

Test No. 18

Purpose: To cyanide Sample E at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of ph and cyanide concentration were made as required.

Feed: 300g of the sample was ground to 85% minus 200 mesh.

Solution Volume: 300ml Pulp Density 50% solids.

Solution Composition: 0.5 g/L NaCn, 0.5 g/L Ca0

pH Range: 9.3 - - 11.2

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.3	-	-	0.20	-	-
0	0.3	-	-	0.20	-	-
0	0.3	-	-	0.2	-	-
1	0.3	0.13	0.17	0.2	-	-
24	0.5	0.10	0.40	0.2	-	-
32	0.7	0.23	0.47	0.4	-	-
48	0.7	0.10	0.60	0.45	0.11	0.34

Reagent Consumption (Kg/tonne of dry feed) NaCN: 2.01
CaO: 1.12

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300 g	ND	ND
Solution	300 ml	ND	ND
Head (calc)	-	ND	ND

DETAILS OF TESTS

Test No. 19

Purpose: To cyanide a mixture of samples A and E at a grind of 90% minus 200 mesh in the presence of actuated carbon.

Procedure: The sample was pulped with a solution of sodium cyanide lime. The cyanidation was carried out on rolls for 47 hours. The pulp was filtered and the residue and carbon assayed. Adjustments of the pit and cyanide were made as required.

Feed: 240g of sample A and 60g of sample E ground to 90% minus 200 mesh.

Solution Volume: 300mL Pulp Density 50% solids.

Solution Composition: 0.5 g/L NaCN

pH Range: 10.7 - - 11.7

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.3	-	-	0.2	-	-
1	0.3	0.09	0.21	0.2	-	-
4	0.55	0.17	0.38	0.2	-	-
8	0.65	0.20	0.45	0.2	-	-
23	0.70	0.12	0.58	0.2	-	-
30	0.80	0.18	0.62	0.2	-	-
47	0.80	0.09	0.71	0.25	0.06	0.19

Reagent Consumption (Kg/tonne of dry feed) NaCN: 2.37
CaO: 0.63

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300 g	0.181	57.6
Solution	300 ml	0.569	0.5
Carbon	6 g	6.569	41.9
Head (calc)	-	0.314	100.0

DETAILS OF TESTS

Test No. 20

Purpose: To determine the specific gravity of sample A.

Procedure: The sample was ground to a fine flour. The specific gravity was determined by the displacement of water caused by a known weight of ore.

Results:

Wt of bottle	41.2 g
Wt of ore and bottle	66.2 g
Wt of ore, bottle and water	107.0 g
Vol of ore, bottle and water	50.0ml

Specific Gravity = 2.71

Test No. 21

Purpose: To investigate the ability of sample E to absorb gold from cyanide solution.

Procedure: The sample was pulped with a pregnant cyanide solution containing gold. The test was carried out on rolls for 20 hours. Samples were taken periodically and analyzed for gold.

Feed: 300g of sample E ground to 80% minus 200 mesh.

Solution Volume: 300 mL Pulp Density 50% solids.

Solution Composition: 0.5 g/L NaCN, 0.5 g/L CaO, 10.2ppm Au

pH Range: 11.5 - - 11.8

Results:

Time Hr	Au ppm	% Au Absorbed
0	10.2	0
1	1.0	90.2
2	0.8	92.2
3	0.5	95.1
20	0.1	99.0

DETAILS OF TESTS

Test No. 22

Purpose: To investigate the flotation response of sample A at a grind of 55% minus 200 mesh.

Procedure: The material was conditioned for one minute then frother was added and the material floated for ten minutes. The rougher concentrate was cleaned for ten minutes without the addition of further reagent or frother.

Feed: 1 Kg of the sample was ground to 55% minus 200 mesh.

Reagent Additions:	0 minutes	0.15	lb/ton AX 350
	10 minutes	0.038	lb/ton AX 350
	frother 2 drops of AF - 88		

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	124.5	1.27
2.	Cleaner Tails	68.5	1.01
3.	Rougher Tails	739.3	0.22

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	128.2	2.23	13.2	67.8
Product 3	845.0	0.16	86.8	32.2

DETAILS OF TESTS

Test No. 23

Purpose: To investigate the flotation response of sample A at a grind of 70% minus 200 mesh.

Procedure: The material was conditioned for one minute then frother was added and the material floated for ten minutes. The rougher concentrate was cleaned for ten minutes without the addition of further reagent or frother.

Feed: 1 Kg of the sample was ground to 70% minus 200 mesh.

Reagent Additions:	0	minutes	0.15	lb/ton AX 350
	10	minutes	0.038	lb/ton AX 350

frother 2 drops of AF - 88

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	82.2	2.64
2.	Cleaner Tails	85.0	1.27
3.	Rougher Tails	805.0	0.096

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	167.2	1.940	17.2	80.8
Product 3	805.0	0.096	82.8	19.2

DETAILS OF TESTS

Test No. 24

Purpose: To investigate the flotation response of sample A at a grind of 85% minus 200 mesh.

Procedure: The material was conditioned for one minute then frother was added and the material floated for ten minutes. The rougher concentrate was cleaned for ten minutes without the addition of further reagent or frother.

Feed: 1 Kg of the sample was ground to 85% minus 200 mesh.

Reagent Additions:				
	0	minutes	0.15	lb/ton AX 350
	10	minutes	0.038	lb/ton AX 350

frother 2 drops of AF - 88

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	124.5	1.27
2.	Cleaner Tails	68.5	1.01
3.	Rougher Tails	739.3	0.22

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	193.0	1.18	20.7	58.2
Product 3	739.3	0.22	79.3	41.8

DETAILS OF TEST

Test No. 25

Purpose: To investigate the flotation response of sample F at a grind of 55% minus 200 mesh.

Procedure: The material was conditioned for one minute then frother was added and the material floated for ten minutes. The rougher concentrate was cleaned for ten minutes without the addition of further reagent or frother.

Feed: 1 Kg of the sample was ground to 55% minus 200 mesh.

Reagent Additions:	0 minutes	0.15	lb/ton AX 350
	10 minutes	0.038	lb/ton AX 350

frother 2 drops of AF - 88

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	44.3	2.77
2.	Cleaner Tails	39.7	0.52
3.	Rougher Tails	888.3	0.20

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	84.0	1.71	8.6	44.5
Product 3	888.3	0.20	91.4	55.5

DETAILS OF TEST

Test No. 26

Purpose: To investigate the flotation response of sample F at a grind of 70% minus 200 mesh.

Procedure: The material was conditioned for one minute then frother was added and the material floated for ten minutes. The rougher concentrate was cleaned for ten minutes without the addition of further reagent or frother.

Feed: 1 Kg of the sample was ground to 70% minus 200 mesh.

Reagent Additions: 0 minutes 0.15 lb/ton AX 350
10 minutes 0.038 lb/ton AX 350

frother 2 drops of AF - 88

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	57.8	2.64
2.	Cleaner Tails	102.9	0.62
3.	Rougher Tails	747.7	0.11

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	160.7	1.35	17.7	72.4
Product 3	747.7	0.11	82.3	27.6

DETAILS OF TEST

Test No. 27

Purpose: To investigate the flotation response of sample F at a grind of 85% minus 200 mesh.

Procedure: The material was conditioned for one minute then frother was added and the material floated for ten minutes. The rougher concentrate was cleaned for ten minutes without the addition of further reagent or frother.

Feed: 1 Kg of the sample was ground to 85% minus 200 mesh.

Reagent Additions:	0 minutes	0.15	lb/ton AX 350
	10 minutes	0.038	lb/ton AX 350
	frother 2 drops of AF - 88		

% Solids: 25; Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	38.4	3.68
2.	Cleaner Tails	35.6	0.80
3.	Rougher Tails	869.8	0.15

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	74.0	2.29	7.9	56.5
Product 3	869.8	0.15	92.1	43.5

DETAILS OF TEST

Test No. 28

Purpose: To investigate the flotation response of sample A at a grind of 55% minus 200 mesh using copper sulphate as a conditioner.

Procedure: The material was conditioned in copper sulphate for one minute. Reagent was added and the material conditioned for a further minute. Frother was added and the material floated for ten minutes at which time further reagent was added. The rougher conc. was cleaned without further reagent addition.

Feed: 1 Kg of the sample A ground to 55% minus 200 mesh.

Reagent Additions:				
	0	minutes	150	g/ton of copper sulphate
	1	minutes	0.30	lb/ton AX 350
	12	minutes	0.076	lb/ton AX 350

frother 5 drops of AF - 88

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	74.7	2.10
2.	Cleaner Tails	58.8	0.53
3.	Rougher Tails	790.5	0.23

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	133.5	1.41	14.1	50.9
Product 3	790.5	0.23	85.6	49.1

DETAILS OF TEST

Test No. 29

Purpose: To investigate the flotation response of sample A at a grind of 70% minus 200 mesh using copper sulphate as a conditioner.

Procedure: The material was conditioned in copper sulphate for one minute. Reagent was added and the material conditioned for a further minute. Frother was added and the material floated for ten minutes at which time further reagent was added. The rougher conc. was cleaned without further reagent addition.

Feed: 1 Kg of the sample A ground to 70% minus 200 mesh.

Reagent Additions:	0	minutes	150	g/ton of copper sulphate
	1	minutes	0.30	lb/ton AX 350
	12	minutes	0.076	lb/ton AX 350

frother 5 drops of AF - 88

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	70.9	1.50
2.	Cleaner Tails	53.2	1.05
3.	Rougher Tails	795.0	0.26

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	124.1	1.31	13.5	44.0
Product 3	795.0	0.26	86.5	56.0

DETAILS OF TEST

Test No. 29

Purpose: To investigate the flotation response of sample A at a grind of 70% minus 200 mesh using copper sulphate as a conditioner.

Procedure: The material was conditioned in copper sulphate for one minute. Reagent was added and the material conditioned for a further minute. Frother was added and the material floated for ten minutes at which time further reagent was added. The rougher conc. was cleaned without further reagent addition.

Feed: 1 Kg of the sample A ground to 70% minus 200 mesh.

Reagent Additions:			
0	minutes	150	g/ton of copper sulphate
1	minutes	0.30	lb/ton AX 350
12	minutes	0.076	lb/ton AX 350

frother 5 drops of AF - 88

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	70.9	1.50
2.	Cleaner Tails	53.2	1.05
3.	Rougher Tails	795.0	0.26

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	124.1	1.31	13.5	44.0
Product 3	795.0	0.26	86.5	56.0

DETAILS OF TEST

Test No. 30

Purpose: To investigate the flotation response of sample A at a grind of 85% minus 200 mesh using copper sulphate as a conditioner.

Procedure: The material was conditioned in copper sulphate for one minute. Reagent was added and the material conditioned for a further minute. Frother was added and the material floated for ten minutes at which time further reagent was added. The rougher conc. was cleaned without further reagent addition.

Feed: 1 Kg of the sample A ground to 85% minus 200 mesh.

Reagent Additions:	0	minutes	150	g/ton of copper sulphate
	1	minutes	0.30	lb/ton AX 350
	12	minutes	0.076	lb/ton AX 350

frother 5 drops of AF - 88

% Solids: 25: Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	79.9	1.68
2.	Cleaner Tails	59.5	0.61
3.	Rougher Tails	850.8	0.26

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	139.4	1.22	14.1	43.6
Product 3	850.8	0.26	85.9	56.4

DETAILS OF TEST

Test No. 31

Purpose: To cyanide the rougher tailings of a mixture of samples A and E ground to 70% minus 200 mesh.

Procedure: The mixed sample was ground to 70% minus 200 mesh. The sample was conditioned for one minute in copper sulphate, a further minute in the presence of AX 350 then frother was added. The material was floated for ten minutes and further reagent added. The tailings were cyanided on bottle rolls for 48 hours. Rougher concentrate was cleaned without reagent additions.

Feed: 1 Kg (800g A + 200g E) of material was ground to 70% minus 200 mesh.

Reagent Additions:	0	minutes	150	g/ton $CuSO_4\ 8H_2O$
	1	minutes	0.15	lb/ton AX 350
	2	minutes	5	drops AF - 88
	12	minutes	0.15	lb/ton AX 350

% Solids: 25; Speed: 1000 rpm

Metallurgical Results:

	Product	Wt g	Au oz/ston
1.	Cleaner Conc.	37.5	2.32
2.	Cleaner Tails	43.8	0.88
3.	Rougher Tails	901.8	0.28

Calculated Grades and Recoveries:

	Wt g	Au oz/ston	Wt %	Au %
Product 1 & 2	81.3	1.54	8.3	33.2
Product 3	901.8	0.28	91.7	66.8

Cyanidation Feed: 300g rougher tailings

pH Range: 10.1 - - 11.6

DETAILS OF TEST

Test #31 CON'T

Reagent Balance:

Time Hr	Sodium Cyanide Added g	Remaining g	Consumed g	Lime Added g	Remaining g	Consumed g
0	0.15	-	-	0.10	-	-
1	0.15	0.144	0.006	0.10	-	-
4	0.15	0.141	0.009	0.10	-	-
12	0.15	0.110	0.040	0.10	-	-
24	0.20	0.050	0.150	0.15	-	-
36	0.30	0.060	0.240	0.20	-	-
48	0.40	0.120	0.280	0.25	0.06	0.19

Reagent Consumption (Kg/tonne of dry feed) NaCN: 0.94
CaO: 0.63

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300 g	0.249	84.2
Solution	300 ml	0.047	15.8
Head (calc)	-	0.296	100.0

DETAILS OF TEST

Test No. 32

Purpose: To determine the distribution of gold in sample A.

Procedure: The sample was ground to 85% minus 200 mesh. The sample was screened using three screens. The fractions were analysed for gold.

Feed: 500g of sample A ground to 85% minus 200 mesh.

Results:

Size Fraction	Wt in Fraction g	Au in Fraction oz/ston
Plus 0.106mm	31.6	0.341
0.074mm - 0.106mm	29.9	0.332
0.045mm - 0.074mm	179.6	0.508
Minus 0.045mm	248.1	0.423

Calculated Results:

	%Wt	Cum %Wt	Au oz/ton	Au %Wt	Au Cum %Wt
Plus 0.106mm	6.45	6.45	0.341	4.97	4.97
0.074mm - 0.106mm	6.11	12.60	0.332	4.58	9.55
0.045mm - 0.074mm	36.70	49.30	0.507	42.00	51.60
Minus 0.045mm	50.70	100.00	0.423	48.40	100.00

DETAILS OF TEST

Test No. 33

Purpose: To determine the distribution of gold in sample F.

Procedure: The sample was ground to 85% minus 200 mesh. The sample was screened using three screens. The fractions were analysed for gold.

Feed: 500g of sample F ground to 85% minus 200 mesh.

Results:

Size Fraction	Wt in Fraction g	Au in Fraction oz/ston
Plus 0.106mm	64.0	0.264
0.074mm - 0.106mm	40.6	0.253
0.045mm - 0.074mm	164.0	0.319
Minus 0.045mm	227.9	0.330

Calculated Results:

	%Wt	Cum %Wt	Au oz/ton	Au %Wt	Au Cum %Wt
Plus 0.106mm	12.8	12.8	0.264	10.9	10.9
0.074mm - 0.106mm	8.2	21.0	0.253	6.6	17.5
0.045mm - 0.074mm	33.0	54.0	0.319	33.8	51.3
Minus 0.045mm	46.0	100.0	0.330	48.7	100.0

DETAILS OF TEST

Test No. 34

Purpose: To cyanide samples A (92%) + E (8%) at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide were made as required.

Feed: 300g of sample was ground to 85% minus 200 mesh.

Solution Volume: 300mL Pulp Density 50% solids.

Solution Composition: 0.5 g/L NaCN, 0.5 g/L CaO

pH Range: 10.5 - - 11.4

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.15	-	-	0.10	-	-
1	0.15	0.14	0.01	0.10	-	-
4	0.15	0.11	0.04	0.10	-	-
24	0.25	0.10	0.15	0.20	-	-
36	0.30	0.12	0.18	0.25	-	-
48	0.35	0.09	0.26	0.30	0.08	0.22

Reagent Consumption (Kg/tonne of dry feed) NaCN: 0.87
CaO: 0.73

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300 g	0.258	62.6
Solution	300 ml	0.154	37.4
Head (calc)	-	0.412	100.0

DETAILS OF TEST

Test No. 35

Purpose: To cyanide samples HW at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide were made as required.

Feed: 300g of sample was ground to 85% minus 200 mesh.

Solution Volume: 300mL Pulp Density 50% solids.

Solution Composition: 0.5 g/L NaCN, 0.5 g/L CaO

pH Range: 10.3 - - 11.1

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.15	-	-	0.15	-	-
1	0.15	0.06	0.09	0.15	-	-
4	0.30	0.15	0.15	0.25	-	-
24	0.45	0.09	0.36	0.45	-	-
36	0.55	0.12	0.43	0.50	-	-
48	0.60	0.14	0.46	0.55	0.08	0.47

Reagent Consumption (Kg/tonne of dry feed) NaCN: 1.53
CaO: 1.56

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300 g	0.011	19.6
Solution	300 ml	0.045	80.4
Head (calc)	-	0.056	100.0

DETAILS OF TEST

Test No. 36

Purpose: To cyanide samples OA at a grind of 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 48 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide were made as required.

Feed: 300g of sample was ground to 85% minus 200 mesh.

Solution Volume: 300mL Pulp Density 50% solids.

Solution Composition: 0.5 g/L NaCN, 0.5 g/L CaO

pH Range: 10.0 - - 11.5

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.15	-	-	0.15	-	-
1	0.15	0.05	0.10	0.15	-	-
4	0.35	0.10	0.25	0.25	-	-
24	0.55	0.02	0.53	0.40	-	-
36	0.65	0.14	0.51	0.50	-	-
48	0.70	0.14	0.56	0.55	0.09	0.46

Reagent Consumption (Kg/tonne of dry feed) NaCN: 1.85
CaO: 1.53

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300 g	0.017	3.1
Solution	300 ml	0.533	96.9
Head (calc)	-	0.550	100.0

DETAILS OF TEST

Test No. 37

Purpose: The cyanide of a mixture of samples OA (92%) and HW (8%) ground to 85% minus 200 mesh.

Procedure: The sample was pulped with a solution of sodium cyanide and lime. The cyanidation was carried out on rolls for 24 hours. The pulp was filtered and the residue washed and assayed. Samples of the leachate were taken on a regular basis. Adjustments of pH and cyanide were made as required.

Feed: 300g of sample was ground to 85% minus 200 mesh.

Solution Volume: 300mL Pulp Density 50% solids.

Solution Composition: 0.5 g/L NaCN, 0.5 g/L CaO

pH Range: 9.5 - - 11.5

Reagent Balance:

Time	Sodium Cyanide			Lime		
Hr	Added g	Remaining g	Consumed g	Added g	Remaining g	Consumed g
0	0.30	-	-	0.15	-	-
1	0.30	0.24	0.06	0.15	-	-
2	0.30	0.22	0.08	0.15	-	-
4	0.30	0.14	0.16	0.20	-	-
8	0.40	0.12	0.28	0.25	-	-
12	0.45	0.14	0.31	0.35	-	-
18	0.45	0.12	0.33	0.40	-	-
24	0.50	0.12	0.38	0.45	0.09	0.36

Reagent Consumption (Kg/tonne of dry feed) NaCN: 1.26
CaO: 0.64

Metallurgical Results:

Product	Amount	Au oz/ston	% Distribution
Residue	300 g	0.2380	4.5
Solution	300 ml	0.5084	95.5
Head (calc)	-	0.5320	100.0

5.0 BOULDER ZONE BULK SAMPLE

Metallurgical testwork was carried out by Orocon Inc. A bulk sample was provided to Orocon out of the stockpile left after the underground program. The sample was randomly taken from this stockpile.

5.1 METALLURGICAL TEST RESULTS

1. The Boulder Zone ore is readily amenable to a cyanidation milling process with recoveries exceeding 95% in ore grade material (rising to 99% in some material).

2. Leach extractions improved with finer grinds over the range tested (to 85% minus 200 - mesh).

3. Reagent consumptions of 1.5 kg./tonne cyanide and 1.2 kg./tonne lime are moderate.

4. Flotation recoveries were good, but this approach was not pursued in view of the excellent results from direct cyanidation.

Orocon would propose to install a Merrill-Crowe gold recovery system at Dome Mountain, because silver levels are quite high. Initial data indicated no particular problem with filtration rates, by confirmation is recommended. A C.I.P. plant capital costs would not be significantly different to the Merrill-Crowe plant.

Details of metallurgical testing are contained in Appendix 1.

6.0 CONCLUSION AND RECOMMENDATIONS

Drifting at the 1370 level on the Boulder Zone was very successful.

1) Gold grade variances between underground sampling and diamond drill indicated is negligible.

2) Revised ore reserves show no change in tonnages and grade.

3) Ore exposures will be stable without support for spans up to 5 metres in most areas.

4) The Boulder Zone ore is readily amenable to a straight cyanidation milling process with recoveries exceeding 95% in ore grade material.

The only one point which should be investigated is the possibility of an oxide cap at the surface contact. The first raise broke through to surface and went through approximately 15 feet of strongly oxidized material. Samples of this material should be sent for metallurgical studies and an accurate volume of this material should be obtained.

7.0 WRITERS CERTIFICATE

KOOS H. SCHIPPERS

ADDRESS: 366 Bay Street, Apt. 2406, Toronto, Ont.
TELEPHONE: (416) 979-0858
EDUCATION: Haileybury School of Mines
Mining Technologist (3 year program)
Graduated June, 1969

South Dakota School of Mines
B.S. Geological Engineer
Graduated June, 1972.

CAREER EXPERIENCE

BELMORAL MINES LTD., Toronto, Ont. Responsible for the planning and implementation of a $15 million underground exploration program at Vedron with ramp access, a $15 million underground exploration program at Broulan with shaft access and a $3 million production and exploration program at Canreos.

April 1987 - March 1988 Vice President, Ontario Operations

ALASKA APOLLO GOLD MINES LIMITED, Vancouver, B.C., Evaluation of the Apollo Gold Mine, a former gold producer (1886-1906) on Unga Island of the Aleutian Island chain, Alaska. Supervision of all the logistical and technical operations required for the dewatering, rehabilitation and sampling of the old mine workings accessed by three shafts and 2 adits. Planning and supervision of a 2,000 feet diamond drilling program utilizing both bulldozer and helicopter support for two drill rigs. The Apollo project is supported by an on-site 20 man remote camp on an island accessed only by boat or light aircraft.

Completed a bulk sample evaluation of a 20 million yard alluvial placer evaluation of a 5 million yard stream channel placer in Arizona. Planned, budgeted and supervised the sampling program completing the program within budget and on schedule. In order to predict production rates and recoveries achievable, designed and constructed a pilot plant capable of processing 7-8 cubic yards of gravel per hour. The plant utilized a small scale production equipment consisting of hopper, control feeder, washing plant, ringer spiral concentrator, a Knelson centrifugal concentrator and sluices.

Evaluate properties submitted to the company and assist the president in acquisition negotiations.

Prepare reports and operating budgets for SEC filing requirements and corporate planning.

Feb.1982 - Feb. 1987 Vice-President Operation
and Project Manager

MINING CORPORATION INC., OF NORANDA MINES, LTD., Engineering control of all United States operations including statistics and costs of current contracts, contract bidding, mine design and feasibility study contracts.

Feb/1981 - Feb/1982 Senior Mining Engineer

NORANDA MINING INC., For the engineering, geology, plant and mine departments. Supervised an extensive rehabilitation of the underground and plant including mechanization of cut and fill mining methods as well as reorganization of geology and engineering departments to functional services departments.

Sept/1987 - Feb/1981 Mine Superintendent

MINING CORPORATION OF CANADA LTD. OF NORANDA MINES, LTD., Administering current contracts, client liaison, job search in district, engineering and feasibility studies for both client and in-house mining projects. Assisted with the feasibility study for the Ontario Project, Park City, Utah which convinced Noranda Mines Ltd. to purchase the property and put in into production.

May/1976 - May/1978 Assistant Manager of Operations -Western Division

PAMOUR PORCUPINE MINES LTD. OF NORANDA MINES, LTD., For engineering, geology and exploration departments for four of the company's five area mines producing 4,000 tons per day. Evaluated cost saving available by ITH drilling and blasting techniques and convinced management to implement new mining methods.

Nov/1973 - Nov/1976 In four years held position of mine geologist, project engineer, shift boss, chief engineer.

NORANDA MINES LTD., All geologist duties of lower area of the mine from the 2,000 level to the 3,600 level including mapping, drill layouts, core logging, interpretation and ore reserves. Also was responsible for surface exploration for an area 50 miles radius from the mine site. Located and mapped the north limb of the Manitouwadge syncline.

June/1972 - Nov/1973 Mine and Exploration Geologist
May/1969 - Sept/1970 Mine and Exploration Geologist

PROFESSIONAL MEMBERSHIP: Society of Mining Engineers of AIME

Koos H. Schippers

K H Schippers

March, 1988

APPENDIX 2 - PERSONNEL

NAME / ADDRESS	POSITION	FIELD WORK
Koos Schippers 366 Bay Street Apt. 2406 Toronto, Ontario	Engineer	May-August,1987
Arnold R. Pollmer RR #2 Site 40 Gabriola Island British Columbia	Engineer	May, July-September,1987
Russell L. Davis 32742 Nicola Place RR #10 Abbotsford, BC	Geologist	July-September,1987
Norman W. Berg 3100 Country Club Dr. Nanaimo, BC	Geologist	August-September,1987
Craig Stewart 108-415 Ginger Dr New Westminister, BC	Geologist	May-September,1987
Herve Hugon 764 Shaw Street Toronto, Ontario	Geologist	August, 1987
Glen Watson c/o A.D.W. Engineering Box 4379 Smithers, BC	Geotechnician	July, 1987
Glenn Forester P.O. Box 91 Smithers, BC	Geotechnician	July-August, 1987
Paul Nessman Smithers, BC	Geotechnician	August-September, 1987

APPENDIX 3 - SUMMARY OF EXPENDITURES

WAGES						
ENGINEERS						
May 01-Sept. 25, 1987	110.00	Days	@	$500.00	/Day	$ 55,000.00
GEOLOGISTS						
July 01-Sept. 30, 1987	70.50	Days	@	$425.00	/Day	29,962.50
Aug. 01-Aug. 31, 1987	17.00	Days	@	$400.00	/Day	6,800.00
Aug. 01-Sept. 30, 1987	45.00	Days	@	$375.00	/Day	16,875.00
May 01-Sept. 30, 1987	114.00	Days	@	$350.00	/Day	39,900.00
						93,537.50
GEOTECHNICIAN						
July 01-Sept. 30, 1987	94.50	Days	@	$225.00	/Day	21,262.50
TOTAL WAGES (451.00 Man Days Total)						$169,800.00

DRIFTING						
VICORE MINING DEVELOPMENTS LTD.						
June 01-15, 1987	323	Feet	@	$415.65	/Foot	$134,254.95
June 16-30, 1987	305	Feet	@	$415.65	/Foot	126,773.25
July 01-15, 1987	282	Feet	@	$415.65	/Foot	117,213.30
July 16-31, 1987	222	Feet	@	$415.65	/Foot	92,274.30
Aug. 01-15, 1987	52	Feet	@	$415.65	/Foot	21,613.80
TOTAL DRIFTING	1,184					$492,129.60

RAISING						
VICORE MINING DEVELOPMENTS LTD.						
* July 16-31, 1987	86.0	Feet	@	$292.50	/Foot	25,155.00
Aug. 01-15, 1987	95.5	Feet	@	$355.00	/Foot	33,902.50
	181.5					
* July 16-31, 1987	40.0	Feet	@	$230.00	/Foot	9,200.00
TOTAL RAISING	221.5					$68,257.50

* July 16 through to August 15, 1987 refers to Raise No. 1 (total 181.5 ft.).
* July 16 through to July 31, 1987 refers to Raise No. 2 (total 40.0 ft.).

SURVEYING						
A.D.W. ENGINEERING						
May 04-17, 1987	157.00	Hrs	@	$16.80	/Hr.	$ 2,637.60
May 18-31, 1987	59.00	Hrs	@	$16.80	/Hr.	991.20
June 01-14, 1987	200.00	Hrs	@	$16.80	/Hr.	3,360.00
June 15-30, 1987	162.50	Hrs	@	$16.80	/Hr.	2,730.00
July 01-15, 1987	83.48	Hrs	@	$16.80	/Hr.	1,402.46
July 16-31, 1987	54.00	Hrs.	@	$16.80	/Hr.	907.20
Aug. 01-15, 1987	68.75	Hrs.	@	$16.80	/Hr.	1,155.00
Aug. 16-31, 1987	108.00	Hrs.	@	$16.80	/Hr.	1,814.40
Sept. 01-15, 1987	117.85	Hrs.	@	$16.80	/Hr.	1,979.88
Sept. 16-30, 1987	58.90	Hrs.	@	$16.80	/Hr.	989.52
TOTAL SURVEYING						$ 17,967.26

ROADS & YARDS CONSTRUCTION	
* WILF'S CONTRACTING	
May 01-31, 1987	$42,532.47
July 01-31, 1987	31,834.27
TOTAL ROADS & YARDS CONSTRUCTION	$ 74,366.74

* Back-up information regarding Times and rates can be supplied on request.

GRAND TOTAL	$822,521.10


1650E
1690E
1720E
1750E
1770
1780
1790
1810
1820E
1840
1850
1860E
1880E
1900E
1920
1930E
1960
1970E
2000E
TOTAL RESERVES
144 997.2 TONNES 16.39 gr. AU per TONNE
80.9 gr. AG per TONNE
GEOLOGICAL BRANCH
ASSESSMENT REPORT
20,974
SURFACE PROFILE ABOVE ORE ZONE
1400 m
1370 m
1335 m
1300 m
LEGEND
TONNES Au gr. per TONNE Ag gr. per TONNE
87-01 A
DIAMOND DRILL HOLE NUMBER
ORE INTERSECTION ELEVATION
UPPER STRUCTURE
87-01 B
LOWER STRUCTURE
0 5 10 20 30 40 50
Metres
PLATE: 7
TEESHIN RESOURCES LTD.
DOME MOUNTAIN PROJECT
BOULDER ZONE
HORIZONTAL PROJECTION OF ORE RESERVES
ON LONGITUDINAL SECTION
SECTION ORIENTED 104°
DATE: DEC. 87
DRAWN BY: H.H.
SCALE: 1:500
FIG:

Teeshin Resources Ltd.

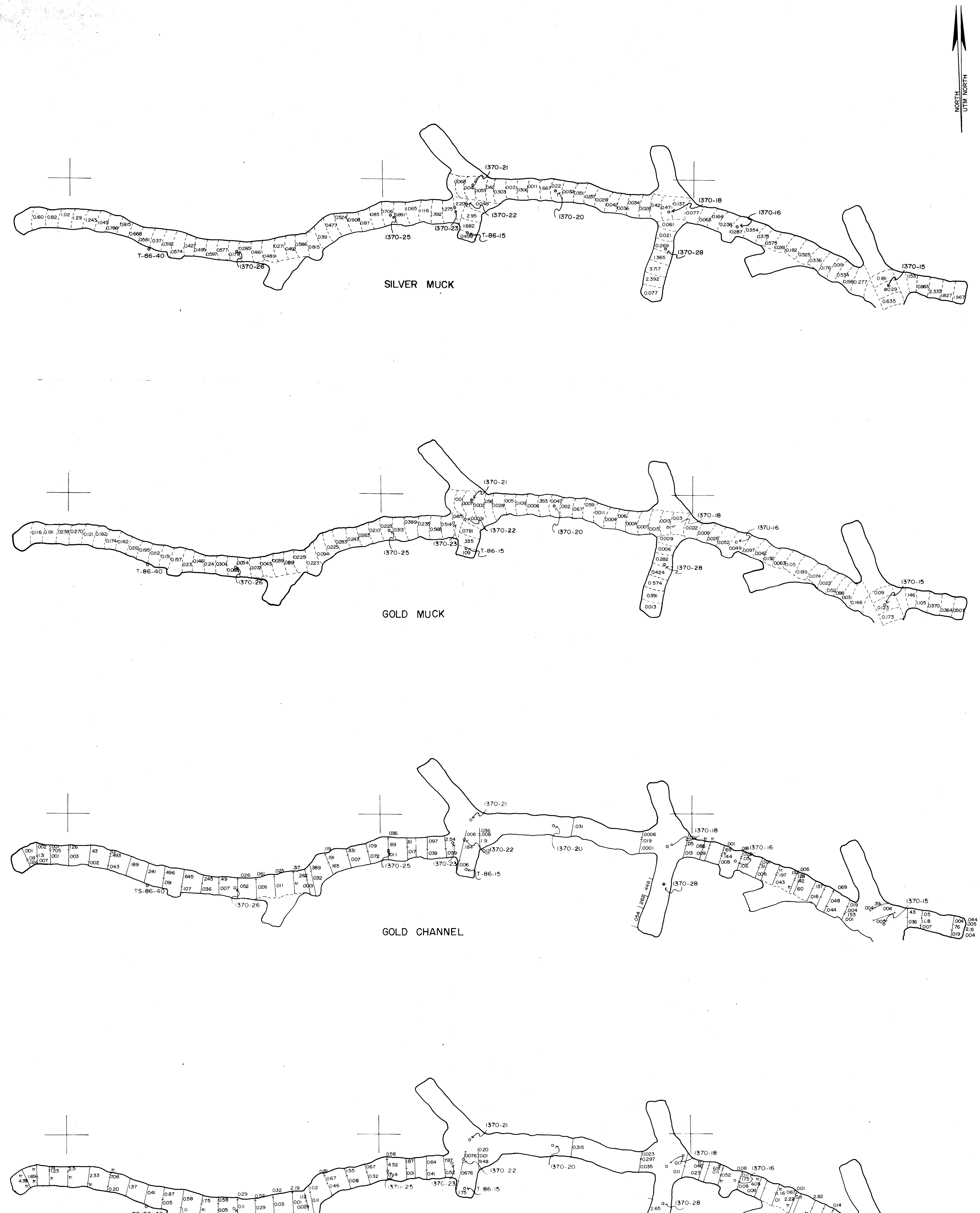
Dome Mountain Project
1370 Level – Geology Plan

Scale : 1 : 500
m 0 10 20 30 40 50 m

Date : May 18 1989

Plate : 2

Drawn By : Junior Mine Services Ltd.



LEGEND

ANDESITE LAPILLI-TUFF ± AGGLOMERATE; MINOR FLOWS

Vm	Maroon
Vg	Green
Vb	Bleached
V3	DACITE LAPILLI-TUFF; GREY
S4	ARGILLITE ± GRAPHITE
Qv	QUARTZ VEIN

0 5 10 15 20 25 Metres

SCALE 1:250

GEOLOGICAL BRANCH
ASSESSMENT REPORT

20,974

PLATE: 3, 4, 5, 6

TEESHIN RESOURCES LTD.

DOME MOUNTAIN PROJECT

PHYSICAL MASTER

Geology by:	Date: JUNE/87	Figure: