REDFERN RESOURCES LTD.

Tulsequah Chief Pre-Feasibility Study

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REDFERN RESOURCES LTD.

TULSEQUAH CHIEF PRE-FEASIBILITY STUDY

1. EXECUTIVE SUMMARY AND CONCLUSIONS

Redfern Resources Ltd. is the 100 percent beneficial owner of the Tulsequah Chief volcanogenic massive sulphide property in northwestern British Columbia. The Tulsequah Chief and adjacent Big Bull mines have previous mining history, jointly accounting for 933,520 ore tonnes produced during the 1950's.

Current geological drill indicated and inferred reserves at the Tulsequah Chief mine total 8.5 million tonnes grading 2.56 grams per tonne gold, 103.42 grams per tonne silver, 1.48 percent copper, 1.17 percent lead and 6.85 percent zinc. Fully diluted mineable reserves at current metal prices are estimated to total 6.93 million tonnes grading 2.40 grams per tonne gold, 93.37 grams per tonne silver, 1.40 percent copper, 1.07 percent lead, and 6.42 percent zinc.

The geological potential for the discovery of additional ore is considered excellent.

Metallurgical balances and coneentrate grades expected to result from a three stage differential flotation process were projected from recent metallurgical testing. The NSR value was calculated to be CAN $115.32 per tonne of mill feed using typical smelter terms and the following metal prices:

US $1.00/lb	copper
US $0.35/lb	lead
US $0.60/lb	zinc
US $375.00/oz	gold
US $4.00/oz	silver

At current metal prices, (April, 1993) NSR value was calculated to be Can. $94.81.

For the purposes of the study the exchange rate between Canadian and U.S. dollars is fixed at one Canadian dollar equals 0.80 US dollars.

Infrastructure is projected to include a deep sea port facility near Juneau, Alaska, a barge landing and fuel depot at Swede Point on Taku Inlet, a 50 kilometer road along the north shore of the Taku River, a major bridge crossing of the Tulsequah River at a point adjacent to the mine, and a 9 megawatt diesel generating plant.

Conceptual mining plans involving open blasthole stoping (73 percent) and shrinkage stoping (27 percent) were developed to form the basis of cost estimates. Evaluation of mining conditions was based on discussions with former mine operators. Further geotechnical study is required to confirm the mining parameters used.

Environmental engineering was largely beyond the scope of this study and will be required for a feasibility study for the project.

Capital and operating costs for a 800,000 tonne per year (2250 tonne per day) mine-mill operation have been estimated with an estimation variance of +/- 20 to 25 percent.

Preproduction capital including $12.2 million working capital is estimated to total Can. $137.7 million .

The preproduction capital estimate is based on the premise that the mine will be deepened by means of a winze during the early stages of production, and ongoing capital during the mine life is estimated to be an additional CAN $22.3 million after an allowance for recouping working capital.

Operating costs are estimated to be CAN $45.89 per tonne milled.

The different economic parameters for the project on a **before tax basis** and assuming all equity financing are summarized as follows:

1) Production Rate 800,000 tonnes per annum (TPA) for 6.93 million tonne reserve

    NSR $115.32 per tonne (at forecast metal price)

    IRR 31.2%

    NPV @ 10% $113 Million (Canadian)

    NPV @ 15% $ 65 Million

2) Production Rate 1,244,000 TPA for a hypothetical increase in mineable reserves to 10.8 million tonnes.

    NSR $115.32 per tonne (at forecast metal prices)

    IRR 41.7%

    NPV @ 10% $224 Million (Canadian)

    NPV @ 15% $141 Million

The project is most sensitive to the price of zinc and an increase in ore reserves distributed spatially to permit increased annual throughput.

The project economics are more sensitive to operating costs than they are to capital costs.

Specific ongoing studies have been recommended in Section 13. These studies will more clearly identify cost centers and permit decreases in estimation variances and contingencies during the feasibility stage.

2.0 INTRODUCTION

2.1 General

Redfern Resources Ltd. is the 100% beneficial owner of the Tulsequah property in northwest B.C. The property contains volcanogenic massive sulphide deposits of Kuroko association, very similar in all respects to the Myra Falls mine deposits of Westmin Resources near Campbell River, Vancouver Island. Previous production of copper, lead, zinc, gold and silver took place in the 1950's at two deposit areas called the Tulsequah Chief and Big Bull mines. Redfern entered into an option agreement with Cominco in 1987 to earn an interest in the property and explore the Tulsequah Chief deposits. This was successful and has culminated in Redfern's acquisition of a 100% interest in the property and definition of a geological drill-indicated reserve at the end of 1992 of 8.5 million tonnes grading 1.48% copper, 1.17% lead, 6.85% zinc, 2.56 g/tonne gold and 103.42 g/tonne silver. To evaluate the economic significance of the Tulsequah deposits and provide a focus for further exploration and development, Redfern commissioned the Pre-feasibility study of this report in early January, 1993.

2.2 Terms of Reference

This Pre-feasibility study is envisaged as a Type II level study consisting of the assimilation of all pertinent aspects of the project and determination of the most probable methods, costs and development options to allow a preliminary evaluation of economic feasibility of the project within an estimation variance of 20 to 25%. Operating and capital cost estimations have been performed by senior estimators with reliance on prior experience with similar operations and verbal quotations from suppliers and contractors. Outline plans were utilized in place of detailed engineering plans for facilities, plant and infrastructure.

Several consulting groups were employed on the various aspects of the study as follows:

Work Area	Responsibility
Project Coordination	Tonto Mining/Redfern Resources
Geology and Reserves	Cambria Geological Ltd
Minable Reserve/Mining	Tonto Mining
Metallurgy	Beattie Consulting Ltd.
Process Plant/Power	G. Hawthorne/Tonto Mining
Infrastructure	Tonto Mining
Financial Analysis	Redfern/Tonto Mining

2.3 Units and Currency

All units of measure are in the metric system. Costs and currency are in Canadian dollars. For the purposes of this study the exchange rate between Canadian and US dollars is fixed at one Canadian dollar equals $0.80 US dollars unless otherwise noted.

3.0 PROPERTY DESCRIPTION AND LOCATION

3.1 Location

The Tulsequah Chief property is located in northwest B.C approximately 100 kilometres south of Atlin, B.C. and 64 kilometres northeast of Juneau, Alaska. The mineral claims constituting the property are located on portions of NTS map sheets 104K/11,12 and 13, centred on latitude 53°43'N and longitude 133°35'W. Access to the property is by air to a gravel airstrip situated on the north side of the Taku River, approximately 8.5 kilometres south of the Tulsequah Chief mine site, and thence by helicopter (Figure 1).

The Tulsequah Chief mine and present exploration camp is on the east bank of the Tulsequah River at an elevation of 120 metres above sea level. The Big Bull mine is located 9 kilometres south of the camp, north of the Taku river and east of the confluence of the Taku and Tulsequah rivers.

3.2 Property Description and Claim Status

Based on the exploration and acquisition history of the various claims comprising the property it has been informally divided into three blocks: Tulsequah Chief, Big Bull and Banker. In each case the blocks consist of both modified grid located and crown granted claims. The configuration and location of the claims for each block is shown in Figure 2. Table 1 provides the size and current status of each of the claims listed by property block and claim type. All claims are valid with most in good standing to the year 2000. Crown granted claims are maintained through yearly tax payments. All claims are recorded in the Atlin Mining Division.

3.3 Ownership and Agreements

Redfern Resources Ltd. holds an undivided 100% interest in the property, subject only to a royalty of $0.10/ton of treated ore on the Tulsequah Chief and Big Bull claim blocks. The sequence of recent events and agreements resulting in Redfern's acquisition of title is summarized as follows:

1. By agreement dated March 25, 1987 between Redfern Resources Ltd. and Cominco Ltd., Redfern was granted the option to acquire a 40% working interest in the original Tulsequah Chief claim block through expenditures of $3,000,000 in exploration on or before December 31, 1990.

   Redfern completed its required expenditures in early 1989, thereby exercising the option and vesting its 40% interest in the Tulsequah Chief claims. Exploration work continued through 1989, 1990 and 1991 on the basis of 60/40 joint venture exploration funding by Cominco and Redfern, respectively. Cominco acted as operator for the Joint Venture from 1987 to June, 1991 with Redfern succeeding as operator thereafter.

2. By a separate option agreement dated December 4, 1991 between Redfern Resources Ltd. and Cominco Ltd., Redfern was granted the option to acquire Cominco's remaining 60% interest in the Tulsequah Chief claim block <u>and</u> 100% of the Big Bull claim block in consideration for the following:

   - payment by Redfern of up to $150,000 of the cost of an environment assessment study and remediation plan designed to satisfy the requirements of a Pollution Abatement Order issued against Cominco Ltd. by the B.C. Ministry of Environment in October, 1989. The study was to be conducted by a consultant acceptable to Redfern and Cominco, under the supervision of Cominco.

   - payment to Cominco of a non-refundable $100,000 deposit which was applicable to the cash payment portion of the Option, if exercised by Redfern.

   - within 30 days of Redfern's receipt of the above Environmental Assessment report, Redfern could exercise the Option by payment to Cominco of a further $100,000 cash, issuance of 1,100,000 shares of Redfern to Cominco and deposit to a trust account the lesser of $1,250,000 or the estimated actual costs of the remediation plan recommended in the Environmental Assessment report plus a 25% contingency.

3. By amending Agreement dated July 17, 1992 between Redfern Resources Ltd. and Cominco Ltd., the terms of the December 4, 1991 Option agreement were amended to reduce Redfern's cash payment obligation on exercise of the Option by $100,000. This was done in consideration of Redfern's agreement to assume the obligations of a previously undisclosed royalty of $0.10 per ton of treated ore from the Tulsequah or Big Bull claim blocks. The amending agreement also provided that Cominco would receive a Warrant for the acquisition of the 1,100,000 shares subject to a voluntary pooling agreement providing for release of the shares on exercise of the warrant on the basis of 300,000 shares immediately and 100,000 shares per month thereafter. Redfern was granted right of first refusal on sale of any of the shares by Cominco. Redfern was also granted the right to operate the remediation plan recommended in the Environmental Assessment report with a 3% management fee entitlement.

   The Environmental Assessment report was completed by Steffen, Robertson & Kirsten (Canada) Inc. in July, 1992. Pursuant to the recommended remediation plan cost estimate, plus a 30% contingency, Redfern deposited $1,154,200 into the trust account. Redfern exercised the Option on July 17, 1992 and completed the payment and Warrant issuance terms. The Joint Venture agreement was terminated on the same date and Redfern was conveyed a 100%

interest in the Tulsequah and Big Bull claim blocks, subject only to the $0.10/ton royalty. Redfern immediately proceeded with a 1992 exploration program, conducted by Cambria Data Services Ltd. as consultant to Redfern.

4. By Agreement dated March 4, 1992 between Redfern Resources Ltd. and Silver Talon Mines Ltd., Redfern was granted the option to acquire a 100% interest in the Banker claim block in consideration for payment by Redfern to Silver Talon of 30,000 Redfern shares and $1.00 cash. Redfern exercised the option on March 18, 1992 through completion of the required payment and acquired a 100% interest in the Banker claims.

TABLE 1. TULSEQUAH CHIEF PROPERTY - CLAIM STATUS

PROPERTY AREA	CLAIM NAME	RECORD NO.	TITLE NO.	UNITS	AREA (ha.)	EXPIRY DATE
Tulsequah Chief	Birds	5224	203794	1	25.00	May 30, 2001
	Pat	5225	203795	1	25.00	May 30, 2001
	Ross	5226	203796	1	25.00	May 30, 2001
	Mary 1	4289	203385	20	500.00	Aug 5, 2001
	Marcie 1	4290	203386	20	500.00	Aug 5, 2001
	Marcie 2	4291	203387	20	500.00	Aug 5, 2001
	Marcie 3	4292	203388	20	500.00	Aug 5, 2001
	Elysa 1	4293	203389	20	500.00	Aug 5, 2001
	Elysa 2	4294	203390	20	500.00	Aug 5, 2001
	Elysa 3	4295	203391	6	150.00	Aug 3, 2001
	Elysa 4	4296	203392	20	500.00	Aug 5, 2001
	<u>Crown Grants:</u>					
	River Fr.	5669			7.99	Jul 2, 1993 [1]
	Tulsequah Bonanza	5668			20.90	Jul 2, 1993 [1]
	Tulsequah Bald Eagle	5676			14.16	Jul 2, 1993 [1]
	Tulsequah Chief	5670			20.90	Jul 2, 1993 [1]
	Tulsequah Elva Fr.	5679			9.70	Jul 2, 1993 [1]
			Tulsequah Chief claims area		3,798.65	
Big Bull	Big Bull Extension	37/21	203965	1	25.00	Jul 18, 2000
	Bruce Fr.	303	203781			Aug 17, 2000
	Bull 2	141/32	203966	1	25.00	Jul 19, 2000
	Bull 3	142/32	203967	1	25.00	Jul 19, 2000
	Bull 4	143/32	203968	1	25.00	Jul 19, 2000
	Bull 8	142	203779	1	25.00	Jul 16, 2000
	Bull 9	179	203780	1	25.00	Apr 25, 2000
	CO 3	997	201802	20	500.00	Mar 4, 2000
	CO 5	998	201803	18	450.00	Mar 4, 2000
	Goat 1	1707	201925	16	400.00	Jul 23, 1994
	Swamp 1	1708	201926	4	100.00	Jul 23, 2000
	Swamp 2	1709	201927	1	25.00	Jul 23, 2000
	Swamp 3	1710	201928	1	25.00	Jul 23, 2000
	Webb 1	2766	202279	20	500.00	Nov 27, 2000
	Webb 4	2769	202282	20	500.00	Nov 27, 2000
	Webb 5	2770	202283	20	500.00	Nov 27, 2000
	Webb 9	2774	202284	10	250.00	Nov 27, 2000
	Webb 10	2775	202285	16	400.00	Nov 27, 2000
	<u>Crown Grants:</u>					
	Big Bull	6303			20.65	Jul 2, 1993 [1]
	Bull No. 1	6304			16.95	Jul 2, 1993 [1]
	Bull No. 5	6306			14.57	Jul 2, 1993 [1]
	Bull No. 6	6305			17.22	Jul 2, 1993 [1]
	Hugh	6308			20.71	Jul 2, 1993 [1]
	Jean	6307			17.02	Jul 2, 1993 [1]
			Big Bull claims area		3,907.12	
Banker	Tallon No. 1	1979	202030	20	500.00	Aug 2, 1993
	Tallon No. 2	1980	202031	9	225.00	Aug 2, 1993
	<u>Crown Grants:</u>					
	Vega No. 1	6155		1	20.90	Jul 2, 1993 [1]
	Vega No. 2	6156		1	17.62	Jul 2, 1993 [1]
	Vega No. 3	6157		1	18.97	Jul 2, 1993 [1]
	Vega No. 4	6158		1	19.85	Jul 2, 1993 [1]
	Vega No. 5	6159		1	14.94	Jul 2, 1993 [1]
	Janet W. No. 1	6160		1	18.95	Jul 2, 1993 [1]
	Janet W. No. 2	6161		1	18.75	Jul 2, 1993 [1]
	Janet W. No. 3	6162		1	16.60	Jul 2, 1993 [1]
	Janet W. No. 4	6163		1	20.76	Jul 2, 1993 [1]
	Janet W. No. 5	6164		1	18.20	Jul 2, 1993 [1]
	Janet W. No. 6	6165		1	19.02	Jul 2, 1993 [1]
	Janet W. No. 7	6166		1	18.78	Jul 2, 1993 [1]
	Janet W. No. 8	6167		1	17.98	Jul 2, 1993 [1]
	Joker	6169		1	16.60	Jul 2, 1993 [1]
			Banker claims area		982.92	
			TOTAL PROPERTY CLAIMS AREA		8,688.69	

[1] Maintained through annual tax payments due July 2 of each year.

4.0 ACCESS, CLIMATE AND LOCAL RESOURCES

4.1 Access and Local Infrastructure

The property is presently accessed by air from Atlin, B.C, 100 kilometres to the North, or from Juneau, Alaska, 64 kilometres to the southwest. Access is also possible by water from Juneau up the Taku inlet and thence up the Taku river which is navigable by shallow draft vessels at certain times of the year. A 4,000 foot long airstrip located north of the Taku river and west of the Tulsequah river accommodates aircraft to DC-3 or Bristol size. A road from the airstrip proceeds to the Polaris-Taku mine located on the west side of the Tulsequah River. There is no direct road access to the Tulsequah mine site at present although the river can be forded at times of low water. Currently, transport to the mine site and other property areas is achieved by helicopter.

The Alaska government has publicly stated its wish to establish road access to Juneau via the Taku corridor and linking with the Canadian highway system at Atlin, B.C. Alaskan legislation has established potential road corridors on both sides of the Taku inlet and river up to the Canadian border.

A 20 person exploration camp is located adjacent to the 5400 level portal. The camp consists of several wood frame bunkhouses, a dry, mess, offices, core shed and storage units. Local power is supplied by two Caterpillar diesel generators of 250 and 125 kva capacity. A Cat D-3 tractor with winch and backhoe attachment is located at the camp site for local road building and earth moving. An ATV is also located on site for local transport.

Underground development at the Tulsequah mine was historically conducted on 9 levels: 5200, 5400, 5600, 5700, 5900, 6100, 6200, 6400 and 6500. Separate portals provided direct access to each of the 5200, 5400, 5900, 6400 and 6500 levels. At present only the 5400 level is used for regular mine access. Underground access and development on the 5400 level is supported by tracked mine equipment including 2 locomotives, 10 rocker dump mine cars and a mucking machine. An Ingersoll Rand compressor (650 cfm) provides compressed air for mine use. Exploration drilling utilizes 2 underground drill rigs: 1 Connors 12HH and 1 Boyles 37A both capable of BQ and NQ wireline drilling. All of the equipment is 100% owned by Redfern Resources.

4.2 Climate, Topography and Vegetation

The property is located in mountainous terrain of the Coast Range of Northwest B.C. Local relief ranges from 50 metres above sea level at river level to a maximum of 1,850 metres on the west side of Mt. Eaton. Slopes are moderate to steep with wide alluvial benches and gravel bars in the vicinity of both the Taku and Tulsequah rivers.

The area is subject to a wide range of climatic conditions with cold winters, moderated at lower elevations by proximity to the coast. Precipitation is moderate to high with a mean annual level estimated at 1900 mm. Approximately half of the precipitation is snow with average snowpack conditions of 1-2 metres at the Tulsequah mine. Numerous permanent glaciers exist in the higher elevations especially to the north and west. The Tulsequah river is primarily fed by glacial melt water of the Tulsequah glacier, which is in recession, and is subject to occasional floods due to release of dammed meltwater from beneath the toe of the glacier. However, due in part to the width of the river at the Tulsequah mine these floods are not catastrophic and usually reach water levels of only 1.2 - 2.0 metres over normal for a few days.

The region is heavily forested with spruce and hemlock to elevations of 1200 metres above sea level. Higher elevations consist of typical sub-alpine to alpine vegetation. Slide and blow-down areas are covered by thick secondary growths of alder and brush.

4.3 Local Resources

In addition to the base and precious metal ores contained in the Tulsequah property, numerous other significant mineral properties exist in the area. In the immediate vicinity these include the past producing Polaris-Taku gold mine (currently in advanced exploration) and the Erickson-Ashby base metal prospect. Farther away, in the Alaska panhandle, is the Greens Creek Mine and the AJ and Kensington gold mines (both in advanced development).

Fish resources are principally centred on the Taku river and major tributaries which are major salmon and trout spawning habitat. The Tulsequah river itself has a negligible to low fishery value due to the high proportion of glacial sediment load and disruptive floods.

Timber resources consist of mature stands of merchantable timber at low elevations. Although no sustained logging development has taken place in the Tulsequah area, minor logging for mine timbers and road construction occurred during the past operation of the Tulsequah Chief, Big Bull and Polaris-Taku mines.

There are no permanent settlements within 50 kilometres of the property. The closest city, Juneau (pop. 30,000), capital of Alaska, is 65 kilometres from the property. Juneau provides deep water port facilities, a stable economy, air and water commercial transportation services and a source of supplies for operations on the property.

5.0 HISTORY

5.1 Exploration History

The Tulsequah Chief deposits were discovered in 1923 by W. Kirkham of Juneau. While prospecting, he located a lens of high grade barite, pyrite, sphalerite, galena and chalcopyrite in a gulley above the present site of the 6500 Level adit. Development of this discovery proceeded through 1923 to 1929 and spurred a wave of prospecting activity in the region. In 1929, V. Manville located the Big Bull deposit and the same year discoveries were made at the Potlatch (Sparling), Banker and Polaris-Taku deposits. The Erickson-Ashby base metal deposit was discovered in 1930.

Cominco acquired the Tulsequah Chief and Big Bull deposits in 1946 and commenced production in 1951. Production continued from both properties through 1957 at an average rate of 530 tons/day. At the end of this time the mine was closed, with reserves in place, due to low metal prices. The deposits lay dormant and unexplored until 1971. Throughout this time period the deposits were considered to be shear-zone controlled mineralization.

In 1971 Cominco geologists re-examined the Tulsequah geology and mineralization and re-interpreted the zones as volcanogenic massive sulphide deposits associated with felsic volcanism and syngenetic processes. Although a number of proposals were advanced over the subsequent decade for renewed exploration no significant work was conducted until 1981. In that year 1:2500 scale mapping was conducted over the Tulsequah Chief and Big Bull deposits by J.P. Sorbara. This was followed in 1982 by airborne Dighem (Cominco) and Questor Input AEM (Redfern) surveys which were unsuccessful in locating conductors over the known deposits. However, both of these surveys did locate weak conductors in areas of felsic volcanic rocks elsewhere on the present property.

In 1981 the Tulsequah area was mapped at 1:50,000 scale by J.G. Payne, J. Nelson and G. Gosson. A separate report and map including age data was published in 1984 by Payne and Nelson. This work was later refined in a 1987 report by J.G. Payne and W.G. Sisson , under Cominco supervision.

Redfern and Comaplex Resources International Ltd. carried out regional exploration on the Tulsequah map sheet in the early 1980's with discovery of a silver/base metal vein deposit south of the Taku river. Subsequently Redfern and Comaplex acquired claims surrounding the Tulsequah Chief deposits with the intent of exploring for extensions of the massive sulphide mineralization using a Kuroko VMS geological model for deposit genesis. Continued discussions between Cominco and Redfern eventually led to the 1987 Option agreement whereby Redfern acquired the right to earn up to a 40% interest in the Tulsequah Chief deposits. Exploration programs to further define and extend the Tulsequah Chief deposits were conducted every year from 1987 to the present in concert with new and amended agreements culminating in Redfern's acquisition of 100% ownership of the present property.

A summary of the exploration work conducted in the 1987 to 1992 period is presented in Table 2 below. Highlights and significant results of the annual exploration programs follow:

- The 1987 program completed 1:10,000 geological mapping over the Tulsequah property and surrounding area with 1:2500 surface mapping on the Tulsequah claim block and 1:1000 scale mapping over the area of the deposits. Surface drilling consisted of five holes totalling 3,523.8 metres. Holes TC-87-1 and TC-87-5 intersected "ore" grade mineralization in the down dip extensions of the G and H zones. The remaining three holes intersected thick altered sections of the mineralized horizon with sub-economic grades and widths.

- In 1988 mapping was restricted to 1:2500 scale follow-up of selected areas of felsic volcanics outside the Central Area and 1:1000 scale mapping of the area west of the Chief Fault near the 5200 portal. The 5400 level was rehabilitated over a length of 884 metres and geologically mapped at 1:500 scale. Two surface holes totalling 485.9 metres evaluated the down-dip extension of the 5200 portal alteration zone. An additional 11 underground holes (3,045 metres) were drilled from the 5400 Level to evaluate lower portions of the old workings. Eight of these intersected significant intervals of mineralization. Water quality sampling was also conducted.

- Exploration in 1989 consisted of further rehabilitation and re-ballasting of track in the 5400 Level plus 174 metres of new drifting on the 5400 level crosscut. Specific gravity determinations were completed on 1987, 1988 and 1989 drill hole intersections. Environmental water quality sampling continued at the mine site. Drilling from underground stations on the 5400 Level consisted of 4,890 metres in 10 holes designed to test the down-dip extensions of the sulphide lenses. Eight holes intersected significant mineralized intervals with increasing thickness at depth in the H and AB lenses. These zones were tested down to 540 metres below the 5400 Level. Redfern calculated a reserve to the 3500 Level of 5.27 million tonnes (5.8 million tons) grading 1.6% Cu, 1.3% Pb, 7.0 % Zn, 2.74 g/tonne Au (0.08 oz/ton) and 99 g/tonne Ag (2.9 oz/ton).

- During 1990 the focus continued to be the expansion of reserves at depth in the H and AB sulphide lenses. The 5400 Level crosscut was extended by 183 metres of drifting and cutting of drill stations. Eight holes were drilled from underground (including 1 abandoned hole) for a total of 5,980 metres. This drilling obtained the thickest mineralization intersected to date in the deep portions of the H lens. Cominco calculated a reserve of 6.3 million tonnes (6.93 million tons) grading 1.58% Cu, 1.33% Pb, 7.59% Zn, 2.75 g/tonne Au (0.08 oz/ton) and 115 g/tonne Ag (3.35 oz/ton). Redfern included some additional lower grade intersections and calculated a reserve of 7.3

million tonnes (8.0 million tons) grading 1.55% Cu, 1.22% Pb, 6.81% Zn, 2.74 g/tonne Au (0.08 oz/ton) and 109 g/tonne Ag (3.19 oz/ton).

- In 1991 Redfern acquired operator rights and contracted Cambria Geological Ltd. consultants to continue with a smaller underground program to confirm and expand the deep mineralization in the H and AB lenses. Drilling consisted of 3,088.8 metres in 6 holes, all of which intersected the targeted zones. Also completed was a comprehensive computer-assisted re-interpretation of the deposit geology, structure and geometry of sulphide zones. This work included re-logging of critical holes, specific gravity determinations of all mineralized intervals and construction of a 3-D model. Cambria also calculated a sectional reserve totalling 7.6 million tonnes (8.36 million tons) grading 1.62% Cu, 1.19% Pb, 6.51% Zn, 2.88 g/tonne Au (0.084 oz/ton) and 116.6 g/tonne Ag (3.4 oz/ton).

- The 1992 exploration program was again managed by Cambria Geological Ltd. on behalf of Redfern. As part of the continuing re-interpretation, all pre-1991 core was re-logged for stratigraphic correlation and surface mapping conducted on a 20 line-kilometre grid over the Tulsequah Chief mine area. The Big Bull area was also mapped. Environmental water sampling and meteorological programs were implemented and continued as recommended in the SRK report completed in June, 1992. Portions of the 5200 and 5400 Levels were mapped and surveyed. Surveying of prior drill collars and tie points were integrated into the national grid system.

  An aggregate of 4,579 metres of drilling in 13 holes was completed from the 5400 Level. Two of the holes were deepened extensions of previous drill holes. Holes were targeted to expand the existing reserves in the H and AB zones and probe the up-dip extensions in the H and G zones. The drilling has been integrated with the results of all past drilling to estimate a new geological reserve for the Tulsequah Chief deposits. This reserve now totals 8.5 million tonnes grading 1.48% copper, 1.17% lead, 6.85% zinc, 2.56 g/tonne gold and 103.42 g/tonne silver.

TABLE 2. SUMMARY OF EXPORATION 1987 - 1992

	U/G DEV.		SURFACE DRILL.		U/G DRILLING		
YEAR	LEVEL	METERS	HOLES	METERS	HOLES	METERS	SURFACE WORK
1987			5	3,523.8			Geological mapping
1988	5400	884(r) rehab	2	485.9	11	3,045.0	Geological mapping, soil sampling, water sampling
1989	5400	174			10	4,890.0	S.G. analysis,water sampling
1990	5400	183			8	5,980.0	Water sampling
1991					6	3,088.8	S.G. analysis, computer modelling
1992					13	4,579.0	Geological mapping, core re-logging, computer modelling, survey establishment.
TOTAL			7	4,009.7	48	21,582.8	

5.2 Past Production

Cominco Ltd. acquired the Tulsequah Chief and Big Bull deposits in 1946 and commenced production in 1951. Cominco adapted the mill used for the former Whitewater (Polaris-Taku) gold mine and continued mine operations until 1957 when the mines were closed due to low metal prices. Production rates over this period averaged 482 tonnes (530 tons) per day with total production over the life of the operation of 1,029,089 tons. Of this amount, 625,781 tons derived from the Tulsequah Chief deposits and 403,308 tons came from the Big Bull mine. Average overall grade of the ore was 1.59% Cu, 1.54% Pb, 7.0% Zn, 0.112 oz/ton Au and 3.69 oz/ton Ag. The metal output totalled 14,756 tons Cu, 11,439 tons Pb, 54,910 tons Zn, 95,340 oz. Au and 3,329,938 oz. Ag. This equates to recoveries of 88% Cu, 94% Pb, 87% Zn, 77% Au and 89% Ag.

Outstanding reserves at shutdown, calculated by Cominco, were 780,000 tons at Tulsequah Chief grading 1.3% Cu, 1.6% Pb, 8.0% Zn, 0.07 oz/ton Au and 2.9 oz/ton Ag. The Big Bull deposit reserves were estimated at 50,000 tons grading 0.5% Cu, 1.3% Pb, 5.6% Zn, 0.07 oz/ton Au and 1.8 oz/ton Ag.

5.3 Previous Work, Investigations and Test Work

5.3.1 Petrography and metallurgical testing:

The most complete petrographic study of the sulphide ore suite was conducted in 1991 by J.A. Mcleod of Cominco's engineering research laboratory. He examined 48 specimens collected from 7 drill intersections of the Tulsequah sulphide zones and characterized the sulphide species of the ore suite. The principal sulphide minerals were identified as pyrite, sphalerite, chalcopyrite, tennantite and galena. The main gangue constituents are barite, sericite and quartz. Tennantite was identified as a significant component of the copper grade. Gold was observed as both native grains and in electrum. Some of the gold was observed in large grains amenable to separate gravity concentration. Due to the relative coarseness of the principal sulphides adequate liberation was predicted at a nominal 200 mesh grind. The presence of minor arsenic and antimony contents were noted within tennantite.

In 1992, preliminary metallurgical test work was completed on a 50 kg. split from a 100 kg composite sample of ore material from the Tulsequah Chief deposits. The work was conducted by Bacon Donaldson and Associates under the supervision of M. Beattie. The metallurgical test work was designed as a preliminary evaluation of the copper, lead and zinc concentrate production. Starting conditions mimicked those employed in historical production at Tulsequah. The composite was crushed to -6 mesh, blended and split into 2 kg. charges for individual tests. Analysis of the composite yielded values of 1.64% Cu, 1.28% Pb, 8.48% Zn, 12.6% Fe, 17.9% total S, 0.14% As, 0.06% Sb, 2.57 g/t Au and 132.7 g/t Ag.

The test work results indicated flotation recoveries in excess of 90% for all metals. Approximately 25% of the gold reported to a gravity concentrate. The cleaned copper concentrate graded 21% Cu. Pb concentrate contained 44% Pb and additional test work was recommended to improve cleaning performance. Zinc concentrate contained 46.7% Zn and additional reagent optimization was recommended to improve grades and recovery. The remaining gold not recovered by gravity primarily reported to the copper concentrate (48%) with a subordinate amount in the lead concentrate (5%). Silver primarily reports to the copper concentrate (65%) with about 15% going with the lead.

Metallurgical testwork was greatly expanded in 1993 with completion of a more comprehensive series of flotation tests on new composite samples. This work was conducted by M. Beattie and was successful in demonstrating a relatively simple three stage separation with consecutive flotation and cleaning of a lead concentrate, copper concentrate and zinc concentrate in that order. This work was completed preparatory to this pre-feasibility study and is discussed in more detail under section 8, Mineral Processing.

5.3.2 <u>Economic studies/evaluations:</u>

In 1992 Redfern commissioned a project review and preliminary economic evaluation completed by H. Wober of Mine Development Engineering. This study incorporated a review of the geology and reserves, possibilities for infrastructure and plant development, metallurgical data, estimate of capital and operating costs for production rates of 1500, 2000 and 2250 tonnes per day, and an appraisal of the economic performance on a DCFROR basis.

The study was qualified as a preliminary evaluation with a 35-40% estimation variance. The evaluation included a very conservative re-appraisal of the geological reserves using a much higher NSR basis cutoff to derive a probable minable reserve of 5.0 million tonnes with an overall NSR value of Can $120.00/tonne. No detailed mine plan or development studies were undertaken to develop this reserve.

Results of the study are summarized as follows:

Production Rate Case	1500 t/day	2000 t/day	2250 t/day
Initial Capital ($US, millions)	100.00	122.40	127.60
Working Capital (")	9.73	11.20	11.75
Operating costs ($US/tonne)	53.09	48.85	47.83
After tax IRR (%)	9.8%	13.7%	14.7%
After tax NPV @10% ($US, millions)	-1.0	16.0	20.0

Although the evaluation indicates that positive economics are achievable it employs a very conservative approach to all estimates and seriously condenses the reserve by using a very high NSR cutoff of $80/tonne. This cutoff is higher than the average gross value of many Canadian massive sulphide properties in past or present production. The study also uses an exchange rate of Can $1.00 = US $0.88. Operating costs are also relatively high when compared to operating data for most massive sulphide deposits at similar production rates. For the above reasons the evaluation was considered a worst case scenario.

5.3.3 Environmental Studies

The most complete environmental assessment of the Tulsequah property was completed by Steffen Robertson & Kirsten Consulting Engineers in 1992 as a pre-requisite to Redfern's exercise of the option agreement with Cominco and acquisition of a 100% interest in the property. This report includes a complete evaluation of the results of past environmental water quality sampling, current status of mine water and surface drainage and the storage of mine waste rock and ore on surface and underground. The report also provides a comprehensive reclamation plan for remediation of acid water and metal discharge to the receiving environment. The remediation plan primarily involves consolidation and safe storage of the mine waste rocks located at the 5200 and 5400 portals and ultimate bulkhead emplacement at each of the lower portals to allow mine flooding and impoundment of the mine waters. The latter actions are contemplated as a "walk-away' solution in the event that the mine is not put into development. Development of the Tulsequah Chief deposits will require entry into the full environmental and mine permitting process and a revised reclamation plan.

6. GEOLOGY AND RESERVES

6.1 Summary

The Tulsequah Chief property contains two past producing polymetallic massive sulphide mines of Kuroko affinities. These two deposits, the Big Bull and Tulsequah Chief, were mined by Cominco during the period 1949 - 1957 and together produced 933,520 tonnes grading 1.59% copper, 1.54% lead, 7.0% zinc, 3.84 g/tonne gold and 126.51 g/tonne silver. Significant exploration activities since 1987 have been principally directed at delineation of additional reserves in the depth extensions of the Tulsequah Chief mine.

The Tulsequah Chief property is predominantly underlain by folded, faulted and metamorphosed pre-Permian age, volcanic dominated rocks of the Mount Eaton Group. The north trending regional Chief fault separates the Mount Eaton Group on the east from the similar but more highly deformed and metamorphosed Tulsequah Group rocks on the west. The Chief fault is located on the west side of the property and extends north from the confluence of the Taku and Tulsequah rivers parallel to the Tulsequah River and passing 800 meters west of the main Tulsequah Chief deposit.

Sulphide mineralization at the Tulsequah Chief Mine consists of pyrite and lesser amounts of sphalerite, chalcopyrite, galena, tetrahedrite and native gold associated with chert, barite and gypsum. The deposits occur both as thinly bedded and debris flow facies at two distinct levels within the mine stratigraphy corresponding to two discreet cycles of felsic volcanism. The footwall to the basal felsic sequence consists of undifferentiated basaltic rocks of Unit 1. The lower felsic unit (Unit 2) of dacitic flows, tuffs and breccias contains the bulk of the known sulphide mineralization which occurs as up to three principal stacked lenses. These are best developed in the Central Mine Block as the H, AB_1 and AB_2 horizons. In the West Mine Block the horizons are collectively termed the F zone and similarly in the East Mine Block they are referred to as G zone. These zones comprise all of the presently defined reserve at the Tulsequah Mine. Above the lower felsic unit is the mine series mafic volcanic Unit 3. This is overlain by the upper felsic Unit 4 which hosts the previously mined sulphide deposits of the I zone within the East Mine Block. The entire mine stratigraphy is capped by the hanging wall series basalts and sediments of Unit 5.

A polygonal reserve estimate has integrated the results of recent exploration drilling programs since 1987 (21,666 meters in 44 holes) with the previous production and drilling records of 1957 and earlier. Total indicated and inferred reserves are 8,500,592 tonnes grading 1.48% copper, 1.17% lead, 6.85% zinc, 2.56 g/tonne gold and 103.42 g/tonne silver.

6.2 Property Geology

The Mount Eaton Group consists of a bimodal suite of volcanics and associated sediments which have been folded, faulted and metamorphosed to upper greenschist/lower amphibolite grade. Two regional belts of felsic volcanic dominated rocks occur within the dominantly mafic stratigraphy. The lower of these two felsic belts hosts the known volcanogenic massive sulphide deposits at the past producing Tulsequah Chief and Big Bull mines. In the vicinity of the Tulsequah Chief mine the rocks are folded into northwest plunging, overturned to steeply west-dipping parasitic folds on the west limb of the regional Mount Eaton anticline. Faults subparallel to the axial planes of these folds dissect the mine stratigraphy and divide the deposit into three fault bounded blocks. The West Mine Block lies west of the 4400E fault, the Central Mine Block is situated between the 4400E and 5300E faults and the East Mine Block is east of the 5300E fault.

The stratigraphy of the Mount Eaton Group rocks has been broadly subdivided into three major lithologic formations. The mafic volcanic succession of Unit 1 consists of undifferentiated basalt and basaltic andesite flows, flow breccias and hyaloclastic tuffs. Unit 2 lower felsic volcanic-sedimentary rocks consist primarily of dacite flows, breccias and tuffs which grade to clastic sediments and chert north and east of the Tulsequah Chief mine. This unit also hosts the exhalite and massive sulphide zones at the Tulsequah Chief and Big Bull mines. Unit 3 represents the upper felsic volcanic-sedimentary sequence which, in the south, consists primarily of dacite flows and tuffs. In the central and northern portions of the property the unit is dominated by limestone, clastic sediment and dacite tuff.

West of the Chief fault lie rocks of the Tulsequah Group which may be metamorphosed and more strongly deformed equivalents of the Mount Eaton Group. These schists, phyllites and marbles derive from metamorphosed intermediate to mafic volcanics and limestones which have been affected by up to four phases of deformation.

Intrusive rocks of the Coast Plutonic belt impinge on the east boundary of the property. A large Tertiary age quartz monzonite intrusion occupies the area east of Mount Eaton. Numerous Tertiary Sloko rhyolite and quartz feldspar porphyry dykes intrude the Mount Eaton Group rocks and are often emplaced along earlier fault structures.

6.3 Mine Geology and Mineralization

Drilling and surface mapping in the immediate vicinity of the Tulsequah Mine has allowed finer subdivision of the Mount Eaton Group rocks into three major series: Footwall Series (Unit 1), Mine Series (Units 2 to 4) and the Hanging wall Series (Unit 5). The Footwall and Hanging wall Series are dominated by mafic volcanics and associated sediments. The Mine Series units are correlated with the lower felsic volcanic belt on a property scale. The basic stratigraphy of the mine geology is shown in the following table:

UNIT	DESIGNATION	LITHOLOGICAL DESCRIPTION ZONES	SULPHIDE
7	Tertiary Intrusions	Sloko rhyolite; quartz feldspar porphyry dyke	
6	Tulsequah Group	Chlorite schist, phyllite, marble after metamorphosed intermediate to mafic volcanics and limestone	
5	Hanging wall Series	Basalt flows, dykes; basalt ash tuff, interflow tuff;tuffaceous argillite, siltstone, minor chert	
4	Mine Series - Upper Felsic Volcanic Horizon	Dacite flow, flow breccia, lapilli to ash tuff; minor basalt flow and interflow tuff; laminated tuffaceous sediment and argillite; sericite/silica/pyrite altered flows and tuffs, chert, gypsum, barite,massive sulphides.	I
3	Mine Series - Mafic Volcanic Horizon	Feldspar-phyric and aphanitic basalt flows/sills plus bleached silica/sericite/pyrite altered varieties; basalt ash to lapilli tuff and tuffaceous sediments	
2	Mine Series - Lower Felsic Horizon	Feldspar phyric dacite flow, flow breccia, ash to lapilli tuff; basalt flows, pillowed flows, lapilli and ash tuffs and interflow tuff; hematite/albite/sericite altered flows and tuffs; banded chert, gypsum, barite, massive sulphides and debris flow variants	F,G,H AB_2 AB_1
1	Footwall Series	Quartz amygdaloidal basalt flows, breccias and lapilli tuffs plus bleached silica/ sericite/pyrite altered varieties; laminated tuffaceous sediments, minor chert	

Sulphide mineralization occurs within both the Upper and Lower Felsic Horizons of Units 4 and 2 respectively. The extent of sulphide mineralization in the Upper Felsic Horizon is restricted to the previously mined I Zone in the upper deposits located east of the 5300E fault. Mineralization in the I Zone thins and pinches out into pyrite facies within 300 meters northeast of the fault. Most of the sulphide mineralization of previous production and the present reserves occurs in up to three separate horizons within the Lower Felsic Horizon. These are best developed in the Central Mine Block as the AB_1, AB_2 and H Zones. The lowermost AB_1 horizon is separated from the AB_2 and H horizons by a variable thickness of dacite flows, tuffs and tuffaceous sediments. The main H-AB horizon contains the bulk of the sulphide zone reserves. For much of the deposit the H and AB_2 horizons are distinct and separated by altered debris flow and locally unaltered dacite and basalt flows. East of section E2580i the zones merge and form a continuous thick sulphide zone termed the H zone. West of this section the H and AB_2 horizons are progressively separated by a thickening wedge of dacite flows and lapilli tuffs, suggestive of a felsic centre to the west. Outside of the Central Mine Block the distinction between the three sulphide horizons in the Lower Felsic Horizon becomes less clear. For this reason the equivalent of the H-AB zones in the East Mine Block are termed the G Zone and in the West Mine Block they are referred to as F Zone.

The sulphide mineralization in the H-AB lenses and equivalents consists of thin banded to massive pyrite, sphalerite, galena and chalcopyrite. Important accessory minerals consist of tetrahedrite-tennantite and native gold. Gangue minerals are chiefly composed of barite, chert and gypsum. The proportion of the ore-forming sulphides varies rapidly such that three distinct sulphide facies are recognized: copper facies consists of massive to semi-massive pyrite with subordinate chalcopyrite (1-10%) and lesser sphalerite and galena. Zinc facies consists of lower total sulphides on average (<60%) and contains more sphalerite, galena and barite. Pyrite facies is composed primarily of pyrite with minor economic sulphides. Laterally the sulphide rich zones may grade into lower sulphide content exhalite sequences of chert, pyrite, gypsum and altered sericitic tuff.

6.4 Sampling, Assays and Density Factors

Sampling was conducted on all mineralized drill intersections and submitted to Acme Laboratories in Vancouver for ICP analysis. All samples with Cu, Pb, Zn or As greater than 1%, Ag greater than 30 ppm and Au greater than 1000 ppb were assayed. Sampling was conducted on half sawn core in all cases.

Specific gravity determinations were conducted on all 1990 and 1992 drill samples. Previous specific gravity (SG) determinations for the 1987 to 1989 core were re-checked by weighing the half split core. All SG data was compiled into a database with the sample assay information for calculation of weighted assay intervals.

6.5 Geological Model and Continuity

The Mine Series rocks at Tulsequah Mine have been folded into a series of easterly verging anticline-syncline parasitic fold pairs on the west flank of the regional Mount Eaton anticline. These folds have axial planes striking 166 ° and dipping 79 ° W. Fold axes plunge 56 ° at 329 °. The folds are tight with interlimb angles of 40 to 50 °, fold amplitudes of 30 to 50 meters and frequency of 50 meters.

Faulting plays an important role in the deposit geometry. The 4400E and 5300E faults are subparallel to the axial planes of the parasitic folds and probably are related to the same deformational episode. The 4400E fault is steep east dipping to near vertical and has a prominent topographic expression. It merges with a fault system extending 8 kilometres south to the Big Bull Mine area. Underground the fault is marked by up to 1 meter of clay gouge and the presence of Sloko dyke rocks. Within the mine area, stratigraphy is not significantly displaced across the 4400E fault - maximum offset is in the order of 50 meters in a dextral sense. The 5300E fault has a more north-northwesterly strike, vertical dip and has a number of associated splays which are intersected in drill core. The apparent displacement is usually less than 70 meters in a right lateral sense. It is suspected, however, that the 5300E fault may have a considerable component of vertical movement due to the marked facies and thickness changes across the fault.

The sulphide mineralization is associated with the onset of felsic volcanism, principally in the Lower Felsic cycle. Layering and banding of the deposits and the presence of thick chert exhalite suggests deposition in a quiescent basin on the margins of the felsic domes. The presence of local debris flows imply some deposition on unstable slopes. Debris flow facies mineralization is especially prevalent above the H horizon suggesting local moderate to high topographic relief on the margins of the basin. Within the central portions of the sulphide horizons the continuity of ore grade mineralization is high. Intercalation with fringe and marginal chert and pyrite rich facies occurs on the margins of the basin and perhaps in areas of local palaeographic highs.

The superimposition of fold deformation on the Tulsequah Chief deposit resulted in attenuation of the limbs and thickening in the fold hinge areas. For this reason continuity of thickness and grade of the sulphide mineralization is highest parallel to the plunge of the fold axes and diminishes laterally across the fold axes. This feature plus the fault discontinuities were used to guide the interpretation and reserve calculation.

6.6 Reserve Estimation Method and Classification

After determination of the orientation and geometry of the controlling fold structures, East facing sections were prepared, inclined parallel to the axial plane of the folds at 10 meter intervals. Similarly a set of North facing sections oriented perpendicular to the plunge of the fold axes and orthogonal to the east sections were also constructed. These north facing inclined sections were used to interpret the true thickness and geometry of the geologic units and fold structures. This information was meshed with the east inclined sections and interpreted sections drawn. Lastly a set of horizontal level plans were produced at 50 meter intervals for ease of visualization and mine planning purposes. All of the old pre-1957 drill information was utilized to assist interpretation and ensure a cohesive model of the deposit from the upper levels through to the deepest penetrations. However, no new reserve calculations were carried out in the area of the old workings and established reserve blocks. All of the reserve calculations were conducted on the sulphide deposit below the 5200 level and the prior Cominco reserve.

A polygonal estimation method was utilized for volume estimation and reserve calculation. To allow for the variable thickening and attenuation of the folded sulphides in the Central Mine Block the drill hole pierce points were projected onto a longitudinal section oriented parallel to the fold axis and perpendicular to the axial plane. Each hole was assigned an elliptical area of influence oriented with the long axis of the ellipse parallel to the fold axis. For reserve classification purposes drill-indicated reserves used an ellipse with a major semi-axis of 40 meters and a minor semi-axis of 25 meters. Drill-inferred reserves were obtained by extending the ellipse dimensions to a major semi-axis of 80 meters and a minor semi-axis of 50 meters.

Only drill holes satisfying a minimum NSR cutoff value of US $40/tonne over a minimum true width of 3.0 meters were included in the reserve calculation. The NSR calculation employed is detailed in the appended full geological report. For volume and tonnage calculations in the Central Block the sectional thickness (axial plane parallel) of the drill hole intersection was multiplied by the interval SG, and area of influence for each of the drill-indicated and drill-inferred classification polygons. In the Eastern Block on the G Zone the deposit is subplanar and a conventional polygon technique was used on a longitudinal section parallel to the plane of the G-Zone. In this case the true thickness of the intersection was multiplied by the interval SG and polygon areas.

6.7 Geological Reserve Summary

Based on the reserve estimation methodology the following geological reserve was calculated and tabulated by zone and reserve classification:

Geological Reserve Summary Table

ZONES	TONNES	Cu%	Pb%	Zn%	g/tonne Au	g/tonne Ag	$US/ TONNE NSR
DRILL INDICATED							
H,AB_2	2,514,225	1.82	1.14	7.16	2.81	106.46	112.71
AB_1	206,269	0.71	1.72	9.34	2.17	144.37	111.25
G	535,905	0.92	0.99	4.20	2.23	82.53	73.52
TOTAL	3,256,399	1.60	1.16	6.81	2.68	104.92	106.17
DRILL INFERRED							
H,AB2	3,152,707	1.67	1.07	7.11	2.64	98.77	107.73
AB1	441,078	0.71	1.76	9.57	2.27	156.17	115.11
G	942,792	0.88	0.95	3.97	2.14	79.23	70.11
TOTAL	4,536,577	1.42	1.11	6.70	2.50	100.29	100.63
OLD RESERVE	707,616	1.30	1.60	8.00	2.40	116.50	109.56
TOTAL ALL CLASSES	8,500,592	1.48	1.17	6.85	2.56	103.41	103.49

6.8 Exploration Potential

The Tulsequah Mine area remains highly prospective for addition of new reserves. The main deposit of the H-AB and G zones remains open to depth in both the Central and Eastern Mine Blocks. Strong footwall alteration and thickening of the Lower Felsic sequence towards the south and west suggest that extensions of the main sulphide horizons or new lenses may exist in the west portion of the Central Mine Block and in the West Mine Block. In this area the F Zone anticline west limb trends towards the south. Thickened sections of the sulphide zone may occur in the nose of the F Zone anticline or in additional parasitic folds to the west and south.

The presence of economic sulphide mineralization in the Upper Felsic Horizon (Unit 4) has been demonstrated by the I Zone mineralization in the East Mine Block. Intense and widespread alteration in the Upper Felsic Horizon has been noted in the vicinity of the 5200 level portal near the Chief fault. This area has been inadequately tested by two holes and merits additional drilling and geophysics.

Outside of the Tulsequah Chief Mine are a number of prospective targets related to the extensions of the ore-bearing felsic horizons. The most outstanding of these is the Big Bull Mine area which has not been drilled since termination of production in 1957. The remainder of the lower felsic belt between the Tulsequah Chief and Big Bull mines has only seen reconnaissance mapping and minor prospecting. Grid emplacement, surface mapping and geophysical surveys are required to define potential sulphide mineralization along this belt. In similar fashion the upper felsic formation west of Big Bull and south of the Tulsequah Chief mine also merits reconnaissance exploration.

7.0 MINING

7.1 General

Assessment of mining and ground conditions as they relate to this study was largely derived from discussions with previous operators and with representatives of Cambria Geological Ltd., managers of the Tulsequah Chief exploration program during 1991 and 1992. Snow accumulation precluded an underground tour at the time of this study and formalized geotechnical data is sparse.

A combination of shrinkage stoping and blasthole stoping was done in previous operations between 1951 and 1957. In general rock conditions were reported to be fairly competent, and by comparison are considerably more favorable than those at the otherwise similar volcanogenic massive sulphide deposits of Westmin Resources Limited's Myra Falls operations on Vancouver Island.

7.2 Mining Methods

This study is predicated on a combination of large diameter blasthole/VCR stoping (73 percent) and shrinkage stoping (27 percent). Shrinkage and blasthole stope outlines were blocked out on a working set of level plans spaced at 50 meter vertical intervals. Selected conceptual level plans are presented in Figures 6, 7, and 8. Criteria for designating ore blocks into shrinkage or blasthole categories included ore widths and hangingwall rock type.

Shrinkage stopes were indicated where a rock unit designated 2a by Cambria containing gypsum and barite was present in the hangingwall and/or where average widths less than four meters were encountered. Pattern rockbolting using a 1.2 meter x 1.2 meter pattern for shrinkage stope hangingwalls was previous practice and has been included in the operating cost estimate. Shrinkage stopes were laid out with a central access raise driven to the next level and one "in stope" timbered manway. Good ore widths and apparent good continuity coupled with an ore density factor of 3.6 should provide good stoping productivity estimated at 50 tonnes per manshift, which includes drilling, blasting, ground support, the mucking of swell to a nearby orepass or remuck station and the installation of a timbered manway.

Blasthole stopes are generally assumed for wider ore sections and were laid out as 30 to 40 meter long primary stopes separated along strike by 30 meter long secondary stopes. Due to deformation of the ore bearing strata and experience of previous operators, it was deemed prudent to include provision for cablebolting the hangingwall from drill sublevels and hangingwall crosscuts excavated specifically for that purpose. Primary stopes will be filled with weakly cemented classified mill tailings and mine development waste prior to extraction of adjacent secondary stopes. In order to maximize ore recovery and reduce possible backfill dilution in secondary stopes, the mining plan incorporates intermediate extraction levels designed to cut the height of secondary stopes to half that of primary stopes.

Vertical spacing of sublevels for blasthole drilling will be a function of the regularity of ore contacts and final determination of the degree of regularity cannot be made without completing closely spaced definition drilling and/or ore development headings. For the purpose of this study, drill sublevels were considered to be spaced at 45 meter vertical intervals (sill to sill) and to be slashed to the contacts where the ore was less than 7 to 8 meters in horizontal width. In wider sections parallel ore headings separated by narrow pillars will be excavated. Sublevel configurations are designed to allow a predominance of parallel production blastholes rather than ring drilling.

Production blastholes would be of sufficient diameter (estimated at 165mm) to reduce deviation to acceptable limits. Crater blasting techniques would be used to open slots. It is envisaged that following creation of slots in primary stopes, the remainder of the stope will be blasted in a conventional blasthole manner, excavating vertical slices. Secondary stopes could require more support of the hanging wall and footwall and cemented fill pillars by broken ore. Therefore it has been estimated that most blasting in secondary stopes will be of VCR type, excavating horizontal layers of ore and drawing off swell only as required prior to final muck drawdown.

7.3 Rock Quality and Geotechnical Work

RQD measurements which were taken on some core holes indicate "fair" wall rock conditions. Formal geotechnical studies have not been initiated. Therefore many of the assumptions regarding rock quality used in this report are based on discussion with former staff members of the Cominco operations during 1951-57. It is reported that in general, open shrinkage and blasthole stopes stood up quite well with the exception of hanging wall sloughing where rocks contain significant amounts of barite or gypsum. Pattern rockbolting of shrinkage stope walls was standard practice in previous operations. Shrinkage stopes were typically 30 meters high (equivalent to the old level interval) and at least one stope had a strike length of 150 meters with no internal pillars indicated. Existing drift and crosscut headings are reported to be largely unsupported except where local timbering was required through a few faults and shear zones.

Based on the information available, vertical stope height for primary blasthole stopes above the -500 m elevation was estimated at about 90 meters with sill pillars separating stopes in the vertical plane (see Figure 9). For the purpose of this study, these sill pillars were deemed to be non-recoverable. Below the -500 meter elevation primary stope heights were arbitrarily reduced to 45 meters to compensate for anticipated increases of insitu stress. Secondary blasthole stopes and shrinkage stopes were designed to have 45-50 meter stope heights throughout. In actual practice, stope height should vary with local conditions.

Backfilling of about 50 percent of stoped voids on a delayed basis was included in the estimate as it is believed that fill will be required to stabilize wall rocks, thus maximizing ore recovery and minimizing dilution both in secondary blasthole stopes and in areas where mining of parallel ore lenses is called for.

7.4 Mineable Reserves, Dilution and Recovery

The diluted mineable ore reserve is estimated to contain 6,932,500 tonnes grading 1.40% copper, 1.07% lead, 6.42% zinc, 2.40 grams per tonne gold, and 93.37 grams per tonne silver. In addition to this reserve, a lower grade extractable mineral inventory containing 785,500 diluted tonnes grading 0.42% copper, 0.61% lead, 2.69% zinc, 1.27 grams per tonne gold, and 59.75 grams per tonne silver exists. The low grade inventory forms part of the geological ore reserve, but has been excluded from the mineable reserve because, after factoring in dilution, NSR values for this material are unsatisfactory at current metal prices. The low grade inventory is located in H-AB_2 zone - holes 90-24 and 91-30; in G zone - holes 88-4, 88-5, and 92-44, and in AB_1 zone - hole 91-31. An increase in metal prices could result in mining of parts of the low grade inventory and some sill pillars considered uneconomic in this study.

Mineable ore reserve outlines were based on geological interpretations by Cambria Geological Ltd. and include both drill indicated and drill inferred ore. Additional diamond drilling is required to upgrade all of these reserves into "probable" or "proven" categories. Tonto Mining estimates that doubling of the existing number of core holes penetrating the ore bodies could be required to firm up the ore reserve. The cost of this drilling has been included in this estimate as part of a third drilling stage, ie. stope definition drilling.

Several areas occur where discrete ore lenses separated by narrow (3 to 5 meters) bands of waste were interpreted. Some of these areas might not be mined as separate open stopes. Examination of these areas on an individual basis was done to either (i) include both ore lenses and intervening waste into the mineable reserve, or (ii) exclude the less attractive ore grade lens as unmineable.

In general, sill pillars were treated as unmineable although in practice it is possible that some of them could be recovered economically if ground conditions are better than anticipated. Mining extraction in mineable lenses was estimated on an individual block by block basis and averages about 85%. In some cases, ore pillars were left adjacent to major faults and dikes.

Those portions of the "old" Cominco reserve which represented pillars in old stopes were treated as unmineable for the purposes of this study and the "old" Cominco reserve was downgraded to a total of 424,350 diluted tonnes located in areas largely undisturbed by previous mining.

Dilution in shrinkage stopes was estimated at 20 percent by weight based on reserve tabulations compiled by previous operators. Dilution in blasthole stopes was assessed on a section by section basis and is estimated generally at 15 percent by weight (20 percent by volume). Locally, blasthole dilution was raised to 20 percent by weight in the vicinity of major faults or dacitic dikes crosscutting the orebody.

Average grade of dilution material was calculated to be 0.11% Cu, 0.11% Pb, 0.57% Zn, 0.42 g/t Au and 9.12 g/t Ag.

7.5 Mine Design and Planning

A conceptual mining plan has been developed to provide a basis for cost estimating. Several other options for mining exist and should be examined more thoroughly during final feasibility studies and detailed mine design engineering.

Major mine access will be via a decline collared near the conceptual mill location (see Figure 4). The proposed mill site/decline collar location will be located at about the +100 meter elevation on a gently sloping flank of Mount Eaton. The location is currently covered by standing forest and appears to be protected from any potential avalanche threat by a "nose" on the mountain slope above the site.

It is envisaged that preproduction mine development between the "0" elevation (sea level) and -150 meter elevation will be completed from the main decline prior to collaring of a production hoisting winze. Some rehabilitation, slashing, and extension of the existing 5400 level (+100 meter elevation) and 5200 level (+50 meter elevation) are required for development of the H zone and G zone orebodies. These levels will be maintained as "tracked" access openings.

Figures 5, 6, and 7 indicate a generalized development layout. The main access ramp from the future winze location to the mill will be driven sufficiently large in cross section to house a conveyor as well as allow trackless equipment to pass.

During the first two full years of operation, ore will be trucked up the ramp in order to provide positive cash flow to help finance shaft sinking and associated mine deepening operations. Most of the initial mine production will come from the 'H' zone between the +100 m elevation and the -150 elevation which will help to maximize head grade. Minor amounts of ore will come from the better 'G' zone areas and 'H' zone ore development below the -150 meter level.

Preproduction development ore is expected to provide two months of mill feed as a cushion to help start up.

Should the shaft sinking/mine deepening program run beyond design time schedule (Figures 10 and 13), additional ore could be mined and trucked from H zone below -150 elevation and from G zone.

The mine deepening program includes sinking and equipping of a timbered winze, 2.75 m x 6.1 m in cross section and collared at approximately the -75 m elevation. Crosscuts from the winze to the orebody at the -350, -500, -650 and -700 meter elevations will be excavated. A crusher station near the footwall of the orebody at the -700 elevation will be established. Conveyors at -700 elevation and in the main ramp between the winze and mill will be installed. Drilling and reaming of three series of boreholes to provide ventilation return airways, orepasses and wastepasses will be done as well. The winze is designed primarily as a three compartment hoisting facility with two cage over skip combinations. Following commissioning of the winze and underground crushing and conveying system, all ore will be fed through the central orepass system to the crusher, and underground trucking will be required only for haulage of mine development waste rock.

The footwall ramp will be developed downwards as the main access corridor in advance of production mining, and for the purpose of this study has been included as a stope development expenditure.

Some consideration was given to alternate mine layouts. Use of trackless equipment without extension of the footwall ramp could be effected by excavating a winze with a sufficiently large cross section (3.5 m x 6.1 m) to lower large pieces of equipment. It is likely that level intervals would have to be established every 50 meters for this option, and there would be less flexibility to vary stope heights and provide access to the orebodies at various elevations when compared to the layout selected. The second option could reduce development costs by up to one million dollars over a six year period, but front end costs for sinking a larger winze are estimated to be $2 million higher than for option 1.

A third option could be the use of track mounted equipment which would also preclude extension of the footwall ramp.

These options are worthy of further consideration at the time of final feasibility, and should take into account the likely future requirements of the mine over and above the current drill indicated and inferred reserves.

Definition diamond drilling is required to sufficiently delineate ore zones and determine stope boundaries for blasthole mining. This diamond drilling would largely be done from drill stations along the footwall decline and from footwall haulage drifts located at the -350, -500, and -650 meter elevations. These haulage drifts will be excavated during production year two as part of the mine deepening program. For the assumed sublevel spacing, it is estimated that 3500 meters of additional drilling is required per 50 vertical meter block. This translates to approximately 125 ore tonnes per meter drilled.

For the mining option selected in this study, mine ventilating air will be forced into the main ramp, downcasting via the winze and footwall ramp, exhausting initially through a series of boreholes near the extremity of the G zone, thence through the abandoned Cominco winze and old upper mine workings.

Water inflow quantities into the upper workings of the old mine are estimated to be in the range of 12 to 15 liters per second and are reported to occur largely because of surface waters seeping into near-surface workings. It is also reported that most of this drainage could be collected by construction of a small dam on the old 5900 level from where it could be fed by gravity lines to waste water disposal.

Future water inflow quantities into the new mine workings are unknown. For estimating purposes, the mine dewatering system is designed to handle about 25 liters per second. Mine discharge water will in all likelihood be acidic, sediment laden, and unsuitable for immediate discharge into the environment. It is proposed to introduce mine discharge water into the mill tailings line, then treat reclaim water as required.

An underground shop located near the winze collar has been incorporated into the cost estimates.

7.6 Mine Construction and Production Schedule

A general mine construction and production schedule is shown in Figure 10.

Mine production is estimated at 2250 tonnes per day based on ground conditions sufficiently competent for mining open blasthole stopes. It is believed that higher throughputs are not likely achievable unless additional ore reserves can be developed along strike resulting in more tonnage per vertical meter and additional numbers of working headings.

The source of millfeed is estimated to be derived of 11 percent from development, 24 percent from shrinkage stoping, and 65 percent from blasthole stoping.

Blasthole stopes are estimated to yield 1000 tonnes per day each when in full production, and shrinkage stopes an average of 200 tonnes per day each over the stope life.

The mine operating schedule is based on crews working a four week on the job and two week out turnaround; thus three full crews are required. Two eight hour shifts per day and a seven day work week are planned.

7.7 Mining Equipment

Mining equipment includes a fleet of mobile trackless and minor track equipment consisting of the following:

- Down-the-hole hammer production blasthole drill (1)
- Two boom hydraulic development drill jumbo (2)
- Rockbolt/cable bolt drill jumbo (1)
- 6 Cubic Yard LHD (4)
- 3.5 Cubic Yard LHD (1)
- 30 Ton Underground Truck (5)
- Underground Road Grader (1)
- Two passenger underground utility vehicle (2)
- Eight passenger underground utility vehicle (1)
- Jackleg/Stoper Drill (15)
- Auxiliary Ventilation Fan (5)
- 1 1/2 Ton Battery Locomotive (2)
- Various electric submersible pumps (8)
- Miscellaneous tuggers, slusher hoists, flatcars

The equipment has been considered as new purchases except for four 30 ton trucks, the road grader, battery locomotives, tuggers and slusher hoists which are considered to be reconditioned. Several of the trucks could become redundant after the initial two years of operation unless required for exploration drives not related to the current ore reserve.

It has been assumed that one of the underground diamond drills currently at the site would be available for definition diamond drilling.

Stationary equipment installed in the mine includes the following:

- Main exhaust fan (175,000 - 200,000 cfm) (1)
- Main dewatering pump (4)
- 42 inch x 48 jaw crusher (1)
- Conveyors (1070 mm) (approx. 1400 lineal meters)
- Double drum production hoist (1000-1200 hp; (1)
  complete with sheaves, dump scrolls,
  loading pocket hardware, ropes and conveyances)

The hoist is assumed to be purchased as a reconditioned unit and installed by the shaft contractor to be used during sinking operations. A reconditioned jaw crusher in addition to the one listed above is assumed purchased for temporary surface duty during production years one and two and is included in the mill cost estimate. The reconditioned jaw crusher would subsequently become available for any mine expansion or for ore from outside sources such as the Big Bull mine.

7.8 Auxiliary Equipment and Services

Average compressed air requirements for the mine equipment are estimated to be 2200 c.f.m. Two 2200 c.f.m. electric compressors (one operating plus one for standby and unusual loads) have been estimated for mine requirements. The mill would have its own dedicated compressor. Mine compressors would be located in one bay of the powerhouse building.

Mine ventilating air will require heating during the coldest winter months. An 8 to 10 million Btu per hour oil fired mine air heater located near the main portal is estimated for this purpose.

Mine process water will, in all likelihood, be drawn from one or more of the three creeks flowing into the Tulsequah River within 1 1/2 kilometers of the portal. There is also a possibility that natural inflow to the upper levels could be tapped as a source of mine process water. Due to the topography, a gravity feed system is planned.

7.9 Waste Rock Disposal

Separation and disposal of acid generating waste rock will be necessary. Significant differences in acid generating potential occur between the various strata and more testwork should be done to clearly identify those strata which are acid consuming or acid neutral. Prior to the commencement of preproduction, a clear program aimed at physically separating acid generating waste rock from non-acid generating waste rock should be instituted. Wherever practical and possible, mine development should be laid out in rocks with low acid generating potential.

A surface waste rock disposal area is suggested in Figure 4. This would, in reality be two separate areas - one for acid generating rock, and one for non acid generating rock. Storage of waste materials should become the responsibility of an environmental officer.

Subsequent to completion of initial stoping, backfill will be introduced into the mine. It is recommended that following that point in time, all acid generating waste rock excavated in the mine be dumped into stoped out voids to form a portion of the backfill material . In addition to this plan, an allowance has been made in the cost estimate for trucking acid generating waste rock produced during the initial phase back into the mine. This would include any deleterious rock left on surface by previous operators.

Upon final abandonment of the operation, the mine would be flooded thereby eliminating any ongoing acid generation problem resulting from mine waste rock.

7.10 Surface Facilities

Surface facilities for the underground mine will be kept to a minimum. The changehouse, dry, safety and mine rescue equipment, first aid station, and offices for mine supervision and technical staff will be located in a general service building adjacent to the mill/portal location.
The bulk of repairs to underground equipment will be carried out in the underground service shop A small repair shop in the general service building will primarily be used for repairing surface equipment.

7.11 Mine Capital Cost Estimates

Mine capital requirements are estimated as follows:

Item	Estimated Cost	Time Frame
Preproduction Mine Development	$19,880,000	During 16 mos. prior to production
Mine Equipment (1)	4,504,000	During 12 mos. prior to production
Mine Equipment (2)	2,630,000	Production year No. 2
Spare Parts	811,000	During 3 mos prior to production
Winze Excavation & Equipping (incl.hoist)	13,233,000	Production year 1
U/G crush-convey, main pumping, orepass, wastepass, ventilation systems	6,513,000	Production year 2
Equipment Replacement	6,400,000	Prod. years 2 & 3 @ 700,000 pa Prod. years 4-8 incl. @ 1,000,000 pa
Total	**$53,971,000**	

Preproduction capital constitutes $25,195,000 of the above.

The foregoing excludes any capital expenditures devoted to further exploration in the mine area on the basis that such expenditures will be written off against future additional ore reserves. The mine plant is sized to permit modest amounts of exploration drifting and diamond drilling without further expansion.

The preproduction mine development cost of $19,880,000 is distributed as follows:

. Contractor mobilization/demobilization	$450,000
. Contractor site set up	90,000
. Contractor direct costs including markup	
- decline portal trench & structure	150,000
- 1,200 m conveyor decline	1,686,000
- 4,695 m drift/ramp	5,352,000
- 600 m drawpoint headings	666,000
- 430 m track drift	489,000
- 1,100 m track drift rehab	132,000
- 470 m conventional raise	416,000
- Raisebore mobilization	65,000
- 170 m raisebore	213,000
- 6,250 m^3 misc. excavation	375,000
- 10,000 m diamond drilling	400,000
- install permanent ventilation bulkheads	20,000
. Contractor indirect costs including markup	8,841,000
. Equip underground shop	75,000
. Cable bolt 2,000 m	40,000
. Production blasthole drilling 2,000 m	76,000
. Company staffing during preproduction period	344,000
TOTAL	**$19,880,000**

7.12 Mine Operating Cost Estimate

Mine operating costs are estimated as follows:

Item	Cost/Annum	Cost/Tonne Mined
Definition diamond drilling	$ 211,000	0.26
Stope development	2,476,000	3.10
Stoping & backfilling	6,683,000	8.35
Mucking, crushing, hoisting & convey.	1,672,000	2.09
Mine services & construction	1,389,000	1.74
Equipment maintenance labour	1,408,000	1.76
Disposal of acid-gen. waste rock	135,000	0.17
Supervision, technical & clerical	1,266,000	1.58
Mine ventilation air heating	250,000	0.31
Total	**$15,490,000**	**$19.36**

Equipment repair parts are included under the appropriate mining activity. Electrical power is listed as a separate cost item in Section 9 - Infrastructure. Direct labour is estimated to comprise 46 percent of the above totals.

There is no provision for further mine exploration within the above listed operating costs, by reason that such costs will be written off against resulting reserve additions.

7.13 Mine Operating Workforce

	On Site	Total Workforce
- Supervision	4	6
- Technical, clerical	8	12
- Miners	20	30
- Mechanical, electrical	13	19
- Operators, construction, diamond drillers, labourers	24	37

7.14 Discussions and Recommendations

Mine operating costs have been carefully compiled based on certain assumptions regarding ground conditions and water inflows. It is recommended, that prior to final feasibility, more detailed geotechnical assessment including a rock mass quality investigation be carried out. The operating cost estimate which at this time is estimated within +/- 20% accuracy could be refined to +/- 10-15% accuracy pending results of such investigation.

Capital costs, estimated to +/- 20% to +/- 25% accuracy at this time are also based to a large extent on assumptions regarding rock conditions and water inflow. Another variable affecting capital costs is the configuration of the surface plant layout insofar as it affects haulage requirements and logistics. A detailed site investigation verifying mill, service buildings and portal locations is in order. The site investigation should identify environmental concerns, soil stability, surface water courses, effects of snow and rain, and ease of local access.

8.0 MINERAL PROCESSING

8.1 Introduction

This property was previously operated, at 500 tons per day, by Cominco Ltd during the period 1951-57. The metallurgical reports which were published during this period represent an interesting historical review. However, the metallurgical response of the current lower Tulsequah deposit is better than that of the upper Tulsequah and Big Bull deposits, which were previously comingled.

The purpose of this section of the report, and a separate laboratory study by Beattie Consulting Ltd, is to provide a technical review of the process options in light of changing technology and processing economics subsequent to the Cominco operation, and to address only the lower Tulsequah deposit.

The financial aspects of this section provides capital (+/-20%) and operating cost estimates for the proposed 2,250 tonne per day concentrator.

8.2 Historical Review

Published literature (see references 1 and 2) describes the first 2 years of the Cominco operation in considerable detail.

The reports indicate that only 70 to 76% of the three base metals reported to their respective flotation concentrates although essentially 90% of each metal was "recovered".

Although the historical review is interesting from a technical perspective, the metallurgical response of the current lower Tulsequah Chief deposit differs from the blended upper Tulsequah Chief/Big Bull ores. As well, the future ore responds well to primary differential lead/copper flotation so that a larger proportion of the gold reports to the lead concentrate than was previously the case.

In the previous operation, 3 - 4% of the gold was lost in the flotation tailing solution due to dissolution by sodium cyanide which was used for selectivity control.

The historical reports indicated that very coarse gold was frequently collected in the rod mill discharge launder, suggesting the benefit of a gravity gold recovery circuit within the grinding circuit.

8.3 Mineralogy

The mineralogy of the lower Tulsequah deposit has been described in a 1991 microscopy study, by Cominco (reference 5).

The sulphide minerals: in order of decreasing abundance, include:

pyrite
sphalerite
chalcopyrite
tennantite
galena

The gangue minerals are dominated by barite, with lesser sericite and quartz.

The 1992 Bacon, Donaldson & Associates (BDA) report (Reference 4) indicates the following analysis of the feed sample:

Element	Assay
Au	2.6 g/t
Ag	133. g/t
Cu	1.6 %
Pb	1.3 %
Zn	8.5 %
Fe	12.6 %
As	0.1 %
Sb	0.06 %
S	17.9 %

Gold is present as both electrum and native gold. Silver occurs as argentiferous tennantite.

8.4 Process Metallurgy

8.4.1 Metallurgical Testing

A laboratory study, by BDA in 1992 (reference 4), indicates that the copper and lead flotation metallurgy will be less challenging in the proposed operation than was experienced by Cominco, presumably because the lower Tulsequah Chief ore is more amenable to processing than was the blended upper Tulsequah Chief and the Big Bull open pit ore.

Three bench scale tests were performed on diamond drill core reject material from the Redfern exploration program. Although the testing program was not sufficiently detailed to provide an emphatic metallurgical statement, it would appear that +80% of each of the copper and lead will report to their respective concentrates grading 25% Cu and 60% Pb, respectively.

Additional, and much more comprehensive testing was performed in 1993 by Beattie Consultants (reference 6), which demonstrated the following:

(1) The ore responds to differential flotation to sequentially produce lead, copper, and zinc flotation concentrates.

(2) Approximately 40% of the gold reports to a rougher gravity concentrate. Although the ultimate gravity circuit product could not be determined due to the small size of the sample which was tested, it is anticipated that at least a high grade saleable gravity concentrate will be produced. Ideally, this product will be upgraded at the site to produce bullion.

(3) Gold dissolution during flotation, due to the addition of sodium cyanide, was reported to be 1.5%. This is significantly lower than the 4% which was reported during the previous operation.

(4) The flotation process control is anticipated to be less challenging than that of the former operation due to the ease with which the lead is floated away from the copper during the rougher stage.

(5) The zinc concentrate grade, at 50% Zn, is relatively low, almost entirely due to dilution from pyrite. Subsequently, optical microscopy indicated that the pyrite is essentially free, so future testing is likely to produce zinc concentrates exceeding 55% Zn with only a minor zinc loss.

8.4.2 Metallurgical Flowsheet

The role of gold in the future flowsheet has a much greater effect on the economics than it did in the 1950's, so there is a considerable incentive to maximize gold recovery into either bullion or high grade gravity concentrates.

Cominco's use of gravity concentration within the grinding circuit, followed by differential flotation to produce separate copper, lead, and zinc concentrates was appropriate, and will remain so. However, more effective gravity concentrators are now available than was the case in the 1950's, and should be investigated on startup of the plant. This investigation should include the use of centrifugal concentration on the flotation feed in an attempt to recover fine, free gold.

8.5 Process Flowsheet

The process flowsheet differs significantly from the former operation, due to changes in process machinery and the different flotation response of the lower Tulsequah orebody.

8.5.1 Crushing

For the first two years of operation, minus 900 mm (36") ore will be hauled to the surface jaw crushing plant where it will be reduced to minus 250 mm (10") with a 1.1 m X 1.2 m (42" X 48 ") jaw crusher.

Subsequently, an underground hoist will be in service, and ore will be crushed underground to the same product size.

Since the mine will operate 7 days per week, and since the crushing will be relocated underground within 2 years, it is proposed that the coarse ore inventory be contained in a 500 tonne capacity bin, and that crushed ore be inventoried in a covered stockpile with a live capacity of 5,000 tonnes (total capacity of 15,000 tonnes with dozing).

The jaw crusher has been sized based upon the relatively coarse anticipated feed, and will result in a relatively high crushing rate of 500 dmt/hr. Once the mine has operated for a few months, the as-mined rock size can be better assessed, and perhaps the size of the proposed underground jaw crusher can be altered.

Subject to a better analysis of the site snow conditions, it is proposed that the crushed ore storage be of the covered A-frame type with open ends for dozer access.

8.5.2 Grinding

The high sulphide content of the ore suggests that it will be amenable to SAG mill/ball mill grinding. An allowance has been made for the addition of steel balls to the "auto" mill, although the media consumption rate is somewhat speculative.

This circuit will operate with two grinding mills: a 1,200 HP 6.4 m X 2.1 m (21' X 7') auto mill and a 1,000 HP 3.4 m X 4.3 m (11' X 14') ball mill.

The grinding circuit cyclones will feed directly to the lead flotation circuit.

8.5.3 Gravity Circuit

Since it is somewhat difficult to fully design a gravity circuit based upon 2 kg laboratory batch tests, it it proposed that sufficient equipment be provided in the completed plant to "maximize" gold recovery in the gravity concentrate or concentrates.

A jig will be installed in either or both the auto mill discharge or cyclone underflow stream. The rougher concentrate will be tabled continuously to produce a cleaner gravity concentrate, which will be cleaned on day shift only, to produce a gravity concentrate(s).

As well, a rougher centrifugal concentrator will be installed on the flotation feed stream. A second stage centrifugal concentrator will clean the concentrate producing a site smeltable concentrate or a high grade saleable concentrate.

8.5.4 Flotation Circuit

Sequentially, lead, copper, and zinc will be floated into separate saleable concentrates.

The reagent requirements are presented in 8.7 "Design Criteria", and are typical for this type of circuit.

The flotation machines will consist of tank cells for rougher service and column cells as cleaners.

8.5.5 Dewatering

Previously, the concentrates were thickened and vacuum filtered, then stored in uncovered bins at the site to accommodate the annual 4 month shipping season.

In all likelihood, at least some of the previous concentrate moisture contents were in excess of the safe transportable limit for ocean freight. It is therefore proposed to dry both the lead and copper concentrates with waste heat from the generators and to pressure filter the zinc concentrate.

The high grade gold gravity concentrate will be vacuum filtered, then bagged in 1.3 tonne woven poly bags for shipping.

Based upon an average generator load of 6.4 megawatts, 2,200,000 btu's per hour of heat is available in the stack system for concentrate drying, and a similar amount is available in the cooling jackets. The stack waste heat is capable of removing 0.4 - 0.6 tonnes of water per hour, which is sufficient to reduce the lead and copper concentrate moisture contents to about 6%. This suggests that some of the stack heat would be available for partial zinc concentrate drying.

The forecast concentrate moistures are shown below:

Product	Filter Cake Moisture		Dryer Product
	Cominco	Forecast	
Copper conc	10.6%	10.6%	6.0%
Lead conc	9.0%	9.0 %	6.0%
Zinc conc	8.8%	7.5 %	no drying

Concentrate Production Forecast

Product	Production DMT	
	DMT/day	DMT/Year
Gravity conc	2.2	820
Copper conc	104	37,800
Lead conc	36	13,100
Zinc conc	259	94,400

8.5.6 Hydraulic Backfill

The mining procedure will benefit from the placing of backfill equivalent to 50 % of the removed volume, or 35 % of the mined tonnage.

It is proposed that this be provided by removing a pyrite concentrate within the flotation circuit as well as operating a modest hydraulic backfill cycloning circuit on the flotation tailing slurry.

This will considerably diminish the acid generation potential of the material which reports to the tailing pond, while depositing the majority of the sulphides into sites which will be below the water table on abandonment.

The composition of the backfill will be as follows:

Product	Quantity - tonnes/day	% Mill Feed
Pyrite Conc	400	18
Cycloned fill	380	17
Total	780	35

Although the grinding requirement is for a relatively fine 75% - 200 mesh, the cycloned fill demands are modest. The circuit is anticipated to be able to produce the required quantity of fill at a typical percolation rate exceeding 75 mm/hour. This will be done by passing the entire flotation tailing stream through a primary hydrocyclone circuit, followed by a second stage for fine particle removal.

The laboratory testing has indicated that the normal operation of the flotation circuit will produce a pyrite "concentrate", which will provide suitable feed for backfilling. It is, therefore proposed that no additional production facilities be provided for pyrite, but that a 6 m (20') diameter thickener be installed for increasing the density of this product to 55 - 60 % solids.

The mining schedule will require no backfilling for the first 6 months of operation, but subsequently, filling will be continuous and without any requirement for surface storage of backfill.

A cement silo and feeder will be provided for addition to the backfill as required.

8.5.7 Tailing Disposal

The flotation tailing (4,700 cu m/day or 860 USGM) will be fed by gravity through a 3 km, 200 mm (8") diameter polyethylene pipe to a side-hill type tailing pond.

The pond will be equipped with a second 200 mm (8 ") polyethylene reclaim pumping system. The reclaim pumping system will be located on the mill side of the pond on original ground and will require periodic relocation.

It is anticipated that the mine discharge water will also be fed to the tailing pond at 1,400 cu m/day (200 - 300 USGM). This may require a separate discharge line.

8.6 Metallurgical Statement

The following metallurgical forecast is based on the referenced Beattie Consulting report, subsequent optical microscopy and ongoing laboratory testing adjusted to reflect the somewhat lower forecast mine grade.

Product	Wt %	Assays g/t		Assays %			
		Au	Ag	Cu	Pb	Zn	Fe
Gravity conc	0.087	800	1,070	-	?	-	-
Copper conc	4.7	5	1,029	25.7	0.3	5	28
Lead conc	1.6	65	1,511	5.0	60	5	6
Zinc conc	9.8	1	76	0.4	0.1	57	7
Tailing	83.8	0.4	15	0.09	0.08	0.5	11
Feed	100	2.4	93	1.40	1.07	6.42	10

Product	Wt Distribution %					
	Au	Ag	Cu	Pb	Zn	Fe
Gravity concentrate	29	1	-	-	-	-
Copper concentrate	10	52	87	2	5	14
Lead concentrate	43	26	5	91	1	1
Zinc concentrate	3	8	3	1	87	7
Subtotal concentrate	85	87	95	94	93	22
Tailing	15	13	5	6	7	78
Feed	100	100	100	100	100	100

The above forecast is predicated on achievement of a zinc concentrate grading 57% with 87% zinc recovery to the zinc concentrate.

8.7 Design Criteria

Production Criteria

Annual processing rate	800,000	dmt
Monthly " "	68,000	"
Daily " "	2,250	"
Annual operating time	94 %	
Operating week	7 days per week /24 hr per day	

Crushing

Coarse ore bin	500	dmt
Feed size	900	mm
Product size	250	mm
Ore SG	3.4	
Bulk density	2.1	dmt/cu m
Ore moisture	4	%
Crushing rate	500	dmt/hr

Grinding

Product size	P80 = 100 microns	75 % pass 200 mesh
Work Index	13 kwh/tonne (est)	
Ore SG	3.4	
COF density	35 % solids	

Grinding Steel Requirements

Auto mill (0.11 kg/kwh @9.5 kwh/t)		1.05 kg/t
Ball mill (0.07 " @	8.0 ")	0.56 "
Total	7.5	1.61 "

Conditioning/Flotation Circuit

Lead conditioning	5 minutes
" rougher flotation	12 "
" 1 st cleaner flotation	? "
" 2 nd " "	? "
Copper conditioning	5 "
" rougher flotation	18 "
" 1 st cleaner flotation	? "
" 2 nd " "	? "
Zinc conditioning	10 "
" rougher flotation	15 "
" 1 st cleaner flotation	? "
" 2 nd " "	? "
" 3 rd " "	? "

Reagent Requirements

Lime	700	g/t
Sodium sulphite	1,300	"
Zinc sulphate	500	"
Sodium cyanide	215	"
Aeroflot 242	25	"
Sodium ethyl xanthate	120	"
MIBC	35	"
S5100	40	"
Copper sulphate	400	"
Dowfroth 250	12	"

Lead Rougher Concentrate Regrinding

Product size P80 = 20 microns : 85 % - 400 mesh

Concentrate Dewatering

Copper	Cake moisture	10.6 %
	Dryer product	6.0 %
Lead	Cake moisture	9.0 %
	Dryer product	6.0 %
Zinc	Cake moisture	7.5 %

Tailing Disposal

Flotation tailing density	40 % solids
Tailing terminal density	75 % "
Tailing specific gravity	3.0

8.8 Environmental

8.8.1 Acid Generation Potential

Both the ore and the tailing exhibit relatively high acid generation potential, as shown below:

Sample	S %	Acid Pot.	Neut Pot.	Net Neut Pot.	Paste pH
Ore (est)	18	590	3	- 587	
Tails (1)	14	447	3	- 444	7.2

Note: (1) Copper / lead / zinc tailing.

This characteristic must be recognized and addressed in the design stage to minimize both short and long term potential environmental damage and liability.

It is therefore proposed to recover most (est +85%) of the pyrite in the flotation circuit for use as hydraulic backfill. As well, a conventional hydrocyclone backfill circuit will be operated to augment the pyrite fill material. The solids which report to the tailing pond, will therefore represent only about 46 % of the mill feed weight and will have an anticipated ABA as shown below.

Sample	S %	Acid Pot.	Neut Pot.	Net Neut Pot.
Plant tlg	2	64	3	- 61

Although this material will retain an acid potential, it will be substantially diminished relative to the copper / lead / zinc flotation tailing.

8.8.2 Cyanide Destruction

Although sodium cyanide will be added to the flotation circuit, the addition rate is sufficiently low that the effluent solution to the tailing pond will not contain sufficient cyanide to justify the operation of a cyanide destruction circuit.

The Beattie laboratory testing program has indicated a cyanide concentration in the tailing solution of 0.005 ppm CNt.

8.9 Site Services

8.9.1 Power Supply

Power will be generated in a diesel electric power plant of which 2.8 mw (30/kwh/tonne) is attributable to the milling operation. The generators will be located adjacent to the concentrate dewatering circuit to facilitate use of the stack heat for concentrate drying.

Diesel fuel will be stored primarily at the Swede Point barge landing site, and only sufficient fuel for 2 day's operation will be stored at the millsite.

8.9.2 Fresh Water Supply

Domestic and process makeup water will be supplied to a steady head tank at the millsite by gravity from an adjacent creek at a rate of 820 cu m/day (150 USGM), of which 115 USGM will be for process requirements, the balance for domestic purposes.

This system will require the storage of sufficient "reclaim " water in the tailing pond prior to startup to satisfy the immediate water needs. It is estimated that 10,000 cu m will have to be impounded.

8.9.3 Reclaim Water Supply

Reclaim water will be pumped directly from the tailing pond to the concentrator reclaim water tank system.

This system will accommodate 3,500 cu m /day (640 USGM).

8.9.4 Assaying

Assaying will be performed using fire assaying for the higher grade gold and silver products and atomic adsorption for the balance of the metals.

It is anticipated that the assaying will not be complicated, but the workload will be relatively high because of the three product flotation circuit.

In addition to the normal sample preparation equipment, the assay laboratory will be equipped with a single fire assay furnace, a microbalance, and two AA units. The second AA unit will both provide backup capacity and improve the flow of multi-element analyses.

The anticipated workload in the assay office is as shown below:

Daily Assaying Requirements

Service Area	Au	Ag	Cu	Pb	Zn	Fe	Total
Mine	-	-	10	10	10	-	30
Geology	5	5	5	5	5	-	25
Mill	8	8	20	20	20	12	88
Total	13	13	35	35	35	12	143

The assay office will be used for environmental assaying as well.

8.9.5 Laboratory Testing

It is anticipated that during the startup period, an on-site metallurgical testing laboratory will be used to optimize the process metallurgy as quickly as possible.

8.10 Manpower

The proposed work schedule is a two week on and two week off rotation and 12 hour shifts. The estimated manpower requirements for the millsite are as follows:

Classification	Number	
Superintendent	1	
Assistant superintendent	1	
Operations foremen	2	
Shift lead hands	4	
Mill operators	12	
Metallurgist	2	
Clerk	2	
Subtotal operations	24	
Chief assayer	1	
Assayers / buckers	5	
Subtotal assaying	6	
Maintenance foreman	2	
Electricians	-	(included in plant budget)
Instrument mechanics	2	
Mechanics	6	
Subtotal	10	
Total	40	

8.11 Capital Cost Estimate

For purposes of this study, the capital cost has been based upon the construction of a newly designed and constructed plant using reconditioned equipment where possible.

A capital cost estimate provided by Proton Engineering and Construction, of Burnaby, B.C. at our request for comparison purposes, totals $ 30 million, equivalent to $13,000 / tonne per day.

An itemized capital cost estimate has been prepared by the writer for the proposed plant, as well which compares fairly well to Proton's estimate.

The possible purchase of the Westmin Premier mill has been suggested by Redfern. From a technical perspective, this is a viable option, since both plants are of about the same capacity and both utilize semi-autogenous grinding. The Premier cyanidation circuit will be redundant and would be replaced with flotation and concentrate dewatering circuits, for which ample room exists inside the Premier mill.

At some price, the Premier mill will represent an attractive acquisition for the Tulsequah project.

Another possible option is preconstruction of a barge-mounted mill. Considerable study beyond the scope of this report is required to determine a viable site for this option because of exposure to Tulsequah River outburst floods.

8.11.1 Capital Cost Summary

The capital cost estimate for the proposed 2,250 mtd concentrator is shown below.

Typically, the capital cost of the processing equipment represents 25 % of the capital cost of the milling plant. This factor has been applied in this analysis. This factor includes: design, procure, site preparation, construction, construction management, but not operating capital.

Due to the location, additional costs will undoubtedly be incurred for freight, transportation, employee costs, and equipment mobilization and demobilization. A location allowance of $ 2 million has been made for this consideration.

Item Description	$ C
Concentrator	
Equipment - coarse ore handling	345,000
" - grinding	2,900,000
" - gravity	150,000
" - flotation	605,000
" - dewatering	1,070,000
" - backfill circuit	125,000
" - tailing line	100,000
" - reclaim water system	81,000
Subtotal - equipment	5,376,000
Direct construction cost	21,504,000
Location allowance	2,000,000
Tailing dam (1)	3,680,000
Total	27,184,000

Note: (1) Of which $ 1,030,000 can be deferred to year 5.

8.11.2 Equipment

Crushing

Item Description		$
Coarse ore feeder		70,000
Jaw crusher - 42 " X 48 "		200,000
Belt conveyor - 36" X 150'		75,000
Subtotal - crushing		345,000

Grinding

Item Description		$
Vibrating feeders (2)		50,000
Belt conveyor		24,000
Autogenous mill - 21' X 7'	1,200 hp	2,000,000
Ball mill - 11' X 14'	1,000 hp	800,000
Slurry pumps		20,000
Cyclones		6,000
Subtotal - grinding		2,900,000

Gravity Circuit

Item Description		$
Jig		15,000
Table		15,000
Gravity Concentrations		120,000
Subtotal - gravity concentrators		150,000

Flotation

Item Description		$
Conditioning tanks		25,000
Flotation machines - lead		120,000
Flotation machines - copper		160,000
Flotation machines - zinc		200,000
Regrind mill - 6' X 6'		50,000
Pumps		20,000
Reagent mixing and feeding		30,000
Subtotal - flotation		605,000

Dewatering

Item Description	$
Thickener - lead conc	25,000
" - copper conc	40,000
" - zinc conc	70,000
Disc filter - lead & copper conc	40,000
Pressure filter - zinc conc	750,000
Dryers - lead & copper conc	90,000
Belt conveyors	40,000
Slurry pumps	15,000
Subtotal - Dewatering	1,070,000

Hydraulic Backfilling

Item Description	$
Thickener - pyrite conc	75,000
Cyclones	6,000
Slurry pumps	16,000
Cement silo	20,000
Cement feeder	8,000
Subtotal - backfill	125,000

Tailing Line

Item Description	$
Piping - 10,000 ft @ $ 10 / ft	100,000
Subtotal - tailing line	100,000

Reclaim Water System

Item Description	$
Pumps	75,000
Piping	6,000
Subtotal - reclaim water system	81,000

8.11.3 Tailing Dam

The proposed tailing site has sufficient capacity for in excess of 10 years operation at a maximum dam height of 9.5 m, of which 1.5 m will be freeboard.

Ideally, the tailing pond will be a "tight" construction so that on abandonment, the sulphide minerals will be deprived of oxygen and will not oxidize. This can be accomplished by either flooding the completed pond in perpetuity or providing a vegetative cover.

To mitigate against this potential difficulty, the pyrite in the flotation tailing, after the first 6 months of operation, will be used as underground backfill, where it will be flooded. The removal of this fraction, the hydraulic backfill, and the flotation concentrates will result in somewhat less than 50 % of the mill feed reporting to the tailing pond.

For purposes of this study, it is proposed that the pond be entirely lined, although if suitable till can be located on site, the lining can be deleted. This is an important cost item, which deserves careful consideration.

Data:

(1) Pond area	250,000	sq m
(2) Dam length	1,000	m
(3) Pond width - max	500	m
(4) Annual tailing	380,000	tonnes
(5) Settled density	1.9	tonnes/cu m
(6) Annual tailing vol.	200,000	cu m
(7) 10 year volume	2,000,000	cu m

Item	$ C
Site preparation	1,250,000
Membrane	750,000
Dam construction - year 1	650,000
Sub total - starter dam	2,650,000
Additional membrane - year 5	30,000
Dam construction - year 5	1,000,000
10 year total	3,680,000

Costs attributable to membrane installation include $780,000 for the membrane itself plus $700,000 additional site preparation costs due to the stringent base requirements for the membrane. A detailed site investigation may indicate that the pond integrity can be assured without utilization of a membrane.

8.12 Operating Cost

The following operating costs are summarized below for the proposed 2,250 mtd concentrator.

Item	$/tonne	$/year
Salaries (1)	2.67	2,200,000
Reagents (2)	3.49	2,870,000
Grinding media (3)	1.23	1,010,000
" mill liners (4)	0.12	98,600
Power (5)	-	-
Operating supplies	0.50	410,600
Maintenance supplies	1.00	821,200
Freight (6)	0.98	804,800
Total	10.02	8,215,200

Note:

(1) Mill labour, 40 employees at an average of $ 55,000 / year (ie. $ 20 / hr with 25 % loading). Includes the property assaying labour costs.

(2)

Item	$ / kg	Consumption g/t	$ / tonne
Lime	0.20	700	0.14
Sodium sulphite	0.87	1,300	1.13
Zinc sulphate	1.04	500	0.52
Sodium cyanide	1.50	215	0.32
Aeroflot 242	7.50	25	0.19
Sodium ethyl xanthate	2.50	120	0.30
MIBC	2.05	35	0.07
S5100	8.50	40	0.34
Copper sulphate	1.15	400	0.46
Dowfroth 250	2.01	12	0.02
Total		3,340	3.49

(3) Grinding balls

Item	$ / kg	Consumption g/t	$ / tonne
Auto mill (100 mm)		1,050	
Ball mill (50 mm)		560	
Total	0.77	1,610	1.23

(4) Mill Liners 10 % of grinding media.

(5) Power included in property - general costs @ $3.00/tonne.

(6) Freight

Item	$ / kg	Consumption g/t	$ / tonne
Reagents		3.3	
Grinding steel		1.6	
Total	0.20	4.9	0.98

8.13 Marketing Of Concentrates

Approximately 18 % of the original feed will report to the four concentrates, as shown below:

Parameter	Au Conc	Cu Conc	Pb Conc	Zn Conc
DMT/year	714	38,600	13,200	80,500
Au - g/t	800	5	65	1
Ag - g/t	1,070	1,029	1,511	76
Cu - %		25.7	5	0.4
Pb - %	+ 45(?)	0.3	60	0.1
Zn - %		5 *	5	57
Fe - %		28	6	7
S - %		25	19	36
As - %		0.9 *	1.1	0.01
Sb - %		0.2 *	0.2	< 0.01
Insol - %		< 3	< 5	< 3
Cd - %				0.2
Cl - %				
F - %				
Hg - ppm		30	6	130 *
Moisture		6.0	6.0	7.5

The concentrates appear to be of an adequate chemical quality, although some elements (see* above) will incur smelter penalties and not all smelters will accept the zinc concentrate because of the elevated mercury content.

The copper, lead, and zinc concentrates will be trucked in bulk to a dedicated barge site, at Swede Point and will be shipped by barge to a dedicated ocean shipping site near Juneau.

Net smelter return calculations have been based on the metallurgical statements presented in Section 8.6. Two sets of metal prices as tabulated below are compared in this section. The first column reflects current metal prices while the second column is predicated upon longer term average prices and forms the "base case" for sensitivity analyses presented in Section 12.

Metal	Current Price Case		Forecast Base Case	
Au	$US 338.00	per oz.	$US 375.00	per oz.
Ag	$US 3.90	per oz.	$US 4.00	per oz.
Cu	$US 0.90	per lb.	$US 1.00	per lb.
Pb	$US 0.35	per lb	$US 0.35	per lb.
Zn	$US 0.48	per lb	$US 0.60	per lb.

The gold concentrate, if bullion, will be flown to Vancouver for sale. The expected high grade second gravity concentrate will be bagged (1.3 tonnes/bag) and shipped to market.

The contribution of the various concentrates to the NSR is tabulated below.

	NSR - $C/DMT of ore (FOB Juneau)	
PRODUCT	Current Prices	"Base Case" Prices
Copper Conc.	27.05	$30.28
Lead Conc.	21.91	23.23
Zinc Conc.	36.69	51.78
Gold Conc.	9.16	10.03
TOTAL	$94.81	$115.32

Currency exchange $1.00 US = $1.265 for current case
= $1.25 for base case

8.14 Conclusions & Recommendations

8.14.1 Processing Plant

A 2,250 tonne per day gravity/flotation concentrator is proposed for this project.

The plant will utilize SAG/ball mill grinding followed by flotation to recover sequentially, lead, copper, and zinc concentrates.

Gold will be concentrated by gravity concentrators within the grinding circuit (using jigs and tables) and in the flotation feed slurry stream using a centrifugal concentrator. The recent industrial application of centrifugal concentrators will almost certainly result in the production of a greater proportion of the gold in some combination of high grade gravity concentrate and bullion. This is difficult to investigate at the laboratory scale, but the required equipment should be installed and optimized once the plant is in operation.

Undoubtedly, increasing gold recovery in the gravity circuit will decrease the gold grade of the lead concentrate, but any decrease will be more than offset by increases in the overall NSR.

8.14.2 Process Metallurgy

The 1993 laboratory testing program by Beattie indicates significantly improved process metallurgy compared to the previous Cominco operation. This is due to improved flotation response of the lower Tulsequah ore and is not due to new technology.

8.14.3 Metallurgical Testing

(1) Subsequent to the distribution of the Beattie report, optical microscopy has determined that the pyrite which represents 22 percent of the locked cycle zinc concentrate is free and can be rejected to tailing. The Beattie forecast metallurgy has been revised to indicate probable recoveries at 57 percent zinc grades in zinc concentrate.

Additional laboratory testing is required to confirm these opinions derived from optical microscopy.

8.14.4 Manpower

The manpower requirements are somewhat greater than normal for the supervisory areas, since absent supervisors must be replaced. This has increased the total number of employees by 3.

The operator requirement is normal, however, since a coverage of 24 hours is required in any case.

8.14.5 Capital Cost

The capital cost for the concentrator is as shown below:

Item	$
Concentrator	23,500,000
Tailing dam and pond	3,680,000

8.14.6 Operating Costs

The operating cost is estimated to be $ 10.02/tonne at a throughput of 800,000 tonnes per year, not including power, at $ 3.00/tonne, which is included in the plant cost estimate.

8.14.7 Marketing

The anticipated NSR at current metal prices is $ C94.81 /tonne of mill feed, FOB Juneau. Anticipated NSR at forecast "base case" metal prices is $115.32.

Complete analyses are contained in Beattie, 1993.

NSR Calculation Worksheet Tulsequah Chief Deposit
Revised terms, Revised diluted grade, Zinc concentrate grade 57%

BASE CASE METAL PRICES

Metal price US$		Metal price CAN$	
		Exchange	0.80
US$Cu	$1.00	CAN$Cu	$1.25
US$Pb	$0.35	CAN$Pb	$0.44
US$Zn	$0.60	CAN$Zn	$0.75
US$Ag	$4.00	CAN$Ag	$5.00
US$Au	$375.00	CAN$Au	$468.75

Metallurgical Balance – Weight % Grades

	Wt%	Cu	Pb	Zn	Fe	S	Sb	As	Au	Ag
Gravity	0.09%	0.00%	0.00%	0.00%	NA		NA	NA	800.00	1100.00
Cu Conc	4.68%	25.70%	0.27%	5.40%	28.00%		0.20%	0.80%	5.00	1029.00
Pb Conc	1.63%	5.00%	60.00%	5.20%	6.00%		0.10%	0.50%	65.00	1550.00
Zn Conc	9.81%	0.45%	0.12%	57.00%	6.00%		0.00%	0.80%	1.00	65.00
Tailing	83.80%	0.09%	0.08%	0.59%	9.60%		0.00%	0.02%	0.40	15.00
Feed	100.00%	1.40%	1.07%	6.42%	10.00%		0.01%	0.05%	2.40	93.37

Metallurgical Balance – % Distribution

	Wt%	Cu	Pb	Zn	Fe	S	Sb	As	Au	Ag
Gravity	0.09%	0.00%	0.00%	0.00%	NA		NA	NA	28.33%	1.00%
Cu Conc	4.68%	85.82%	1.18%	3.93%	13.09%		85.00%	89.52%	9.74%	51.52%
Pb Conc	1.63%	5.83%	91.46%	1.32%	0.98%		14.83%	17.35%	44.17%	27.08%
Zn Conc	9.81%	3.15%	1.10%	87.05%	5.88%		0.00%	0.00%	4.09%	6.83%
Tailing	83.80%	5.39%	6.27%	7.70%	80.45%		30.47%	26.75%	13.97%	13.46%
Feed	100.00%	100.18%	100.00%	100.01%	100.40%		130.30%	133.82%	100.30%	99.89%

Copper Concentrate Sale

	Metal Price	R/C	Net price	Unit	
Au	$468.75	$6.80	$461.95	Troy Oz.	$5.98
Ag	$5.00	$0.45	$4.55	Troy Oz.	$0.40
Cu	$1.25	$0.12	$1.13	pound	$0.11

Payables Metal	# Units	Deduction	Remaining Units	Pay%	Payable Units	Net Payment	Equivalent US$/DMT
Au	0.146	0.000	0.146	96.0%	0.140	$64.66	$56.91
Ag	30.01	0.00	30.01	94.0%	28.21	$128.35	$112.95
Cu	514.00	20.00	494.00	100.0%	494.00	$558.22	$491.23
Total Payables						$751.23	$661.08
Deductions							
Basic treatment						$102.27	$90.00
Pb + Zn penalty						$10.00	$8.80
As + Sb penalty						$20.00	$17.60
Total deductions						$132.27	$116.40
NSR FOB Smelter (Total Payables – total deductions)						$618.96	$544.69
Concentrate freight from Juneau						$30.23	$26.60
NSR FOB Juneau						$588.73	$518.08
NSR FOB Juneau per short dry ton mill feed						$27.52	$24.22
NSR FOB Juneau per metric dry tonne mill feed						$30.28	$24.22

Lead Concentrate sale

	Metal Price	R/C	Net price	Unit	
Au	$468.75	$6.80	$461.95	Troy Oz.	$5.98
Ag	$5.00	$0.25	$4.75	Troy Oz.	$0.22
Cu	$1.25	$0.20	$1.05	pound	$0.18
Pb	$0.44		$0.44	pound	

Payables Metal	# Units	Deduction	Remaining Units	Pay%	Payable Units	Net Payment	Equivalent US$/DMT
Au	1.896	0.029	1.866	95.0%	1.773	$819.09	$720.80
Ag	45.20	1.00	44.20	95.0%	41.99	$199.46	$175.53
Cu	100.00	10.00	90.00	40.0%	36.00	$37.80	$33.26
Pb	1200.00	0.00	1200.00	95.0%	1140.00	$498.75	$438.90
Total Payables						$1,555.10	$1,368.49
Deductions							
Basic treatment						$220.00	$193.60
As + Sb penalty						$10.00	$8.80
Total deductions						$230.00	$202.40
NSR FOB Smelter (Total Payables – total deductions)						$1,325.10	$1,166.09
Concentrate freight from Juneau						$30.23	$26.60
NSR FOB Juneau						$1,294.87	$1,139.49
NSR FOB Juneau per short dry ton mill feed						$21.12	$18.59
NSR FOB Juneau per metric dry tonne mill feed						$23.23	$18.59

Zinc Concentrate Sale

	Metal Price	R/C	Net price	Unit
Au	$468.75	$9.38	$459.38	Troy Oz.
Ag	$5.00	$0.35	$4.65	Troy Oz.
Zn	$0.75	$0.00	$0.75	pound

Payables Metal	# Units	Deduction	Remaining Units	Pay%	Payable Units	Net Payment	Equivalent US$/DMT
Au	0.029	0.050	0.000	96.0%	0.000	$0.00	$0.00
Ag	1.90	3.00	0.00	96.0%	0.00	$0.00	$0.00
Zn	1140.00	160.00	980.00	100.0%	980.00	$735.00	$646.80
Total Payables						$735.00	$646.80

Deductions		
Basic treatment	$218.18	$192.00
Hg penalty	$6.00	$5.28
Total deductions	$224.18	$197.28
NSR FOB Smelter (Total Payables – total deductions)	$510.82	$449.52
Concentrate freight from Juneau	$30.72	$27.03
NSR FOB Juneau	$480.10	$422.49
NSR FOB Juneau per short dry ton mill feed	$47.07	$41.42
NSR FOB Juneau per metric dry tonne mill feed	$51.78	$41.42

Gravity Concentrate Sale

	Metal Price	R/C	Net price	Unit
Au	$468.75		$468.75	Troy Oz.
Ag	$5.00		$5.00	Troy Oz.

Payables Metal	# Units	Deduction	Remaining Units	Pay%	Payable Units	Net Payment	Equivalent US$/DMT
Au	23.330	0.000	23.330	97.0%	22.631	$10,608.05	$9,335.08
Ag	32.08	0.00	32.079	95.0%	30.48	$152.38	$134.09
Total Payables						$10,760.43	$9,469.17

NSR FOB Smelter	$10,760.43	$9,469.17
Concentrate freight from Juneau	$29.00	$25.52
NSR FOB Juneau	$10,731.43	$9,443.65
NSR FOB Juneau per short dry ton mill feed	$9.12	$8.03
NSR FOB Juneau per metric dry tonne mill feed	$10.03	$8.03

Total Concentrate Sales Summary

	NSR FOB Juneau/DMT	
Gravity Conc.	$10.03	$8.08
Copper Conc.	$30.28	$24.22
Lead Conc.	$23.23	$18.59
Zinc Conc.	$51.78	$41.42
TOTAL	$115.32	$92.26

NSR Calculation Worksheet Tulsequah Chief Deposit
Revised terms, Revised diluted grade, Zinc concentrate grade 57%

CURRENT METAL PRICES

Metal price US$		Metal price CAN$	
		Exchange	0.79
US$Cu	$0.90	CAN$Cu	$1.14
US$Pb	$0.35	CAN$Pb	$0.44
US$Zn	$0.48	CAN$Zn	$0.61
US$Ag	$3.90	CAN$Ag	$4.93
US$Au	$338.00	CAN$Au	$427.57

Metallurgical Balance – Weight % Grades

	Wt%	Cu	Pb	Zn	Fe	S	Sb	As	Au	Ag
Gravity	0.09%	0.00%	0.00%	0.00%	NA		NA	NA	800.00	1100.00
Cu Conc	4.68%	25.70%	0.27%	5.40%	28.00%		0.20%	0.90%	5.00	1029.00
Pb Conc	1.63%	5.00%	60.00%	5.20%	6.00%		0.10%	0.50%	65.00	1550.00
Zn Conc	9.81%	0.45%	0.12%	57.00%	6.00%		0.00%	0.00%	1.00	65.00
Tailing	83.80%	0.09%	0.08%	0.59%	9.60%		0.00%	0.02%	0.40	15.00
Feed	100.00%	1.40%	1.07%	6.42%	10.00%		0.01%	0.05%	2.40	93.37

Metallurgical Balance – % Distribution

	Wt%	Cu	Pb	Zn	Fe	S	Sb	As	Au	Ag
Gravity	0.09%	0.00%	0.00%	0.00%	NA		NA	NA	28.33%	1.00%
Cu Conc	4.68%	85.82%	1.18%	3.93%	13.09%		85.00%	89.52%	9.74%	51.52%
Pb Conc	1.63%	5.83%	91.46%	1.32%	0.98%		14.83%	17.35%	44.17%	27.08%
Zn Conc	9.81%	3.15%	1.10%	87.05%	5.88%		0.00%	0.00%	4.09%	6.83%
Tailing	83.80%	5.39%	6.27%	7.70%	80.45%		30.47%	26.75%	13.97%	13.46%
Feed	100.00%	100.18%	100.00%	100.01%	100.40%		130.30%	133.62%	100.30%	99.89%

Copper Concentrate Sale

	Metal Price	R/C	Net price	Unit	
Au	$427.57	$6.80	$420.77	Troy Oz.	$5.98
Ag	$4.93	$0.45	$4.48	Troy Oz.	$0.40
Cu	$1.14	$0.12	$1.02	pound	$0.11

Payables Metal	# Units	Deduction	Remaining Units	Pay%	Payable Units	Net Payment	Equivalent US$/DMT
Au	0.146	0.000	0.146	96.0%	0.140	$58.90	$51.22
Ag	30.01	0.00	30.01	94.0%	28.21	$126.47	$109.98
Cu	514.00	20.00	494.00	100.0%	494.00	$503.14	$437.51
Total Payables						$688.51	$598.71
Deductions							
Basic treatment						$102.27	$88.93
Pb + Zn penalty						$10.00	$8.70
As + Sb penalty						$20.00	$17.39
Total deductions						$132.27	$115.02
NSR FOB Smelter(Total Payables – total deductions)						$556.24	$483.69
Concentrate freight from Juneau						$30.23	$26.29
NSR FOB Juneau						$526.01	$457.40
NSR FOB Juneau per short dry ton mill feed						$24.59	$21.38
NSR FOB Juneau per metric dry tonne mill feed						$27.05	$21.38

Lead Concentrate sale

	Metal Price	R/C	Net price	Unit	
Au	$427.57	$6.80	$420.77	Troy Oz.	$5.98
Ag	$4.93	$0.25	$4.68	Troy Oz.	$0.22
Cu	$1.14	$0.20	$0.94	pound	$0.18
Pb	$0.44		$0.44	pound	

Payables Metal	# Units	Deduction	Remaining Units	Pay%	Payable Units	Net Payment	Equivalent US$/DMT
Au	1.896	0.029	1.866	95.0%	1.773	$746.07	$648.76
Ag	45.20	1.00	44.20	95.0%	41.99	$196.67	$171.02
Cu	100.00	10.00	90.00	40.0%	36.00	$33.79	$29.38
Pb	1200.00	0.00	1200.00	95.0%	1140.00	$504.74	$438.90
Total Payables						$1,481.27	$1,288.06
Deductions							
Basic treatment						$220.00	$191.30
As + Sb penalty						$10.00	$8.70
Total deductions						$230.00	$200.00
NSR FOB Smelter(Total Payables – total deductions)						$1,251.27	$1,088.06
Concentrate freight from Juneau						$30.23	$26.29
NSR FOB Juneau						$1,221.04	$1,061.77
NSR FOB Juneau per short dry ton mill feed						$19.92	$17.32
NSR FOB Juneau per metric dry tonne mill feed						$21.91	$17.32

Zinc Concentrate Sale

	Metal Price	R/C	Net price	Unit
Au	$427.57	$8.55	$419.02	Troy Oz.
Ag	$4.93	$0.35	$4.59	Troy Oz.
Zn	$0.61	$0.00	$0.61	pound

Payables Metal	# Units	Deduction	Remaining Units	Pay%	Payable Units	Net Payment	Equivalent US$/DMT
Au	0.029	0.050	0.000	96.0%	0.000	$0.00	$0.00
Ag	1.90	3.00	0.00	96.0%	0.00	$0.00	$0.00
Zn	1140.00	160.00	980.00	100.0%	980.00	$595.00	$517.44
Total Payables						$595.06	$517.44

Deductions		
Basic treatment	$218.18	$189.72
Hg penalty	$6.00	$5.22
Total deductions	$224.18	$194.94
NSR FOB Smelter(Total Payables – total deductions)	$370.88	$322.50
Concentrate freight from Juneau	$30.72	$26.71
NSR FOB Juneau	$340.16	$295.79
NSR FOB Juneau per short dry ton mill feed	$33.35	$29.00
NSR FOB Juneau per metric dry tonne mill feed	$36.69	$29.00

Gravity Concentrate Sale

	Metal Price	R/C	Net price	Unit
Au	$427.57		$427.57	Troy Oz.
Ag	$4.93		$4.93	Troy Oz.

Payables Metal	# Units	Deduction	Remaining Units	Pay%	Payable Units	Net Payment	Equivalent US$/DMT
Au	23.330	0.000	23.330	97.0%	22.631	$9,676.12	$8,414.02
Ag	32.08	0.00	32.079	95.0%	30.48	$150.35	$130.74
Total Payables						$9,826.47	$8,544.76

NSR FOB Smelter	$9,826.47	$8,544.76
Concentrate freight from Juneau	$29.00	$25.22
NSR FOB Juneau	$9,797.47	$8,519.54
NSR FOB Juneau per short dry ton mill feed	$8.33	$7.24
NSR FOB Juneau per metric dry tonne mill feed	$9.16	$7.24

Total Concentrate Sales Summary

	NSR FOB Juneau/DMT	
Gravity Conc.	$9.16	$7.24
Copper Conc.	$27.05	$21.38
Lead Conc.	$21.91	$17.32
Zinc Conc.	$36.69	$29.00
TOTAL	$94.80	$74.94

9. INFRASTRUCTURE

9.1 Access and Transportation

Previous operators were able to stockpile concentrates during Tulsequah River outburst floods and periods of adverse weather conditions, then barge these concentrates down the Taku River from the point of confluence with the Tulsequah. These concentrates were produced from a mill feed of 400 to 500 short tons per day.

Due to the proposed larger scale of operations and the possibility that continuous barge operations on the Taku River above tidewater will require dredging and might not be environmentally acceptable, construction of a five meter wide gravel surfaced road from the mine site to tidewater at Swede Point is proposed . The proposed route is about 50 kilometers in length and will involve a major crossing of the Tulsequah flood plain at a point adjacent to the mine.

Possible road routes were reconnoitered by helicopter during February, 1993. Routes along the south side of the Taku River appear to involve excessive cost due to numerous steep bluffs and many major avalanche chutes which encroach upon the river. The route along the north shore of the Taku appears to be less challenging except for bridging the Tulsequah River.

Figure 3 indicates the proposed road route. A barge terminal will be constructed at Swede Point for the purpose of transshipping concentrate and supplies. Contract barge operators located in Juneau will provide a daily barge service between Juneau and Swede Point.

Further land-based investigations of the proposed road route are required. These will identify construction measures required to cross swampy ground adjacent to Twin Glacier Lake, to cross in front of the toe of "Hole in the Wall" glacier, to provide adequate protection from two avalanche chutes situated on Mounts Strong and Kluchman, and to bridge the Tulsequah.

It is anticipated that the Tulsequah bridge will be constructed on steel pilings driven into the glacial outwash of the flood plain. The bridge will be a single lane structure and will be designed to withstand the effects of floodwaters and debris which occur during the annual outburst flood.

Some consideration was given to a tramline crossing of the Tulsequah. While this approach should be considered in more depth, apparent disadvantages of a tramline system include the need to transport personnel and supplies as well as concentrate, the possibility of interference to the system from wind and the failure to provide a structure capable of supporting a pipeline in the event that mill tailings are required to be deposited on the west side of the Tulsequah River.

9.2 Deep Sea Port Facility

A facility capable of loading concentrate into ocean going ships will be required. Juneau is a deep sea port which at this time is largely dedicated to ocean liner traffic, local container traffic and fishermen.

It is proposed to construct a covered storage building with storage capacity for 30,000 tonnes of concentrate and a ship loading conveyor system capable of handling 400 tonnes per hour. The facility will also include a 300 meter long wharf constructed on pilings and be capable of berthing a 30,000 ton ship.

A front end loader will be used to empty concentrate containers from incoming barges and to stack and reclaim concentrate. It is envisaged that a small crew will be stationed at the shiploading facility to handle concentrate, load ships, perform minor maintenance, and carry out minor expediting duties for the mine.

A detailed investigation of the availability of existing port facilities such as those at the Green's Creek Mine could indicate some capital cost savings to the project. Another option could involve barging concentrates to existing port facilities at Prince Rupert or Stewart, B.C. Investigations of these options were not carried out for this pre-feasibility study.

9.3 Power Supply and Electrical Distribution

The options investigated included the generation of diesel power on site and the construction of a 69KV powerline to connect with the Alaska Electric Light and Power Company's grid at Bishop Point. An assessment of local hydroelectric potential within a 25-30 km radius of the site was not carried out, and at minimum, a preliminary assessment of this potential should be made prior to final feasibility.

At this time, economics favour on site diesel power generation. The capital expenditure required to construct a 9 megawatt plant at the mine site is estimated to be $7,150,000. Four 2250 KW generators, three of which will operate and one of which will be a spare, are included.

The powerline option would likely involve laying at least 20 km of submerged cable in Taku Inlet. Encroachment of the Taku glacier, severe topography, and exposure to the "Taku" winds make an overland route in the upper Taku Inlet all but impossible. The expenditure required to construct a powerline to Bishop Point is estimated to be $34,000,000 due largely to the high cost of submerged cable. In addition, a backup diesel plant at the mine site would almost certainly be required. The purchase of power from Alaska Electric Light and Power is uncertain at this time because they cannot guarantee sufficient power for mine/mill operations on a year round basis although they are currently assessing expansion in light of possible increased demand. Without considering financing charges, it is calculated that a minimum mine life of eleven years is required to justify capital expenditure of a powerline to Bishop Point. Interruptions to power supply and interest charges could deteriorate that parameter considerably.

Mine site power will be generated at 4160 volts and fed to the main 2000 amp switch gear. Five switches will transfer power to the mill, the surface plant, the mine hoist, the general mine distribution, and to the main exhaust fan. Appropriate transformers, load breaks and cables are provided for. The mine general electrical distribution will be fed through a 400 amp ground fault breaker.

Total connected load is anticipated to be 7.5 to 8.0 megawatts with average demand of 6.4 megawatts.

Waste heat from the power plant will be captured to provide heating for mill dryers and partial heating for surface buildings.

9.4 Fuel Storage

The need for adequate fuel storage is reinforced by the probability of on site power generation.

Swede Point appears to be the appropriate location for the main storage depot. Juneau is supplied by barges from Seattle and Swede Point is reported to be a viable unloading point for such barges.

Estimated daily fuel consumption at the mine site is 12,500 U.S. gallons during summer months and 15,500 gallons during winter. Storage facilities for 160,000 US gallons at Swede Point and a further 30,000 US gallons at the mine site have been estimated. Fuel would be transferred from Swede Point to the mine site by concentrate trucks as a backhaul.

9.5 Water Supply

It is anticipated that potable water, process water, and fire fighting water will be obtained from the creeks near the mine. Discussion with previous operators indicates that minimum winter flows in most of these creeks could be sustained at 150-200 US gallons per minute. This requires further substantiation.

9.6 General Service Building

The general service building will house administrative, technical, and supervisory offices, mine rescue first aid station, mine dry and changehouse, warehouse, and a small shop for surface equipment and electrical repairs. The approximate size is 2250 square meters divided between two floors.

9.7 Employee Housing and Transport

It is anticipated that site operations will be conducted by a site crew flown in and out on a regular rotation. For the purpose of this study, it was assumed that mine crews will work a four week in - two week out rotation, mill and surface crews will work twelve hour shifts on a two week in- two week out rotation, and administrative personnel will generally work two weeks in and one week out.

The existing air strip will be lengthened sufficiently to allow "Hercules" assisted mobilization and upgraded with navigational aids. It has been assumed that a charter "Convair 580" or similar service will make one flight weekly on a Vancouver - Smithers - mine site - return basis. This route is similar to the one formerly used by Cheni Gold Mines Inc.

Another possible option which has not been explored could be housing of employees in Juneau, Alaska. This arrangement would require special treatment by Canadian and U.S. Customs and Immigration Services and is beyond the scope of this study.

It is also possible that the marshalling point could be located closer to the site than Vancouver if a location with a suitable skilled labour pool is determined. Locations such as Smithers might be investigated.

A 200 man camp will be constructed on the flank of Mount Eaton a short distance north of the mine site. Other locations are worthy of consideration, but advantages of the proposed site include short travel time to work, less road building, and the possibility of common facilities such as sewage disposal for the man camp and general services building.

Domestic and putrescible wastes will be disposed of by force fired incinerator. Inert refuse will be disposed of in land fill.

Capital and operating costs for the sewage disposal system and incinerator are included in the appropriate infrastructure and surface operating sections.

9.8 Surface Plant and Equipment

Site surface equipment will consist of the following:

- Front End Loader (CAT 966 or Equivalent) (2)
- Road Grader (1)
- Backhoe (1)
- Boom Truck (1)
- Bulldozer (D-6 or Equivalent) (1)
- Concentrate Haul Tractors & Trailers (3)
- Personnel Carrier (1)
- Ambulance (1)
- Crew Cab (1)
- Pick up trucks (including mine, mill & admin.) (12)

The Swede Point landing will consist of a short causeway to permit barge landing, an open yard for temporary storage of concentrate and freight containers, the fuel depot and a very small building housing communication equipment and first aid supplies. Moving of containers by front end loader is contemplated.

9.9 Capital Cost Estimate

Infrastructure capital requirements are estimated as follows:

Item	Estimated Expenditure
Access and Site Roads Including Minor Crossings	$8,205,000
Tulsequah Bridge	7,500,000
Site Preparation	500,000
Deep Sea Port Facility	6,380,000
Site Power Generation Plant & Elec. Distribution	8,360,000
General Service Building, Man Camp & Communications	2,710,000
Fuel Storage	410,000
Water Supply and Distribution	250,000
Landing Strip Upgrade	250,000
Subtotal:	34,565,000
Preproduction man camp operating costs	2,746,000
Surface equipment purchase	2,564,000
Spare parts for surface equipment	461,000
TOTAL:	**$40,336,000**

9.10 Surface Operations Operating Cost Estimate

The estimated cost of surface operations is as follows:

Item	Cost per Annum	Cost per Tonne Milled
Plant Supervision	166,000	0.21
Site and Surface Equipment mtc.	1,164,000	1.46
Power Generation	5,857,000	7.32
Concentrate Haul to Swede Point	728,000	0.91
Daily Barging	1,226,000	1.53
Juneau Port Facility	143,000	0.18
Total	**$9,284,000**	**$11.61**

Estimated power generation costs are based on diesel fuel cost of $0.302 per litre (Canadian) landed at the Swede Point fuel depot by barge from Seattle. This is based on the current landed price at Juneau plus Canadian G.S.T. and P.S.T. and a small increment for demurrage. Every 1 cent (Canadian) variance in fuel cost will result in a power generation cost variance of $0.20 per tonne milled at 800,000 tonnes per annum.

Labour costs are estimated to comprise approximately 19 percent of the above totals.

Daily freight from Juneau will be carried by contract barge to Swede Point, where it will be backhauled by the concentrate trucks.

10. ENVIRONMENTAL AND PERMITTING

Requirements and cost of the permitting process and mine reclamation are beyond the scope of this study.

This study does include descriptions and cost estimates of several mitigating measures for environmental protection. These measures include the following:

- deposition of all acid generating waste rock including existing material left by previous operators into mined out voids which will lie below the water table after abandonment
- treatment of mine waste water during the operation
- production of a waste sulphide concentrate which will form part of the mine backfill
- a membrane - lined tailings pond
- a reclaim system for mill tailings water
- normal environmentally acceptable operating procedures regarding fuel storage and transport, refuse disposal, sewage treatment, etc.

11. STAFFING AND ADMINISTRATION

11.1 Management Requirements

All management functions will be performed on-site under the direction of a mine manager and in his absence, the mine superintendent.

Accounting, clerical, and secretarial personnel total five people of whom three to four are on site at any given time.

11.2 Manpower Requirements

Manpower requirements are estimated as follows:

Category	On Site	Total Workforce
• Management, accounting, purchasing, safety and environmental	7	11
• Mine		
- Supervision	4	6
- Technical, clerical	8	12
- Miners	20	30
- Mechanical, electrical	13	19
- Operators, construction, diamond drillers, labourers	24	37
• Mill		
- Supervision, metallurgy, clerical	5	10
- Assay and sample preparation	3	6
- Operators	8	16
- Maintenance	4	8
• Surface Plant		
- Supervision	1	2
- Journeymen	5	10
- Heavy Equipment Operators	3	6
- Truck Drivers	3	6
- Labourers	2	3
- Juneau Port	2	2
Subtotal	**112**	**184**

In addition to the above, it is estimated that camp operations (catering subcontractor) will require twelve people, bringing the total number of personnel employed at the operation to 196.

Wages for hourly rated personnel were estimated by category ranging from a low of $13.00 per hour for surface labourers to a high of $33.00/hour (including incentive bonus) for miners. Fringe benefits were estimated at 25 percent and an underground overtime allowance averaging six percent was added. Camp catering costs were estimated to be $30 per man shift which is additional to the aforementioned costs.

11.3 Warehousing

Warehousing space will be provided in the general service building. Due to the relatively low annual snowfall, an outside storage yard is suitable for bulky items such as mine timbers.

11.4 Communications

A satellite based communication system similar to the one in use at the SNIP mine is assumed. The system features multi-channel operations and can be tied to a local Vancouver, B.C. telephone number.

11.5 Safety

The mine safety director will be responsible for all safety and environmental aspects at the mine. This person will also coordinate first aid and mine rescue training. An assistant safety director will assume full responsibility when the safety director is off site.

The mine site first aid room will comply with British Columbia WCB Regulations (#5 first aid room).

11.6 Administration Cost Estimate

Administrative costs are estimated as follows:

Item	Cost Per Annum	Cost Per Tonne Milled
General Office Salaries and Supplies	$400,000	0.50
Purchasing and Warehouse	146,000	0.18
Safety and Environmental	363,000	0.45
Man Camp Operations	1,570,000	1.97
Air Transport	900,000	1.13
Communications	144,000	0.18
Insurance, Consultants and Audit	190,000	0.24
Juneau Land Lease & Property Tax	164,000	0.21
Total	**$3,877,000**	**$ 4.85**

No provision is made for corporate (head office) overhead.

Labour costs including catering staff are estimated to comprise approximately 30 percent of the totals above.

11.7 Working Capital

Working capital requirements are estimated to be four months operating expense, totalling $12,237,000.

12. FINANCIAL ANALYSIS

12.1 Summary of Capital Costs

Estimated capital costs can be summarized as follows:

12.1.1 Preproduction

ITEM	TOTAL
Access and Site Roads	$ 8,205,000
Tulsequah Bridge	7,500,000
Preproduction Mine Development	19,880,000
Preproduction Mine Equipment Purchase	4,504,000
Spare Parts - Mine & Surface Equipment	1,272,000
Site Preparation	500,000
Site Power Generation Plant & Elec. Distribution	8,360,000
Mill	23,500,000
Tailings Impoundment	2,650,000
General Service Bdg., Man Camp & Communications	2,710,000
Fuel Storage	410,000
Water Supply and Distribution	250,000
Landing Strip Upgrade	250,000
Deep Sea Port Facility	6,380,000
Surface Equipment Purchase	2,564,000
Preproduction Man Camp Operating Costs	2,746,000
Subtotal:	**$91,681,000**
Eng. Design, Procurement & Management @ 14%	12,835,000
Subtotal:	**$104,516,000**
Contingency 20%	20,903,000
Subtotal Preproduction Capital:	**$125,419,000**
Working Capital (4 months)	**$137,656,000**
TOTAL PRODUCTION EXPENDITURE	**$137,656,000.**

Cost of permitting and reclamation bond is not in the scope of this study.

12.1.2 Ongoing Capital Expenditure

ITEM	TOTAL
Mine Deepening - Production Years 1 & 2	$ 19,746,000
Additional Mine Equipment-Production Year 1 & 2	2,630,000
Subtotal Mine Development & Equipment	**22,376,000**
Tailing Dam	1,030,000
Contingency 20%	4,681,000
Subtotal	**28,087,000**
Equipment Replacement - Years 2 - 8	6,400,000
Exploration Development & Drilling	Not Included
TOTAL ONGOING CAPITAL - Production Years 1-8	**$ 34,487,000**

12.2 Schedule of Capital Requirements

The capital requirement scheduled expenditures are estimated as follows:

ITEM		TOTAL
Pre-Production Year - 2		$ 21,350,000
Preproduction Year - 1		104,069,000
Subtotal Preproduction		**$125,419,000**
Working Capital		12,237,000
Production Year -	1	16,800,000
"	2	10,751,000
"	3	700,000
"	4	1,000,000
"	5	2,236,000
"	6	1,000,000
"	7	1,000,000
"	8	1,000,000
"	9	(12,237,000)
Subtotal Ongoing Capital		**$ 34,487,000**
TOTAL ESTIMATED CAPITALIZED EXPEND.		**$159,906,000**

12.3 Summary of Operating Costs

The following is a summary of estimated operating costs at a milling rate of 800,000 tonnes per annum:

ITEM	COST PER ANNUM	COST PER TONNE MILLED
Site Administration	$ 1,406,000	1.76
Air Charter	900,000	1.13
Man Camp Operations	1,570,000	1.96
Subtotal Administrative	**3,876,000**	**4.85**
Stope Definition & Development	2,687,000	3.36
Stoping	6,683,000	8.35
Mucking, Crushing, Hoisting & Conveying	1,672,000	2.09
Mine Services & Construction	1,389,000	1.74
U/G Equip. Maintenance (labour)	1,408,000	1.76
Acid. Gen. Wasterock Disposal	135,000	0.17
Mine Supervision, Tech. & Clerical	1,266,000	1.58
Mine Ventilation Air Heating	250,000	0.31
Subtotal Mine	**15,490,000**	**19.36**
Mill - Salaries & Wages	2,200,000	2.75
Reagents	2,792,000	3.49
Grinding Media	984,000	1.23
Mill Liners	96,000	.12
Operating Supplies	400,000	0.50
Maintenance Supplies	800,000	1.00
Freight	784,000	0.98
Subtotal Mill	**8,056,000**	**10.07**
Surface Plant Supervision	166,000	0.21
Site & Surface Equipment Mtce.	1,164,000	1.46
Freight & Concentrate Haul to Swede Pt.	728,000	0.91
Barging	1,226,000	1.53
Juneau Port Facility	143,000	0.18
Site Power Generation	5,857,000	7.32
Subtotal Surface Ops.	**9,284,000**	**11.61**
TOTAL OPERATING COST	**$36,706,000**	**$45.89**

12.4 Cash Flow Forecast

Table 3 presents a cash flow forecast based on projected longer term metal prices and on an exchange rate of $1.00 U.S. equals $1.25 Canadian.

Table 4 presents a cash flow forecast based on current metal prices and the current exchange rate of $1.00 U.S. equals $1.265 Canadian.

12.5 Sensitivity Analysis

Graphs 1 and 2 indicate project sensitivities to various fluctuations in metal prices and to changes in capital and operating costs for the current ore reserve.

Sensitivity to increased ore reserves at the same annual production rate (800,000 tonnes) is indicated by graph 3. This situation would reflect adding reserves as active mining progresses.

Adding significant reserves along strike could create a situation warranting installation of greater mine/mill capacity. Table 5 is a projected case for a 10.8 million tonne mineable reserve with a daily throughput of 3,500 tonnes. Individual capital costs have been prorated by industry-accepted factors and operating costs have been adjusted in areas where economies of scale would make significant differences.

12.6 Discussion

Graphs numbered 1 and 2 indicate that metal price and cost sensitivities rank in descending order as follows:

1) zinc price (greatest sensitivity)
2) operating costs
3) copper price
4) gold price
5) capital costs
6) silver price
70 lead price (least sensitivity)

The internal rate of return (pretax) for the project is relatively insensitive to increased reserves if the mining rate is held constant at 800,000 tonnes per annum (see Graph No. 3).

The internal rate of return and cumulative project cash flow could be beneficially impacted by increasing the mineable reserve <u>AND</u> increasing annual throughput (see Table 5). An increase in annual throughput would require additional capital costs (estimated in Table 5) and a distribution of new ore reserves along strike which would support additional mine working places.

TULSEQUAH CHIEF PRETAX CASH FLOW MODEL

800,000 TONNES/ ANNUM

– CANADIAN CURRENCY UNLESS OTHERWISE NOTED

AU PRICE($US)	$375.00	NSR	$115.32
AG PRICE($US)	$4.00		
CU PRICE($US)	$1.00		
PB PRICE($US)	$0.35		
ZN PRICE$US)	$0.60		

		YEAR												
		–3	–2	–1	1	2	3	4	5	6	7	8	9	TOTAL
REVENUE														
TONNES MILLED x 1000		0	0	0	800	800	800	800	800	800	800	800	560	6960
AU HEAD g/t					2.44	2.44	2.32	2.32	2.32	2.32	2.32	2.32	2.32	2.35
AG HEAD g/t					90.7	90.7	91.18	91.18	91.18	91.18	91.18	91.18	91.18	91.07
CU HEAD %					1.58	1.58	1.31	1.31	1.31	1.31	1.31	1.31	1.31	1.37
PB HEAD %					1.03	1.03	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04
ZN HEAD %					6.56	6.56	6.31	6.31	6.31	6.31	6.31	6.31	6.31	6.37
NET SMELTER RETURN x$1000					92,256	92,256	92,256	92,256	92,256	92,256	92,256	92,256	64,579	802,627
CAPITAL EXPENDITURES x $1000														
PERMITTING & BONDING		not inclu	0	0	0									not incl
INFRASTRUCTURE			11607	25704	0									37311
PREPROD. MINE DEVELMT			4000	15880	0									19880
MINE PLANT & EQUIPMENT			0	8340	0	2630								10970
MILL & TAILINGS			0	26150	0				1030					27180
DES. ENG, MGMT & PROC			2185	10650	0									12835
MINE DEEPENING			0	0	14000	5746						0	0	19746
SUBTOTAL			17792	86724	14000	8376	0	0	1030	0	0	0	0	127922
CONTINGENCY 20 %			3558	17345	2800	1675	0	0	206	0	0	0	0	25584
CAPITAL REPLACEMENT			0	0	0	700	700	1000	1000	1000	1000	1000	0	6400
WORKING CAPITAL					12237								–12237	0
TOTAL CAPITAL x $1000		0	21,350	104,069	29,037	10,751	700	1,000	2,236	1,000	1,000	1,000	(12,237)	159,906
OPERATING EXPENDITURES	**$/UNIT**													
STOPE DEVELOPMENT	3.36		0	0	2688	2688	2688	2688	2688	2688	2688	2688	1882	23386
MINE PRODUCTION	16.00		0	0	12800	12800	12800	12800	12800	12800	12800	12800	8960	111360
MILLING	10.07		0	0	8056	8056	8056	8056	8056	8056	8056	8056	5639	70087
SURFACE OPERATIONS	4.29		0	0	3432	3432	3432	3432	3432	3432	3432	3432	2402	29858
ELECTRICAL POWER	7.32		0	0	5856	5856	5856	5856	5856	5856	5856	5856	4099	50947
AIR TRANSPORT	1.13		0	0	904	904	904	904	904	904	904	904	633	7865
MANCAMP	1.96		0	0	1568	1568	1568	1568	1568	1568	1568	1568	1098	13642
SITE ADMINISTRATION	1.76		0	0	1408	1408	1408	1408	1408	1408	1408	1408	986	12250
	45.89 /tonne													
TOTAL OP COSTS x $1000			0	0	36,712	36,712	36,712	36,712	36,712	36,712	36,712	36,712	25,698	319,394
OPERATING PROFIT x $1000		0	0	0	55,544	55,544	55,544	55,544	55,544	55,544	55,544	55,544	38,881	483,233
PROJECT CASH FLOW x $1000		0	(21,350)	(104,069)	26,507	44,793	54,844	54,544	53,308	54,544	54,544	54,544	51,118	323,326
CUMULATIVE CASH FLOW x $1000		0	(21,350)	(125,419)	(98,912)	(54,119)	725	55,269	108,577	163,121	217,665	272,209	323,326	

INTERNAL RATE OF RETURN	31.21%
NPV @12.5%	86,250
NPV @ 15%	64,762

TABLE 3

TULSEQUAH CHIEF PRETAX CASH FLOW MODEL

800,000 TONNES/ ANNUM

– CANADIAN CURRENCY UNLESS OTHERWISE NOTED

AU PRICE($US)	$338.00	NSR	$94.81
AG PRICE($US)	$3.90		
CU PRICE($US)	$0.90		
PB PRICE($US)	$0.35		
ZN PRICE($US)	$0.48		

		YEAR												
		–3	–2	–1	1	2	3	4	5	6	7	8	9	TOTAL
REVENUE														
TONNES MILLED x 1000		0	0	0	800	800	800	800	800	800	800	800	560	6960
AU HEAD g/t					2.44	2.44	2.32	2.32	2.32	2.32	2.32	2.32	2.32	2.35
AG HEAD g/t					90.7	90.7	91.18	91.18	91.18	91.18	91.18	91.18	91.18	91.07
CU HEAD %					1.58	1.58	1.31	1.31	1.31	1.31	1.31	1.31	1.31	1.37
PB HEAD %					1.03	1.03	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04
ZN HEAD %					6.56	6.56	6.31	6.31	6.31	6.31	6.31	6.31	6.31	6.37
NET SMELTER RETURN x$1000		0	0	0	$75,848	$75,848	$75,848	$75,848	$75,848	$75,848	$75,848	$75,848	$53,094	$659,878
CAPITAL EXPENDITURES x $1000														
PERMITTING & BONDING	not included		0	0	0									not incl
INFRASTRUCTURE			$11,607	$25,704	$0									$37,311
PREPROD. MINE DEVELMT			$4,000	$15,880	$0									$19,880
MINE PLANT & EQUIPMENT			$0	$8,340	$0	$2,630								$10,970
MILL & TAILINGS			$0	$26,150	$0				$1,030					$27,180
DES. ENG, MGMT & PROC			$2,185	$10,650	$0									$12,835
MINE DEEPENING			$0.00	$0.00	$14,000	$5,746	$0	$0	$0	$0	$0	$0	$0	$19,746
SUBTOTAL			$17,792	$86,724	$14,000	$8,376	$0	$0	$1,030	$0	$0	$0	$0	$127,922
CONTINGENCY 20 %			$3,558	$17,345	$2,800	$1,675	$0	$0	$206	$0	$0	$0	$0	$25,584
CAPITAL REPLACEMENT			$0	$0	$0	$700	$700	$1,000	$1,000	$1,000	$1,000	$1,000	$0	$6,400
WORKING CAPITAL					$12,237								($12,237)	$0
TOTAL CAPITAL x $1000		0	$21,350	$104,069	$29,037	$10,751	$700	$1,000	$2,236	$1,000	$1,000	$1,000	($12,237)	$159,906
OPERATING EXPENDITURES $/UNIT														
STOPE DEVELOPMENT	3.36		0	0	2688	2688	2688	2688	2688	2688	2688	2688	1882	23386
MINE PRODUCTION	16.00		0	0	12800	12800	12800	12800	12800	12800	12800	12800	8960	111360
MILLING	10.07		0	0	8056	8056	8056	8056	8056	8056	8056	8056	5639	70087
SURFACE OPERATIONS	4.29		0	0	3432	3432	3432	3432	3432	3432	3432	3432	2402	29858
ELECTRICAL POWER	7.32		0	0	5856	5856	5856	5856	5856	5856	5856	5856	4099	50947
AIR TRANSPORT	1.13		0	0	904	904	904	904	904	904	904	904	633	7865
MANCAMP	1.96		0	0	1568	1568	1568	1568	1568	1568	1568	1568	1098	13642
SITE ADMINISTRATION	1.76		0	0	1408	1408	1408	1408	1408	1408	1408	1408	986	12250

	$45.89 /tonne													
TOTAL OP COSTS x $1000		0	0	0	36,712	36,712	36,712	36,712	36,712	36,712	36,712	36,712	25,698	319,394
OPERATING PROFIT x $1000		0	0	0	39,136	39,136	39,136	39,136	39,136	39,136	39,136	39,136	27,395	340,483
PROJECT CASH FLOW x $1000		0	(21,350)	(104,069)	10,099	28,385	38,436	38,136	36,900	38,136	38,136	38,136	39,632	180,577
CUMULATIVE CASH FLOW x $1000		0	(21,350)	(125,419)	(115,320)	(86,935)	(48,499)	(10,363)	26,537	64,673	102,809	140,945	180,577	

INTERNAL RATE OF RETURN	18.77%
NPV @12.5%	27,196
NPV @ 15%	14,204

TABLE 4

TULSEQUAH CHIEF PRETAX CASH FLOW MODEL 1,244,000 TONNES/ ANNUM

– CANADIAN CURRENCY UNLESS OTHERWISE NOTED

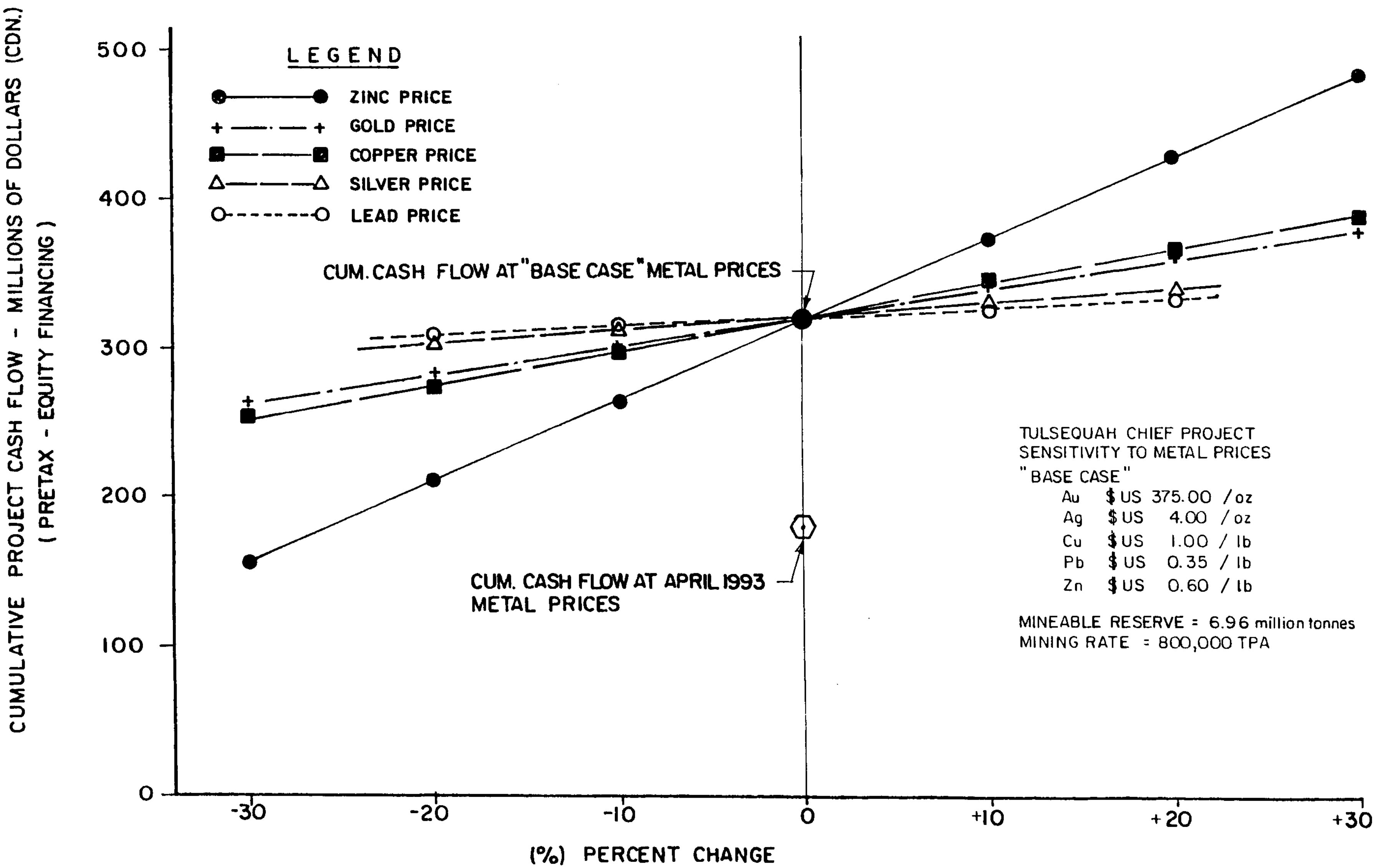
AU PRICE($US)	$375.00	NSR	$115.32
AG PRICE($US)	$4.00		
CU PRICE($US)	$1.00		
PB PRICE($US)	$0.35		
ZN PRICE($US)	$0.60		

		YEAR												
		–3	–2	–1	1	2	3	4	5	6	7	8	9	TOTAL
REVENUE														
TONNES MILLED x 1000		0	0	0	1244	1244	1244	1244	1244	1244	1244	1244	875	10827
AU HEAD g/t					2.44	2.44	2.32	2.32	2.32	2.32	2.32	2.32	2.32	2.35
AG HEAD g/t					90.7	90.7	91.18	91.18	91.18	91.18	91.18	91.18	91.18	91.07
CU HEAD %					1.58	1.58	1.31	1.31	1.31	1.31	1.31	1.31	1.31	1.37
PB HEAD %					1.03	1.03	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04
ZN HEAD %					6.56	6.56	6.31	6.31	6.31	6.31	6.31	6.31	6.31	6.37
NET SMELTER RETURN x$1000					143,458	143,458	143,458	143,458	143,458	143,458	143,458	143,458	100,905	1,248,570
CAPITAL EXPENDITURES x $1000														
PERMITTING & BONDING		not includ	0	0	0									not incl
INFRASTRUCTURE			11607	30019	0									41626
PREPROD. MINE DEVELMT			6200	24724	0									30924
MINE PLANT & EQUIPMENT			0	12973	0	2630								15603
MILL & TAILINGS			0	33284	0				1030					34314
DES. ENG, MGMT & PROC			2493	14140	0									16633
MINE DEEPENING			0	0	14000	5746						0	0	19746
SUBTOTAL			20300	115140	14000	8376	0	0	1030	0	0	0	0	158846
CONTINGENCY 20 %			4060	23028	2800	1675	0	0	206	0	0	0	0	31769
CAPITAL REPLACEMENT			0	0	0	1000	1000	1500	1500	1500	1500	1500	0	9500
WORKING CAPITAL					18000								–18000	0
TOTAL CAPITAL x $1000		0	24,360	138,168	34,800	11,051	1,000	1,500	2,736	1,500	1,500	1,500	(18,000)	200,115
OPERATING EXPENDITURES	**$/UNIT**													
STOPE DEVELOPMENT	3.36		0	0	4179.84	4179.84	4179.8	4179.8	4179.8	4179.8	4179.8	4179.8	2940	36379
MINE PRODUCTION	16.00		0	0	19904	19904	19904	19904	19904	19904	19904	19904	14000	173232
MILLING	9.26		0	0	11519.4	11519.4	11519	11519	11519	11519	11519	11519	8103	100258
SURFACE OPERATIONS	3.89		0	0	4839.16	4839.16	4839.2	4839.2	4839.2	4839.2	4839.2	4839.2	3404	42117
ELECTRICAL POWER	7.32		0	0	9106.08	9106.08	9106.1	9106.1	9106.1	9106.1	9106.1	9106.1	6405	79254
AIR TRANSPORT	1.13		0	0	1406	1406	1406	1406	1406	1406	1406	1406	989	12235
MANCAMP	1.39		0	0	1729.16	1729.16	1729.2	1729.2	1729.2	1729.2	1729.2	1729.2	1216	15050
SITE ADMINISTRATION	1.24		0	0	1542.56	1542.56	1542.6	1542.6	1542.6	1542.6	1542.6	1542.6	1085	13425

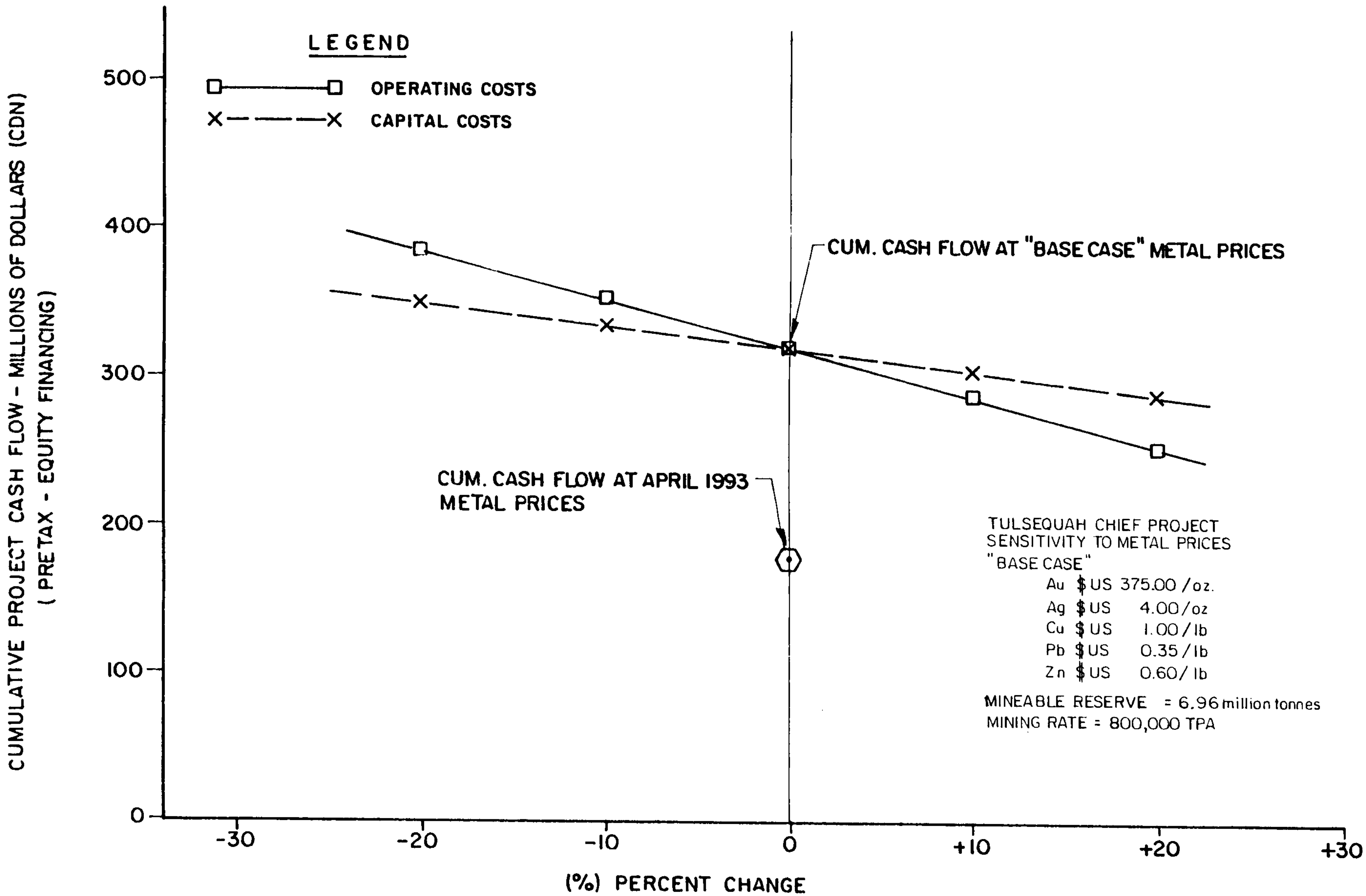
	43.59 /tonne													
TOTAL OP COSTS x $1000			0	0	54,226	54,226	54,226	54,226	54,226	54,226	54,226	54,226	38,141	471,949
OPERATING PROFIT x $1000		0	0	0	89,232	89,232	89,232	89,232	89,232	89,232	89,232	89,232	62,764	776,621
PROJECT CASH FLOW x $1000		0	(24,360)	(138,168)	54,432	78,181	88,232	87,732	86,496	87,732	87,732	87,732	80,764	576,506
CUMULATIVE CASH FLOW x $1000		0	(24,360)	(162,528)	(108,096)	(29,915)	58,317	146,049	232,545	320,278	408,010	495,742	576,506	

INTERNAL RATE OF RETURN	41.69%
NPV @12.5%	177,857
NPV @ 15%	140,705

TABLE 5

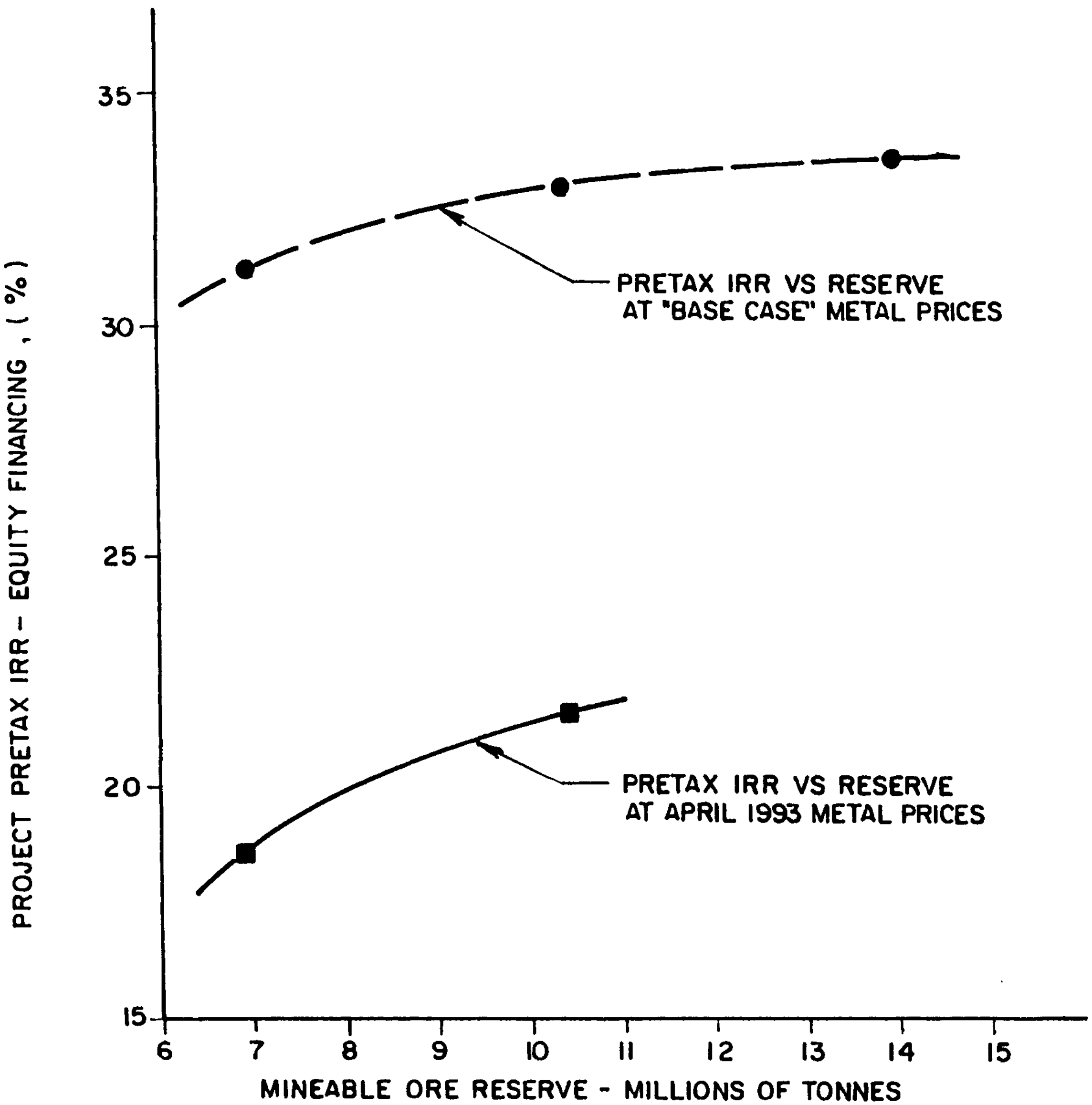


GRAPH Nº. 1
METAL PRICES VS PROJECT CASH FLOW



GRAPH Nº. 2

CAPITAL & OPERATING COSTS VS CASH FLOW



TULSEQUAH CHIEF PROJECT
SENSITIVITY TO INCREASING MINEABLE RESERVE AT CONSTANT MINING RATE (800,000 TPA)

"BASE CASE" METAL PRICES

Au	$US	375.00/oz
Ag	$US	4.00/oz
Cu	$US	1.00/lb
Pb	$US	0.35/lb
Zn	$US	0.60/lb

GRAPH №. 3

MINEABLE RESERVE AT CONSTANT MINING RATE
VS
PRETAX INTERNAL RATE OF RETURN

13. CONCLUSIONS AND RECOMMENDATIONS

- Current geological drill indicated and inferred reserves at the Tulsequah Chief mine total 8.5 million tonnes grading 2.56 grams per tonne gold, 103.42 grams per tonne silver, 1.48 percent copper, 1.17 percent lead, and 6.85 percent zinc.

- Fully diluted mineable reserves are estimated to total 6.93 million tonnes grading 2.40 grams per tonne gold, 93.37 grams per tonne silver, 1.40 percent copper, 1.07 percent lead, and 6.42 percent zinc.

- Preproduction capital expense for a 2,250 tonne per day mine/mill complex is estimated to be $CAN 137.7 million. It is estimated that an additional $22.3 million capital expenditure during the nine year operating life of the project will be required. Estimation variance is +/- 20 percent.

- Operating costs are estimated to total $CAN 45.89 per ore tonne. Estimation variance is +/- 20 percent.

- Net smelter return at forecast metal prices tabulated below is estimated to be $CAN 115.32 per ore tonne milled.

Metal		Forecast Price		Current Price	
Au	$US	375.00 per oz.	$US	$338.00	per oz.
Ag	$US	4.00 per oz.	$US	3.90	per oz.
Cu	$US	1.00 per lb.	$US	0.90	per oz.
Pb	$US	0.35 per lb.	$US	0.34	per lb.
Zn	$US	0.60 per lb.	$US	0.48	per lb.
NSR	$CAN	115.32 per tonne	$ CAN/94.81 per tonne		

- Current reserves could yield a 31 percent pre tax rate of return and capital payback in three years at the above forecast metal prices. Sensitivities to cost and metal price fluctuation are presented in Section 12.

- Revenues and project economics are particularly sensitive to the zinc price, the grade of zinc in zinc concentrate, and zinc recovery to zinc concentrate. Further metallurgical test work aimed at verification and enhancement of zinc concentrate grades and zinc recoveries is recommended.

- The potential for increasing reserves is considered excellent. An increase in reserves, particularly along the strike or in nearby deposits such as the Big Bull mine have the potential of increasing daily throughput by providing additional mine working areas. This could have a beneficial impact on project economics.

- The classification of the existing reserve should be upgraded by infill drilling to provide a firmer basis for economic projections. The level of confidence in reserve estimation could be significantly increased by doubling the number of drillhole penetrations in the upper half of the existing reserve. This type of program would form a good test comparison against the current reserve estimate.

- Cost estimation variances could be reduced to a range of +/- 10 to 15 percent by upgrading the reserve classification and performance of the investigations tabulated below which are recommended as the project is advanced towards the feasibility stage. Some of these studies also have potential for identification of cost saving measures. Recommended additional investigations include the following:

  - geotechnical studies aimed at confirming mining methods, stope size, ground support requirements and water inflows.
  - land based inspection of the road route, tailings impoundment area, millsite, and Tulsequah crossing location. Elimination of a membrane lining in the tailings pond could reduce preproduction capital by $1.5 million.
  - complete environmental and socio economic studies. These are required for project permitting and will also clearly identify physical measures which are required.
  - confirmation of possible government assistance such as the proposed Taku road linking Juneau, Alaska and Atlin, British Columbia.
  - investigation of the possible availability of existing sea port facilities for lease or purchase.
  - appraisal of local hydro-electric potential within a 30-40 km radius of the minesite.
  - preparation of a detailed mill cost estimate when project strategy, construction methodology and metallurgical test work are sufficiently defined. Definition of project strategy would involve possible use of existing mills or concentrate shipping facilities

Appendices

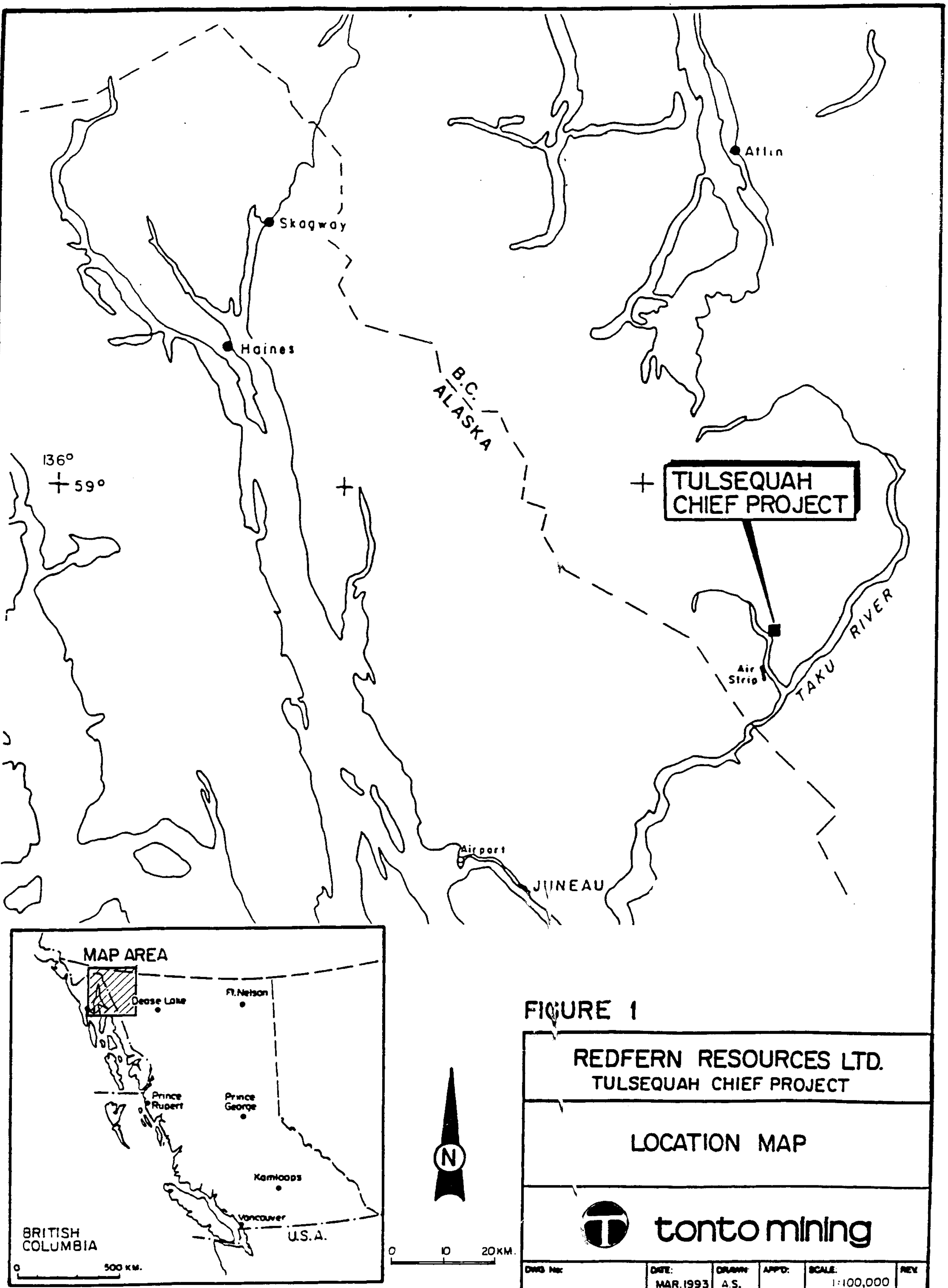


FIGURE 1

REDFERN RESOURCES LTD. TULSEQUAH CHIEF PROJECT					
LOCATION MAP					
tonto mining					
DWG No:	DATE: MAR. 1993	DRAWN: A.S.	APP'D:	SCALE: 1:100,000	REV.

CHONG

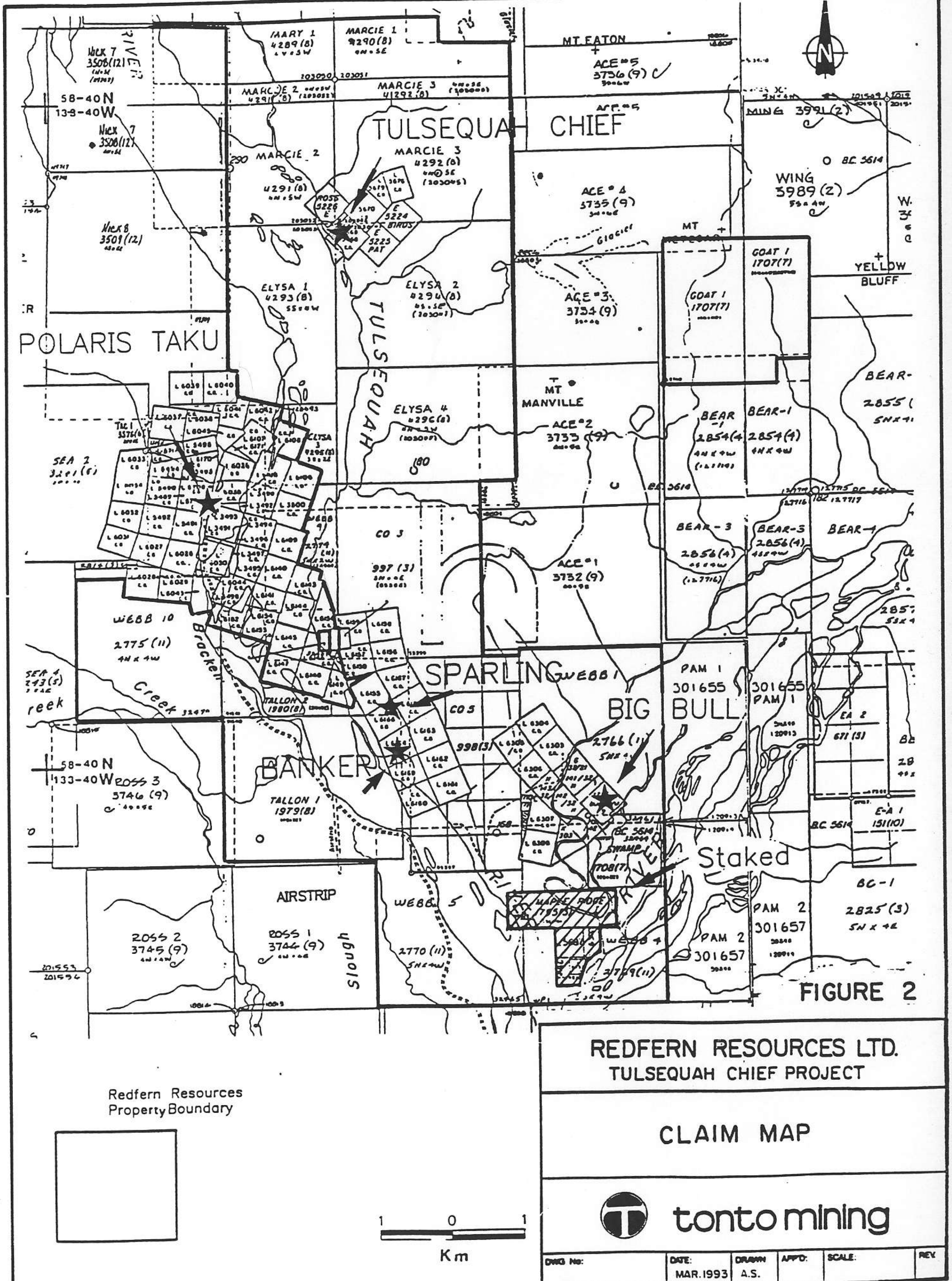
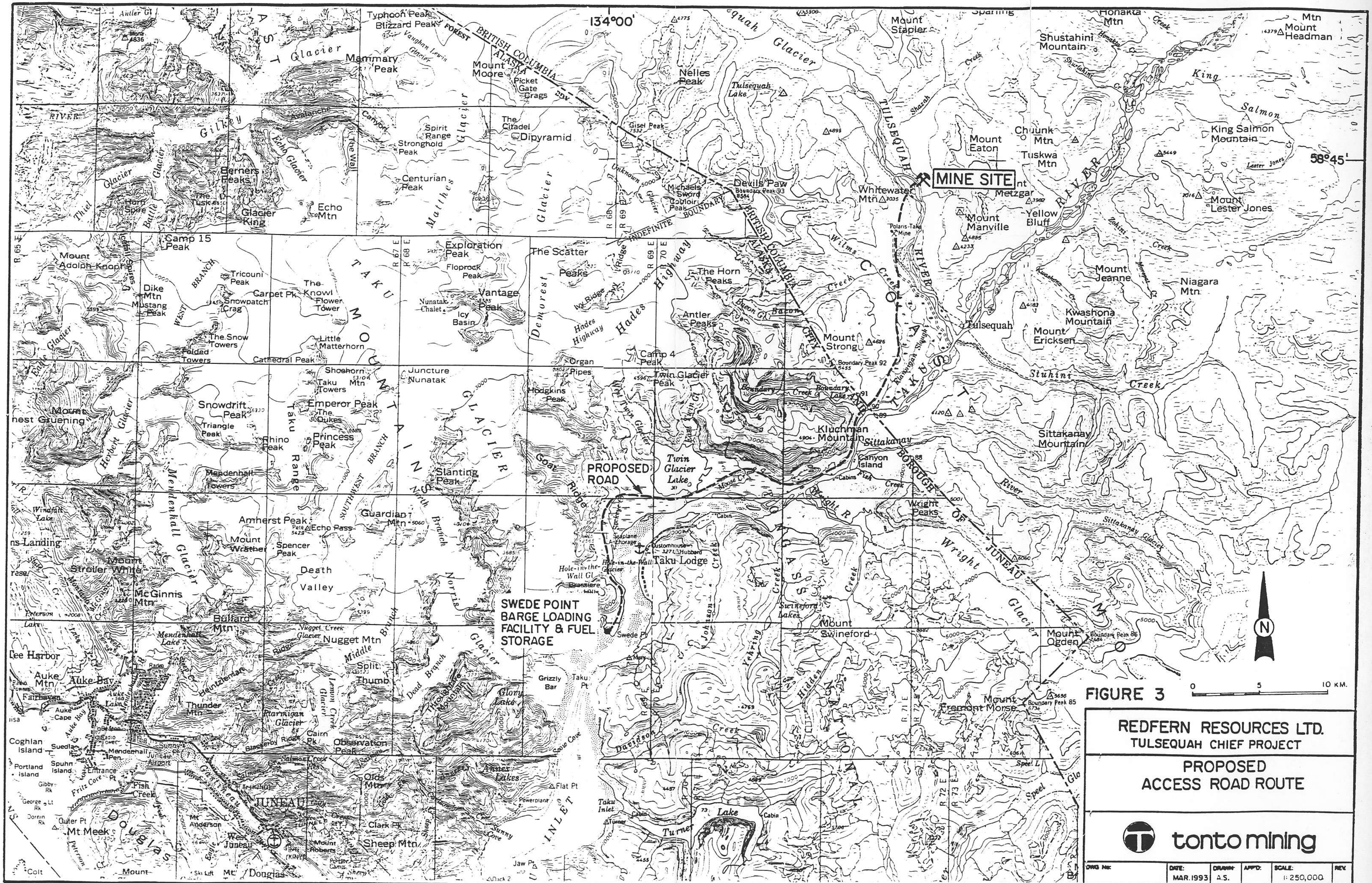


FIGURE 2

CHONG


MINE SITE
PROPOSED ROAD
SWEDE POINT BARGE LOADING FACILITY & FUEL STORAGE
134°00'
58°45'
Tulsequah
Taku Lodge
JUNEAU
Twin Glacier Lake
Canyon Island
Polaris-Taku Mine
0 5 10 KM.
N
FIGURE 3
REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT
PROPOSED ACCESS ROAD ROUTE
tonto mining
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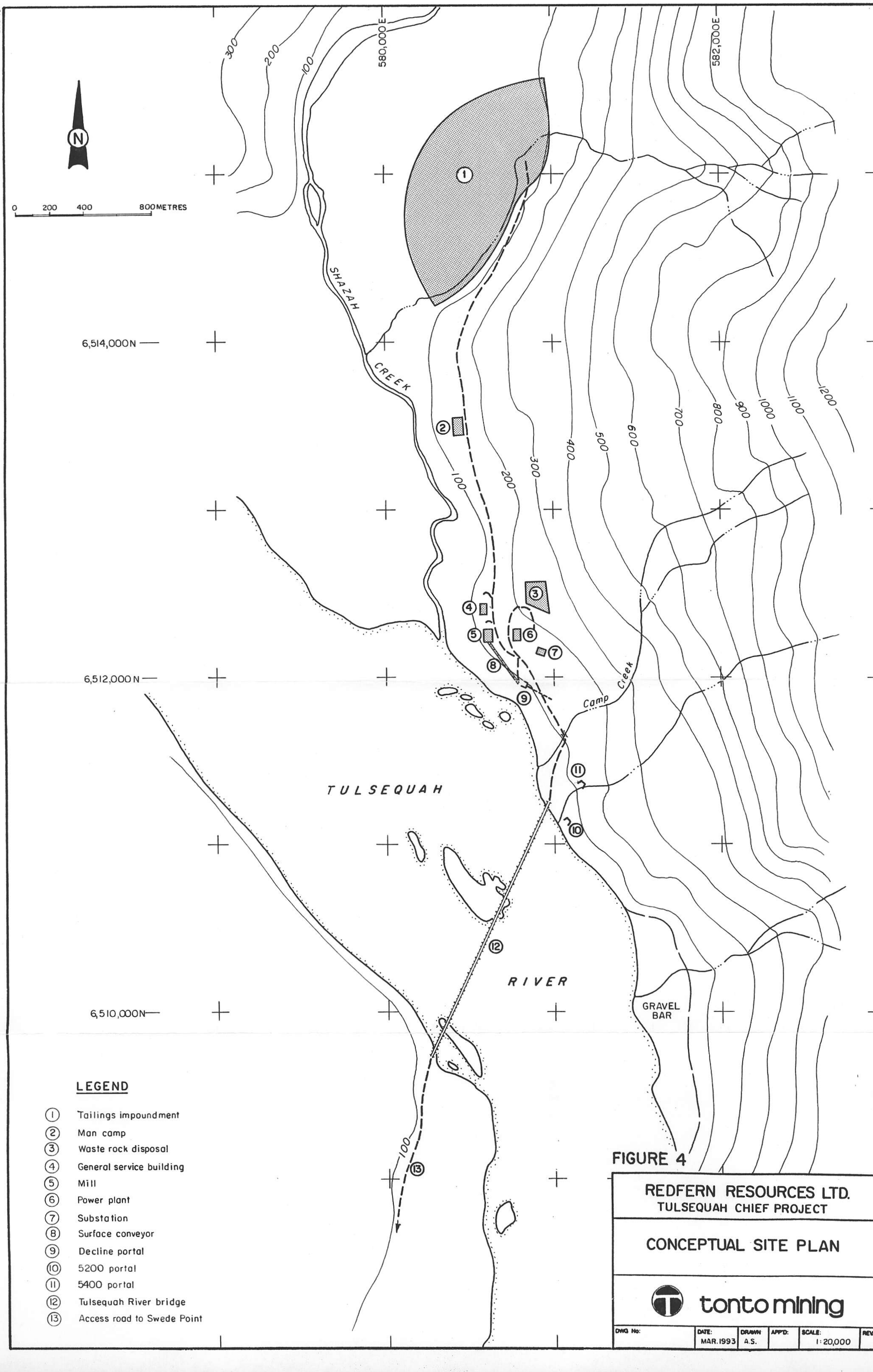
DRAWN: A.S.
APP'D:
SCALE: 1:250,000
REV.
CHONG



LEGEND

1. Tailings impoundment
2. Man camp
3. Waste rock disposal
4. General service building
5. Mill
6. Power plant
7. Substation
8. Surface conveyor
9. Decline portal
10. 5200 portal
11. 5400 portal
12. Tulsequah River bridge
13. Access road to Swede Point

FIGURE 4

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

CONCEPTUAL SITE PLAN

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	MAR. 1993	A.S.		1:20,000	

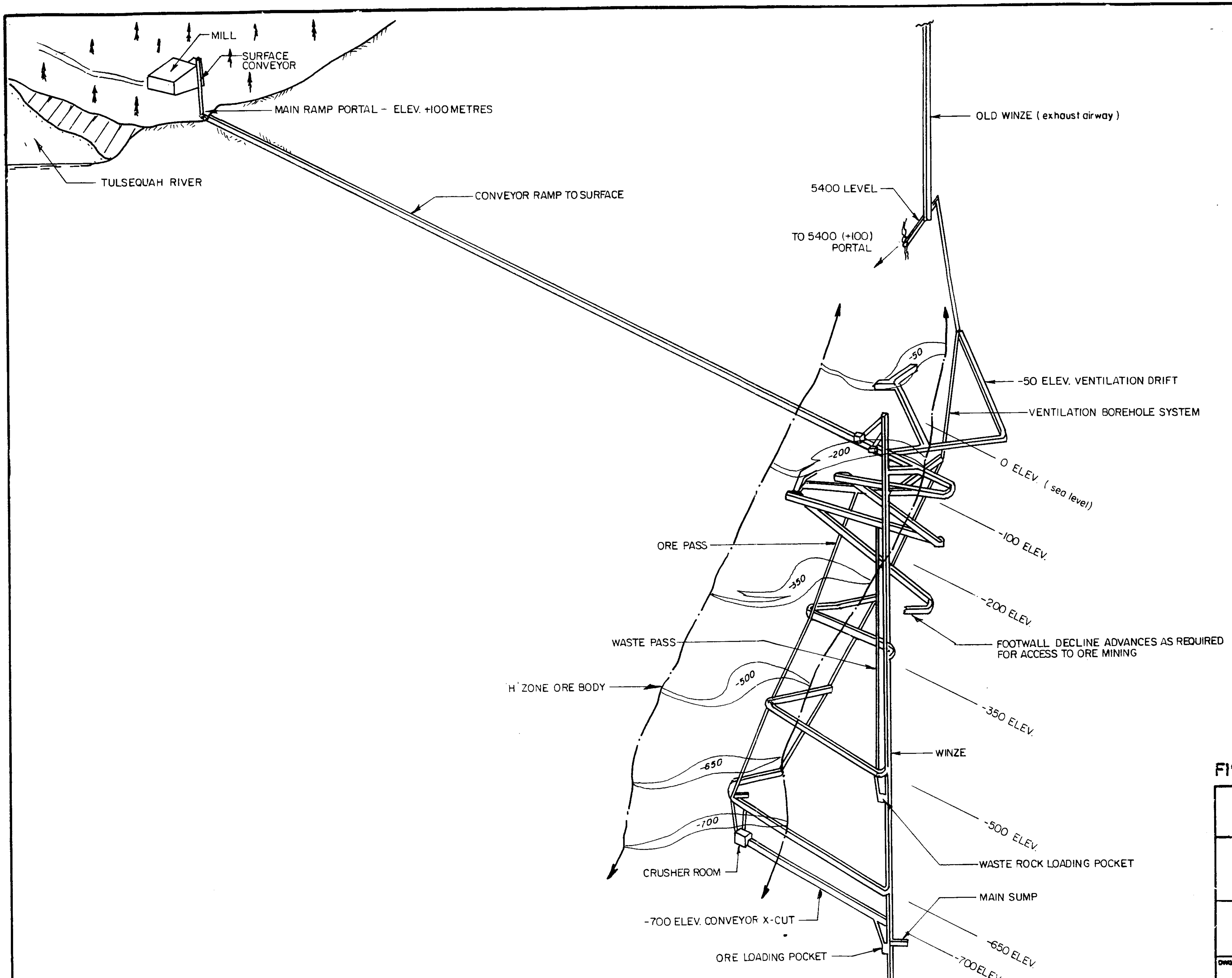

FIGURE 5

REDFERN RESOURCES LTD. TULSEQUAH CHIEF PROJECT
SCHEMATIC MINE DIAGRAM LOOKING APPROXIMATELY N 25°E
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	MAR. 1993	A.S.		N.T.S.	

CHONG

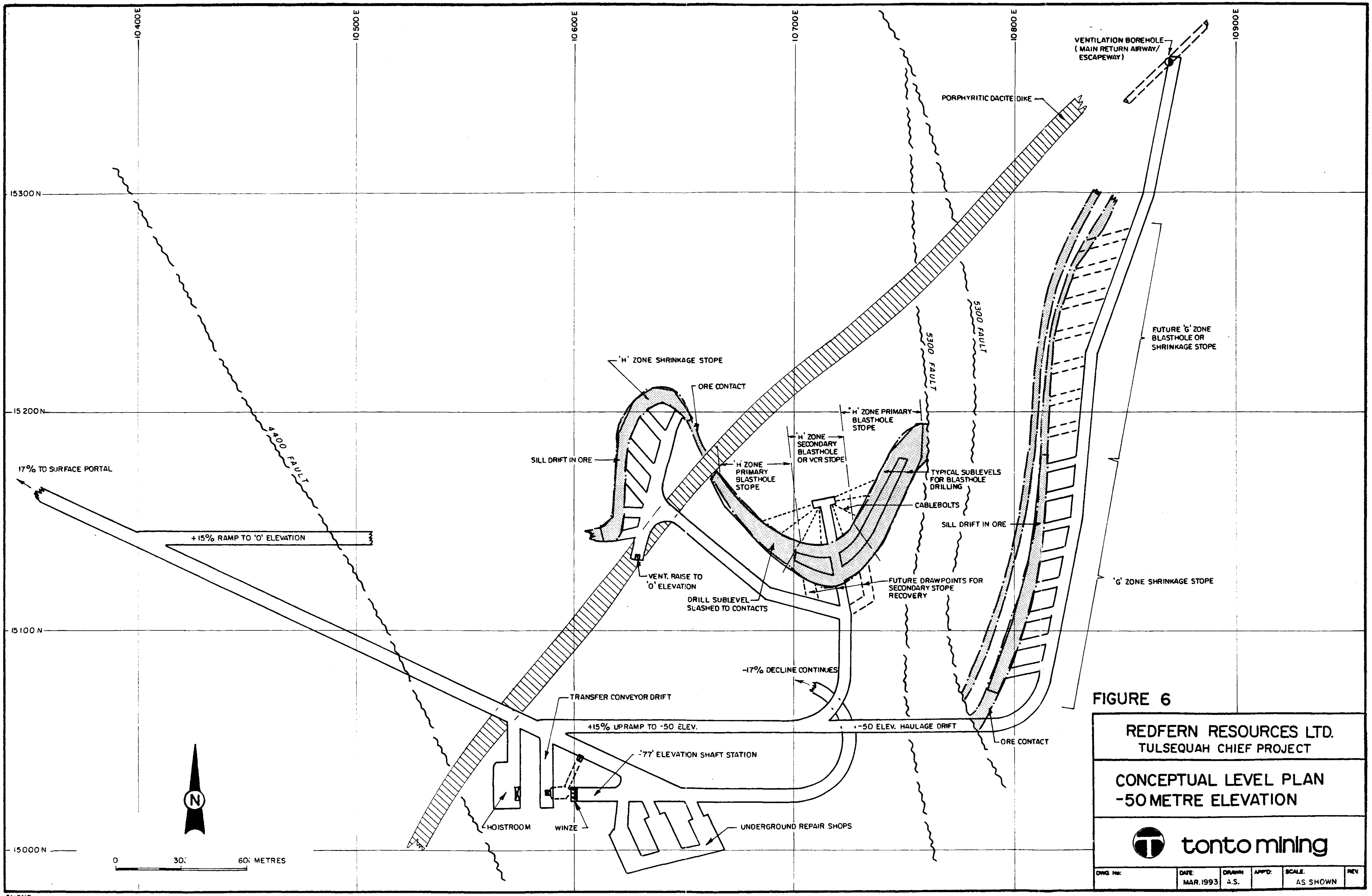

FIGURE 6

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

CONCEPTUAL LEVEL PLAN
-50 METRE ELEVATION

tonto mining

CHONG

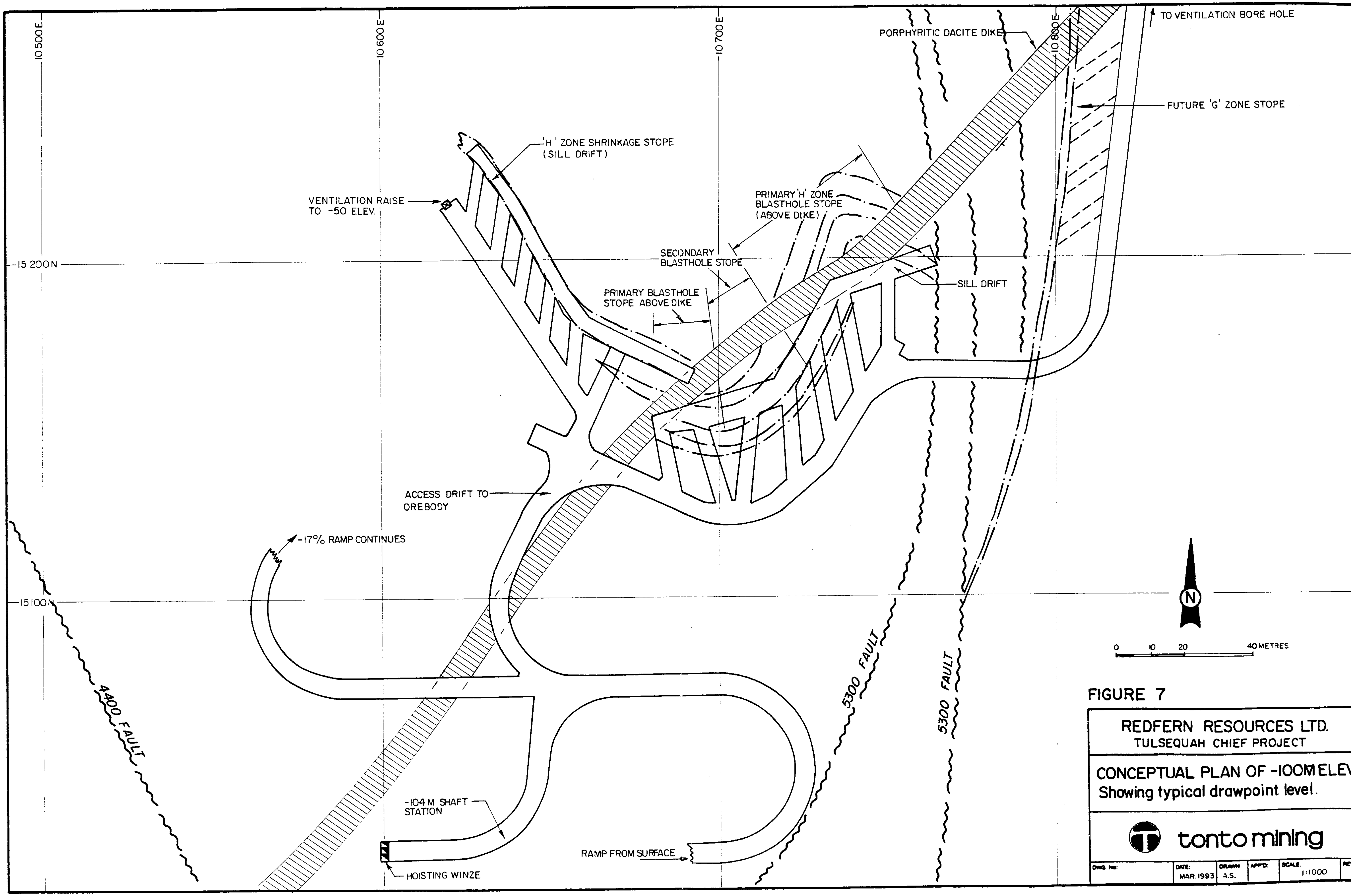

FIGURE 7

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

CONCEPTUAL PLAN OF -100M ELEV.
Showing typical drawpoint level.

tonto mining

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	MAR. 1993	A.S.		1:1000	

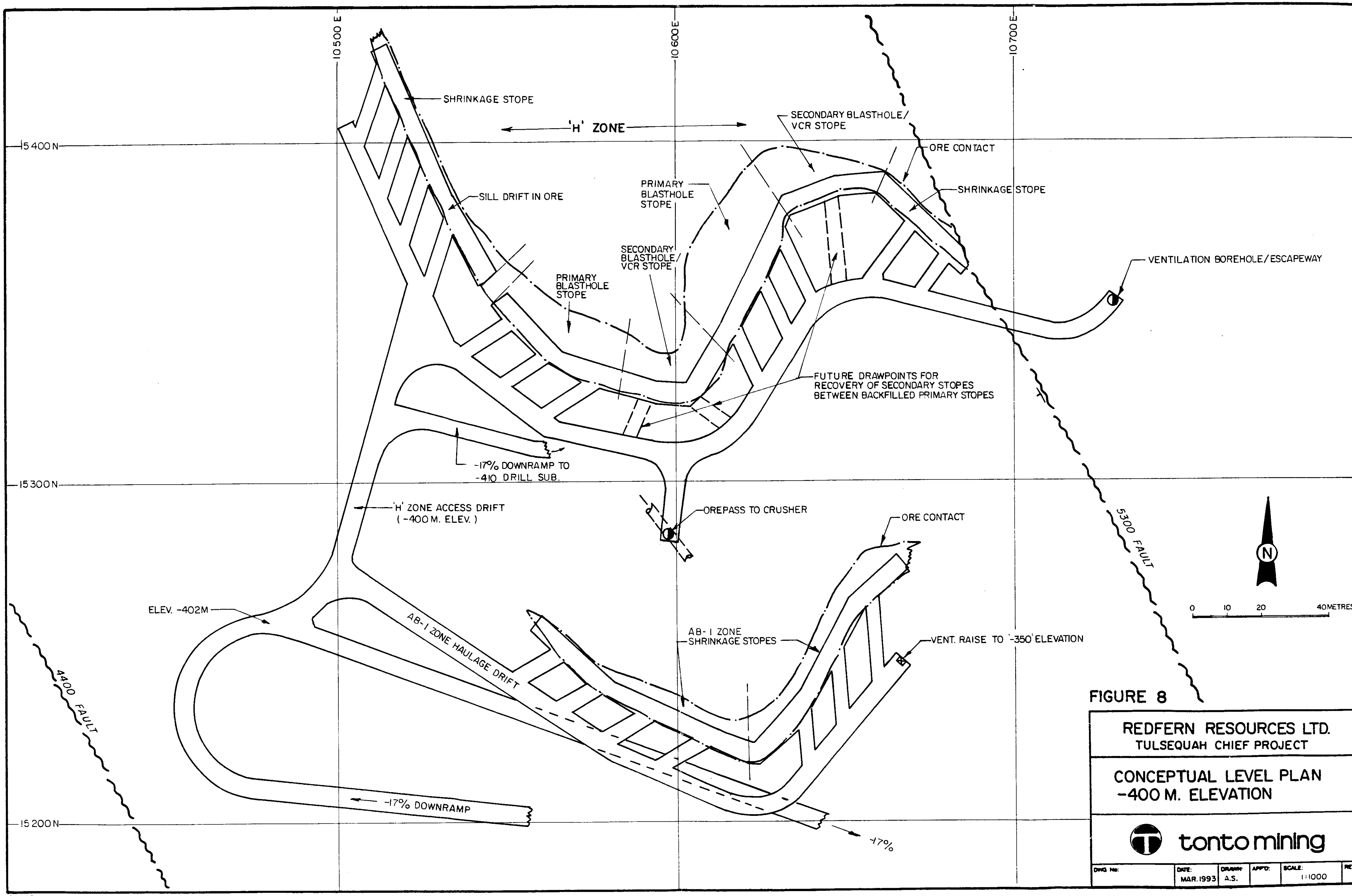

FIGURE 8

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

CONCEPTUAL LEVEL PLAN
-400 M. ELEVATION

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	MAR. 1993	A.S.		1:1000	

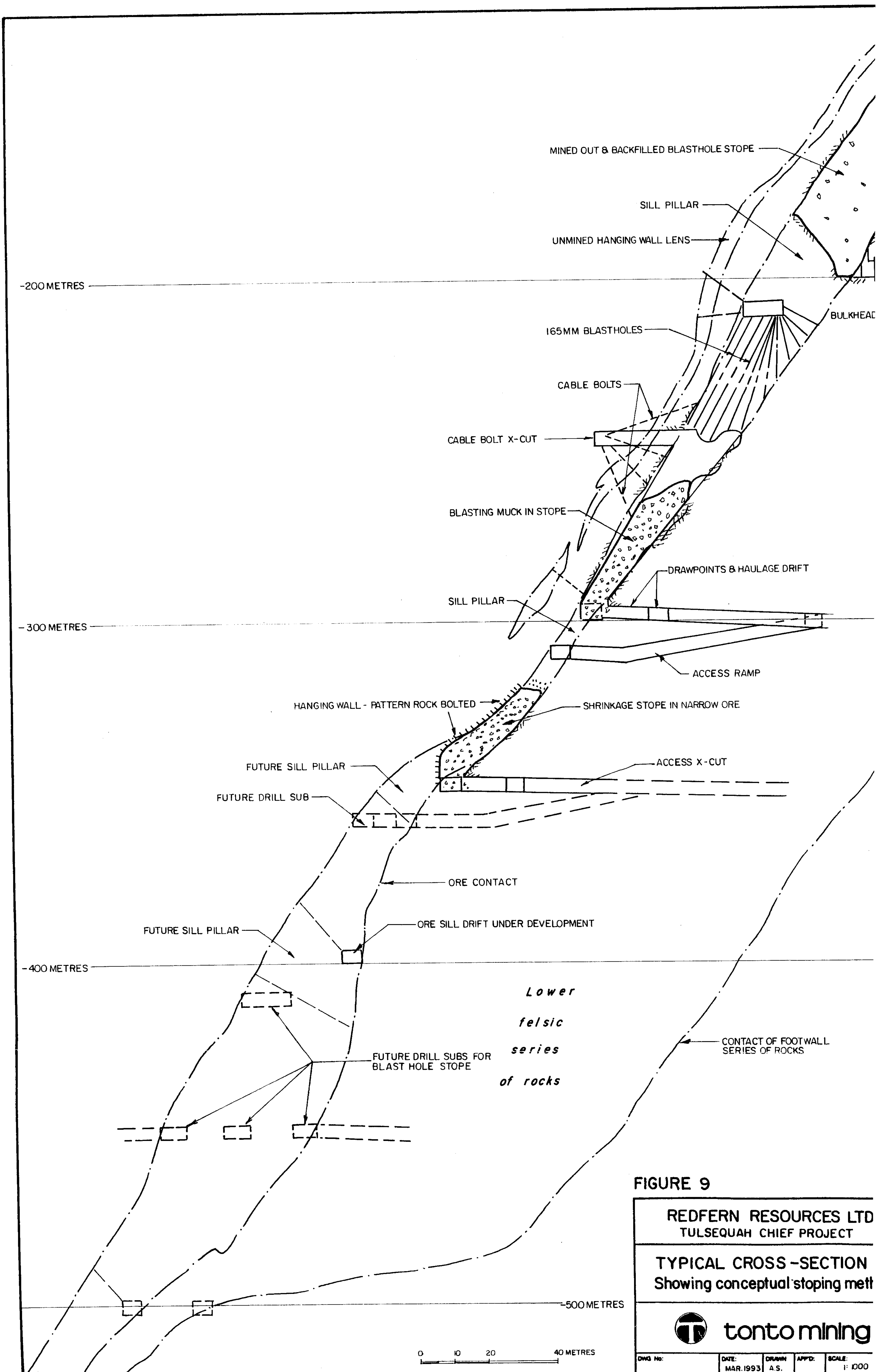

FIGURE 9

REDFERN RESOURCES LTD
TULSEQUAH CHIEF PROJECT

TYPICAL CROSS-SECTION
Showing conceptual stoping met

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	MAR.1993	A.S.		1:1000

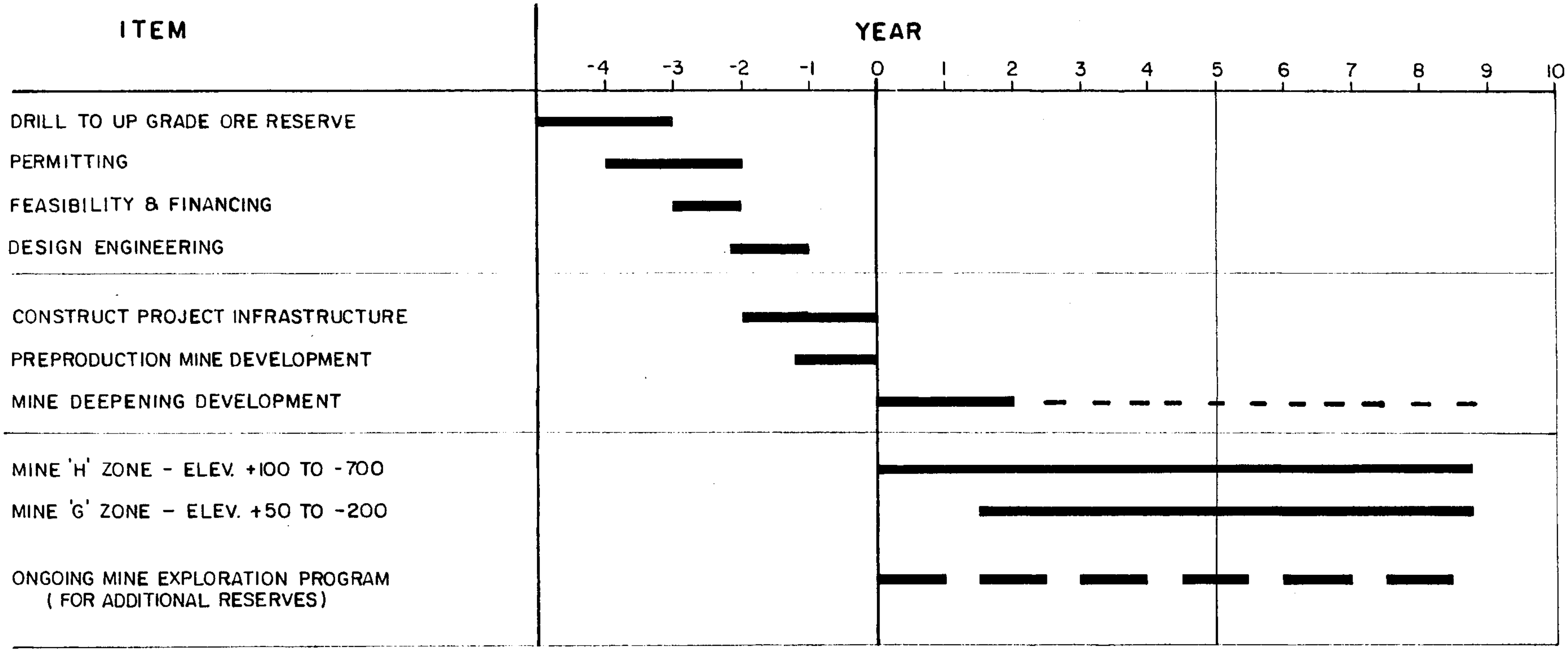

FIGURE 10

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

MINE CONSTRUCTION AND PRODUCTION SCHEDULE

tonto mining

DWG No:	DATE:	DRAWN:	APP'D:	SCALE:	REV.
	MAR. 1993	A.S.		—	

CHONG

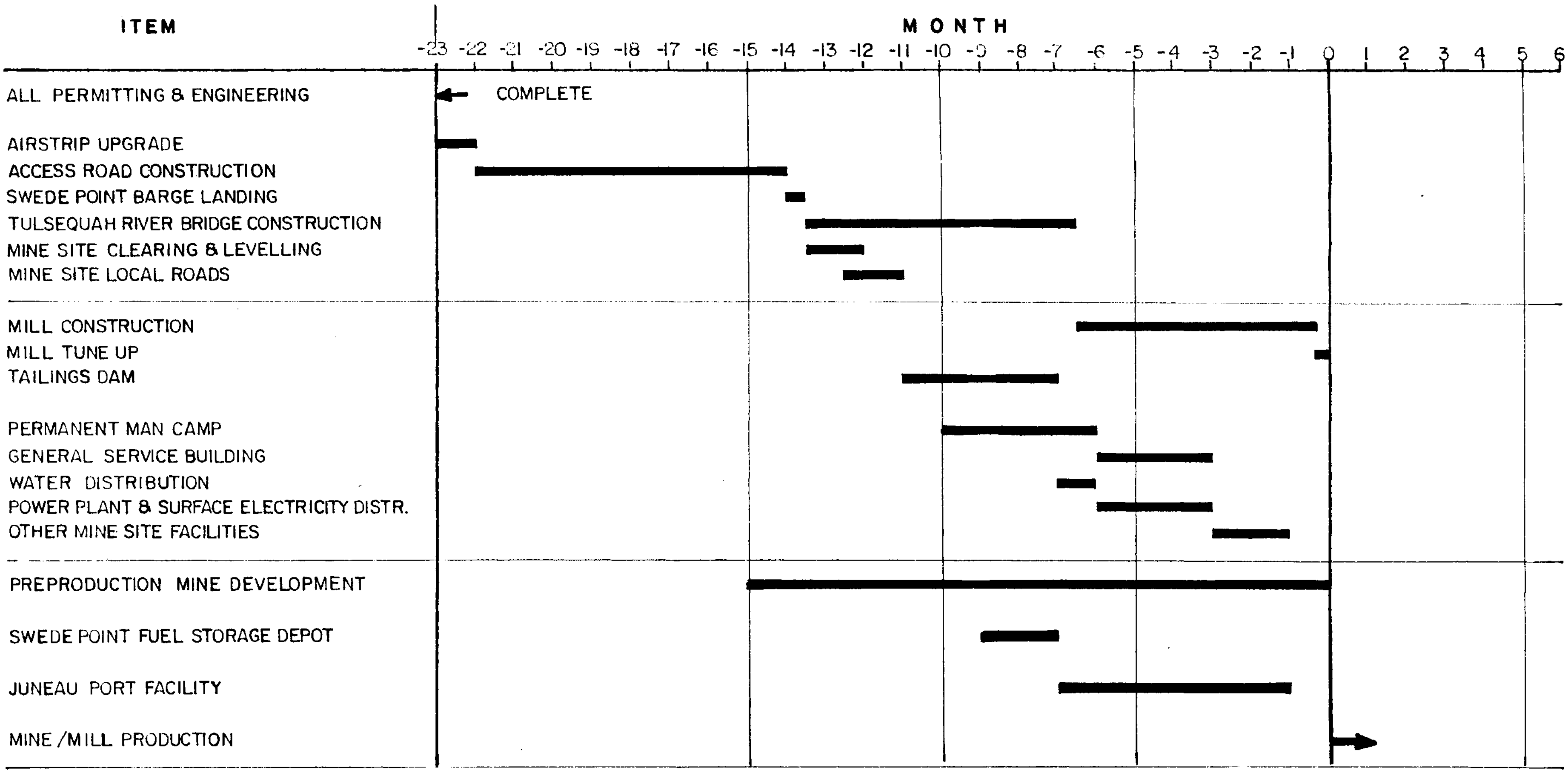

FIGURE 11

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

PREPRODUCTION SCHEDULE

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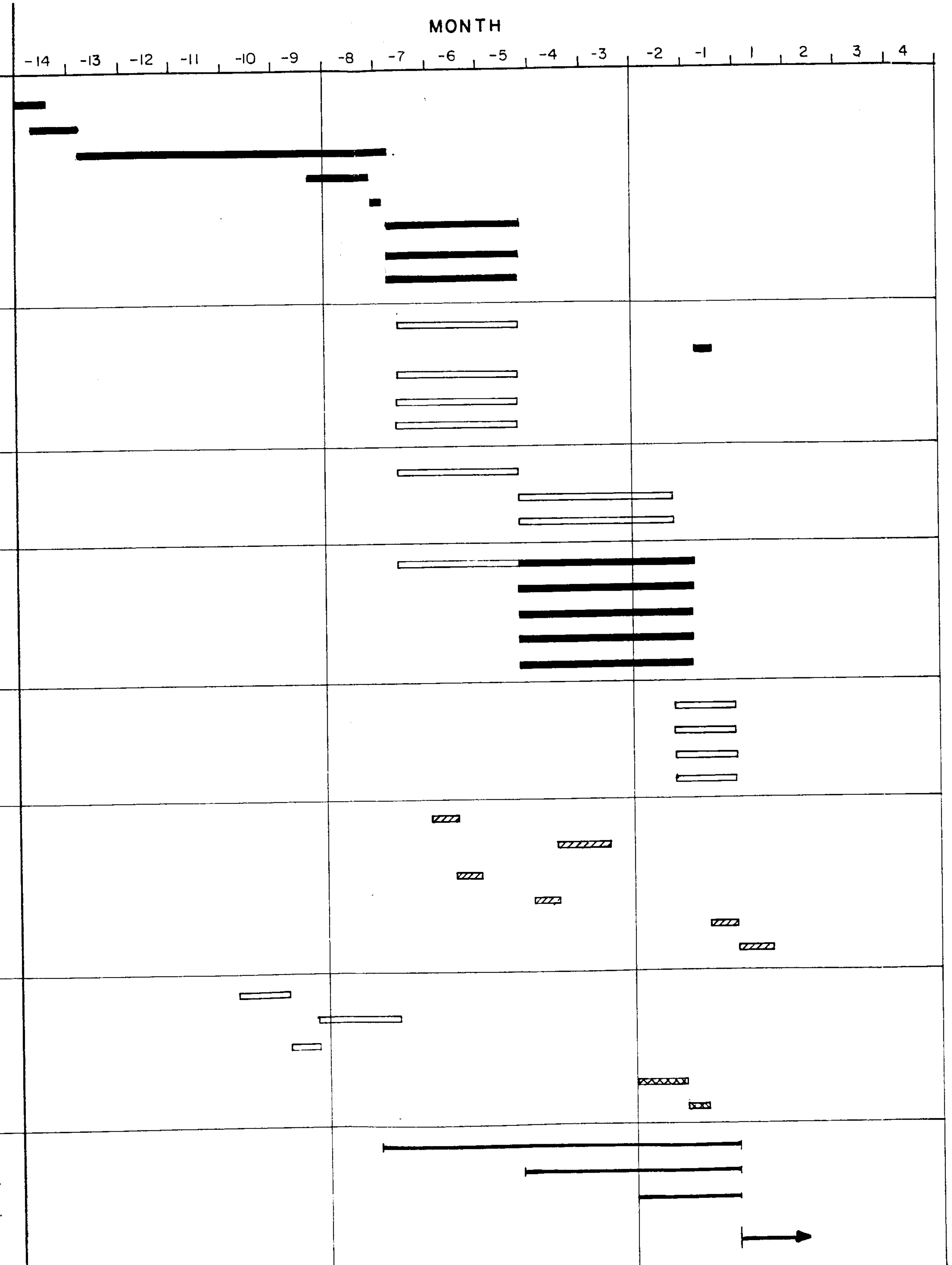
DWG No:	DATE:	DRAWN	APP'D:	SCALE:	REV.
	MAR. 1993	A.S.		—	

ITEM	X-SECTION SIZE (M)	QUANTITY	UNITS
CONTRACTOR SET UP			
DECLINE PORTAL		40	L.M.
CONVEYOR DECLINE	3.5x6.5	1190 EQUIV.	L.M.
RAMP TO O ELEV.	4 x 5.3	240	L.M.
DRIFT TO -10 ELEV.	4 x 5.3	60	L.M.
O ELEV. HAULAGE & 'H' SILL DRIFT	4 x 5.3	640	L.M.
O ELEV. 'H' DRAWPOINTS	4 x 4	180	L.M.
-10 ELEV. 'H' SILL DRIFT	4 x 5.3	220	L.M.
HOISTROOM ACCESS X-CUT	4 x 5.3	20	L.M.
HOISTROOM EXCAVATION		2000	CU. M.
TRANSFER CONVEYOR X-CUT	4 x 5.3	30	L.M.
-50 ELEV. SHAFT STATION	6 x 6	40	L.M.
UNDERGROUND REPAIR SHOPS		4300	M3
RAMP TO -50 ELEV.	4 x 5.3	100	L.M.
-50 ELEV. HAULAGE & SILL DRIFTS	4 x 5.3	1060	L.M.
-50 ELEV. DRAWPOINTS	4 x 4	160	L.M.
MAIN DECLINE TO -225 ELEV.	4 x 5.3	975	L.M.
-100 ELEV. SHAFT ACCESS	4 x 5.3	80	L.M.
-100 ELEV. OREBODY ACCESS	4 x 5.3	70	L.M.
-100 ELEV. HAULAGE & 'H' SILL DRIFT	4 x 5.3	650	L.M.
-100 ELEV. 'H' DRAWPOINTS	4 x 4	230	L.M.
-130 ELEV. ACCESS X-CUT	4 x 5.3	80	L.M.
-130 'H' ORE DRILL SUBS	4 x 5.3	160	L.M.
-150 ELEV. 'H' HAULAGE & SILL DRIFT	4 x 5.3	180	L.M.
-150 ELEV. 'H' DRAWPOINTS	4 x 4	30	L.M.
O ELEV. TEMPORARY VENT. RAISE	1.8 x 2	60	L.M.
-50 ELEV. SHRINKAGE STOPE RAISES	1.8 x 2	120	L.M.
-50 ELEV. VENT. RAISE	1.8 x 2	60	L.M.
-100 ELEV. SHRINKAGE STOPE RAISE	1.8 x 2	60	L.M.
-150 ELEV. VENT. RAISE	1.8 x 2	60	L.M.
ROPE RAISE & DUMP RAISES	1.8 x 2	110 EQUIV.	L.M.
5400 LEVEL TRACK DRIFT	2.75 x 275	160	L.M.
5200 LEVEL TRACK DRIFT	2.75 x 2.75	270	L.M.
REHAB 5200 LEVEL	2.75 x 2.75	1100	L.M.
RAISE BORE 5400 TO -50 AIRWAY	2.4 Ø	170	L.M.
INSTALL PERMANENT VENT. FAN		LOT	
DEFINITION DIAMOND DRILL		10,000	L.M.
CABLE BOLT HANGING WALL		2000	L.M.
PRODUCTION BLASTHOLE DRILLING		2000	L.M.
COMMENCE STOPE PRODUCTION			



LEGEND

- Contractor development crew Nº 1
- Contractor development crew Nº 2
- Raise crew
- Raise bore crew
- Company crews

FIGURE 12

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

UNDERGROUND PREPRODUCTION DEVELOPMENT SCHEDULE

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DWG No:	DATE: MAR. 1993	DRAWN: A.S.	APP'D:	SCALE: –	REV:

CHONG

ITEM	X-SECTION SIZE (M)	QUANTITY	UNITS
SHAFT CONTRACTOR SET-UP			
PREP. / FOUNDATIONS / HOIST INST.			
SINK / EQUIP SHAFT	2.75 x 6.1	763	L.M.
EXC. STATIONS, SUMPS, POCKETS			
INST. LOADING POCKET		1	
CHANGEOVER CONVEYANCES		LOT	
-350 M ELEV. TRACK X-CUT & DRIFT	2.75 x 2.75	455	L.M.
-500 M. ELEV. TRACK X-CUT & DRIFT	2.75 x 2.75	500	L.M.
-650 M. ELEV. TRACK X-CUT & DRIFT	2.75 x 2.75	560	L.M.
-700 M. ELEV. TRACK X-CUT	2.75 x 4	375	L.M.
MAIN SETTLING SUMPS	3 x 3	50	L.M.
CRUSHER ROOM EXCAVATION		1700	CU. M.
CRUSHER ACCESS RAISE	1.8 x 2	45	L.M.
CRUSHER ORE PASS RAISE	1.8 x 2	45	L.M.
CRUSHER / CONVEYOR INSTALLATION		LOT	
INSTALL PORTAL CONVR. - FEEDER		1000	L.M.
VENTILATION BOREHOLES			
-50 TO -200 M. ELEV.	2.4 Ø	160	L.M.
-200 TO -350 M. ELEV.	2.4 Ø	160	L.M.
-350 TO -500 M. ELEV.	2.4 Ø	160	L.M.
-500 TO -650 M. ELEV.	2.4 Ø	160	L.M.
-650 TO -700 M. ELEV.	1.8 Ø	55	L.M.
ORE PASS BOREHOLES			
-200 TO -350 M. ELEV.	1.8 Ø	160	L.M.
-350 TO -500 M. ELEV.	1.8 Ø	160	L.M.
-500 TO -650 M. ELEV.	1.8 Ø	160	L.M.
WASTE PASS BOREHOLES			
-200 TO -350 M. ELEV.	1.8 Ø	160	L.M.
-350 TO -500 M. ELEV.	1.8 Ø	160	L.M.

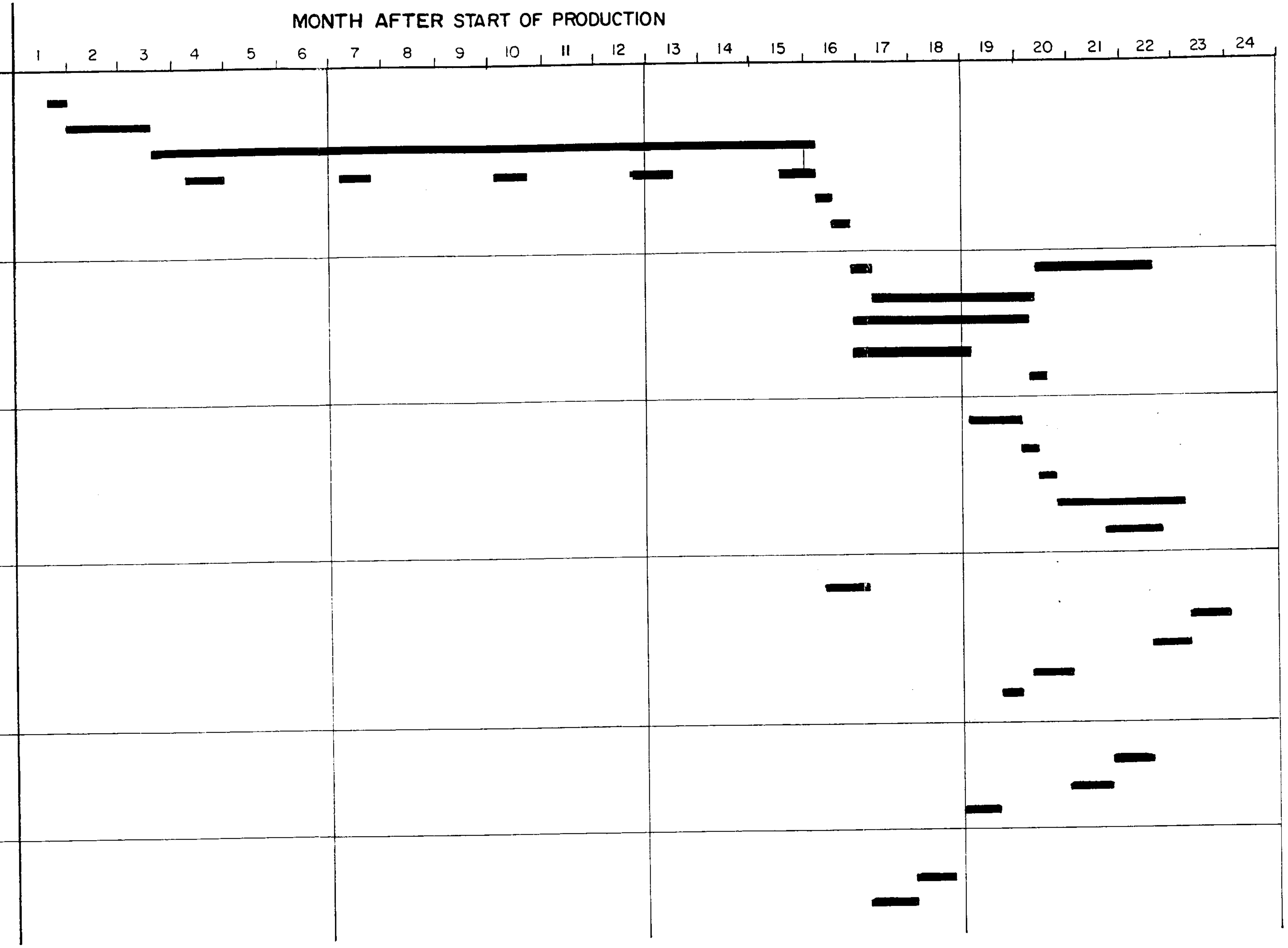

FIGURE 13

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

MINE DEEPENING SCHEDULE

tonto mining

DWG No:	DATE: MAR. 1993	DRAWN: A.S.	APP'D:	SCALE: –	REV:

CHONG

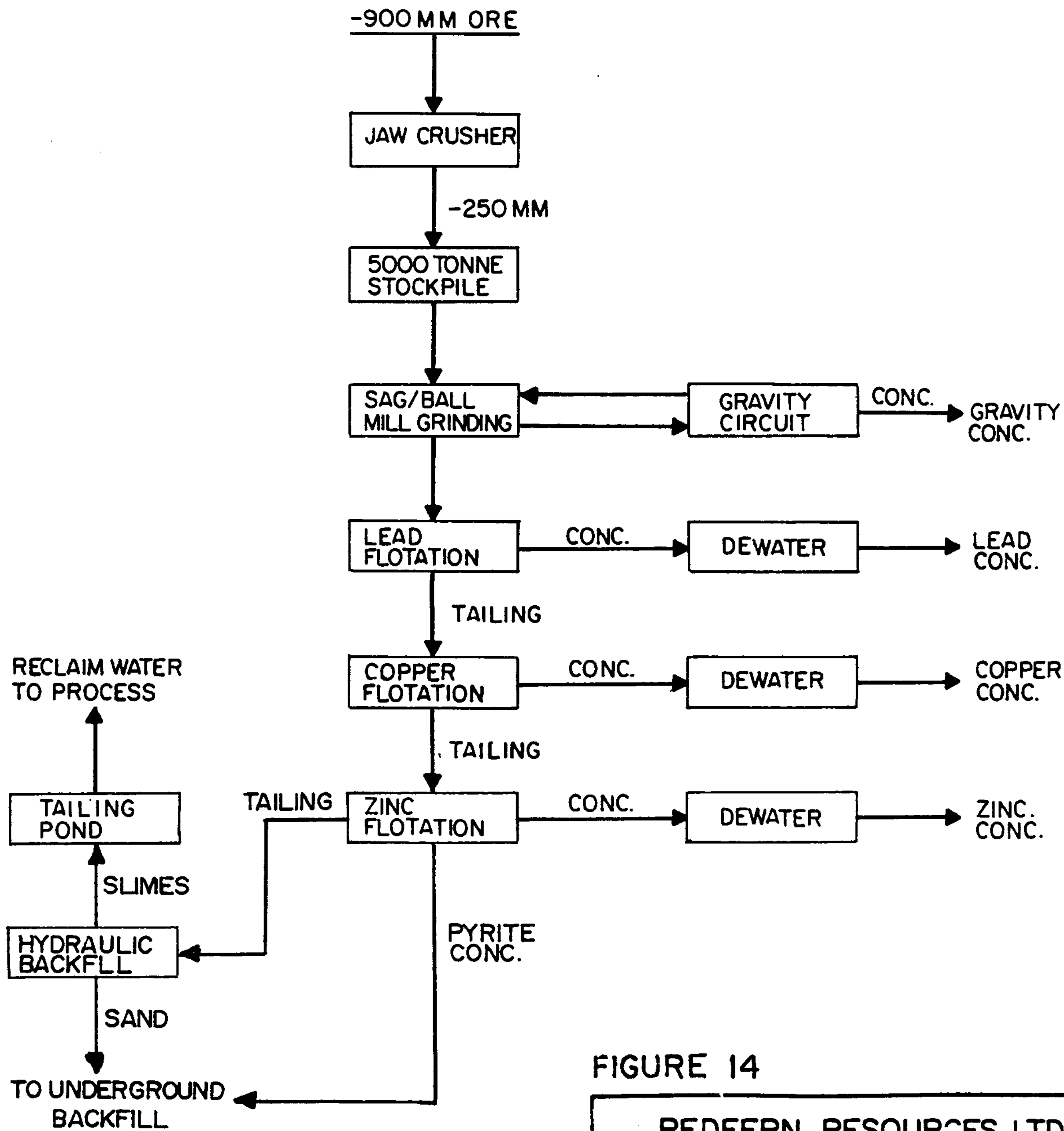

FIGURE 14

REDFERN RESOURCES LTD.
TULSEQUAH CHIEF PROJECT

TULSEQUAH
PROCESS FLOWSHEET

tonto mining

DWG No:	DATE:	DRAWN	APP'D:	SCALE:	REV.
	MAR. 1993	A.S.			

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CERTIFICATE OF DISCLOSURE

1. The attached pre-feasibility study on the Tulsequah Chief Project, dated April 30, 1993, was prepared at the request of Redfern Resources Ltd.

2. The study has been completed by experienced personnel of Tonto Mining, A Division of Dynatec Mining Limited, Burnaby, British Columbia, a company that provides mine contracting and engineering services to the mining industry.

3. The company believes the results and conclusions of the study to be realistic and practical.

4. The company and its employees involved with this study have no interest, nor do they expect to receive any interest, direct or indirect in Redfern Resources Ltd.

5. The company hereby grants permission for Redfern Resources Ltd. to use this report in any company documents including, but not limited to, statements of material facts and prospectus fillings

Dated at Burnaby, B.C. this 30th day of April 1993.

Robin McLatchy, P. Eng.
Chief Engineer,

RDM/wc