Ladner Creek
92HKW003

Tom Schroeter



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# THE CAROLIN MINES

# LADNER CREEK

# GOLD CONCENTRATOR

BY

R. SAMUELS

MILL SUPERINTENDENT

Present to:

CIM

Sixth Annual District 6 Meeting
Victoria, B.C.
October 31, 1981

## LOCATION:

Carolin's Ladner Creek project site is in southern British Columbia approximately 150 km. E.N.E. of Vancouver (Figure 1). It is located on the south west fork of Ladner Creek in the rugged North Cascade Mountains. Access to the site is from the town of Hope (elevation 136 ft.) via a 30.3 km. gravel road. A good portion of the initial 23.3 km. is on the old Kettle Valley Railway grade which follows the Coquihalla River and rises up to the elevation 1300 ft. at the entry of Ladner Creek. The final 7 km. of road along Ladner Creek is steeply graded: rising quickly to the concentrator elevation of 2650 ft.

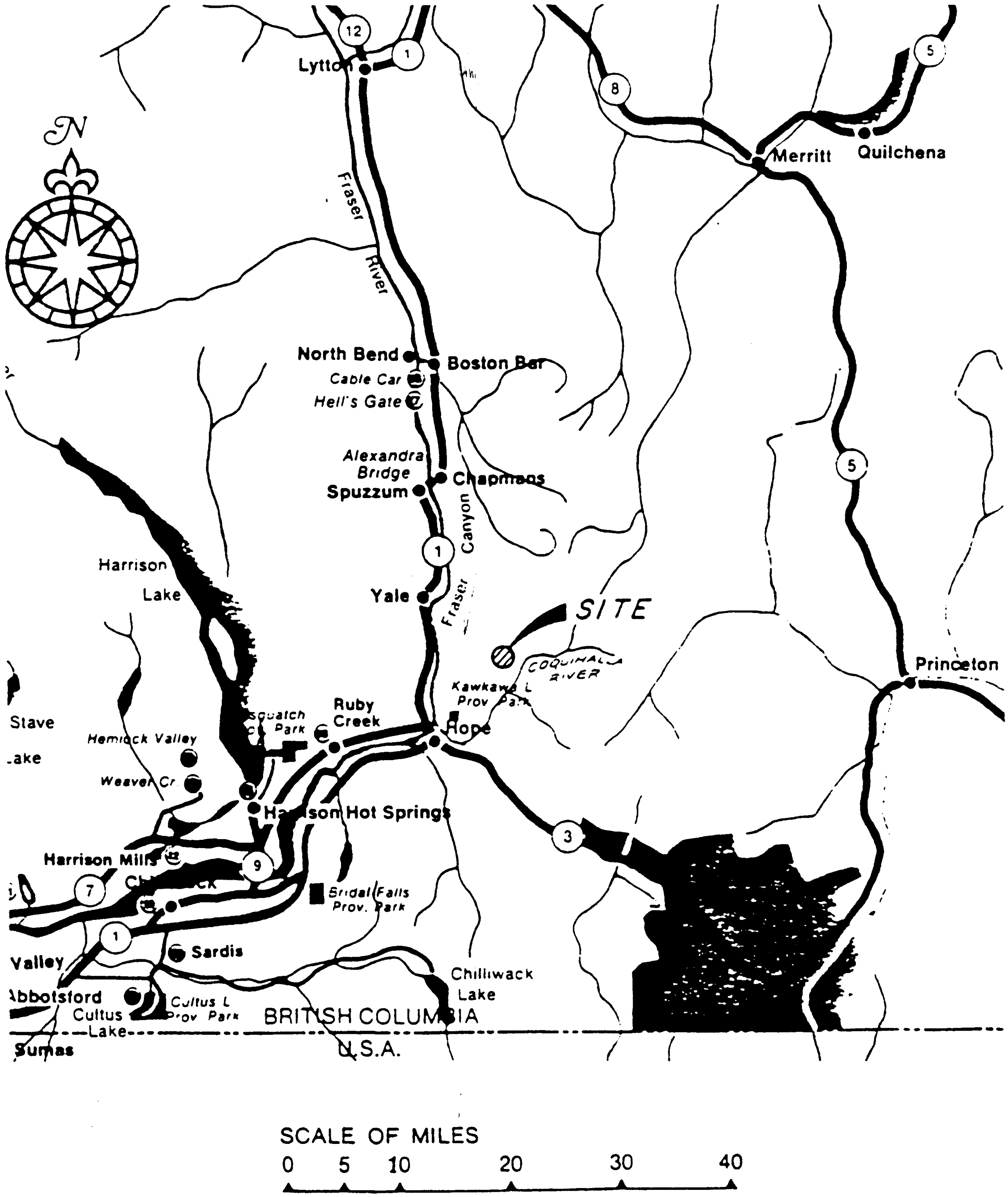
The Coquihalla Highway, a major artery which will link Hope and Merritt, is currently under construction on the east side of the access valley. Completion of the route will have significant impact of transportation to the site.

## HISTORY:

The immediate area contains a number of lode gold deposits known as the Coquihalla Gold Belt. Discovery of the Gold Belt was actually a by-product of the 1850 Fraser River Gold Rush. Placer miners working the tributaries of the Fraser found gold in the Coquihalla River and follow up prospecting revealed lode gold deposits.

The first recorded workings were on the Ward Claim on Siwash Creek, and from 1913 to 1942 sporadic production occurred along the Gold Belt from the Emancipation, Aurum, Georgia #2, Pipestem and Idaho mines. The total recorded production was 3,913 oz. from unknown tonnage - the Aurum has a recorded production of 500 oz. from 500 tons.

LOCATION Figure 1



The area remained relatively dormant until Carolin Mines Ltd. acquired 8 crown granted claims (which included the Aurum and Idaho zones) from Summit Mining in 1972. That same year, additional claims were staked adjacent to the crown grants.

Drill programs in 1974 by Precambrian Shield and by Carolin in 1975 and 1976 outlined a zone of gold bearing mineralization under the Idaho showing. A 1200 ft. decline adit was driven in 1978 and underground drilling further defined the ore zone. Later in the year Carolin reached agreement with a consortium of petroleum companies and a mining exploration company to finance a feasibility study of the project. The participants, known as the Aquarius Group included Ocelot Industries Ltd., Great Basins Peteroleum Ltd., Windjammer Power and Gas Ltd. all of Calgary and Aquarius Resources Ltd. of Edmonton.

The feasibility report (completed in May 1979) was prepared by Kilborn Engineering B.C. Ltd. with input from Ker, Priestman & Associates Ltd., Golder Associates and Britton Research Ltd. Based on the study, the Aquarius Group committed to spend $19.3 million to put the Idaho zone mine into production for a 50% joint venture interest. Carolin Mines Ltd. manages the joint venture, while Kilborn Engineering was retained as construction managers.

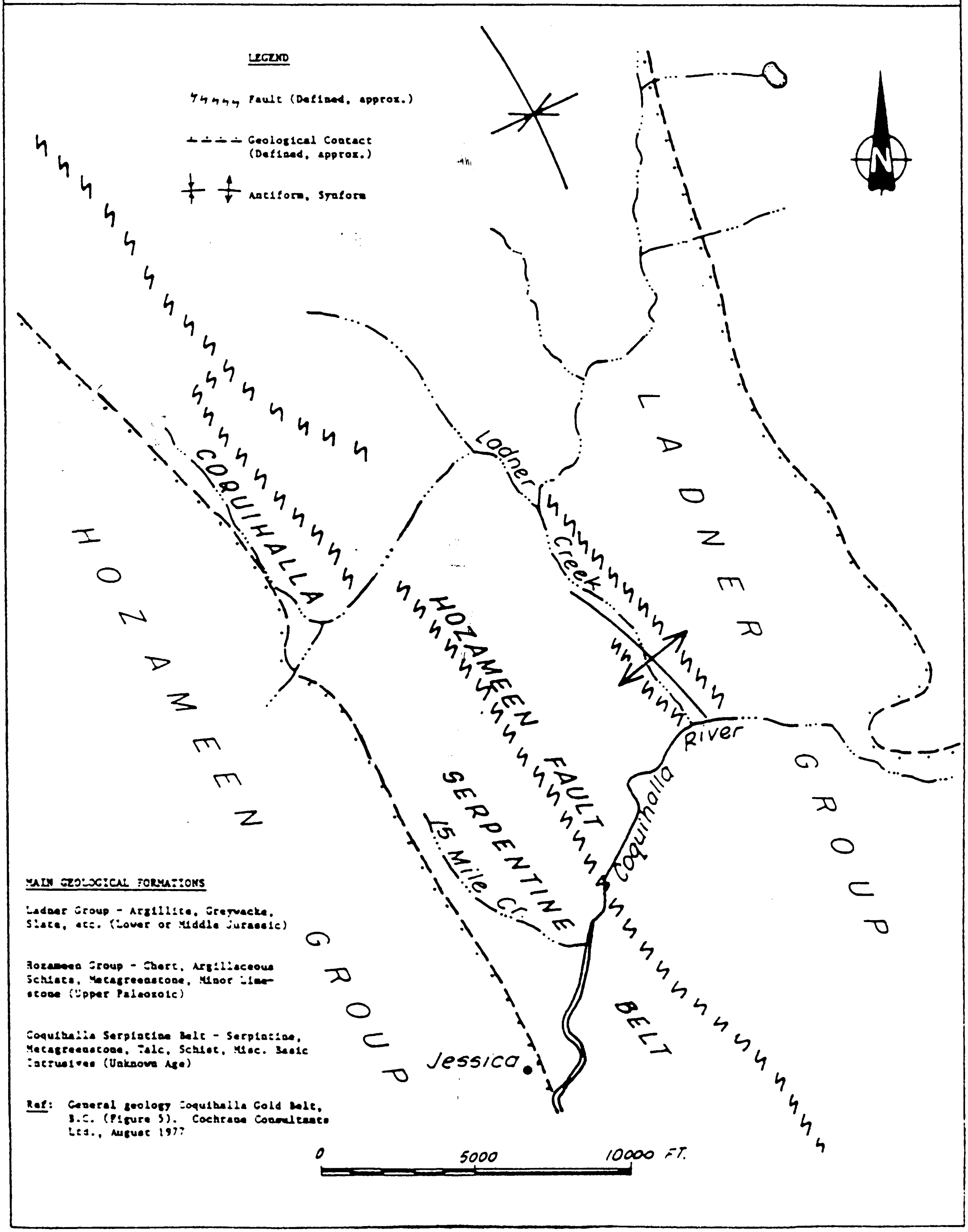
GEOLOGY & ORE RESERVES:

The Cascade Mountains in the deposit area are composed of a mixture of sedimentary and volcanic rocks strongly folded by heat and pressure.(Figure 2)

The principal geological feature of the area is the Hozameen Fault (composed of talc and carbonate, varying from 10 cm. to 2 m. in width) which separates the main geological formations. The Aurum Mine was within the fault zone. West of the fault is the Coquihalla Serpentine Belt consisting of serpentine, metagreenstone, talc, schist and miscellaneous basic intrusives.

# GEOLOGICAL MAP

**Figure 2**



Further west lies the Hozameen Group comprising chert, argillaceous schists, metagreenstone and minor limestone. The eastern contact of the Hozameen Group is the western limit of all significant gold mineralization found to date.

East of the Hozameen Fault is discontinous unit of extrusive greenstone followed by the Ladner Group of sediments (comprising of greywacke, argillite and graphitic argillites or slates). The Ladner Group are host rocks for the main occurences in the Coquihalla Gold belt, including the Idaho deposit.

There are two main controls to mineralization in the Idaho zone:

1. Quartz viens associated with steeply dipping faults sub-parallel to the Hozameen Fault.

2. Gently dipping zones of replacement and mineralization within certain sedimentary beds of the Ladner Group.

The ore itself is not visually striking, averaging less than 3% by volume sulphide. The principal sulphides, in decreasing order of abundance are pyrrhotite, pyrite, and aresenopyrite. Also present are traces of chalcopyrite, magnetite and spahlerite. No specific sulphide is the major host of gold: it is very fine and disseminated in the group of sulphides, but also includes minor occurences of visual native gold.

In the feasibility study, drill indicated ore reserves (based on a cut-off of 0.08 ounces per ton) were estimated at approximately 1,650,000 sdt. grading 0.14 ounces of gold per ton. Silver content is 0.03 ounces per ton. In light of higher gold prices, subsequent calculations using a 0.05 ounces per ton cut-off grade increased the reserve to approximately 2,100,000 sdt. grading 0.12 ounces per ton. This would provide more than 4 years of mill feed.

It is probable that ore reserves will be increased by future exploration drilling. The Idaho zone is open and untested in two directions: most notable is down plunge to the north.

MINING:

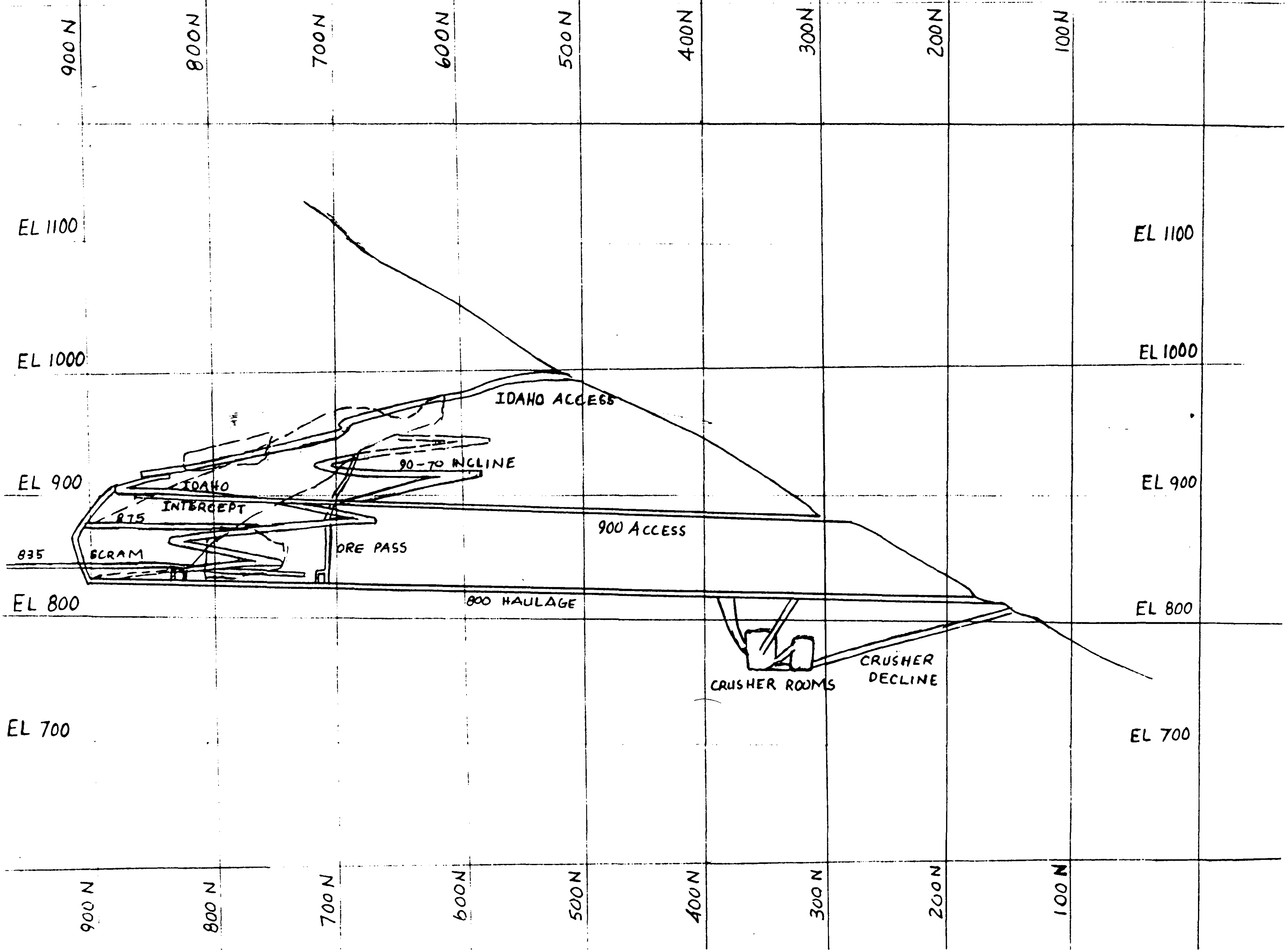
Development mining commenced in December 1979 with portals collared at the 900 m. (2953 ft.) and 820 m. (2690 ft.) levels. (Figure 3). The 900 access has dimensions 3.5 m. wide x 4 m. high to allow entry of 5 cu.yd. L.H.D. equipment. It is driven horizontal for a distance of 915 m. from which ramps turn upward and downward to establish drawpoints and drilling sub-levels at the stopes. Maximum ramp inclination is 18%. The northern extremity of this drift intercepts the Idaho decline. This exploration decline, because of its location is not suitable for mining development but is being utilized for ventilation, and as an escapeway.

The 820 m. portal is a 3 m. x 3m. access driven for use as a tracked haulage level. It is driven in a straight line at an inclination in favour of loaded trains of 0.4%. It is approximately 1015 m. in length and will eventually be advanced to the north as the main exploration level. It is linked to the intermediate levels and the Idaho decline by a vent raise, thus allowing air movement throughout the workings.

The crusher portal is collared at elevation 795 m. and the 4 m x 3.4 m. decline slopes at 18½% to elevation 780 m. where it enters the cone and jaw crusher rooms. The jaw room is joined to the 820 m. haulage level by an 1800 t coarse ore storage pocket and a raise required for ventilation and a secondary access.

Long hole open stoping has been chosen as the mining method. The mining has started at the northerly end of the present known ore zone and will proceed sequentially upward towards the surface. This approach offers the probability of the quickest return of capital because of marginally higher ore grade, maximum size of stopes, and minimum amount of stope development. When the stopes are exhausted they will be backfilled with a cemented coarse fraction of mill tailings.

IDAHO ZONE
FIGURE 3



Due to the differences in size of the various lenses and their irregular wall outlines, it is not possible to introduce a standard stoping pattern. Each stope must be individually engineered.

Production drilling of the 2 inch diameter longholes is done with a Simba H 221 electric over hydraulic drill at a current rate of 500 ft. per drill shift. Ring burden is 1.5 m. and toe spacing is 3.5 m. (5 ft. x 11½ ft.). This will break 2.35 short tons per foot drilled with an initial power factor of 0.6 lb. of explosive per short ton broken. Changes in both burden and spacing of drill holes can be expected after initial blasting results have been evaluated.

Ore recovery from the stope drawpoints will be done with four Jarco JS 500 - 5 cu. yd. scooptrans and runs to the ore passes will range from 120 m. to 240 m. Ore is then gathered on the 820 m. level from the ore pass chutes and transported to a coarse ore pocket above the primary crusher. The haulage equipment includes a 10 ton electric trolley or battery operated locomotive and a fleet of 165 cu. ft. capacity side dumping ore cars.

The mine will operate 2 shifts a day (8-4) five days a week to produce enough ore to sustain a continuous seven day per week milling operation at 1500 sdt. per day.

MILLING:

Based on design throughput, the Carolin Mill is the largest gold mill in Western Canada, and ranks sixth amongst the largest gold concentrators in Canada as shown in the following graphic.

| COMPANY | TONNAGE sdt. | GRADE oz/t Au |
|---|---|---|
| Pamour-Porcupine | 3000 | 0.082 |
| Pamour-Schumacher | 3000 | 0.082 |
| Lamaque | 2100 | 0.131 |
| Dome | 1985 | 0.206 |
| Agnico Eagle | 1500 | 0.191 |
| Carolin | 1500 | 0.14 |
| Sigma | 1400 | 0.195 |
| Kerr Addison | 1300 | 0.256 |
| Camflo | 1250 | 0.129 |

| | | |
|---|---|---|
| Giant Yellowknife | 1250 | 0.21 |
| Belmoral | 1200 | 0.14 |

Most of Canada's gold mills have been in production for many years, which allows this modern facility to be unique in construction and design. Never the less, the metallurical flowsheet has remained conventional, particularly in the area of cyanidation and gold precipitation...the Merrill-Crowe process is employed.

Location of the crusher is not typical. The steep valley and lack of working area prompted a decision to construct the total crushing operation underground. The high cost of excavating rock limited the facility to two stage open circuit crushing. ( Figure 4 )

Minus 24 inch material is recovered from the coarse ore pocket with a Ross chain feeder. The controlled tonnage is fed to a 36 inch x 48 inch Birdsborough Buchanan jaw crusher. Minus 5 inch crusher discharge is conveyed to, and sized by a 5 ft. x 8 ft. rod deck vibrating screen with 5/8" openings. The screen oversize is then reduced to minus 5/8 inch by a 5½ ft. shorthead Symons cone crusher. A combined product of screen undersize and cone crusher discharge is conveyed up the decline portal, over the valley to a 2000 ton draw fine ore bin. The design called for two 2,000 ton fine ore bins but construction of one has been delayed until the summer of 1982. To compensate, the coarse ore pocket was enlarged to 1800t from 500 t and a crushing crew will be required one shift on the weekend to maintain continuous milling.

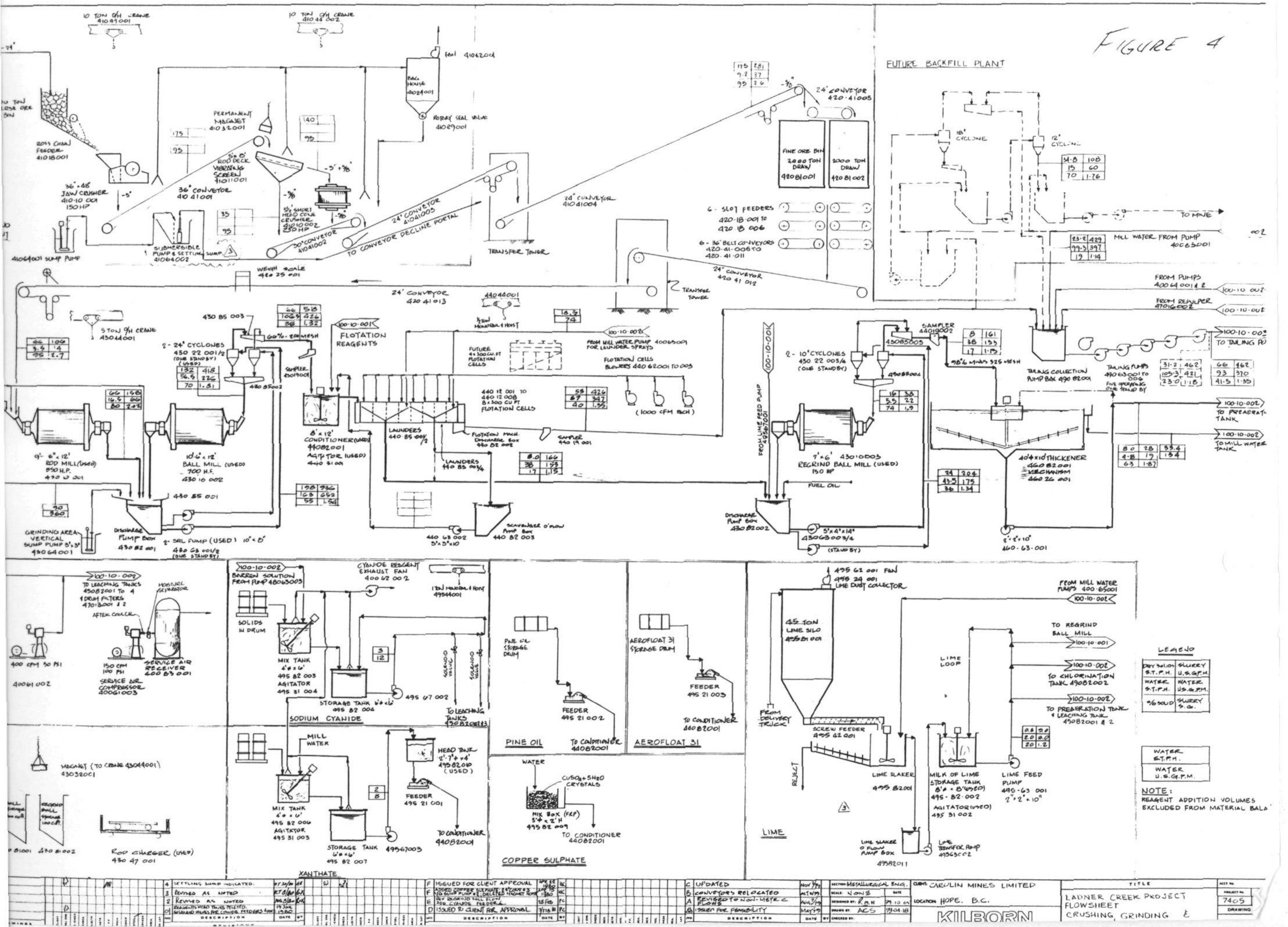
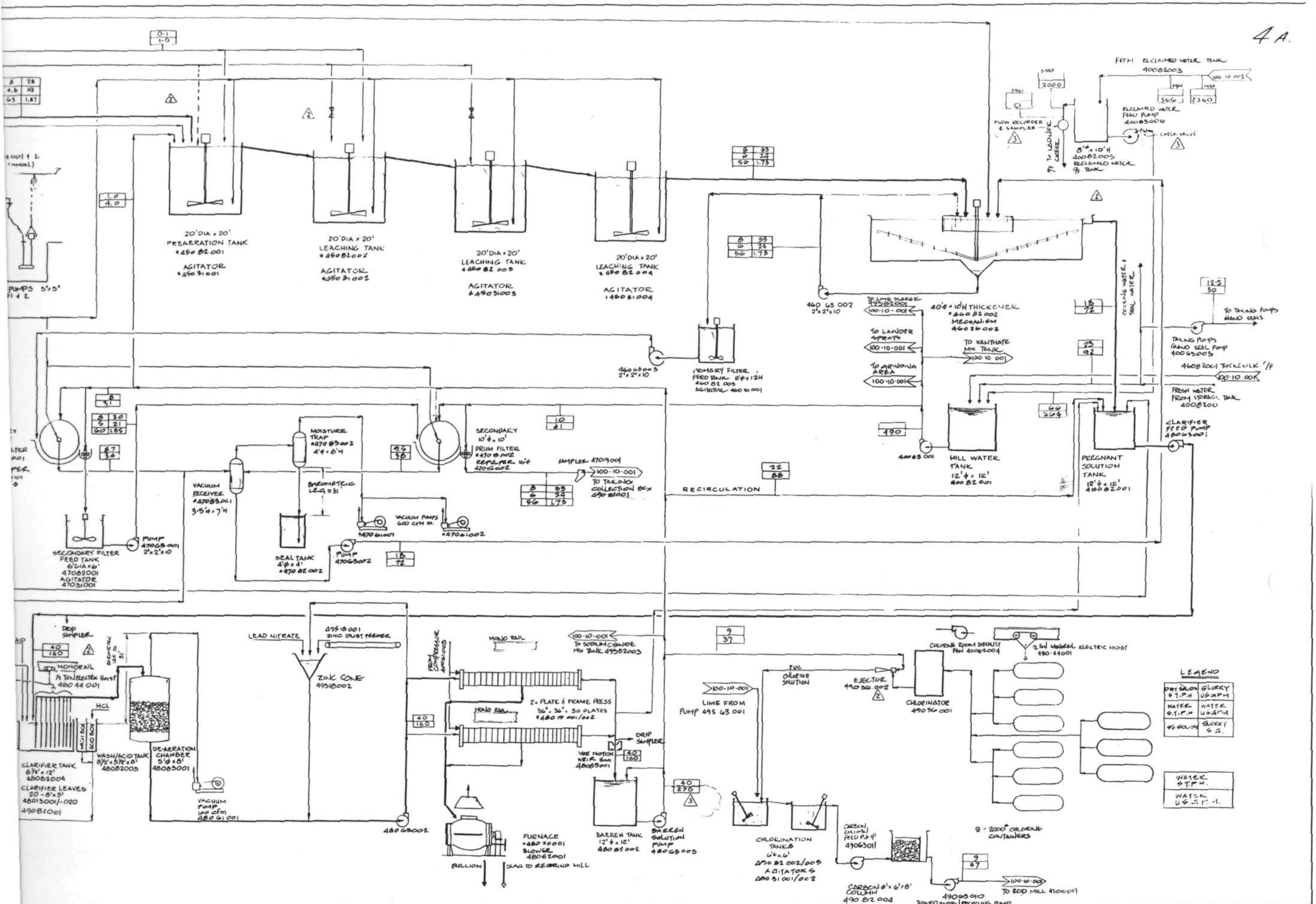


FIGURE 4
FUTURE BACKFILL PLANT
10 TON O/H CRANE 41041001
10 TON O/H CRANE 41048002
BAG HOUSE 4102400l
ROTARY SEAL VALVE 41029001
PERMANENT MAGNET 41032001
ROSS CHAIN FEEDER 41018001
36" x 48" JAW CRUSHER 41010 001 150 HP
5' x 8' ROD DECK VIBRATING SCREEN 41011001
36" CONVEYOR 41041001
5½' SHORT HEAD CONE CRUSHER 41010002 250 HP
30" CONVEYOR 41041002
24" CONVEYOR 41041003
TO CONVEYOR DECLINE PORTAL
24" CONVEYOR 41041004
24" CONVEYOR 420-41005
FINE ORE BIN 2000 TON DRAW 42081001
2000 TON DRAW 42081002
6 - SLOT FEEDERS 420-18-001 TO 420 18 006
6 - 36" BELT CONVEYORS 420-41-006 TO 420-41-011
24" CONVEYOR 420 41 012
TRANSFER TOWER
SUBMERSIBLE PUMP & SETTLING SUMP 41064002
41064001 SUMP PUMP
WEIGH SCALE 420 25 001
24" CONVEYOR 420 41 013
5 TON O/H CRANE 43044001
2 - 24" CYCLONES 430 22 001/2 (ONE STANDBY)
60% - 200 MESH
FLOTATION REAGENTS
SAMPLER 43019001
FUTURE 4 x 300 CU FT FLOTATION CELLS
FROM MILL WATER PUMP 40065001 FOR LAUNDER SPRAYS
FLOTATION CELLS BLOWERS 440 62001 TO 003
(1000 CFM EACH)
440 12 001 TO 440 12 008 8 x 300 CU FT FLOTATION CELLS
8' x 12' CONDITIONER (USED) 44082001
AGITATOR (USED) 440 31 001
LAUNDERS 440 85 001/2
LAUNDERS 440 85 003/4
FLOTATION MACH. DISCHARGE BOX 440 82 002
SAMPLER 440 19 001
9' - 6" x 12' ROD MILL (USED) 850 H.P. 430 10 001
10'-6" x 12' BALL MILL (USED) 700 H.P. 430 10 002
GRINDING AREA VERTICAL SUMP PUMP 3" x 3" 43064001
DISCHARGE PUMP BOX 430 82 001
2 - SRL PUMP (USED) 10" x 8" 430 63 001/2 (ONE STANDBY)
SCAVENGER O'FLOW PUMP BOX 440 82 003
440 63 002 5" x 5" x 10
FROM LIME FEED PUMP 49563001
2 - 10" CYCLONES 430 22 003/4 (ONE STANDBY)
SAMPLER 44019002
98% MINUS 325 MESH
7' x 6' 430 10 003 REGRIND BALL MILL (USED) 150 HP
FUEL OIL
DISCHARGE PUMP BOX 430 82 002
5" x 4" x 14" 43063003/4 (STAND BY)
TAILING COLLECTION PUMP BOX 490 82 001
TAILING PUMPS 490 63 001 TO 006 FIVE OPERATING ONE STAND BY
40' x 10' THICKENER 460 82 001 MECHANISM 460 26 001
2" x 2" x 10" 460-63-001
TO MINE
MILL WATER FROM PUMP 40065001
FROM PUMPS 400 64 001 & 2
FROM REPULPER 47016002
TO TAILING POND
TO PREAERATION TANK
TO MILL WATER TANK
TO LEACHING TANKS 45082001 TO 4 & DRUM FILTERS 47013001 & 2
MOISTURE SEPARATOR
AFTER COOLER
400 CFM 50 PSI 40061002
150 CFM 100 PSI SERVICE AIR COMPRESSOR 40061003
SERVICE AIR RECEIVER 40083001
MAGNET (TO CRANE 43044001) 43032001
ROD CHARGER (USED) 430 47 001
BARREN SOLUTION FROM PUMP 48063003
CYANIDE REAGENT EXHAUST FAN 400 62 002
1 TON MONORAIL & HOIST 49544001
SOLIDS IN DRUM
MIX TANK 6' x 6' 495 82 003 AGITATOR 495 31 004
STORAGE TANK 6' x 6' 495 82 004
495 67 002
TO LEACHING TANKS 45082001/3
SODIUM CYANIDE
MILL WATER
HEAD TANK 2'-7" x 4' 49582010 (USED)
FEEDER 495 21 001
MIX TANK 6' x 6' 495 82 006 AGITATOR 495 31 003
STORAGE TANK 6' x 6' 495 82 007
49567003
TO CONDITIONER 44082001
XANTHATE
PINE OIL STORAGE DRUM
FEEDER 495 21 002
TO CONDITIONER 44082001
PINE OIL
AEROFLOAT 31 STORAGE DRUM
FEEDER 495 21 003
TO CONDITIONER 44082001
AEROFLOAT 31
WATER
$CuSO_4 + 5H_2O$ CRYSTALS
MIX BOX (FRP) 3'φ x 2'H 495 82 009
TO CONDITIONER 44082001
COPPER SULPHATE
495 62 001 FAN
495 24 001 LIME DUST COLLECTOR
45 TON LIME SILO 495 81 001
FROM DELIVERY TRUCK
SCREW FEEDER 495 62 001
REJECT
LIME SLAKER 495 82001
LIME SLAKER O FLOW PUMP BOX 49582011
LIME TRANSFER PUMP 49563002
LIME
FROM MILL WATER PUMPS 400-65001
TO REGRIND BALL MILL
LIME LOOP
TO CHLORINATION TANK 49082002
TO PREAERATION TANK & LEACHING TANK 45082001 & 2
MILK OF LIME STORAGE TANK 8' x 8' (USED) 495-82-002 AGITATOR (USED) 495 31 002
LIME FEED PUMP 495-63-001 2" x 2" x 10"
LEGEND
DRY SOLIDS S.T.P.H. | SLURRY U.S.G.P.M.
WATER S.T.P.H. | WATER U.S.G.P.M.
% SOLID | SLURRY S.G.
WATER S.T.P.H.
WATER U.S.G.P.M.
NOTE: REAGENT ADDITION VOLUMES EXCLUDED FROM MATERIAL BALA
4 SETTLING SUMP INDICATED.
3 REVISED AS NOTED
2 REVISED AS NOTED
F ISSUED FOR CLIENT APPROVAL
E REV. REGRIND BALL MILL FLOW, FOR CYANIDE FEEDERS.
D ISSUED TO CLIENT FOR APPROVAL
C UPDATED
B CONVEYORS RELOCATED
A REVISED TO NON-METRIC & FLOWS
ISSUED FOR FEASIBILITY
SECTION METALLURGICAL ENG.
SCALE NONE
DESIGNED BY R.M.N.
DRAWN BY ACS
CLIENT CARLIN MINES LIMITED
LOCATION HOPE. B.C.
KILBORN
TITLE
LADNER CREEK PROJECT
FLOWSHEET
CRUSHING, GRINDING &
7405
DRAWING

4A.




| NO. | DESCRIPTION | DATE | BY |
|---|---|---|---|
| 3 | CHECK VALVE, SAMPLER & FLOW REC ADDED IN RECLAIM WTR SYSTEM | OCT/30/80 | |
| 2 | REVISED AS NOTED | OCT 8/80 | |

REVISIONS

| NO. | DESCRIPTION | DATE | BY |
|---|---|---|---|
| 1 | RECLAIMED WATER 9⅝ TANK & PUMP ADDED PRESS FILTER TO PREGNANT SOL TANK LINE ADDED | 19 JUNE 1980 | |
| E | ISSUED FOR CLIENT APPROVAL | APR 22 1980 | |
| E | REVISED CLARIFIER OVERFLOW, DELETED 3 CHLORINE CONTAINERS | APR 15 1980 | |
| D | REV CARBON COLUMN & PUMP REV No. CHLORINE CONTAINERS | 28 FEB | PC |

REVISIONS

| NO. | DESCRIPTION | DATE | BY |
|---|---|---|---|
| C | ISSUED TO CLIENT FOR APPROVAL | 7/12/80 | PC |
| B | UPDATED | NOV 79 | |
| A | REVISED TO NON-METRIC FLOWS | AUG 79 | |
| 0 | ISSUED FOR FEASIBILITY | MAY 79 | |

REVISIONS

SECTION: METALLURGICAL ENG.
DESIGNED BY: R.M.N 79-10-05
DRAWN BY: A.C.S 79-04-18
CHECKED BY:
APPROVED BY: D.R.B 79-10-

CLIENT: CAROLIN MINES LIMITED
LOCATION: HOPE, B.C.
KILBORN

TITLE: LADNER CREEK PROJECT FLOWSHEET LEACHING, DEWATERING & PRECIPITATION

PROJECT NO. 7405
DRAWING NUMBER 100-10-03

Ore is recovered from the fine ore bin with three slot feeders discharging to three parallel belts, one of which is variable speed for fine control of the rodmill feed. The feeder belts discharge to a fixed speed belt which, in turn feeds the minus 5/8 inch ore into the rodmill. A weightometer monitors tonnage on the rodmill feed belt for metallurgical accounting purposes.

Primary grinding is done with a 9½ by 12 ft. Dominion rodmill with an inventory of 3½ inch rods. The rodmill discharge, along with the discharge from a secondary 10½ by 12 foot Dominion ballmill is pumped to a 24 inch cyclone in open circuit with the rodmill and closed circuit with the ballmill. The ore, with a Bond Work Index of 12.0 kw. per short ton is ground to 66% minus 200 mesh. A jig has not been installed in the grinding circuit at this time, but possible losses of coarse gold particles in flotation will be evaluated immediately.

After conditioning at pH 8.7 with 0.33 lb per ton copper sulphate, 0.33 lb per ton potassium amyl xanthate, 0.01 lb. per ton Aerofloat 31 and 0.08 lb. per ton pine oil a bulk rougher concentrate 12% by weight of the feed is recovered in 4 - 300 cu. ft. flotation cells. The metallurgical testwork showed only 91% of the gold was recovered in flotation and therefore, 50% more retention time than was actually used in the laboratory tests was designed in the circuit. The slurry is scavenged in 4 - 300 cu. ft. cells and the scavenger concentrate is discharged back to the conditioner tank. Scavenger tails are pumped to the tailings pond.

At this point, all that remains is 170-180 tons per day of bulk concentrate assaying 1.2 oz. per ton of gold. The flowsheet now becomes typical of most small gold mill operations. Rougher concentrate reports to a 10 inch cyclone and the underflow goes to a 7′ by 6′ regrind mill for reduction to 98% minus 325 mesh. Regrinding is done with high lime addition to neutralize the acid products formed by pyrrhotite which would otherwise consume excessive cyanide. Fuel oil is also added in the regrind mill to deactivate any carboniferous material in the concentrate. The ore body does contain minor amounts of this material.

The reground and thickened pulp (63% solids) in pumped to a 20 ft. dia. x 20 ft. preaeration tank. Aeration for an initial 24 hour period oxidizes the pyrrhotite which reduces cyanide consumption in the leach tanks.

Retention time for the pulp in the agitated leach tanks with cyanide reagent and dispersed air is 72 hours. Anticipated reagent consumptions are 1.28 lb. per ton sodium cyanide and 2.6 lb. per ton lime. Gold dissolution in the testwork was 91 % for an overall recovery of 83%. Experience has shown that better recoveries can be achieved in a plant environment vs. bench testing and we believe 83% overall recovery is a conservative figure for the feasibility study.

The study evaluated other alternatives, such as direct cyanide leaching without suphide floatation and heap leaching. Recoveries by heap leaching were very low and not economic. Recoveries by direct cyanidation were higher than the sulphide floatation procedure but not significant enough to offset the large capital costs for additional agitator and drum filter capacity.

The carbon in pulp process was not seriously considered. The technology at the time of Carolin's study in 1979 was not as advanced as it is today, and at that time it was primarily used where slimy, direct cyandidation ores were difficult to filter. To date, the author is not aware of carbon in pulp technology being applied to high grade pregnant solutions produced by sulphide concentrates.

The pregnant solution is recovered from a 40 foot thickner as overflow after it decants from the last leach tank. The thickner underflow is pumped to a primary 10 ft. by 10 ft. drum filter where additional pregnant solution is recovered. The filter cake is repulped in barren solution and subsequently filtered in a secondary 10 x 10 drum filter. The filter cake is repulped and charged to tailings. All filt rates return to the pregnant solution thickner where they are recovered as pregnant solution overflow. The overflow is pumped to a leaf clarifer to ensure the removal of finely divided, suspended solids and passed through a deaeration tower. Zinc dust and lead nitrate are added to the deaerated solution and the gold/zinc precipitate is removed from the now barren solution by one of two 36 inch Perrin filter presses. The dried gold/zinc precipitate is then smelted to bullion in a single chamber oil fire smelting furnace. At design throughput the plant will be recovering 150-175 ounces of gold and 30-40 ounces of silver per day.

To avoid leach circuit contamination a bleed from the barren stream must be continually removed. Cyanide must be destroyed prior to disposal of the solution to tailings. Carolin has elected to install an alkali chlorination system to oxidize and destroy cyanide. Chlorine gas is bubbled through the solution at a maintained alkaline pH. After retention in two 6 x 6 fiberglass tanks the treated barren is passed through a carbon column for extraction of residual chlorine and other undesirable ions. It is felt that the carbon system will prove to be a profitable back-up for collecting gold not precipitated by zinc, in addition to its environmental advantage.

Solid tailing disposal will be a significant operating cost. There are no downstream impoundment sites where effluent can be discharged by gravity from the mill.

Flotation and leach tailings are collected and then pumped through a series of 6 x 4 Warman pumps. A final discharge pressure of approximately 350 psi. boosts the tailings up a 6 inch pipeline through approximately 600 feet of head to the tailings impoundment area.

The tailings dam is an earthfill structure, constructed mainly of compacted boulder clay from a borrow pit local to the site. First stage construction includes a downstream zone of free draining shot rock, which will become a central chimney drain in the final design. The two zones are separated by filter cloth. In the summer and fall of 1981, approximately 350,000 cu.yards of earthfill were placed on the dam to net 1 year of storage capacity. In future the dam will be raised annually to meet the production requirements. Final dam crest will be 148 ft. above the valley floor. It requires 1,140,000 cu. yds. of earthworks would provide 6.67 years of tailings storage. Seepage is controlled by a small downstream sump and pumping system. Pumps on a floating barge reclaim clarified water and pump it up to a head tank where it flows by gravity back to the mill to meet much of the process water demand.

Make up water for the process, and fire storage are pumped from Ladner Creek. The potable water system is charged by a seperate upstream infiltration gallary.

COSTS:

The feasibility study estimated the Idaho zone could be brought into production at an estimated cost of $19.3 million. It can be appreciated that there is a significant difference between those 2nd quarter 1972 dollars and their current purchasing power.

Although a good portion of the overrun can be attributed to inflation, there have been other contributing factors. Most notable was a $1.242 million extra cost to place a 10 foot diameter precast cement culvert and divert Ladner Creek past the site. After back filling with ore, it provided much needed living space as well as considerably reduced the environmental impact of silting Ladner Creek whenever construction disturbed the hillside.

A directive from the Department of Mines to shotcrete both crusher rooms was one of the other major expenses that was not covered by the study.

The current projected capital cost exceeds $30 million.

The study indicated a payback of 2.93 years at a gold price of $250.00 Canadian/troy ounce. Fortunately the present price of gold is double and relatively stable, which allows payback within the same time frame.