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CINOLA OPERATING COMPANY LTD.

#402-595 HOWE ST. VANCOUVER, B.C. V6C 2T5

(604) 669-1524

October 7th, 1982.

Mr. P.S. Cross
Executive Vice-President and
Chief Executive Officer
Kerr Addison Mines Limited
P.O. Box 91,
Commerce Court West
Toronto, Ontario
M5L 1C7

Dear Mr. Cross,

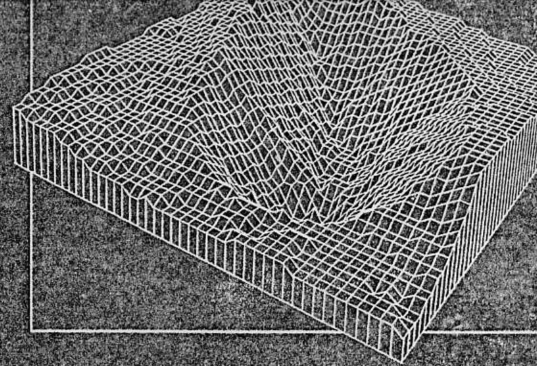
Enclosed please find copy #26 of our Queen Charlotte Gold Project Final Feasibility Study Summary Report as promised to you by Jerry Whiting. Also included is a Consolidated Cinola Mines Ltd. annual report.

Should you require clarification on any part of the report please do not hesitate to contact either myself or Jerry Whiting.

Yours truly,

Peter Kresin

Peter Kresin, P. Eng.
Manager of Metallurgy



Consolidated Cinola Mines Ltd.

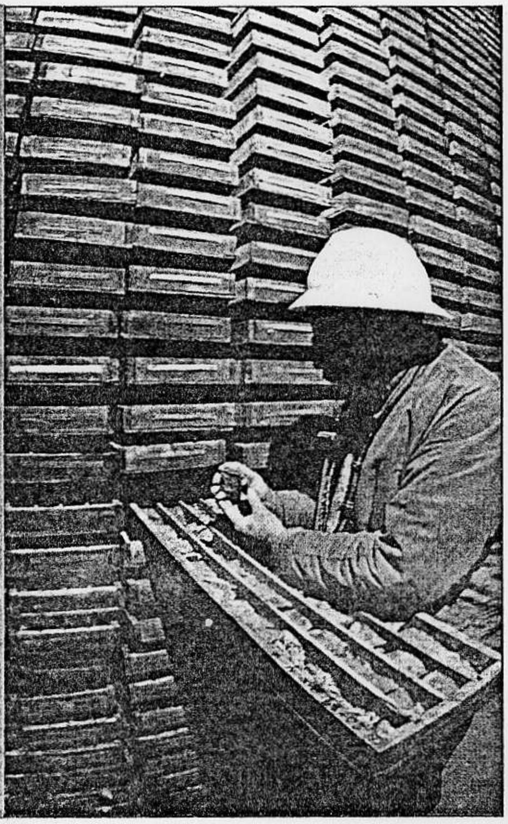
Annual Report 1981

To the Shareholders

With exploration and development of the Queen Charlotte gold deposit complete, we are now convinced that we can initiate the engineering and construction of a major open pit gold mine. The last four years of gathering and analysing data have produced results which we are now costing and optimizing in order to prove the economic feasibility of the project. There are only a few points which require clarification before we have a "bankable" project and can embark on the construction stage, and so the year 1982 will definitely be one of transition.

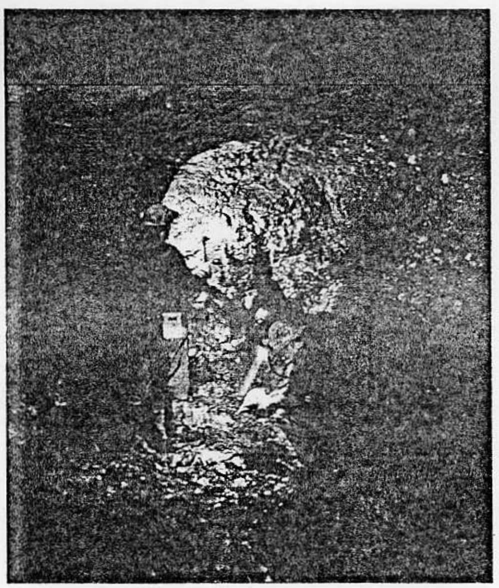
The goals of the Company for 1981 were to complete the data base on the Queen Charlotte gold project and finish the feasibility study. While we were able to accomplish the first objective, some pleasant surprises meant that we had to take a longer and harder look at the project, and so the final feasibility study will not be completed until mid 1982. As the data on the desposit emerged through the year it became very clear that the deposit differed from the original estimation, containing significantly more gold and in more varied concentrations than originally thought. This fact showed the need for flexibility in the 1981 program of bulk sampling and pilot milling.

The underground bulk sample program was designed to confirm surface vertical drill results and supply feed for the 50 ton per day pilot mill. Early in the program, significantly higher grade material was encountered. An underground drill program late in 1981 confirmed the extent and nature of this higher grade material, which will substantially alter the economics of the project. It is



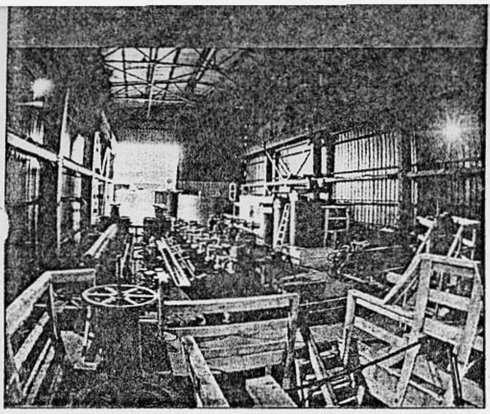
now possible to consider a "high grading" option in the financial analysis, enabling us to mine higher than anticipated grades during the payback period.

Considerable effort and expense went toward testing and optimizing gold recovery. The pilot mill, which operated from May to December was based on preliminary metallurgical results obtained in 1980 from diamond drill core samples. Initially, results were only partially successful, mainly because the mill feed differed significantly from the sample used for circuit design. Ultimately, pilot mill and bench scale testing produced eight flowsheet options for the large processing plant.



Work to date on the feasibility study has eliminated all but three of those options. The optimizing process of the feasibility study continues, and the final flowsheet should be developed in the second quarter of 1982. Our metallurgical engineers are determining the optimum gold recovery, which simply means the highest gold recovery for the lowest possible cost. While work to date has indicated a lower than usual percentage gold recovery, total cost per ounce will be very competitive with other Canadian gold producers, with preliminary data indicating a cost per ounce in the range of $200 (U.S.).

The most important part of the optimizing process is the integration of the supply of consistent mill feed (mining) with the most efficient method of extraction (milling). After studying several different mining and milling options we found that adequate profitability can be obtained only with an operation of 9,000 metric tonnes per day (10,000 short tons

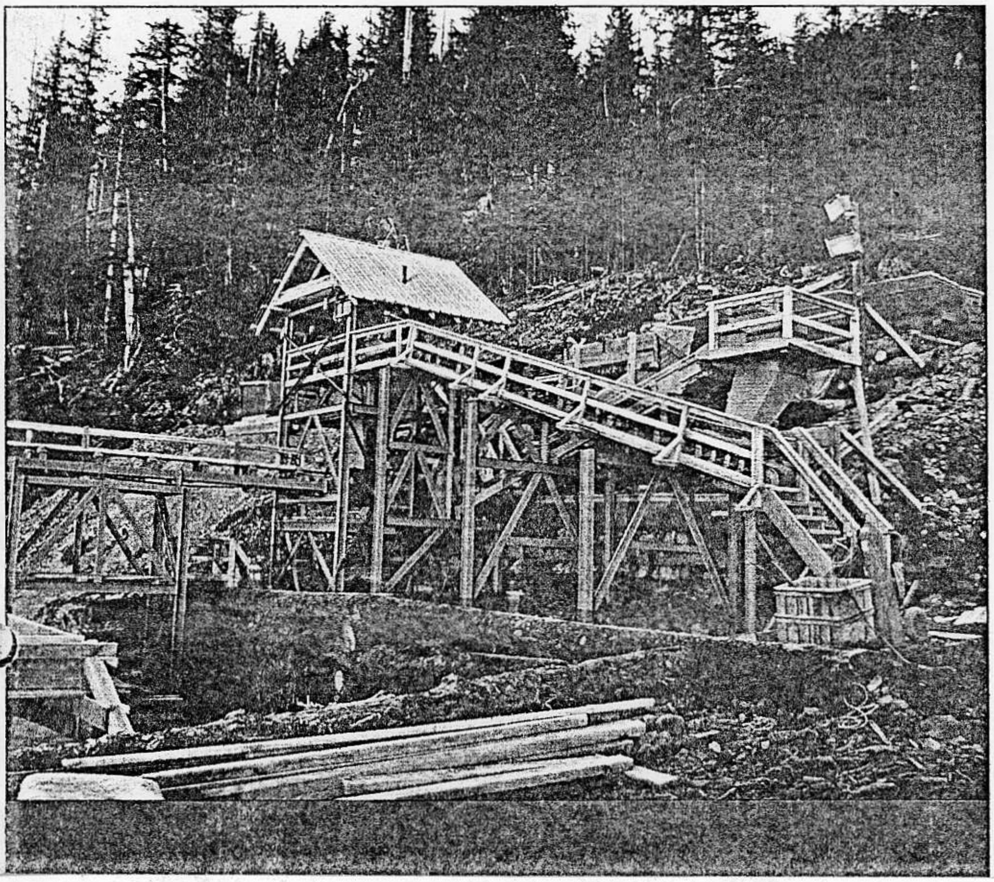


per day) or larger. It now appears very likely that the economic parameters of the final feasibility study will be based on a mill capacity of 13,500 metric tonnes per day (15,000 short tons per day). The other parameters developed to date include a gold recovery of 75 percent, an average grade of mill feed of 0.067 ounces gold per tonne and a gold price of $400 U.S. per ounce.

With respect to the current condition of depressed gold prices, we do not believe that the present price affects the profitability of the project simply because we are not yet producing. We believe that the timing of the project in relation to gold prices could not be better, as it is our opinion that we are nearing the end of the downward trend in price and expect gold at least to keep pace with inflation over the next decade. We believe that by the time the mining operation is on stream we will find ourselves in the uptrend part of the gold cycle, and so our base case gold price of $400 U.S. is considered to represent a conservative outlook.

Perhaps more important than the final feasibility study is the timely submission of the Stage II Environmental Study to the provincial authorities. We have worked diligently on this aspect of the project and are preparing a well researched, comprehensive presentation to the government. Once the Stage II Study is submitted this summer, public input will be invited. Given the benign nature of our flowsheet options and the vast economic benefit of the project to the regional economy of the Queen Charlottes we believe we will have the support of all parties concerned.

The final feasibility study and Stage II submission are scheduled for completion by mid 1982. Data to date indicate that we have a viable project and it is the opinion of the management that construction could commence this year. Our primary objective is to ensure continuity of the project. To this end, our efforts in the financing and permitting aspects of the project should allow for the engineering and construction of a major open pit gold mine.



On Behalf of
the Board of Directors

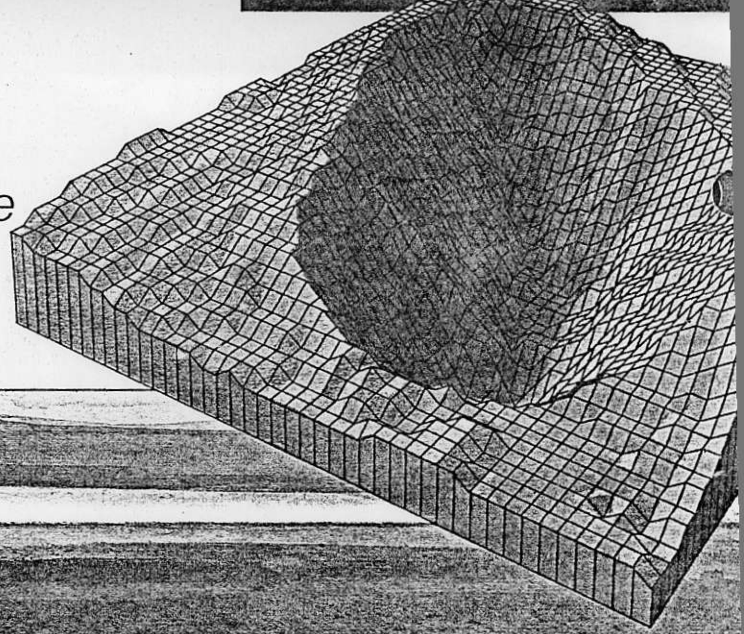
K. G. Sanders

K. G. Sanders, P.Eng.,
President and
Chief Executive Officer.

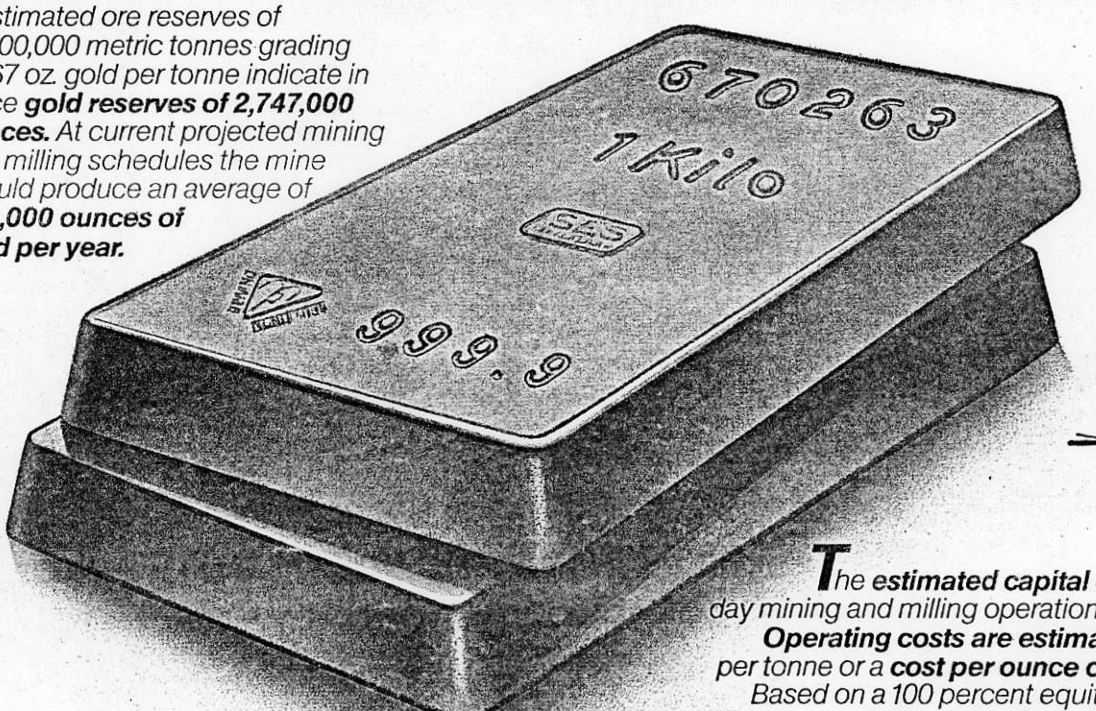
Angelo Tosi

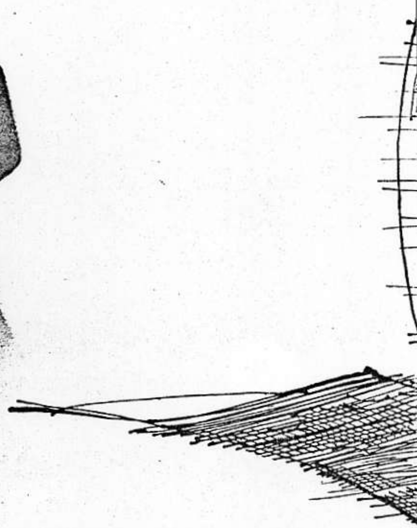
Angelo Tosi,
Chairman of the Board.

As the Final Feasibility Study is being compiled we have been able to develop and narrow the project details. The following is a brief presentation of the economic and operating statistics developed to date.



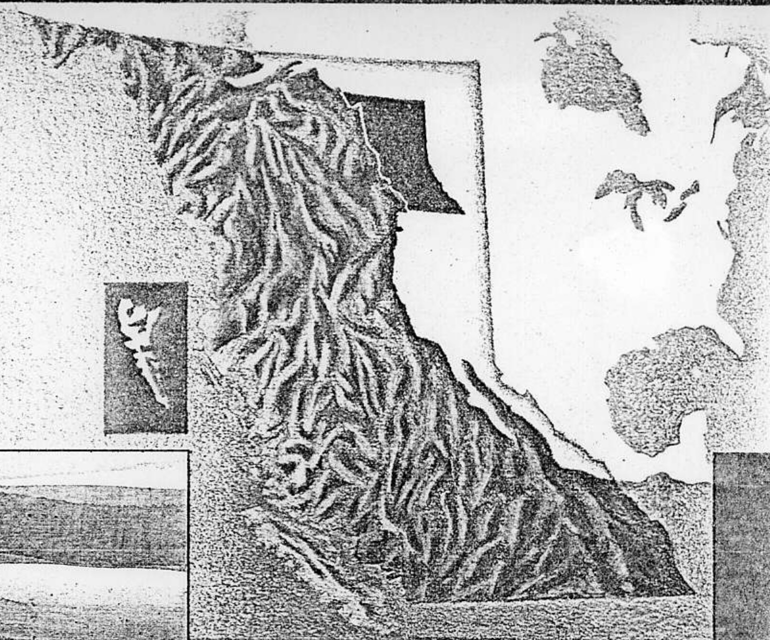
Estimated ore reserves of 41,000,000 metric tonnes grading 0.067 oz. gold per tonne indicate in place ***gold reserves of 2,747,000 ounces.*** *At current projected mining and milling schedules the mine should produce an average of* ***225,000 ounces of gold per year.***

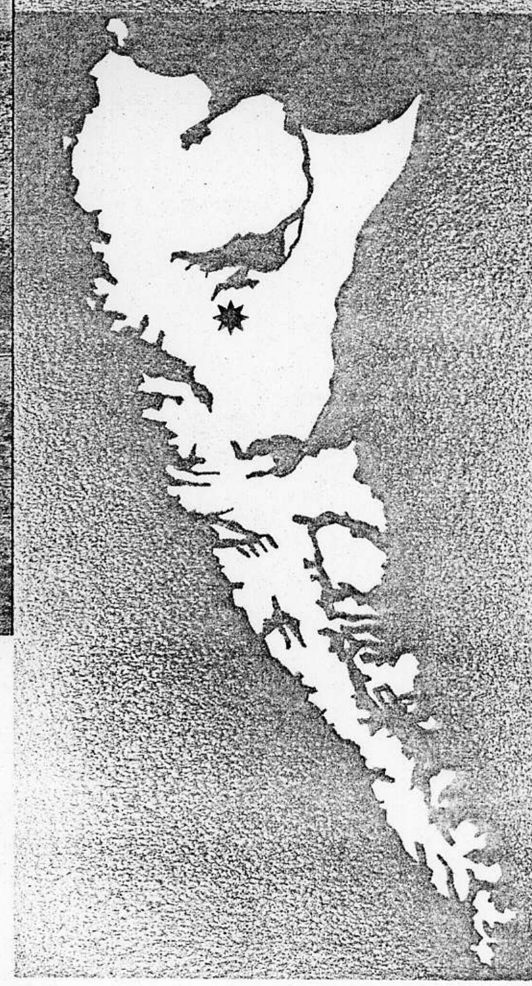




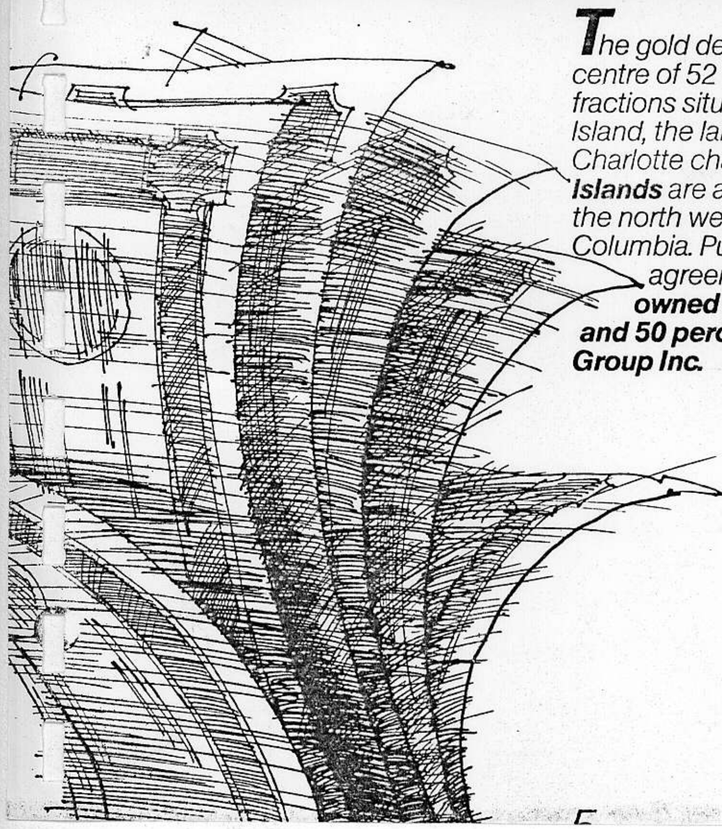
The ***estimated capital cost*** *of a 13,500 tonne per day mining and milling operation is* ***$183,000,000 Canadian. Operating costs are estimated at $9.58 Canadian*** *per tonne or a* ***cost per ounce of approximately $200 U.S.*** *Based on a 100 percent equity financing scenario, the* ***payback period is estimated at 2.7 years.***

The size and shape of the deposit make it very amenable to open pit mining methods. Open pit mining utilizes economies of scale and therefore is relatively inexpensive. It is currently estimated that enough ore will be mined to process **13,500 metric tonnes per day.** A conventional gold recovery circuit will be in place in the mill. Current reserves and mill capacity estimates indicated a **mine life of 8.9 years.**





The gold deposit is located in the centre of 52 mineral claims and fractions situated on central Graham Island, the largest of the Queen Charlotte chain. The **Queen Charlotte Islands** are approximately 80 miles off the north west coast of British Columbia. Pursuant to a Joint Venture agreement, the claims are ***owned 50 percent by Cinola and 50 percent by Energy Reserves Group Inc.***



In the past we have made a policy of not diverting our time, energy and money away from the Queen Charlotte Project. The success of this Company depends completely on our ability to see the Queen Charlotte Project through to commercial production. As production is now in the foreseeable future, we have started to think about the post production development stage of this Company. To this end some low cost highly leveraged positions have been taken in exploration possibilities.

Kelly Gold Mines Inc.

Cinola holds 36 percent of the outstanding stock in Kelly Gold Mines Inc., a Delaware exploration company. Kelly is managed by Cinola personnel.

The Company is engaged in a joint venture with Suneva Resources Ltd. whereby Kelly can earn 50 percent interest in Suneva's Alligator Ridge, Nevada claims by spending three million dollars ($3,000,000) U.S. over three years. The Company is committed to an initial expenditure of three hundred thousand dollars ($300,000).

The key claim group is located on the eastern slope of Alligator Ridge in White Pine County. On the western slope of the ridge, Amselco and Occidental Petroleum are operating an open pit, heap leach gold mine.

Geological mapping and soil sampling by Kelly have located drill targets in addition to those tested by Suneva in 1980. Close proximity to a producing mine and a similar geological environment make this project an excellent exploration bet.

Kelly also holds 62 units on the Queen Charlotte Islands not far from the Cinola deposit. An extensive geochemical program was undertaken in 1981 and has located several good drill targets.

The best target exists approximately ten kilometres southeast of the Cinola deposit. This area is cut by the same fault system which is the structural control of gold mineralization at Cinola. The 1981 program showed a large coincident mercury and arsenic anomaly in this area. Ore grade float has been located in this area.

Westland Syndicate

Cinola holds a 20 percent interest in an exploration syndicate along with four other participants. The purpose of the syndicate is to acquire resource projects, undertake initial development, and spin the projects off for a leveraged type of consideration.

The syndicate has submitted to the Australian Government three exploration permit applications to explore for oil and gas. The areas under application are in the Canning Basin in Western Australia and are believed to be on the same structure as the discovery wells in that area. Each permit application consists of over 40 blocks. A block is 25 square miles. At the time of this report, the applications are still pending.

The syndicate has staked 63 mineral claims in the Thunder Bay, Ontario mining district adjoining the claims owned by the Consolidated Louanna–Cumo joint venture. Consolidated Louanna recently commenced production at close to 200 tons per day. The grade of ore is reported at approximately 0.4 oz. gold per ton.

Ark La Tex Petroleum Corporation

Cinola holds a minor pre-prospectus share position in Ark La Tex Petroleum Corporation. Ark La Tex holdings include a 65 percent net revenue interest in approximately 320 acres of oil and gas leases in the Yates field, Pecos County, Texas. The Yates oil field is one of the most famous large Texas oil fields having produced, since discovery in 1926, over 900,000,000 barrels of oil from its more than 800 wells.

Consolidated Cinola Mines Ltd.

Balance Sheet
As at December 31, 1981

Assets

	1981	1980
Current Assets		
Cash and short-term deposits	$ 952,099	$ 730,363
Accounts receivable (note 4)	15,381	79,623
	967,480	809,986
Investments (note 3)	121,016	—
Mineral Properties and Deferred Costs (note 4)	7,321,118	2,178,246
Fixed Assets (note 5)	50,614	97,854
	$8,460,228	$3,086,086

Liabilities

	1981	1980
Current Liabilities		
Accounts payable and accrued liabilities	$ 71,463	$ 100,888
Long-Term Debt (note 6)	5,357,105	430,638
	5,428,568	531,526

Shareholders' Equity

	1981	1980
Capital Stock (note 7)	3,843,125	3,268,645
Deficit	811,465	714,085
	3,031,660	2,554,560
	$8,460,228	$3,086,086

Approved by the Directors

K. G. Sanders

Director

G. Sanders

Director

Financial Statements

Consolidated Cinola Mines Ltd.

Statement of Deficit
for the year ended December 31, 1981

	1981	1980
Balance—Beginning of Year	$ 714,085	$ 714,085
Write-off of interest in mineral properties	40,000	—
Write-off of mining equipment (note 5)	57,380	—
Balance—End of Year	$ 811,465	$ 714,085

Statement of Deferred Costs
for the year ended December 31, 1981

	1981	1980
Administrative—per Schedule	$ 778,222	$ 312,891
Exploration and Development		
Construction consulting	19,985	—
Westland Syndicate exploration	7,200	—
	27,185	—
	805,407	312,891
Deferred Costs—Beginning of Year	1,133,858	820,967
Deferred Costs—End of Year	$1,939,265	$1,133,858

Schedule of Deferred Administrative Costs
for the year ended December 31, 1981

	1981	1980
Advertising	$ 16,119	$ 7,108
Automobile	15,832	8,522
Corporation capital tax	15,829	—
Depreciation	10,708	19,504
Donations, dues and subscriptions	8,420	7,739
Equipment rental	8,622	4,539
Interest on long-term debt	549,002	—
Legal, accounting and audit	34,803	29,714
Office supplies, postage and delivery	14,811	10,778
Printing and stationery	31,397	27,232
Public relations, travel and promotions	50,881	85,942
Rent	15,429	11,217
Salaries and benefits	188,645	130,619
Sundry	10,414	1,559
Telephone and telegraph	12,433	18,672
Transfer agent fees	17,045	19,116
	1,000,390	382,261
Less: Overhead charges recovered from joint venture	30,000	29,905
Interest income	192,168	39,465
	222,168	69,370
	$ 778,222	$ 312,891

Consolidated
Cinola Mines Ltd.

Statement of Changes in Financial Position

for the year ended December 31, 1981

	1981	1980
Source of Working Capital		
Proceeds from issue of capital stock	$ 574,480	$ 873,030
Long-term debt	4,926,467	430,638
	5,500,947	1,303,668
Use of Working Capital		
Exploration, development and administrative costs	805,407	312,891
Deduct: Depreciation, which does not require use of working capital	10,708	19,504
	794,699	293,387
Fixed asset additions	20,848	45,631
Purchase of investments	121,016	—
Joint venture	4,377,465	393,492
	5,314,028	732,510
Increase in Working Capital	186,919	571,158
Working Capital—Beginning of Year	709,098	137,940
Working Capital—End of Year	$ 896,017	$ 709,098
Represented by:		
Current assets	967,480	809,986
Current liabilities	71,463	100,888
	$ 896,017	$ 709,098

Notes to Financial Statements

for the year ended December 31, 1981

1. Nature of Operations

The company is in the process of developing its Queen Charlotte mineral properties and is of the opinion that these properties contain economically recoverable ore reserves. The recoverability of the amounts shown for mineral properties and related deferred costs is dependent upon the ability of the company to obtain necessary financing to complete the development, and upon future profitable production.

2. Significant Accounting Policies

Deferred Costs

Exploration, development and administrative costs relating to mineral properties are deferred until the properties are brought into production, at which time they are amortized on a unit of production basis. If properties are abandoned or sold, the deferred costs are written off at that time.

Depreciation

Depreciation of office furniture is calculated using the declining-balance method at the annual rate of 20%. Leasehold improvements are amortized at 20% per annum on a straight-line basis. The automobile is depreciated at 25% per annum on a declining-balance method.

Joint Venture

The company proportionately consolidates its share of the assets and liabilities, net earnings or losses of the joint venture.

Loss per Share

Basic loss per share has not been calculated as it is not considered meaningful at this stage in the company's operations.

3. Investments

Kelly Gold Mines Inc.—at equity 1,000,000 shares	$ 72,216
Ark La Tex Petroleum Corp.—at cost 20,000 shares	26,000
Westland Syndicate—at equity 30 units	22,800
	$ 121,016

All the company's investments are involved in exploration and development of natural resources. At December 31, 1981 there were no quoted values for these investments.

4. Mineral Properties and Deferred Costs

	1981	1980
Queen Charlotte Joint Venture	$4,808,103	$ 430,638
Mineral Properties	573,750	613,750
Deferred Costs—per statement	1,939,265	1,133,858
	$7,321,118	$2,178,246

Substantially all of the company's activities are directed to the development of the company's Queen Charlotte mineral properties.

(a) Mineral Properties

The company's mineral properties known as the Queen Charlotte Gold Prospect, are situated in the Queen Charlotte Islands, in the Skeena Mining Division, British Columbia. These properties were acquired for $450,000 cash and 300,000 shares of the company. Energy Reserves Canada, Ltd. ("ERC") has earned a 50% interest in these properties, having contributed $5,000,000 to their development.

(b) Queen Charlotte Joint Venture

Under an agreement dated November 21, 1979 the company and ERC entered into an unincorporated joint venture to explore, develop, mine, process and sell gold and other minerals from the Queen Charlotte Gold Prospect.

Initial contribution by ERC to earn 50% interest		$ 5,000,000
Subsequent contributions to joint venture		
By the company (note 6)	$4,808,103	
By ERC	4,808,103	9,616,206
Total contributions from inception to December 31, 1981		$14,616,206

Contributions over the first $5,000,000 provided by ERC to the joint venture are shared equally between ERC and the company. During the year the company charged $77,878 to the joint venture for costs incurred on the venture's behalf. Included in accounts receivable at December 31, 1981 was $6,048 due from the joint venture.

5. Fixed Assets

	1981			1980
	Cost	Accumulated depreciation	Net	Net
Mining equipment	$ —	$ —	$ —	$57,380
Office furniture	46,402	12,614	33,788	37,568
Leasehold improvements	3,229	969	2,260	2,906
Automobile	16,848	2,282	14,566	—
	$66,479	$15,865	$50,614	$97,854

During the year, mining equipment with a net book value of $57,380 was abandoned.

6. Long-Term Debt

	1981	1980
Loan payable	$4,808,103	$ 430,638
Accrued interest	549,002	—
	$5,357,105	$ 430,638

The long-term debt arises as a result of financing of the company's portion of joint venture costs by Energy Reserves Canada, Ltd. The loan is subject to the Bank of Nova Scotia prime rate of interest plus one percent. The loan and interest are repayable from 50% of the share of operating profits of the joint venture distributed to the company.

The company's share of all assets of the Queen Charlotte Joint Venture has been pledged as security for the repayment of such loan.

7. Capital Stock

	1981	1980
Authorized—		
10,000,000 shares without par value		
Issued and fully paid—		
3,682,688 shares for cash	$3,524,200	$2,949,720
690,438 shares for mineral properties	318,925	318,925
4,373,126	$3,843,125	$3,268,645

During the year ended December 31, 1981, the company issued 500,000 shares for a cash consideration of $550,000 relating to warrants exercised and 1,700 shares for a cash consideration of $24,480 relating to options exercised.

Stock Options

Employees of the company have the following stock options outstanding:

(i) An option to purchase 17,100 shares of the company at $14.40 per share by March 28, 1985.

(ii) One-year option commencing December 11, 1981 to purchase 15,000 shares at U.S. $14.75 per share. One-year option commencing December 11, 1982 to purchase 20,000 shares at U.S. $15.00 per share.

During the year the authorized capital of the company was increased from 5,000,000 to 10,000,000 shares without par value.

8. Commitments

The company is committed to make minimum payments under the terms of lease agreements as follows:

Year ending December 31, 1982	$48,357
1983	48,357
1984	40,273
1985	24,147

The company has entered into five-year employment contracts with four directors, one officer and an employee of the company commencing January 1981 with options to renew them for an additional five years at the discretion of the individuals. The contracts provide for minimum yearly increments in remuneration at a rate which is 5% higher than the cost of living index. At December 31, 1981 the annual salaries under these contracts totalled $163,300.

9. Related Party Transactions

The company has undertaken a number of related party transactions during the year, all of which are disclosed in the notes to these financial statements.

10. Comparative Figures

Comparative figures are based upon financial statements which were reported on by other auditors.

Auditors' Report to the Shareholders

We have examined the balance sheet of Consolidated Cinola Mines Ltd. as at December 31, 1981 and the statements of deficit, deferred costs and changes in financial position for the year then ended. Our examination was made in accordance with generally accepted auditing standards, and accordingly included such tests and other procedures as we considered necessary in the circumstances.

In our opinion, these financial statements present fairly the financial position of the company as at December 31, 1981 and the results of its operations and the changes in its financial position for the year then ended in accordance with generally accepted accounting principles applied on a basis consistent with that of the preceding year.

Coopers & Lybrand

Coopers & Lybrand
Chartered Accountants

Vancouver, B.C.
March 3, 1982

Consolidated Cinola Mines Ltd.

Officers and Directors

Angelo Tosi
Chairman and Director

Kenneth G. Sanders, P.Eng.
President and
Chief Executive Officer

Nola Peterson
Secretary and Director

George Sanders
Vice President and Director

William R. Green, Ph.D.
Director

Reno Calabrigo
Vice President

Cinola Operating Company Ltd.

Kenneth G. Sanders, P.Eng.
Chairman

David W. McSkimmings, P.Eng.
President

Solicitors

Swinton & Company
1300–1090 W. Georgia Street
Vancouver, B.C.

Milgrim, Thomajan, Jacobs & Lee
405 Lexington Avenue
New York, N.Y. 10017

Bank

Toronto Dominion Bank
Hastings and Hornby
839 W. Hastings St.
Vancouver, B.C.

Registrar and Transfer Agent

Crown Trust Company
700–750 W. Pender Street
Vancouver, B.C.

320 Bay Street
Toronto, Ontario

Investment Bankers

Drexel Burnham Lambert
60 Broad Street
New York, N.Y.

Exchange Listing

Vancouver Stock Exchange
Symbol CSZ

Over-the-counter
NASDAQ symbol CCIMF

Head Office:
402–595 Howe Street,
Vancouver, B.C.

Registered Office:
1300–1090 W. Georgia Street,
Vancouver, B.C.

Authorized Capitalization:
10,000,000 common shares

Issued:
4,373,126

Direct Inquiries to:

George Sanders, Vice President
402–595 Howe Street,
Vancouver, B.C. V6C 2T5

CONSOLIDATED CINOLA MINES LTD.
402 - 595 Howe Street
Vancouver, B.C. V6C 2T5

INTERIM REPORT
For the Period Ended
June 30, 1982

August 26, 1982

TO THE SHAREHOLDERS

We are pleased to announce that results of the final feasibility study of the Queen Charlotte Gold Project will be available in September 1982. The exhaustive study has been underway since December of last year when the data base was completed on the property. The data, accumulated over a four year period, includes results from 93,750 ft. of diamond drilling (88,650 from surface and 5100 underground) and 1108 ft. of underground drifting and crosscutting. Some 5,525 tonnes of bulk sample from the adit was treated in the pilot mill.

Although the final numbers of project economics are not ready for release, we have every reason to believe that they will be in the range of the June, 1980 pre-feasibility study by Wright Engineers Ltd. The project, in our opinion, remains a viable and bankable one. We expect the final feasibility will confirm this position.

The capital costs of the anticipated 13,500 tonne per day operation are expected to be in the previously stated range of $180 million Canadian. Once we are in receipt of the feasibility study documents the joint venture partners can proceed to finalize project financing arrangements. To this end both partners, jointly and separately, have been pursuing several financing possibilities. It is the opinion of the Board Of Directors that with recent declining interest rates and buoyant precious metal prices this project will be more easily financed than was anticipated several months ago.

We are pleased to announce the addition to management of Mr. John J. Crowhurst who has recently accepted an appointment to the Board Of Directors. Mr. Crowhurst is very well known is the Canadian mining industry and his expertise in financing and operations is a welcome addition at this stage of the Company's development. Formerly an Executive Vice President with McIntyre Mines, Mr. Crowhurst is currently Chief Operating Officer of Pegasus Gold Ltd.

Looking to the second half of 1982 and on to next year, we are confident that financing will be arranged to take this project to mine development stage. All our data to date indicates that the Queen Charlotte Project will be among the largest gold mines in North America.

On Behalf Of The Board
Of Directors

G. Sanders

George Sanders,
Vice President And Director

CONSOLIDATED CINOLA MINES LTD.

BALANCE SHEET

JUNE 30, 1982

(UNAUDITED)

	June 30, 1982	June 30, 1981
ASSETS		
CURRENT		
Cash	$ 729,210	$1,086,011
Accounts receivable	21,073	24,336
Loans to directors	73,000	-
	823,283	1,110,347
FIXED (Net of accumulated depreciation of $21,765)	44,713	104,730
MINERAL CLAIMS	573,750	613,750
INVESTMENTS	126,358	72,217
JOINT VENTURE	5,782,296	2,850,260
DEFERRED EXPLORATION, DEVELOPMENT AND ADMINISTRATIVE EXPENSES	2,533,361	1,432,829
	$9,883,761	$6,184,133
LIABILITIES		
CURRENT		
Accounts payable	$ 87,458	$ 56,236
LONG TERM DEBT	6,764,643	2,998,857
	6,852,101	3,055,093
SHAREHOLDERS' EQUITY		
SHARE CAPITAL		
Authorized: 10,000,000 shares, no par value		
Issued: 4,373,126	3,843,125	3,843,125
DEFICIT	(811,465)	(714,085)
	3,031,660	3,129,040
	$9,883,761	$6,184,133

CONSOLIDATED CINOLA MINES LTD.

STATEMENT OF CHANGES IN FINANCIAL POSITION

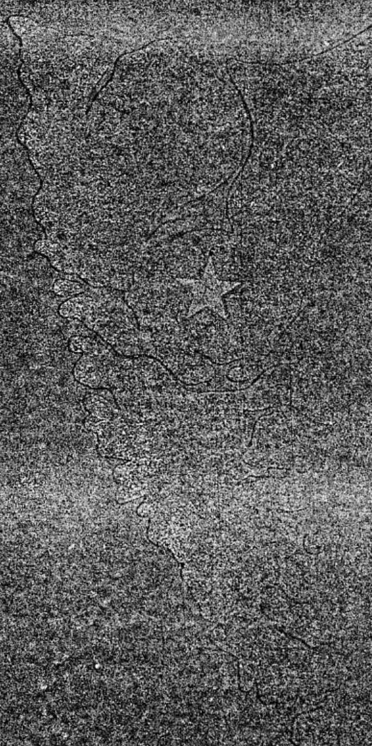
FOR THE SIX MONTHS ENDED JUNE 30, 1982

(WITH COMPARATIVES FOR THE SIX MONTHS ENDED JUNE 30, 1981)

(UNAUDITED)

	1982	1981
SOURCE OF FUNDS:		
Issue of share capital	$ -	$ 574,480
Long-term debt	1,407,538	2,568,218
Interest income	65,337	99,382
	1,472,875	3,242,080
APPLICATION OF FUNDS:		
Administration expense	219,365	219,800
Purchase of automobile	-	16,848
Exploration and development costs	975,016	2,439,606
Interest on long-term debt	433,345	148,597
Investments	5,341	72,217
	1,633,067	2,897,068
INCREASE (DECREASE) IN WORKING CAPITAL	(160,192)	345,012
WORKING CAPITAL - BEGINNING OF PERIOD	896,017	709,099
WORKING CAPITAL - END OF PERIOD	$ 735,825	$1,054,111
REPRESENTED BY:		
Current Assets	$ 823,283	$1,110,347
Current Liabilities	87,458	56,236
	$ 735,825	$1,054,111

QUEEN CHARLOTTE GOLD PROJECT



CINOLA OPERATING
COMPANY LTD.

COPY 26 of 30

QUEEN CHARLOTTE
GOLD PROJECT

FINAL FEASIBILITY STUDY

Summary

SEPTEMBER 1982

BY

CINOLA OPERATING COMPANY LTD.
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Vancouver, B.C.

(604) 669-1524

VOLUME II - TECHNICAL DESCRIPTION (SUMMARY)

TABLE OF CONTENTS

0.0	SUMMARY	0-1
0.1	Introduction	0-1
0.2	Geology	0-2
0.2.1	Regional Geologic Setting	0-2
0.2.1.1	Rock Types	0-2
0.2.1.2	Structure	0-2
0.2.2	Deposit Geology	0-2
0.2.2.1	Exploration History	0-3
0.2.2.2	Host Rocks	0-4
0.2.2.3	Structure	0-4
0.2.2.4	Mineralogy	0-5
0.2.2.5	Alterations	0-6
0.2.3	Correlation of Geology and Geologic Reserve Estimates	0-7
0.3	Reserve Estimations	0-10
0.3.1	Evaluation of Data Base	0-10
0.3.1.1	Types of Data	0-10
0.3.1.2	Comparison Between Vertical and Inclined Drill Hole Samples and Adit Muck Samples	0-10
0.3.2	Interpretation of Gold Distribution	0-12
0.3.3	Geologic Reserve Estimation	0-13
0.3.3.1	Inverse Distance Squared Method	0-14
0.3.3.2	Grade-Thickness Contour Method	0-15
0.3.3.3	Kriging Method	0-15
0.3.3.4	Comparison of Kriged and Inverse Distance Squared Estimates	0-18
0.3.3.5	Confidence of Kriged Reserve Estimate	0-18
0.3.3.6	Accuracy of Reserve Estimates	0-19
0.4	Mine Design	0-21
0.4.1	Geotechnical Analysis	0-21
0.4.1.1	Site Investigation	0-22
0.4.1.2	Rock Fabric	0-23
0.4.1.3	Rock Strength	0-24
0.4.1.4	Seismicity	0-25
0.4.1.5	Surface Water	0-26
0.4.1.6	Groundwater	0-28

0.4.2 Mine Design and Equipment Parameters 0-29

0.4.2.1 Design Criteria 0-29

0.4.2.2 Equipment Parameters 0-31

0.4.3 Organization and Manning 0-33

0.4.4 Mineable Ore Reserves for Ultimate Pit Design 0-35

0.4.4.1 Ultimate Pit Generation 0-35

0.4.4.2 Grade Adjustment for Selective Mining Unit 0-37

0.4.5 Mine Operations 0-39

0.4.5.1 Preproduction Pit 0-40

0.4.5.2 Production Scheduling 0-40

0.4.5.3 Tailings Dam Material 0-41

0.4.5.4 Waste Dumps and Ore Stockpiles 0-43

0.4.5.5 Mine Drainage 0-44

0.5 Mill Design 0-46

0.5.1 Facilities Description 0-48

0.5.1.1 Crushing and Storage 0-49

0.5.1.2 Grinding 0-50

0.5.1.3 Flotation, Dewatering and Tailings Disposal 0-51

0.5.1.4 Roasting (Single-Stage) 0-53

0.5.1.5 Acid Plant (Gas Treatment) 0-55

0.5.1.6 Leaching and Decantation 0-58

0.5.1.7 Precipitation and Smelting 0-59

0.5.1.8 Cyanide Removal 0-61

0.5.1.9 Electrical Distribution 0-63

0.5.1.10 Instrumentation and Control 0-63

0.5.1.11 Auxiliary Facilities 0-63

0.5.2 Process Alternatives 0-64

0.5.3 Environmental Considerations 0-65

0.5.3.1 Crushing and Storage (Area 1) 0-65

0.5.3.2 Grinding and Flotation (Areas 2 and 3) 0-66

0.5.3.3 Roasting and Acid Plant (Area 4) 0-73

0.5.3.4 Leaching, Precipitation and Smelting (Areas 6 and 7) 0-75

0.5.3.5 Cyanide Removal 0-76

0.5.3.6 Summary of Process Emissions 0-77

	0.5.4	Metallurgical Processing, Operational Requirements	0-79
		0.5.4.1 Personnel	0-79
		0.5.4.2 Operating Supplies	0-81
0.6	**Power Plant Design**		**0-88**
0.7	**Support Facilities**		**0-90**
	0.7.1	Access Road and Site Preparation	0-90
	0.7.2	Construction Camp and Housing	0-91
	0.7.3	Buildings and Facilities	0-92
0.8	**Project Costs**		**0-95**
	0.8.1	Capital Costs	0-95
	0.8.2	Operating Costs	0-97
0.9	**Economic Analysis**		**0-99**
	0.9.1	Comparison of Prefeasibility and Final Feasibility Values	0-99
	0.9.2	Discussion of the Base Case	0-103
	0.9.3	Sensitivity Analyses	0-111
		0.9.3.1 Revenue Variables	0-113
		0.9.3.2 Unit Preduction Cost Variables	0-115
		0.9.3.3 Capital Variables	0-116
		0.9.3.4 By-Product Prices	0-116
		0.9.3.5 Current-Dollar Scenarios	0-120

VOLUME II - TECHNICAL DESCRIPTION (SUMMARY)
LIST OF TABLES

No.	Title	
0-1	Average Grade of the Higher-Grade Lenses Computed from Two-Metre Assays in Each Lens	0-14
0-2	Comparison of Geologic Reserves for the Higher-Grade Lenses in the Cinola Deposit Determined by the Grade-Thickness Contour, Traverse Distance Squared and Kriging Methods	0-16
0-3	Bench-by-Bench Geologic Reserves for the Traverse Distance Squared and Kriged Block Models	0-17
0-4	Comparison of Kriged Versus Inverse Distance Squared Geologic Reserves for Various Cutoffs	0-20
0-5	Mine Equipment Units in use by Year of Operation	0-34
0-6	Hourly Paid Manpower Requirements - Year 1 Salaried Manpower Requirements - Year 1	0-36
0-7	Mineable Reserves-Refined Ultimate Pit - $1/d^2$ Model	0-38
0-8	Mineable Reserves Reported at Varying Cutoff Grades	0-38
0-9	Mineable Reserves - Refined Ultimate Pit - Geostatistical Model	0-38
0-10	Annual Mine Production Schedule	0-42
0-11	Mine Waste Rock hauled for Tailings Dam Construction	0-43
0-12	Pit Dewatering Statistics	0-45
0-13	Process Equipment Unit List	0-83
0-14	Acid Production Test on Pilot Mill Ore and Flotation Tailings	0-69

0-15	Iron, Zinc, Mercury and Arsenic Leaching Characteristics of Pilot Mill Tailings Samples	0-70
0-16	Pilot Mill Tailings Pond Water Analyses	0-71
0-17	LC50 Static Bioassays Pilot Mill Tailings Pond Water	0-72
0-18	Cinola Operating Company Ltd. Queen Charlotte Gold Project Capital Cost Expenditure Schedule	0-96
0-19	Cinola Operating Company Ltd. Queen Charlotte Gold Project Annual Operating Costs	0-98
0-20	Comparison of Base Case Studies	0-100
0-21	Production Statistics and Operating Profit (Base Case)	0-104/0-105
0-22	Cash Flow Summary (Base Case)	0-109/0-110
0-23	Production Statistics and Operating Profit (Case I, Current Dollar)	0-122/0-123
0-24	Cash Flow Summary (Case I, Current Dollar)	0-124/0-125
0-25	Production Statistics and Operating Profit (Case II, Current Dollar)	0-126/0-127
0-26	Cash Flow Summary (Case II, Current Dollar)	0-128/0-129

LIST OF FIGURES

No.	Title	After Page
0-1	Geographic Location Plan	0-1
0-2	Location Plan	0-1
0-3	Cinola Gold Deposit Local Geology	0-2
0-4	Geologic Plan 150 m Elevation	0-4
0-5A	Geologic Cross-Section B-B'	0-4
0-5B	Geologic Longitudinal Section A-A'	0-5
0-6	Geologic Cross-Section Showing Lenses	0-8
0-7	Recommended Slope Angles	0-21
0-8	Rock Mass Design Strengths	0-25
0-9	Mine Operations Organization Chart	0-34
0-10	Mesh Plot - End of Mining	0-38
0-11	Area 1 Crushing and Storage	0-49
0-12	Area 2 Grinding	0-50
0-13	Area 3 Flotation, Dewatering and Tailings Disposal	0-51
0-14	Area 4 Roasting	0-53
0-15	Area 5 Gas Treatment (Acid Plant)	0-55
0-16	Area 6 Leaching and Decantation/Area 8 Cyanide Removal	0-58
0-17	Area 7 Precipitation and Smelting	0-59
0-18	Metallurgical Process Facility Block Flowsheet	0-65
0-19	Constituent Material Balance Grinding and Flotation	0-66
0-20	Constituent Material Balance Roasting and Acid Plant	0-74
0-21	Constituent Material Balance Leaching, Decantation, Precipitation and Smelting	0-75
0-22	Constituent Material Balance Cyanide Removal	0-76
0-23	Summary Constituent Material Balance	0-77
0-24	Project IRR and NPV (10 PCT) Versus Percentage Changes in Revenue Variables	0-113
0-25	Project IRR and NPV (10 PCT) Versus Changes in Unit Production Cost Variables	0-116
0-26	Project IRR and NPV (10 PCT) Versus Changes in Mill, Power Plant and Tailings Pond Capital	0-117

0.1 INTRODUCTION

0.0 SUMMARY

0.1 Introduction

The Cinola gold project is located on Graham Island, the largest of the Queen Charlotte Islands, British Columbia. As shown on Figures 0-1 and 0-2, the ore deposit is approximately 140 km west of the mainland, 770 km northwest of Vancouver, and about 13 km inland from the east coast of Graham Island.

The islands are served by regularly-scheduled, jet-powered commercial flights into Sandspit, and ferry service from Prince Rupert. Several well-surfaced roads provide good access to the property from Sandspit, Port Clements and surrounding areas.

The deposit, discovered in 1970, was explored by Cominco, Kennco, Quintana, and Silver Standard between 1970 and 1977. Consolidated Cinola Mines, Ltd. (CCM) acquired the property in 1977. Energy Reserves Canada, Ltd. (ERC) acquired 50 percent interest in the property in late 1979. Development of the property, in terms of ore reserve definition, metallurgical research and flowsheet design, project feasibility analysis, and all aspects of environmental assessment and permitting, has been continuous since CCM's acquisition and is presently under the management of Cinola Operating Company Ltd. (COC), a joint-venture of ERC and CCM.

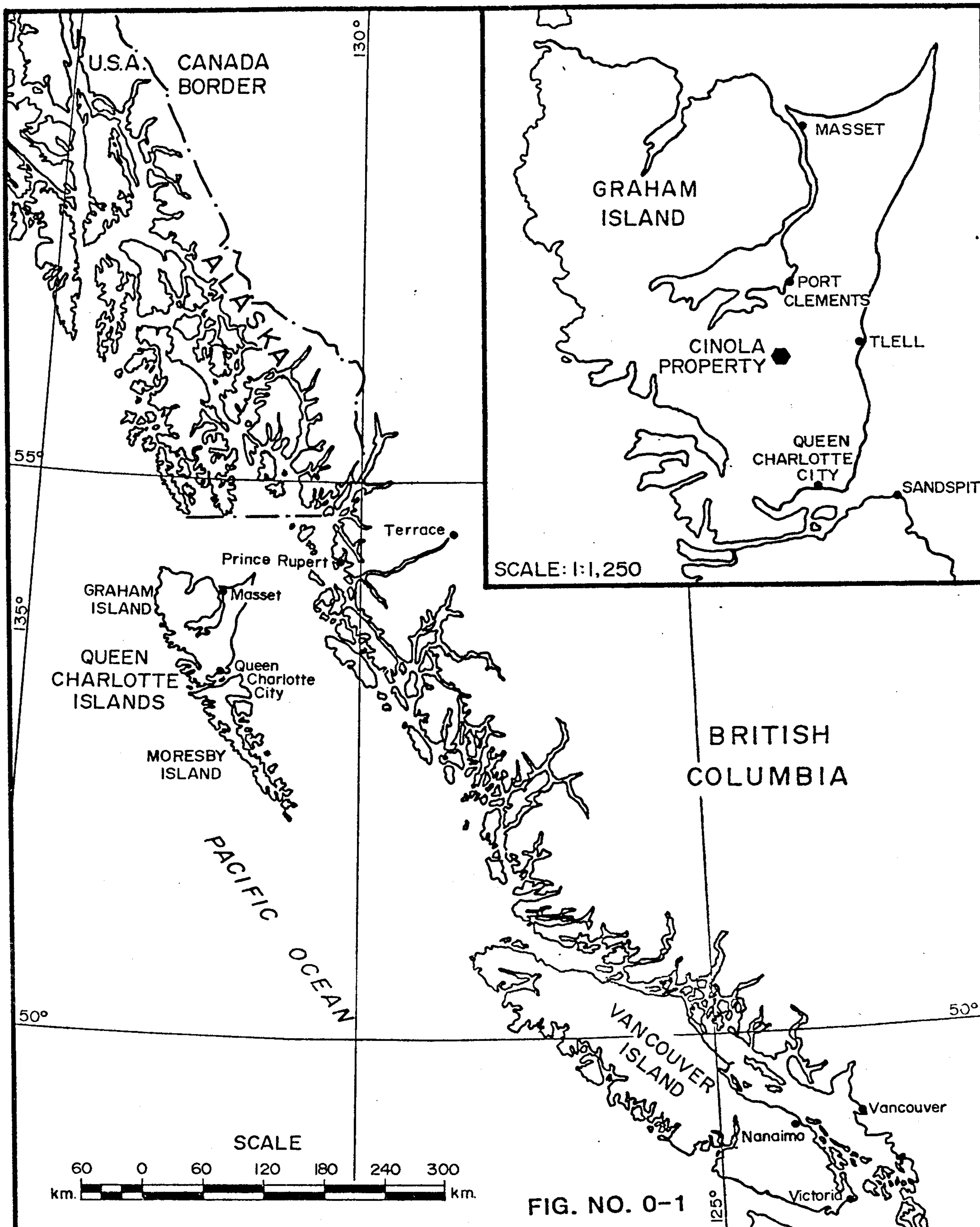

FIG. NO. 0-1

GEOGRAPHIC LOCATION PLAN

CINOLA OPERATING COMPANY LTD.

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CHECKED D.A.	APPROVED D.A.
DWG. NO.	REV.

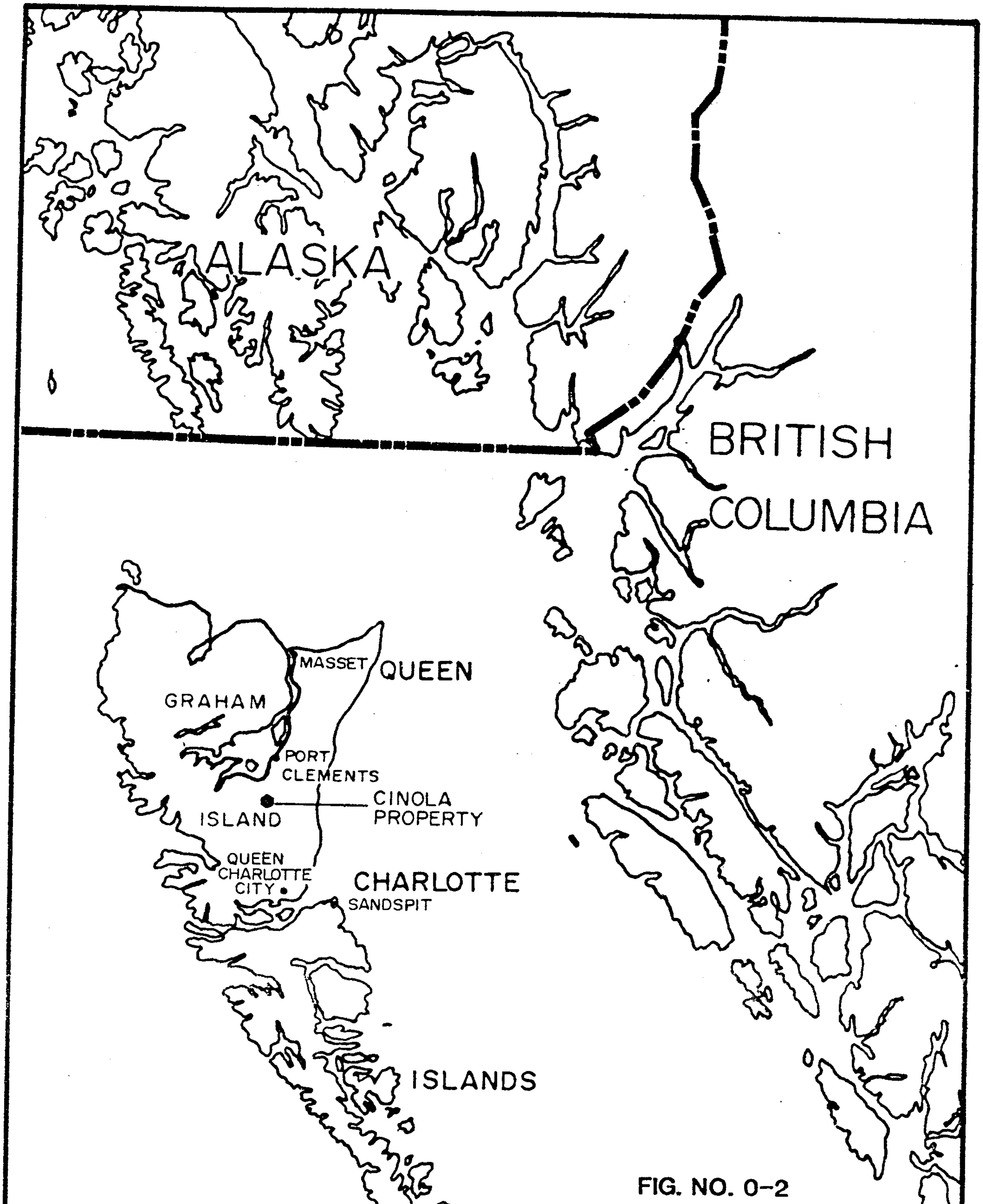

FIG. NO. 0-2

LOCATION PLAN

CINOLA OPERATING COMPANY LTD.

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CHECKED J.D.	APPROVED J.D.
FIGURE NO. 1.3-1	REV.

0.2 GEOLOGY

0.2 Geology

0.2.1 Regional Geologic Setting

The Cinola deposit occurs in the Mesozoic to Tertiary volcanic-sedimentary domain of coastal British Columbia which is characterized by complex, poorly-exposed sediments and volcanics that have been faulted and intruded and altered by a variety of plutonic rocks. More precisely, the Cinola deposit is in Tertiary Skonun Formation sediments. It is bounded on the west by an unconformity with the underlying Cretaceous Haida Formation.

0.2.1.1 Rock Types

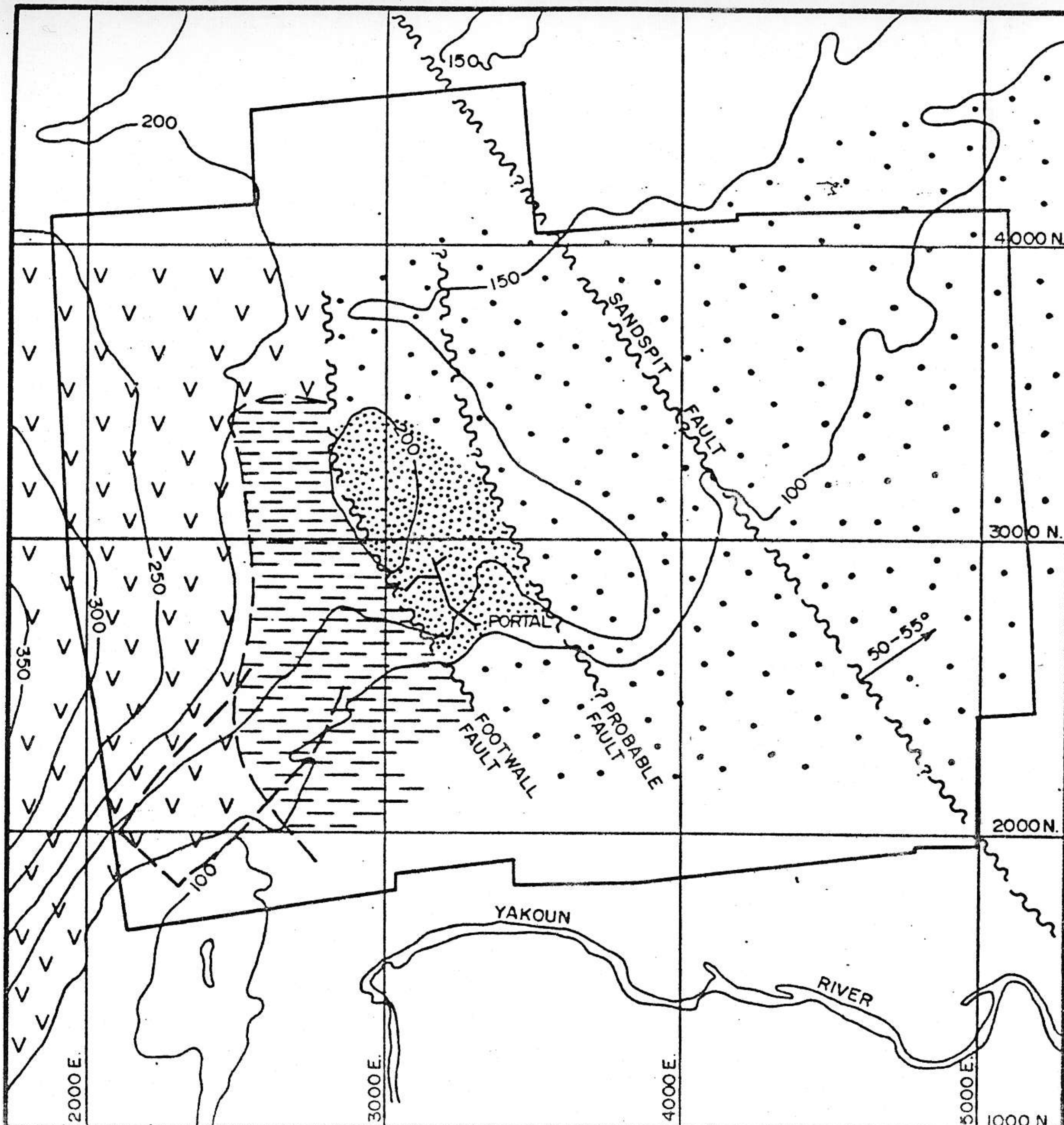
The oldest rocks in the region of the deposit are the black shales of the Cretaceous Haida formation These sediments are weakly metamorphosed, well-bedded pyritiferous black shales of marine origin. The shales extend westward from the deposit to where they disappear beneath younger volcanics of the Masset Formation. (See Figure 0-3). The Haida is unconformably overlain by the clastic sediments of Skonun Formation, the host for the Cinola deposit. The Skonun sediments underlie a vast swampy plain that extends east from the deposit to the coastline.

0.2.1.2 Structure

The major regional structure in the area is the Sandspit fault which extends for tens of kilometres along a northwest trend. This structure is the boundary between the Queen Charlotte lowlands to the east and the Skidegate Plateau to the west. The fault is not exposed in the vicinity of the deposit, but is believed to be from one to more than five km east of the deposit. The Masset Formation outcrops along a prominent south-westerly trending ridge a few hundred metres west of the deposit. The ridge may reflect a structure related to the location of the deposit.

0.2.2 Deposit Geology

Certain characteristics of the Cinola deposit, such as structure, host rock lithology, alteration, and mineralogy are related to the distribution of gold, and therefore influence the estimation of mining reserves and the design of the mine.



CINOLA DEPOSIT - MINERALIZED ZONE

SKONUN FORMATION - LATE TERTIARY SEDIMENTS

MASSET FORMATION - MID TERTIARY VOLCANICS

HAIDA FORMATION - CRETACEOUS SHALES

FIG. NO. 0-3

DWG. ORIGINATOR (CONSULTANT)	
ERTEC ROCKY MOUNTAIN INC.	
CINOLA OPERATING CO. APPROVAL:	
J. F. Delaney	

CINOLA GOLD DEPOSIT
LOCAL GEOLOGY

CINOLA OPERATING COMPANY LTD.

DRAWN:	DATE: 6/9/82
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FIG. NO. 2-1	REV.

0.2.2.1 Exploration History

The Cinola deposit is a large, low-grade gold deposit located in a complex area of sedimentation, structure and alteration. Over a 12-year period, it has been explored by surface sampling and mapping, surface and underground drilling, and underground development of 462 m of adit and crosscuts.

Prior to the acquisition of the deposit by CCM, exploration concentrated on trenching, diamond drilling, and preliminary metallurgical studies. Thirty-four holes totaling 1716 m were drilled. Since acquisition, 165 additional holes totaling 25,329 m were drilled from surface, and 12 holes totaling 1555 m were drilled from the adit. Commencing with the 1977 drill program, core was split at 2 m intervals for assaying. Except for occasional short intervals, high core recovery was reported. Drill hole logs were prepared to describe lithology, alteration, mineralogy, carbon distribution, quartz veins, and to show gold assay values. Split core was stored at the deposit.

In late 1980, COC began underground development of an adit from the south end of the ridge on which the deposit oucrops. The adit was advanced northerly for 294 m. Two crosscuts totaling 148 m, one extending to the east and one to the west, provided a cross-section of the deposit. Two short crosscuts of 20 metres were used as drill stations. During the period from April to December 1981, over 5,000 tonnes of adit muck were treated in the pilot mill. From each round of advance, a sample of crushed material weighing 60 to 70 kg was collected by an automatic sampler. In addition, both horizontal and vertical chip samples were taken from the walls of the headings.

All core samples were fire assayed. Two confirmation holes were assayed by several labs to test repeatability. The majority of the drill core collected since 1979, plus confirmation adit muck samples, were assayed by General Testing Laboratories in Vancouver. Assays for the drilling program in late 1981 and January 1982 were performed in the COC assay lab at the property.

The individual lithologic units within the Skonun Formation tend to be lenticular and difficult to correlate laterally. Thicker packages of conglomerates and sandstone-siltstone have been correlated between drill holes, but reliable marker beds have not been identified within the sediments.

0.2.2.2 Host Rocks

The Skonun Formation sediments that host the deposit (see Figure 0-3) are dominantly coarse, continental clastics that range from pebble conglomerates to siltstones. Some of the sediments are moderately well sorted and bedded; others are poorly sorted, with coarse clasts suspended in the matrix fine-grained material. The clasts are well to poorly rounded, and are dominantly composed of porphyritic intrusive rocks, but granite, basalt, chert, older conglomerates, and volcanics are also present.

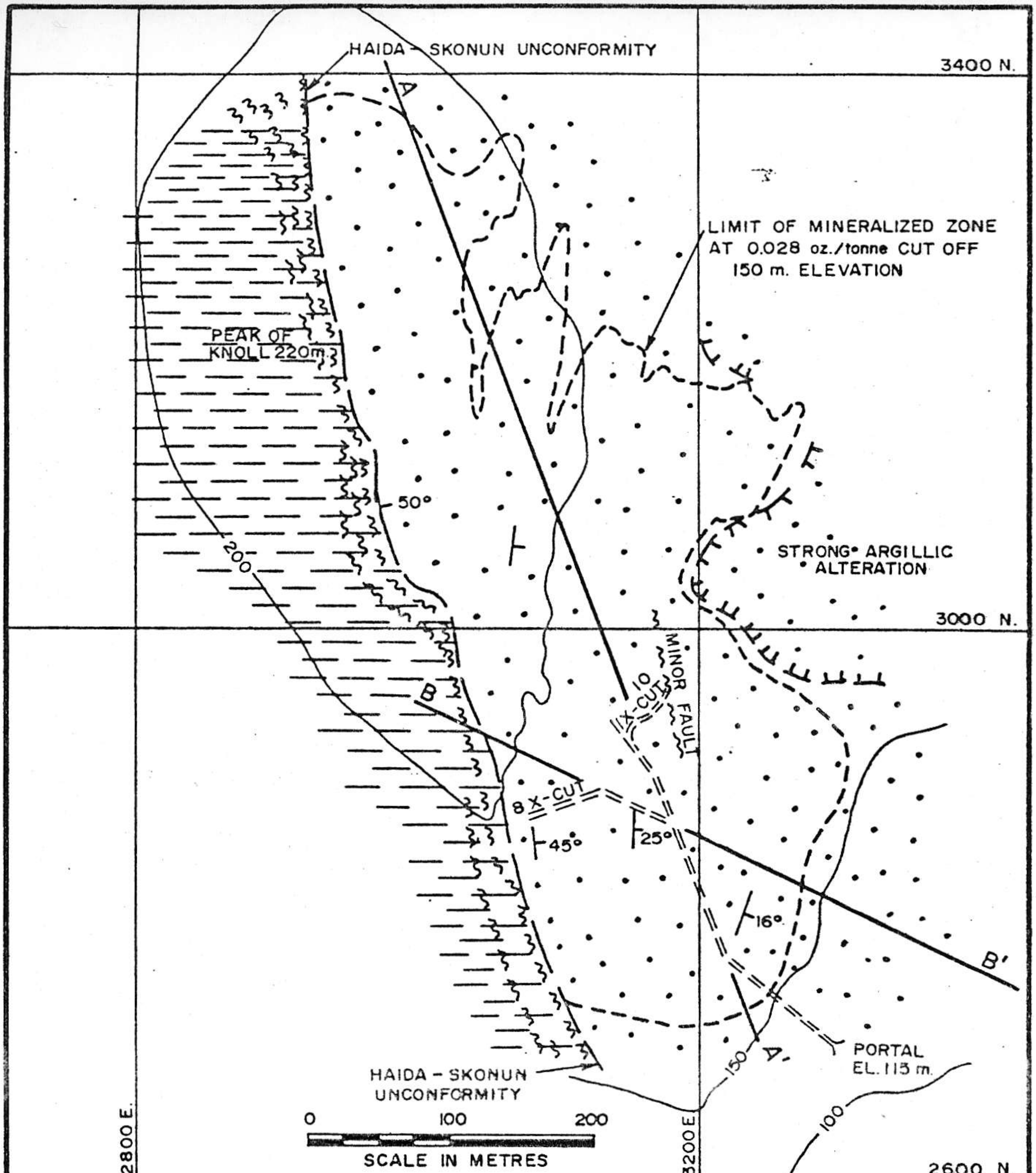
Colors of the Skonun sediments range from dark gray and brown to lighter colors where silicified, and to buff, tan, and pinkish where strongly clay altered. Carbonaceous material, as recognizable plant debris and fragments, can comprise a significant portion of any one sample, but the average concentration is less than one percent.

At the deposit, the Skonun sediments strike north to northeast. The dip of the sediments decreases from approximately 45 degrees adjacent to Specogna fault and unconformity on the west to 25 to 15 degrees toward the east. Figure 0-4 shows the approximate outline of the mineralized zone on the 150 m level where it is widely developed.

Neither intrusive nor volcanic rocks have been positively identified within the deposit, apart from clasts within the sediments. Altered Skonun sediments adjacent to the Specogna fault are identified as rhyolite.

0.2.2.3 Structure

The Specogna fault occurs immediately west of the deposit. The fault strikes approximately north 20 to 25 degrees west, and dips eastward below the footwall of the deposit at 50 to 55 degrees. (See Figure 0-5A cross-section B-B' from



SKONUN FORMATION - SEDIMENTS

HAIDA FORMATION - SHALES, ARGILLITES

20° STRIKE & DIP OF BEDDING

FAULT, FAULT GOUGE

FIG. NO. 0-4

DWG. ORIGINATOR (CONSULTANT)

ERTEC ROCKY MOUNTAIN INC.

CINOLA OPERATING CO. APPROVAL:

GEOLOGIC PLAN
150 m ELEVATION

CINOLA OPERATING COMPANY LTD.

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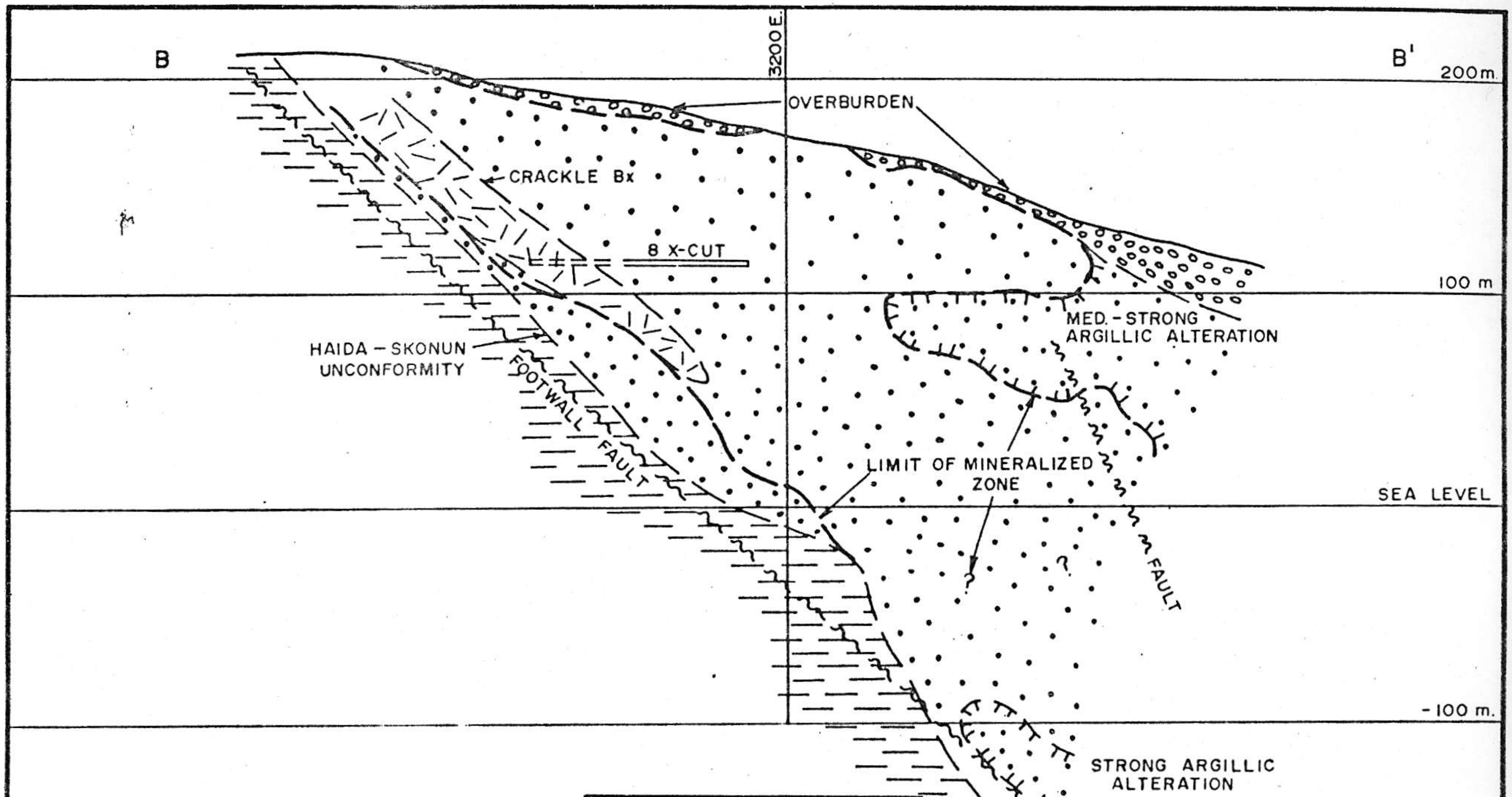
FIG. NO. 2-2	REV.



SKONUN FORMATION - SEDIMENTS

HAIDA FORMATION - SHALES, ARGILLITES

DWG. ORIGINATOR (CONSULTANT)
ERTEC ROCKY MOUNTAIN INC.
CINOLA OPERATING CO. APPROVAL:
J. F. Delaney

0 50 100
SCALE IN METRES

FIG. NO.0-5A

GEOLOGIC CROSS SECTION B-B'

CINOLA OPERATING COMPANY LTD.

DRAWN:	DATE: 6/9/82
CHECKED I.S.	APPROVED
FIG. NO. 2-3	REV.

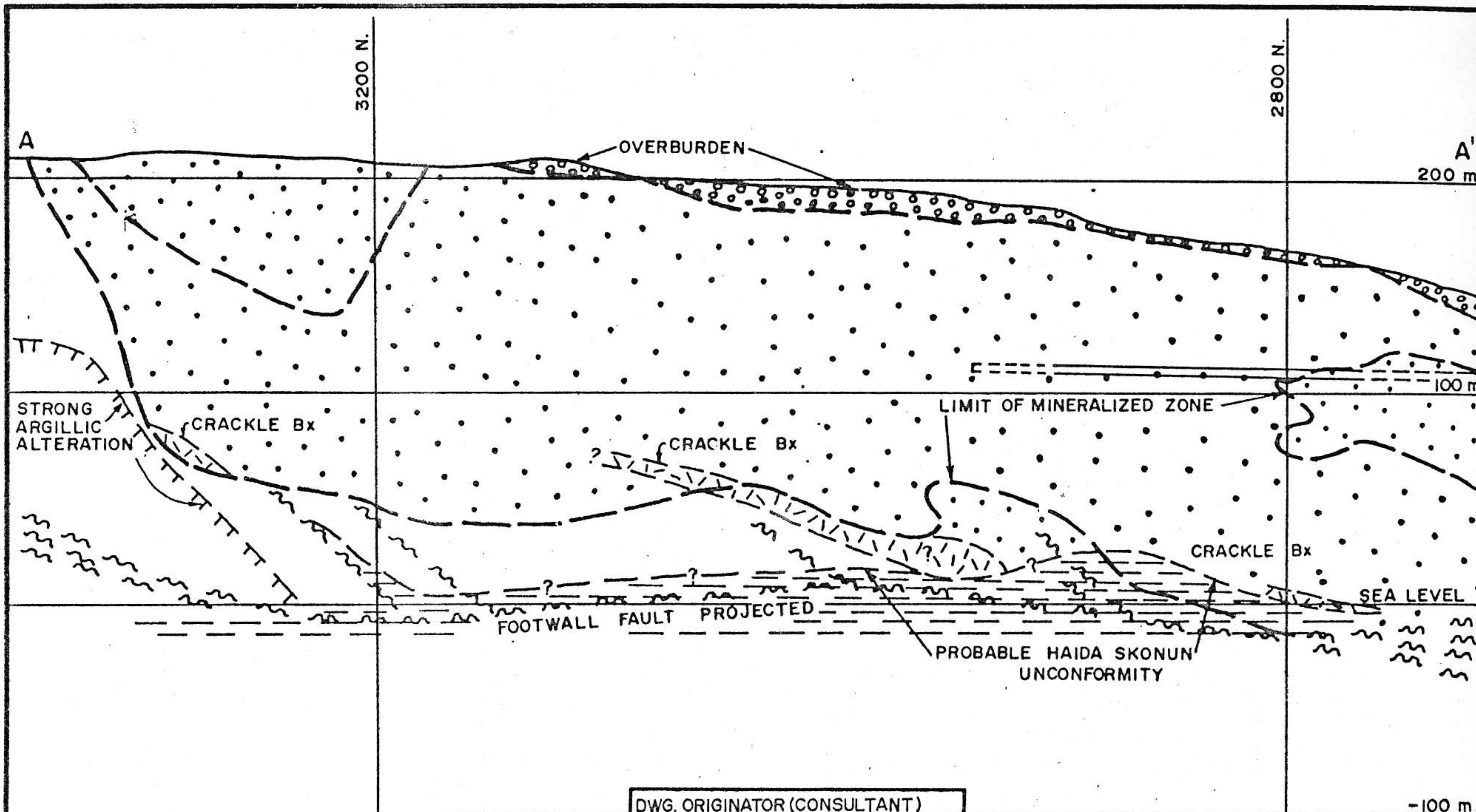
Figure 0-4.) The fault zone is principally in the fractured Haida shales below the Skonun-Haida unconformity and is approximately 30 m wide. Marker beds have not been identified which confirm the direction or amount of displacement. The fault probably experienced repeated movement, one stage of which formed the so-called "crackle breccia" shown in Figure 0-5B. The Haida shale between the fault and the unconformity is silicified to a so-called "argillite".

The structures most important to gold mineralization are a series of quartz-filled veins within the deposit. The major veins strike north 20 to 45 degrees east (average 28 degrees east), and dip up to 20 degrees from the vertical. Vein widths range from a few millimetres to tens of centimetres, but the average width is less than 10 cm. These veins are most abundant in the core of the deposit near the northern part of the adit. Horizontal to shallow dipping veins are also present in the deposit, and are most noticeable in the upper parts of the deposit and to the east of the central core. Surface mapping indicates that vein size changes rapidly over short distances along strike. Similar rapid changes may also occur vertically. Throughout the deposit, quartz veins are associated with higher gold assays, but quartz veins are not necessarily gold-rich.

0.2.2.4 Mineralogy

The clasts of the Skonun sediments are composed of the common rock forming minerals such as quartz and feldspars, mafic minerals including biotite and hornblende, and opaques such as magnetite. Pyrite, associated principally with carbonaceous plant material, was an early mineral formed in the sediments.

In the vicinity of the deposit, mafic minerals and feldspars were selectively converted to some combination of quartz and clay minerals (including kaolinite, sericite, and/or illite), chlorite, and lesser amounts of hematite and epidote. The most intense alteration produced rocks composed almost entirely of quartz or clay.



SKONUN FORMATION - SEDIMENTS

HAIDA FORMATION - SHALES, ARGILLITES

DWG. ORIGINATOR (CONSULTANT)

ERTEC ROCKY MOUNTAIN INC.

CINOLA OPERATING CO. APPROVAL:

J.F. Delaney

FIG. NO. 0-5B

0 50 100

SCALE IN METRES

GEOLOGIC LONGITUDINAL SECTION A-A'

CINOLA OPERATING COMPANY LTD.

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FIG. NO. 2-4	REV.

Vein-related minerals include the gold-bearing phases and the minerals that preferentially occur with them. Gold occurs principally as native gold in submicron flakes in silicified Skonun sediments, and in quartz veins that cut sediments. Larger gold particles are rarely visible except in high-grade (generally greater than 0.5 oz/tonne) veins. Significant gold concentrations in pyrite are suggested by the fact that gold recovery is improved by roasting, but only a few blebs of gold have been seen in pyrite, and then only under high magnification. The association of gold with carbonaceous material is probably of minor and only local importance, because of the low concentration of carbon. Silver occurs with gold as an electrum.

Quartz is the most important vein-related material. The majority of low-grade gold occurs in silicified sediments, and virtually all high-grade mineralization is associated with dark-gray to white and clear quartz veins which have complex color banding and cross-cutting relations. Pyrite and marcasite are important vein-related minerals. They are estimated to comprise about 1.5 percent of the total vein rock mass, and occur as crystals disseminated in silicified and clay-altered sediments concentrated within and along the walls of quartz veins. Although gold-bearing pyrite is apparently an important vein constituent, barren pyrite is common.

0.2.2.5 Alteration

Distribution of gold in the Cinola deposit bears important relations to several alteration types that have affected the host rocks of the deposit.

Silicification of the clasts and matrix of the Skonun sediments is the most pervasive type of alteration within the deposit. It tends to preserve the prior color of the rocks or makes them lighter. Virtually the entire Cinola ore deposit occurs within the silicified sediments. However, the degree of silicification at any spot ranges from partial to complete. Thus, portions of the deposit remain punky and vuggy, but more generally, the deposit is hard and abrasive.

The broad zone of silicification that hosts the deposit extends from the Specogna fault upward and eastward through the Skonun sediments. The most intensive alteration, adjacent to the fault, has converted the Haida shales to so-called "argillite", and the basal Skonun sediments to a strongly bleached and indurated rock referred to as "rhyolite". These silcified sediments are cut by quartz veins and so-called "crackle breccia".

Skonun sediments in or adjacent to the deposit that are not silicified are composed of some combination of clay minerals. Studies have identified kaolinite, sericite or illite, and cholorite. Distinct zones appear be present. East of the deposit, for example, silicification with gold mineralization in the upper part of the sediments gives way to brownish sediments, reportedly composed of kaolinite which are virtually barren of gold. They are characterized instead by carbonaceous plant remains in the finer-grained sediments and disseminated euhedral pyrite. Strongly-bleached, clay-rich zones with up to five percent disseminated pyrite and low-grade gold values have been observed in core samples.

0.2.3 Correlation of Geology and Geologic Reserve Estimates

The geologic reserve estimates presented in the following section are based upon specific assumptions concerning the following geologic parameters:

1. The limits of mineralization
2. Continuity within the mineralized zones
3. Variability within the zones
4. Distribution of subpopulations within the deposit.

The geologic interpretation of the deposit characterizes it as comprised of a central vertical core zone of mineralization approximately 200 m long and 40 to 80 m wide that extends vertically up to 150 m through the complex sediments the Skonun Formation. Within this core, the intensely silicified sediments are cut by a profusion of dominantly vertical quartz veins with which are associated the higher gold concentrations. Within the silicified zone are generally poorly-defined subzones that have been identified and correlated as lenses for purposes of reserve estimation. Extending out from this core zone, several subhorizontal tabular zones of mineralization show the influence of stratigraphic control, but are not associated with any particular rock type. The most promirant of these zones have been assigned lens designations (U, M-1, M-2, etc). See Figure 0-6. Away from the core zone, particularly toward the east, the lenses diminish in thickness and grade and ultimately encounter clay alteration zones with which they display both tabluar and abutting relationships. It is assumed that the formation of the deposit occurred as a series of pulses or stages of mineralization and alteration, with fluids moving up into, and to some extent out through, the core zone to form the surrounding tabular lenses.

All geologic data encountered in this study are compatible with the foregoing interpretation, and it is on this basis that the mineralized lenses were defined and the reserves estimated. More specifically, the compatibility of geologic interpretation and estimation methods can be summarized as follows:

1. Mineralization Limits - The existing drill holes are adequate to identify the limits of the deposit to within a few tens of meters within the Skonun sediments. The absence of isolated patches of significant mineralization support this interpretation. Undoubtedly some isolated patches do exist but are too infrequent and small to have been encountered in existing drill holes. The boundary of the deposit along the fault is more of a problem because of the complexity of the fault zone, and minor amounts of additional mineralization may be added in this area.

2. Continuity - The association of higher-grade mineralization with quartz veins and the irregular distribution of those veins indicates that significant grade variations occur over short distances. On the other hand, drilling indicates that the distribution of higher-grade zones is not erratic or random. Mining results can be expected to corroborate the reserve estimates for larger volumes (lenses), but local zones stand to vary substantially, and require appropriate grade control procedures.

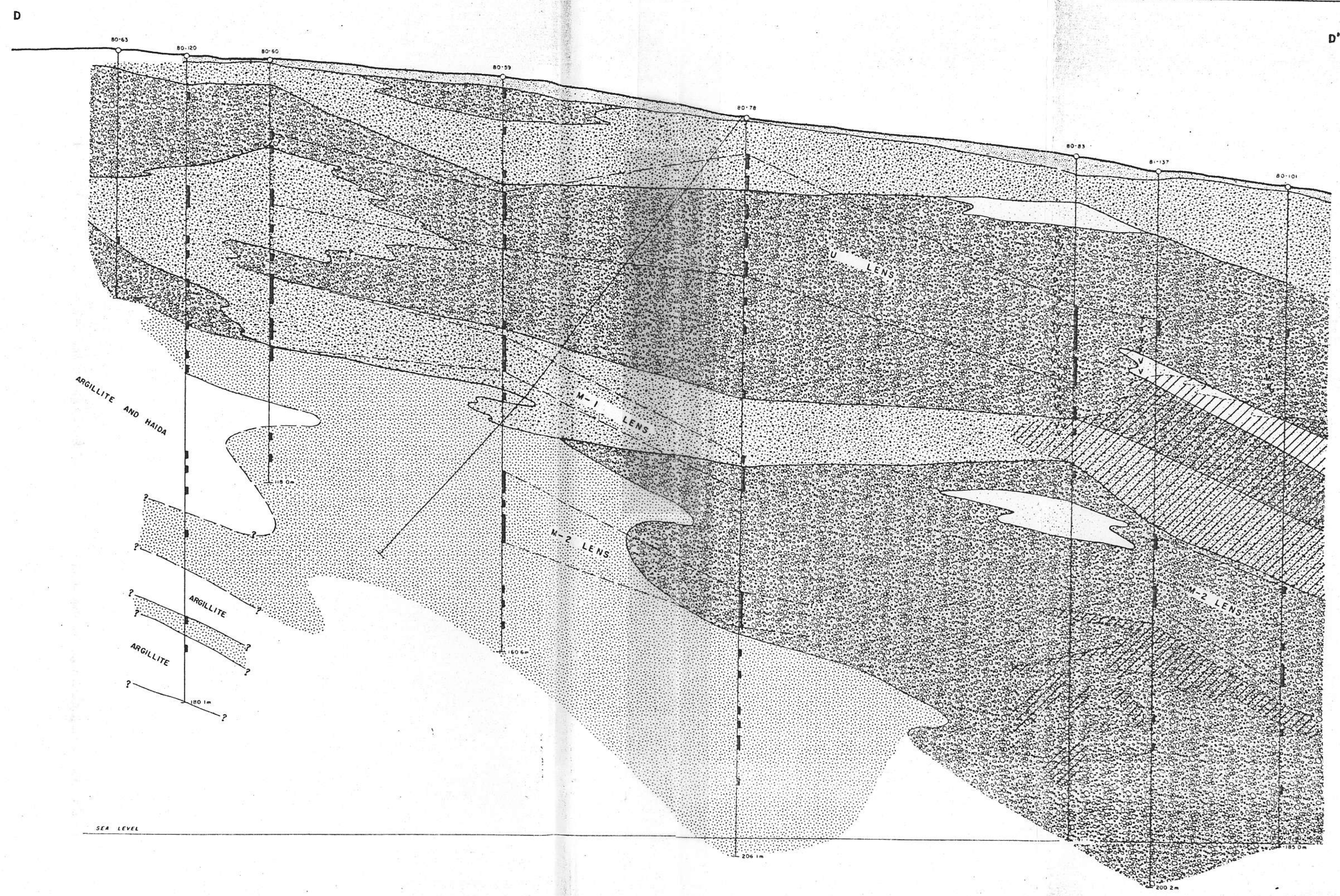

FIG. NO. 0-6

LEGEND

- V QUARTZ VEINS
- QUARTZ BRECCIA
- DRILL HOLE
- > 0.05 OZ./TONNE Au
- BOUNDARY OF MINERALIZED LENS
- OVERBURDEN
- PEBBLE CONGLOMERATE
- SANDSTONE AND SILTSTONE
- MATRIX-SUPPORTED CONGLOMERATE
- "RHYOLITE"
- PERVASIVE KAOLINITIC ALTERATION

VERTICAL AND HORIZONTAL SCALE
0 10 20 30 40 50 METRES

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#402 - 595 HOWE ST.
VANCOUVER, B.C.
V6C 2T5
(604) 669-1524

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QUEEN CHARLOTTE ISLAND GOLD PROJECT
CONSOLIDATED CINOLA MINES LTD. & ENERGY RESERVES (CANADA) LTD

GEOLOGIC CROSS SECTION,
NORTHERN PART OF CINOLA DEPOSIT

DATE: 7/9/82	SCALE: 1:500	DWG. NO. 30 / 15 / 3

3. Variability - No specific geologic assumptions regarding grade variability within the lenses or the lower-grade material are made in this report, but local rapid variations are expected. The reserve estimating technique tends to smooth extreme higher grade and lower grade excursions.

4. Subpopulations - Geologic and grade distribution considerations lead to the delineation of higher-grade lenses and waste zones. These are treated as two subpopulations which tends to minimize the dilution of the higher-grade material. The actual boundaries of these zones are only approximately located, and they may prove to be quite irregular. Careful monitoring during early mining should identify opportunities for grade control.

Therefore, the geologic interpretation of the deposit and the reserve estimation methods are considered appropriate and well matched for the deposit based upon existing data.

0.3 RESERVES

0.3 Reserve Estimations

The inverse distance squared method was used to estimate geologic reserves for mine design and the estimation of minable reserves. The grade-thickness and kriging methods were used as checks on the inverse distance squared method. The geologic reserves estimated for this study are related to specific cut-off grades and particular geologic interpretations.

0.3.1 Evaluation of Data Base

0.3.1.1 Types of Data

Sample data from the Cinola deposit consist of assays of split diamond drill core from vertical angle and horizontal holes; muck samples from each round mined from the adit; and chip samples from the adit. Drilling was done on a generally rectangular grid, with holes designed to be about 40 m apart. The final pattern of drilling was random, with holes varying from 10 m to 50 m apart. In addition to the vertical and inclined surface holes, seven inclined and five horizontal holes were drilled from underground stations in the adit.

The drillhole assay data were composited into 10 m intervals from the top of bedrock. Using the digitized boundaries of bedrock, the Specogna fault, and the clay alteration to the east, assay composites below the footwall fault and within the clay zone were eliminated from the deposit block model estimation.

Ore reserves were estimated using only the drillhole assay data. The block model estimates were subsequently compared with the adit samples to test the consistency of the model.

0.3.1.2 Comparison Between Vertical and Inclined Drill Hole Samples and Adit Muck Samples

Review of gold distribution in the deposit indicates that dominant gold mineralization is essentially vertical, striking approximately north-south, and is composed of dominantly northeast-striking quartz veins. This zone is particularly well developed in the central part of the deposit near the adit. The second well developed continuity, marginal to the central part of the deposit, is subparallel to the stratigraphy and dips from a few to a few tens of degrees toward the east.

A limited number of comparisons (8) between vertical and inclined surface holes, and inclined and horizontal underground holes, and with the adit muck samples, allow the following tenuous generalizations to be made:

1. The vertical holes compare closely with inclined or horizontal holes for three sets of data and show significant discrepancies for five sets. These discrepancies most likely reflect both the inherent variability of gold grades in the deposit and differences in sampling between the two sets of holes.

2. Comparisons for the central part of the deposit continually show vertical holes with lower grades than horizontal and inclined holes or the adit muck samples. As this is the region of most persistent vertical quartz veining, brecciation, and vertical continuity of mineralization, it is likely that the preponderance of vertical holes tends to underestimate the grade of the deposit. It is also possible that the limited underground holes intersected unrepresentatively high-grade ore, and that the underestimation is not as high as is indicated by these comparisons.

3. In the northern and west central parts of the deposit, vertical holes indicate a higher grade than inclined holes. The vertical holes show significantly higher grade for the uppermost zone of mineralization (U lens), which is the most tabular of the mineralized zones. In contrast to the central part of the deposit, therefore, where mineralization is more vertically oriented, vertical holes may produce a better sampling of the peripheral, more tabular, mineralization.

It is likely that ore grades in the central part of the deposit are at least locally higher, and perhaps significantly higher, than has been indicated by vertical drilling. Any future drilling in the central part of the deposit should be inclined or horizontal, and should be oriented approximately east-west at a high angle to the northeast trending quartz veins and the north-northwest trend of vertically continuous mineralization.

0.3.2 Interpretation of Gold Distribution

The high concentration of steeply-dipping veins in the central part of the deposit accounts for the well-developed vertical continuity of mineralization in that area. However, drillhole cross-sections also indicate a tabular habit to the mineralization, particularly in the upper part of the deposit and peripheral to the central part of the deposit.

These two patterns of mineralization are in many cases related to geologic features such as stratigraphic units, alteration zones, and the abundance of quartz veins and breccias (see Appendix A.1 of the main report). These patterns are accepted, therefore, as the best representation of the distribution of gold mineralization. Mineral zones are defined accordingly for purposes of reserve estimation.

The delineation of mineralization lenses or zones is based upon the grade (using 2 m assays) and thickness of gold-bearing intervals. Minimum thicknesses of 4 m and 10 m are used as reasonable for selective and conventional pit benches.

Intervals meeting the following criteria were marked on core logs:

1. Intervals of $\geq$10 m averaging $\geq$0.05 oz/tonne and bounded by 2 m assays of $\geq$0.05 oz/tonne.
2. Intervals of $\geq$10 m averaging $\geq$0.03 oz/tonne but $\leq$0.05 oz/tonne and bounded by 2 m assays of $\geq$0.03 oz/tonne.

3. Intervals of ≥ 4 m but <10 m averaging ≥ 0.05 oz/tonne (generally high-grade intervals within thicker intervals of lower-grade material).

Once compiled on drill logs, the mineralized intervals were correlated between drill holes to produce lenses or zones of mineralization that are inferred to have horizontal and vertical continuity within the deposit. The correlated lenses are shown on Drawings 30 1 27/0 through 30 1 43/0. Lenses M-2 and L-2 are in contact with lens M-1 in many places. The boundaries between them are arbitrarily drawn so as to maximize the volume of ore included in lens M-1. (Note: References to drawings and appendices in this summary are made for completeness; all such material, however, is contained in the main report.)

The top and bottom elevations of each lens in each hole were marked in the computer data base and the values kriged to generate a surface for each. The horizontal extent of each lens meeting one or more of the criteria was outlined on a drill hole plan map. Contour maps were prepared for each lens, showing average grade, thickness, grade-thickness, and structure contours on the top and bottom of the lens. These maps are shown in Drawings 30 1 44/0 through 30 1 68/0, together with a plan map showing drillhole locations, the adit location, and the approximate area of each lens. Table 0-1 reports the average grade calculated for 2 m assays of each lens. The average grade for the five lenses is 0.100 oz/tonne.

The lenses or zones appear to describe the general continuity of mineralization as well as can be done with existing data. It should be noted, however, that heterogeneity of the host rock environment, due to 1) variability of sediment permeability and structural behaviour, 2) the complexity of alteration, and 3) the uneven distribution of mineral-controlling fractures, produces substantial grade discontinuties, even within the lenses.

0.3.3 Geologic Reserve Estimation

The geologic reserve estimate for mine planning was prepared for a block model by the inverse distance squared method. As a check of this method, grades and tonnages of the higher grade lenses were also estimated by both the kriging method and a grade-thickness contour method. The inverse distance squared block model was checked with a geostatistical kriging estimate for the same block model.

TABLE 0-1
AVERAGE GRADE OF THE HIGHER-GRADE LENSES COMPUTED FROM TWO-METRE ASSAYS IN EACH LENS

Lens	Average Grade (oz/tonne)	Total Length of Sample (metres)
U	0.093	1,432
M-1	0.092	2,918
M-2	0.103	440
L-1	0.200	318
L-2	0.107	275
Combined	0.100	5,383

0.3.3.1 Inverse Distance Squared Method

The inverse distance squared method of reserve calculation uses a regular block model for the deposit and assigns values to each block. A maximum of 32 samples is used to evaluate each block, and the samples are weighted according to the inverse square of the distance from the center of the block being evaluated to the center of the sample. The search radius used in the inverse distance squared estimation was determined by calculating variograms for both lens and selected non-lens composites, in horizontal and vertical directions.

Because the inverse distance squared model only considers the nearest 32 samples to the center of a block, the use of 2-m assay values would excessively weight the closest hole to the block. Therefore, the 2-m assays were composited into 10-m intervals, beginning at bedrock for surface holes, and at the collar for underground holes. The 10-m composites then became the sample population from which the inverse distance squared model was estimated. The use of 10-m composites introduced approximately 9 percent dilution, thus lowering the average grade of lenses to 0.090 oz/tonne.

Grade estimates for the mineralized lenses calculated by the inverse distance squared block model are compiled in Table 0-2. This method gives results that are slightly greater in tonnes and lower in grade than the grade-thickness contour method. This is to be expected because of dilution resulting from the 10-m composites used in the inverse distance squared method. The kriging method gives slightly lower grade, as is typical of that method. However, the three methods agree to within four percent, an excellent correlation.

Bench plans showing blocks developed with the inverse distance squared method are included within Drawings 30 1 69/0 through 30 1 94/0. Histograms (Figures A-9 and A-10) for the estimated block grades and logs of the estimated block grades are presented in Appendix A.6 of the main report.

The bench-by-bench geologic reserves for the inverse distance squared and kriged block models, at 0.033 oz/tonne cut-off are given in Table 0-3. As is to be expected, the kriging method tends to estimate higher tonnes (14.3 percent) and lower grades (7 percent). Kriging also estimates 6 percent higher ounces of gold.

0.3.3.2 Grade-thickness Contour Method

The grade-thickness contour method was used to calculate a second estimate of geologic reserves within the higher grade lenses. The grade-thickness product for each lens in each drill hole was calculated and values for each lens contoured with a kriging routine. This method tends to smooth out erratic grade variations and gives a good approximation of total tonnes and grade.

The reserves calculated for the lenses by this method are shown in Table 0-2. As discussed above, estimates by the three methods compare well.

0.3.3.3 Kriging Method

Reserves were independently estimated by the geostatistical kriging method to provide a comparison for the inverse distance squared block model. This estimation method is preferred by many as it provides estimates of confidence in addition to grade and tonnage. A more thorough discussion of the methods used is presented in Appendix A.7.

TABLE 0-2

COMPARISON OF GEOLOGIC RESERVES FOR THE HIGHER-GRADE LENSES IN THE CINOLA DEPOSIT DETERMINED BY THE GRADE-THICKNESS CONTOUR, INVERSE DISTANCE SQUARED AND KRIGING METHODS

Method	Lens	Average Grade (oz/tonne)	Tonnes ($x10^6$)	Total Ounces Au ($x10^6$)
G-T Contour	U	0.094	3.623	
	M-1	0.086	7.034	
	M-2	0.104	0.931	
	L-1	0.155	0.589	
	L-2	0.099	0.609	
	Combined	0.093	12.786	1.189
$1/d^2$ Block Model	U	0.081	3.779	
	M-1	0.085	7.738	
	M-2	0.085	1.406	
	L-1	0.114	0.267	
	L-2	0.099	0.851	
	Combined	0.085	14.041	1.197
Kriged Block Model	U	0.078	3.779	
	M-1	0.083	7.727	
	M-2	0.082	1.406	
	L-1	0.098	0.267	
	L-2	0.082	0.841	
	Combined	0.082	14.020	1.149

TABLE 0-3

BENCH-BY-BENCH GEOLOGICAL RESERVES FOR THE INVERSE DISTANCE SQUARED AND KRIGED BLOCK MODELS

(0.033 oz/tonne Cutoff)

	Inverse Distance Squared		Kriged	
BENCH	TONNES	GRADE*	TONNES	GRADE*
1	145,946	0.035	291,674	0.037
2	546,018	0.049	978,626	0.045
3	1,026,067	0.058	1,350,123	0.053
4	1,597,846	0.058	1,786,682	0.058
5	2,049,013	0.058	2,306,433	0.059
6	2,344,588	0.058	2,680,681	0.058
7	2,683,055	0.057	2,959,518	0.056
8	2,486,728	0.064	2,835,033	0.059
9	2,378,738	0.066	2,768,855	0.061
10	2,218,738	0.060	2,425,328	0.059
11	1,833,282	0.062	2,058,373	0.059
12	1,584,825	0.067	1,672,181	0.064
13	1,652,571	0.061	1,616,792	0.059
14	1,604,012	0.058	1,474,310	0.058
15	1,704,266	0.058	1,745,486	0.056
16	1,786,946	0.054	1,849,461	0.052
17	1,893,114	0.050	1,977,360	0.048
18	1,857,551	0.055	1,971,368	0.050
19	1,809,441	0.054	1,963,135	0.049
20	1,559,209	0.054	1,818,518	0.047
21	791,461	0.061	1,243,170	0.044
22	604,853	0.067	963,168	0.047
23	544,030	0.052	809,244	0.044
24	442,040	0.042	781,280	0.040
25	328,941	0.045	616,781	0.039
26	370,080	0.058	308,400	0.042
TOTAL	37,843,361	0.058	43,251,963	0.054

* Troy oz/tonne

A kriged model of the ore deposit for 20 x 20 x 10-m blocks was constructed using a calibrated variogram model and the technique of log-normal kriging. As with the inverse distance squared method, blocks were marked as being in lens or non-lens material and were kriged using composite samples identified as lens or non-lens material. The same search pattern as utilized in the inverse distance squared method was chosen for determining the kriged estimates. Histograms of the estimated block grade and the log of estimated block grade for combined lens and non-lens material are given in Appendix A.7 (Figures A-20 and A-21). The block bench plan is presented in Drawings 30 1 95/0 through 30 1 120/0. Bench by bench geologic reserves are given in Table 0-3.

0.3.3.4 Comparison of Kriged and Inverse Distance Squared Estimates

Comparison of the block model reserve estimates prepared by kriging and the inverse distance squared methods indicate acceptable agreement. As is usually the case, kriging estimates somewhat lower grades than the inverse distance squared method.

A comparison of the geologic reserves for the two estimation techniques for 6 cutoff grades is detailed in Table 0-4. The effect of higher grades due to more numerous high-grade blocks for the inverse distance squared estimate is reflected in higher grades as the cutoff is raised.

Block grades were estimated by the inverse distance squared and kriging methods for comparison with adit muck samples within the same blocks. The prediction of grade by both estimation methods was very close to the mean grade of material mined in the adit from those blocks. However, individual blocks showed considerable variation in muck grade versus predicted grade.

0.3.3.5 Confidence of Kriged Reserve Estimate

The kriging technique permits the estimation of confidence for each block grade. The average kriged grade of lens blocks is 0.0812 with an approximate 95 percent confidence interval, (i.e., a range from 0.0797 to 0.0828 oz/tonne), while the average kriged grade of non-lens blocks is 0.0263 with an approximate 95 percent confidence interval (range of 0.0261 to 0.0265 oz/tonne).

Kriging variances for blocks in the Cinola deposit indicate that the estimation of block grade is relatively poor along the boundaries of the ore deposit, with the largest estimation variances being observed in the lowest benches where drill core data are sparce. The "poorness" of a particular block estimate reflects a complex interaction of the search radius, the number of data points used in estimation, their proximity and configuration relative to the block centroid, and the variogram model.

0.3.3.6 Accuracy of Reserve Estimates

The accuracy of the reserve estimates depends upon the accuracy of each step in the preparation of the estimates. Because there is no rigorous method to determine the final estimation accuracy, each of the principal steps is briefly reviewed below:

1. Sampling - Drill density and drill hole orientation within the deposit are sufficient to prepare a reserve estimate. However, drilling in the central part of the deposit is inadequate to identify potentially higher-grade material. The reserves presented are most likely minimum reserves which may or may not be significantly increased with further drilling. Additional reserves, particularly at higher production costs, are likely to occur in the footwall zone, possibly at depth below the deposit, in separate zones near the deposit and particularly as additional deposits similar to Cinola in the vicinity.

2. Sample Handling - Tests of core splitting have not been made, but the fine-grained nature of most of the gold suggests this step has not introduced error. Tests of sample splitting indicate good reproducibility.

3. Assays - Comparative assays performed within Cinola lab and other labs indicate approximately a 0.011 oz/tonne bias in the lower grade range with the Cinola lab yielding higher assays than General Testing. Cinola assays, therefore, provide over-estimations for those samples, but few of the exploration holes were analyzed in the Cinola lab.

TABLE 0-4
COMPARISON OF KRIGED VERSUS INVERSE
DISTANCE SQUARED GEOLOGIC RESERVES
FOR VARIOUS CUTOFFS

	Kriged Estimates		Inverse Distance Squared Estimates	
Cutoff*	Tonnes x 10^6	Grade*	Tonnes x 10^6	Grade*
0.02	90.55	0.040	95.02	0.039
0.03	52.44	0.050	47.70	0.052
0.04	27.55	0.064	23.16	0.072
0.05	16.62	0.078	16.27	0.084
0.06	13.31	0.084	13.41	0.090
0.07	9.98	0.090	10.16	0.098

* Troy oz/tonne

4. Geologic Interpretation - As discussed in the following section, the geologic interpretation is believed to be generally compatible with observations, hence no significant error is expected. Higher-grade reserves, however, may be delineated in the core of the deposit with additional drilling.

5. Estimation Method - Geologic reserves for this study were prepared by three methods (grade-thickness contour, inverse distance squared and kriging) for the higher-grade lenses and by two methods (inverse distance squared and kriging) for the block models. As there is no method to evaluate the accuracy of the reserve estimate itself, the closeness of these methods is the best measure of the validity of the inverse distance squared reserves.

In summary, each step in the reserve estimation process was reviewed or designed so that the reserve estimate would be as reliable as possible. The inverse distance squared geologic reserve estimate used for mine design and the estimation of minable reserves in the following section is believed to be as accurate as can be prepared with existing data.

0.4 MINE DESIGN

0.4 Mine Design

0.4.1 Geotechnical Analysis

A two-stage engineering study was carried out to evaluate slope stability for the proposed open pit. The first stage involved data collection at the mine site and included engineering geological core logging, fracture mapping, and field and laboratory testing. In the second stage, these data were used to construct a geomechanical model of the rock mass within the open pit area. With the aid of the model, the stability of slopes associated with the interim and ultimate pits was analyzed.

The proposed pit was divided into four design sectors. These sectors were based on differences in slope orientations and geometries as defined by the pit plan. Within each sector, zones having similar geological fabric and strength properties were defined. The various design sectors and structural domains, together with the recommended inter-ramp slope angles are shown in Figure 0-7.

Slope stability was evaluated at overall, inter-ramp, and bench scales. Currently, full-height slope failures are not expected, because major geological structures and their potential intersections are interpreted as either dipping too steeply to daylight in the slope or are of insufficient size to form a throughgoing failure surface. With the exception of the kaolinized Skonun domain in the east wall of the ultimate pit, and the footwall fault domain in the west wall of the interim pit, the probabilities of failure are acceptably low for inter-ramp slopes. However, because the complex jointing patterns allow for high probabilities of bench backbreak (partial or total bench failure), bench stability significantly affects overall and inter-ramp slope design in all design sectors.

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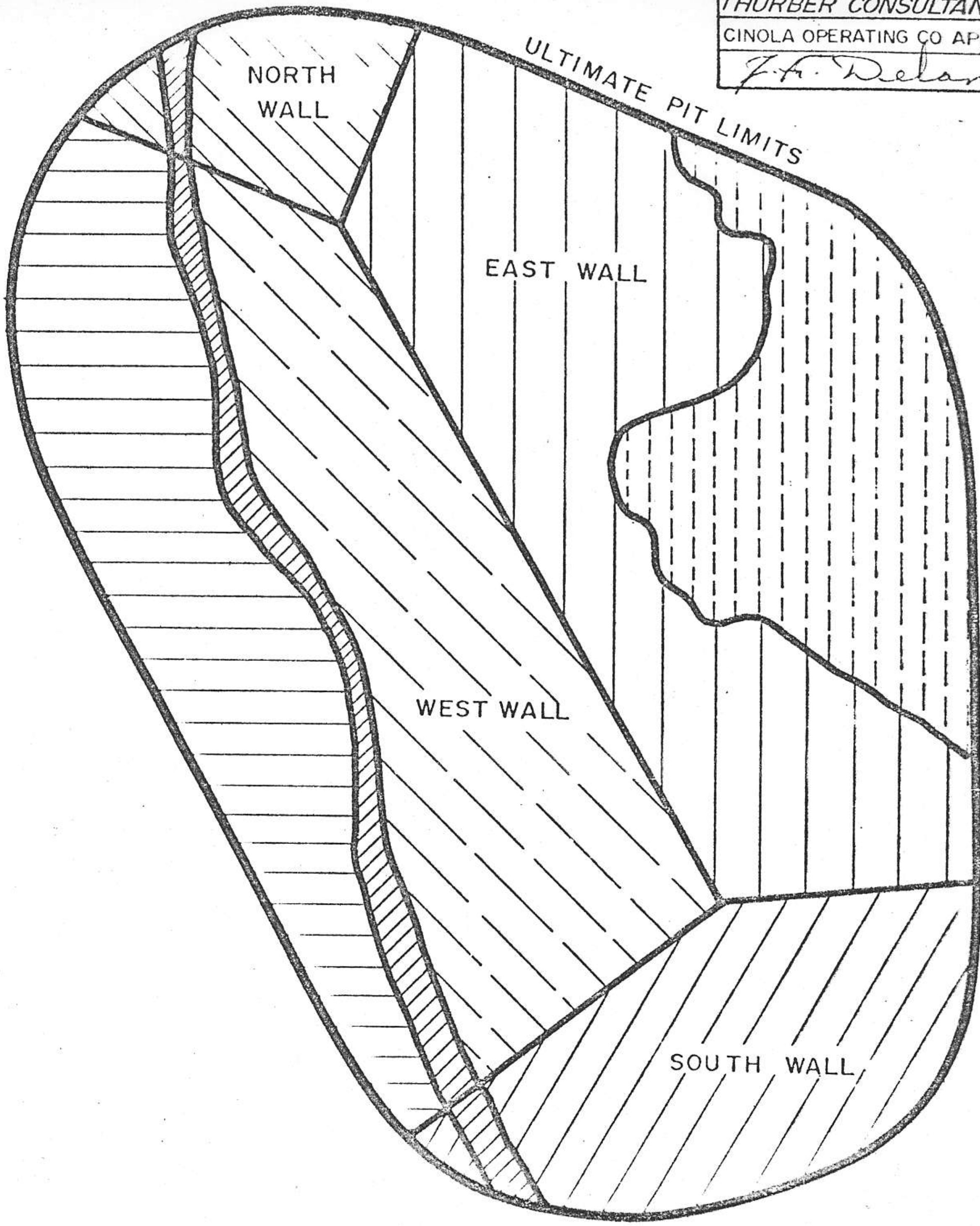

LEGEND

SCALE 1:4000

- INTER-RAMP ANGLE = 55°
- INTER-RAMP ANGLE = 55°
- INTER-RAMP ANGLE = 55°
- INTER-RAMP ANGLE = 55°
- INTER-RAMP ANGLE = 35°
- INTER-RAMP ANGLE = 50°
- SLOPES NOT TO EXCEED 10m IN HEIGHT

FIG. NO. 0-7

RECOMMENDED SLOPE ANGLES	*CINOLA OPERATING COMPANY LTD.*	DRAWN: IK	DATE AUG '82
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		FIGURE 3-7	REV

The footwall fault (Haida shales) (west wall) and the kaolinized Skonun Formation (east wall) can both exhibit soil-like characteristics. Inter-ramp angles in the kaolinized Skonun are designed to ensure overall slope stability. However, from examination of borehole core and field exposures, this material is extremely variable, particularly in its strength properties. Pockets of extremely weak material will be encountered during mining. Therefore, care must be exercised during excavation of the pit slopes.

Bench geometry was evaluated in two stages. First, the size of potential failures that could occur over a range of bench slope angles was examined and berm widths necessary to contain these failures were defined. The second stage involved the estimation of bench backbreak distance that the failures would cause. The recommended berm widths therefore were assessed on the probability of a berm being unable to retain sloughed material due to bench backbreak.

Because bench backbreak significantly influences the inter-ramp angles, steepening of the slopes can be achieved by minimizing blast-induced bench damage. Experimentation with blasting trechniques during the initial years of mining might indicate that blast-induced backbreak is controllable, allowing for steeper inter-ramps than those currently recommended. During this period, the feasibiltiy of double benching should be investigated. Because the critical joint sets have short mean lengths, the impact of backbreak is decreased due to the greater design bench width afforded by 20-m high benches. Operational considerations must be included in determining the economic merit of double benching.

0.4.1.1 Site Investigation

The field investigation consisted of measuring a number of basic engineering parameters both along exploration drill core and at rock exposures. Total core recovery, rock quality designation (RQD), and fracture frequency were recorded in the drill core in addition to routine geological observations. The principal rock exposure was an adit totaling 462 metres in length including crosscuts. This was driven by Cinola to produce pilot plant feed and to provide a cross-section of the deposit for geological mapping and bulk sampling. To obtain data on in situ rock mass characteristics, the adit was mapped in accordance with recommendations of the International Society of Rock Mechanics (1977). Some 575 discontinuities were observed and details on orientation, spacing, length, infill, and surface properties were recorded. Further mapping was carried out at two small quarries adjacent to the proposed mine site.

During the engineering geological logging of the exploration drill core, selected samples were subjected to point load strength testing. In all, 115 tests were conducted. A field evaluation of joint shear strength was undertaken using the method suggested by Barton and Choubey (1979). The method is based on three index properties: joint roughness coefficient, joint wall compressive strength, and residual friction angle. Measurements of those properties were taken at 21 locations within the adit. Where joints were infilled with clay gouge, a pocket penetrometer was used to estimate the undrained shear strength of the gouge.

A number of incompetent rock units were encountered during the investigations. These included the footwall fault (Haida shales), clay-altered Skonun, and joint infill. Samples from each unit were taken and tested in the laboratory for classification purposes and to estimate shear strength parameters.

0.4.1.2 Rock Fabric

The rock fabric consists of geological discontinuities of limited length, too numerous to be mapped individually. At Cinola, joints are the dominant elements of the rock fabric that affect slope stability. Because most the measurements of joints were carried out in the adit, joint length was difficult to estimate due to the limited window of observation (the tunnel width). However, a four-fold classification of fabric origin was developed, and based on actual observation of joint trace length, origin, and prior experience, an order of magnitude for joint length was estimated.

The majority of the joints measured were tight, with no separation between joint walls. However, in the silcified Skonun, individual joints were often found to have clay infill and a wall separation of up to 35 cm.

0.4.1.3 Rock Strength

Failure of rock slopes can occur along pre-defined planes of weakness such as bedding or joints, through intact rock material, or by a combination of both. Therefore, the evaluation of rock strength was divided into three topics: strength along discontinuities, strength of intact rock, and strength of overall rock mass.

Values of these three forms of rock strength were obtained by a combination of field observation and testing, laboratory testing, and from published data. Because of the early stage of the investigation, and the fact that little rock exposure was available, a conservative approach was adopted in assigning strengths to the various rock units.

Based on strength characteristics, two distinct groupings of the various rock units at Cinola were recognized: Group I - strong to very strong rocks whose strength is governed by brittle failure criteria, and Group II - very weak rocks whose strength is governed by soil mass failure criteria. Included in the first group were the silicified Skonun, and the footwall shales and rhyolites. Kaolinized Skonun and material within the footwall fault were placed in the second group. Strength values for Group I rocks were primarily obtained from field testing with the point load apparatus and published data for similar rock types. Values for Group II rocks were obtained from a combination of field and laboratory testing.

In an open pit slope where large-scale or persistent geological discontinuities are not present, an assessment of rock mass strength must be made. This strength depends on the shear and tensile strength of the intact rock, together with the strength of all the inherent planes of weakness within the mass. The potential range of rock mass strength is apparent if one visualizes a massive, unjointed rock such as granite, and compares its strength with that of a heavily-jointed, weak rock such as shale.

Based on published criteria and rock mass classifications it was possible to construct failure envelopes. These are presented in Figure 0-8. They indicate the rock mass strength of each lithology over the range of normal stresses that could be developed in the mine slopes.

0.4.1.4 Seismicity

Earthquake-induced motions produce dynamic loading of pit slopes. The response of a slope to the external forces generated by an earthquake depends largely on the magnitude of ground acceleration, duration of ground motions, rock mass strength, and the slope geometry.

A seismicity analysis based on historical seismicity and regional tectonics was conducted. The results that were pertinent to the pit slope design are summarized as follows:

1. The principal source of earthquake activity is the Queen Charlotte Fault System which is some 60 km to the west of the mine site.
2. The 1-in-10C year earthquake stands to produce a peak acceleration of 28 percent of gravity at the mine site.
3. No recent movement along the trend of the nearby Sandspit Fault System is evident.

Earthquakes are a time-dependent phenomenon. Therefore, a certain probability exists for the occurrence of a specific size of earthquake during a given time period. The expected life of the open pit is about 10 years. Therefore, accelerations of at least 5 percent of gravity can be expected during the life of the mine.

The effect of earthquake-induced accelerations on pit wall stability is not well documented. However, Call (1980) observed that peak acceleration of up to approximately 20 percent of gravity generally does not influence the stability of inter-ramp or overall rock slopes. It is estimated that approximately 78 percent of the earthquakes that might occur during the life of the mine would result in

ROCK MASS DESIGN STRENGTHS

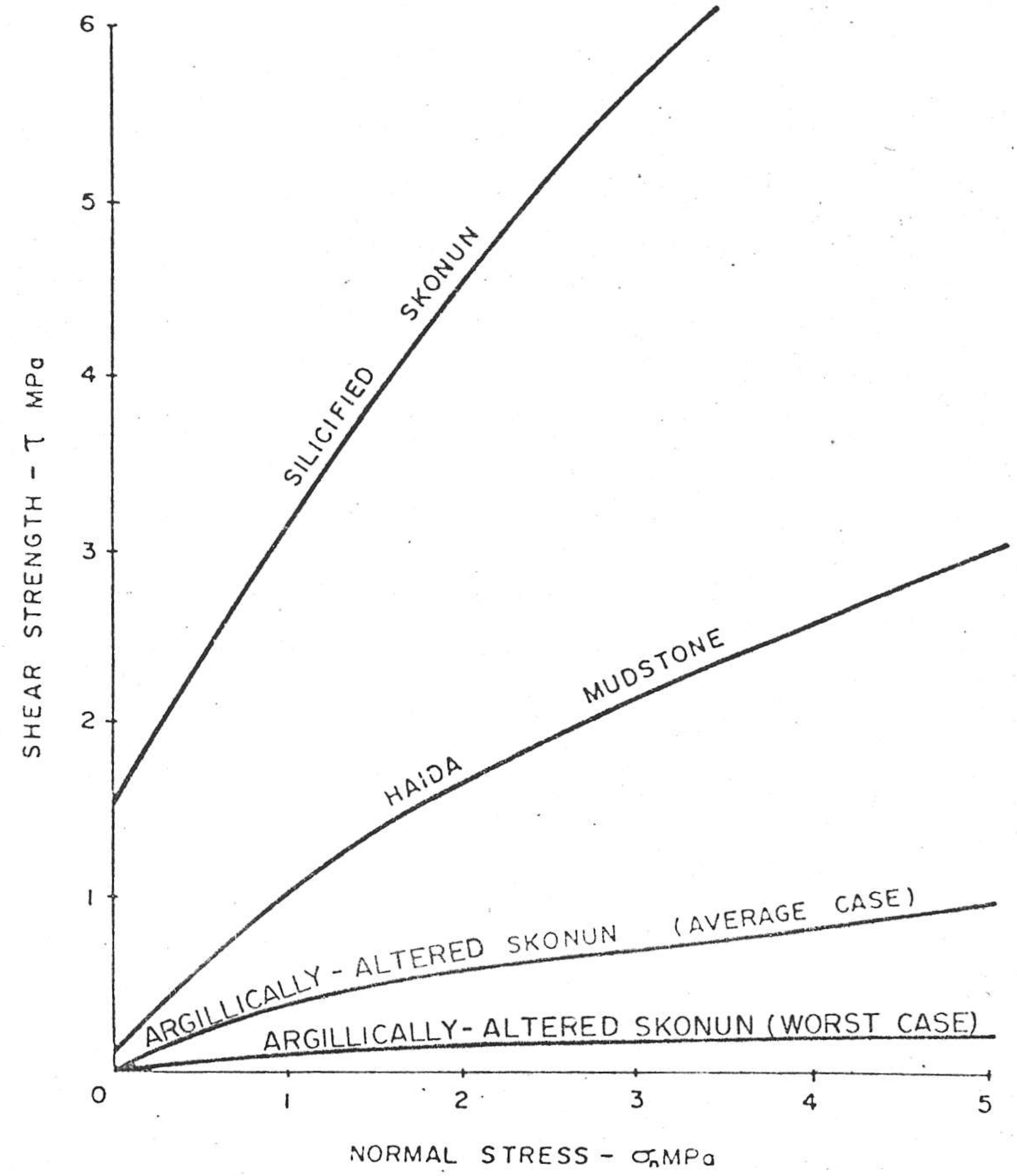

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FIG. NO. 0-8

ROCK MASS DESIGN STRENGTHS

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FIGURE 3-10	REV.

accelerations of less than 20 percent gravity. These would not affect inter-ramp or overall slope stability. Some 12 percent of the earthquakes that might occur would give rise to peak horizontal accelerations of greater than 20 percent gravity. These might result in slope instability. However, due to the high strength of the rock found in the Cinola pit, it is expected that this instability would be minimal, and would be restricted to localized pockets of weak material such as those that occur in the east wall. A more detailed investigation of the effects of seismicity should be carried out during the final pit design studies.

0.4.1.5 Surface Water

Flow management procedures are planned to minimize the impact of the project on the natural ground and surface water regimes. The basic management concept involves the diversion of surface drainage from undisturbed catchment areas, wherever possible, directly to the natural drainage system. Runoff from disturbed areas that is potentially sediment laden, and runoff from undisturbed areas which cannot be diverted, are temporarily retained in sediment ponds to allow clarification and then is discharged to the environment. All sediment pond discharge is monitored to ensure the effluent meets specified quality criteria.

Mining activities are concentrated in the upper portions of the Florence and Barbie Creek drainage basins. The following mine components generate clean surface runoff which is conveyed by diversicn ditches directly to the natural drainage system downstream from the mining activities:

1. The catchment area upslope from the waste rock dump and overburden stockpile
2. The catchment area around the pit
3. The catchment area upslope from the low grade ore stockpile

The following mine components generate runoff which could contain suspended solids:

1. Waste rock dump
2. Overburden stockpile
3. Mill site
4. Crusher area
5. Low grade ore stockpile
6. Open pit during preproduction stripping

Precipitation and seepage accumulating in the pit are collected in excavated sumps and continually pumped out. Much of the sediment settles out in the pit before the water is removed. Most of the pit water is used as makeup water at the mill, with the remainder being discharged to the waste rock dump sediment pond.

The tailings impoundment is almost a closed system. Except for the seepage which occurs through the bottom of the impoundment and the embankment and some surface spillage over the emergency spillways during extreme flood events, all the water that enters the impoundment is either be trapped in the deposited talings, evaporated, or returned to the mill.

It is expected that total mill water requirements can be met by withdrawing water from the pit and from the waste rock sediment ponds. However, during the summer months and during dry years, there is insufficient water available in the pit and the sediment ponds to supply the needs of the mill. This potential deficit is more pronounced in later years when the tailings impoundment seepage and evaporation losses are at a maximum due to the large exposed water surface area. A standby supply of mill water, therefore, is required during these periods. Preliminary indications are that this requirement is as high as 3000 m^3/day averaged over the year and as high as 13,500 m^3/day in any single month. The latter value applies during the final stages of the operation during the dry summer months when minimum flow from the tailings impoundment occurs. Additionally the demand for potable water is currently estimated at 1600 m^3/day.

If possible, groundwater development for the supplemental and potable water supply would be preferred over surface water development due to reduced variations in water quality and quantity. However, because of the apparent lack of a high-yield aquifer in the mine area, surface water development may be required.

Some consideration is given in drawing at least part of the supplementary supply directly from the Yakoun River. The Water Survey of Canada records indicate that the minimum monthly flow-rate during the period 1962-79 on the Yakoun at Port Clement is approximately 120,000 m^3/day. Even allowing for some reduction in this flow-rate by adjusting it to the site location, the impact of pumping some of the total maximum requirements of 15,000 m^3/day would be minimal.

0.4.1.6 Groundwater

The adit was driven about 122 m into the proposed open pit limits. Groundwater flows from the adit since July 1981 were measured between 50 and 100 litres per minute. Furthermore, seven open hole and three sealed piezometers were installed in exploration boreholes. In each unit, observations were made on the position of the water table. Results from falling head tests defined the permeability of the Skonun Formation, and indicated that fault zones are not high-permeability conduits for groundwater flow.

From the above data, the total pumping rate required to dewater the proposed open pit over a specified time period ranges from 3,300 litres per minute to 12,250 litres per minute depending on the total time period over which the mine is dewatered. However, it is anticipated that dewatering volumes probably will not exceed 4,000 litres per minute.

Two principal options exist for pit dewatering. These are sump pumps and perimeter dewatering wells. At this stage of the design, it is anticipated that sump pumps combined with strategically-located, horizontal-toe wells will ensure adequate pit dewatering for both mine operations and pit slope stability.

0.4.2 Mine Design and Equipment Parameters

The approach to planning the open pit initially required a thorough, in-depth analysis of all pertinent aspects of the deposit. The approach was governed by the following mining philosophy:

1. Only proven mining systems were considered. Studies of similar properties and the experience of project engineers were used to the greatest possible extent.
2. Production schedules for ore and waste material were set to provide the most economic conditions based on current costs and gold price.
3. The potential to modify the development of the pit in future years was considered.
4. It was assumed that, during the operating phase, a good grade control program would be in effect to ensure delivery of the required ore grades to the mill.

0.4.2.1 Design Criteria

At the outset of mine design work, a study was initiated to evaluate the impact of varying production rates and cutoff grades on the profitability of the project. As the study was for purposes of comparison only, production rates of 4,500, 9,000, and 13,500 tonnes per day and preliminary data on ore reserves and capital and operating costs were used.

The results from this study conclusively showed that, for the assumed economic parameters (including a $400 US per oz gold price), the highest production rate gave the best rate of return on investment, because the lower production rates were disproportionately burdened by initial capital and higher operating costs. Sensitivity analyses that lowered initial capital by as much as 50 percent for the 4,500 tonne-per-day case indicated that further work based on this production rate was not justified. It was concluded that mine design in the feasibility study would be based on a production rate of 13,500 tpd. Further optimizing studies might be carried out to refine this rate.

The following data detail the criteria upon which work in this report is based:

General

1. Preproduction period - 2 years (Years -2 and -1)
2. Estimated mine life at annual rated capacity - 7 plus years (Years 1 through 8)
3. Ore production during start-up - 400,000 tonnes (Year -1)
4. Full production rate - 4.725 million tonnes of ore per year

Mining Method

The mining method is characterized by the following operational plans:

1. Conventional open pit mining methods are used in both ore and waste zones.
2. Waste material excavation:
   a. Waste material is drilled and blasted using medium-sized diesel/hydraulic-powered rotary blast hole drills and conventional blasting procedures.
   b. Off-highway trucks, loaded by wheeled loaders, haul the waste to a waste dump southwest of the open pit.
   c. Unconsolidated overburden material is stripped using twin powered scrapers and stockpiled. This material is then used to reclaim the waste dumps at a later date.

Ore excavation:

   a. Ore and any internal waste are drilled and blasted using medium-sized diesel/hydraulic powered rotary drills and conventional blasting methods.
   b. Off-highway trucks, loaded by wheeled loaders, haul the ore to a gyratory crusher, located on the north-eastern pit rim. Waste is hauled to the waste dump southwest of the pit.
   c. Low-grade ore is stockpiled until its processing can be justified.

Operating Parameters

Bench height	10 m
Bench face slope angle	80 °
Ultimate pit slopes (inter-ramp slope angles)	
North wall	55 °
South wall	55 °
East wall	35 °
West wall	50 °

Haul Roads

Width overall	25 m
Running surface	20 m
Safety berm and table drain	5 m
Road grade	8 percent
Ramp grade	10 percent

Shift Scheduling

Mine shift scheduling is based upon the following data:

1.	Statutory holidays per year	10
2.	Percentage vacation, sickness, and absenteeism (VSA)	8 percent
3.	Work week (hours)	40
4.	Annual work schedule - 5 days per week	
	Years 1 through 4	3 shifts per day
	Years 5 through 8	2 shifts per day
5.	Scheduled hours per shift	8
6.	Probable maximum annual mine production work shifts	750

0.4.2.2 Equipment Parameters

The following parameters are considered to be practical for the Cinola Mine, considering the prevailing conditions at the Queen Charlotte Islands.

1. All major equipment is assigned a realistic useful life which forms the basis for scheduled equipment replacement.
2. Percentages of utilization for equipment are defined to specify the maximum number of shifts per year that any unit of equipment can reasonably be expected to operate.
3. Effective equipment operating time per shift is assumed to be 350 minutes.
4. Equipment productivities used to estimate operating costs are listed below:

<u>Rotary drills - silicified Skonun</u>	
Hole diameter, mm	229
Drilling rate per shift, m	85
Maximum scheduled shifts per drill year	503
<u>Rotary drills - Haida shale</u>	
Hole diameter, mm	229
Drilling rate per shift, m	126
Maximum scheduled shifts per drill year	503
<u>Wheeled loaders - silicified Skonun/Haida shale</u>	
Bucket capacity, m^3	10
Production dry tonnes per shift	8,100
Maximum scheduled shifts per loader	562
<u>Wheeled loaders - clay altered sediments</u>	
Bucket capacity, m^3	10
Production dry tonnes per shift	6,600
Maximum scheduled shifts per loader year	562
<u>Blasting considerations</u>	
Silicified Skonun --	
Powder factor, kg/tonne	0.27
Explosive loading (explosive type as a percent of total hole charge)	
- Slurry	30
- ANFO	70

Haida shale --	
Powder factor, kg/tonne	0.18
Explosive loading (sxplosive type as a percent of total hole charge)	
- ANFO	100

An outline of mine equipment units in use each year of operation is given in Table 0-5.

0.4.3 Organization and Manning

An organization chart is included in Figure 0-9. The mine operations department is organized into two basic functional sections; production and engineering, each run by a general mine foreman and a chief mining engineer, and who report directly to the mine superintendent. Mine maintenance is overseen by a master mechanic who reports directly to the project's maintenance superintendent. The master mechanic maintains a functional relationship with the mine superintendent and oversees the contractor personnel responsible for maintaining the mine equipment.

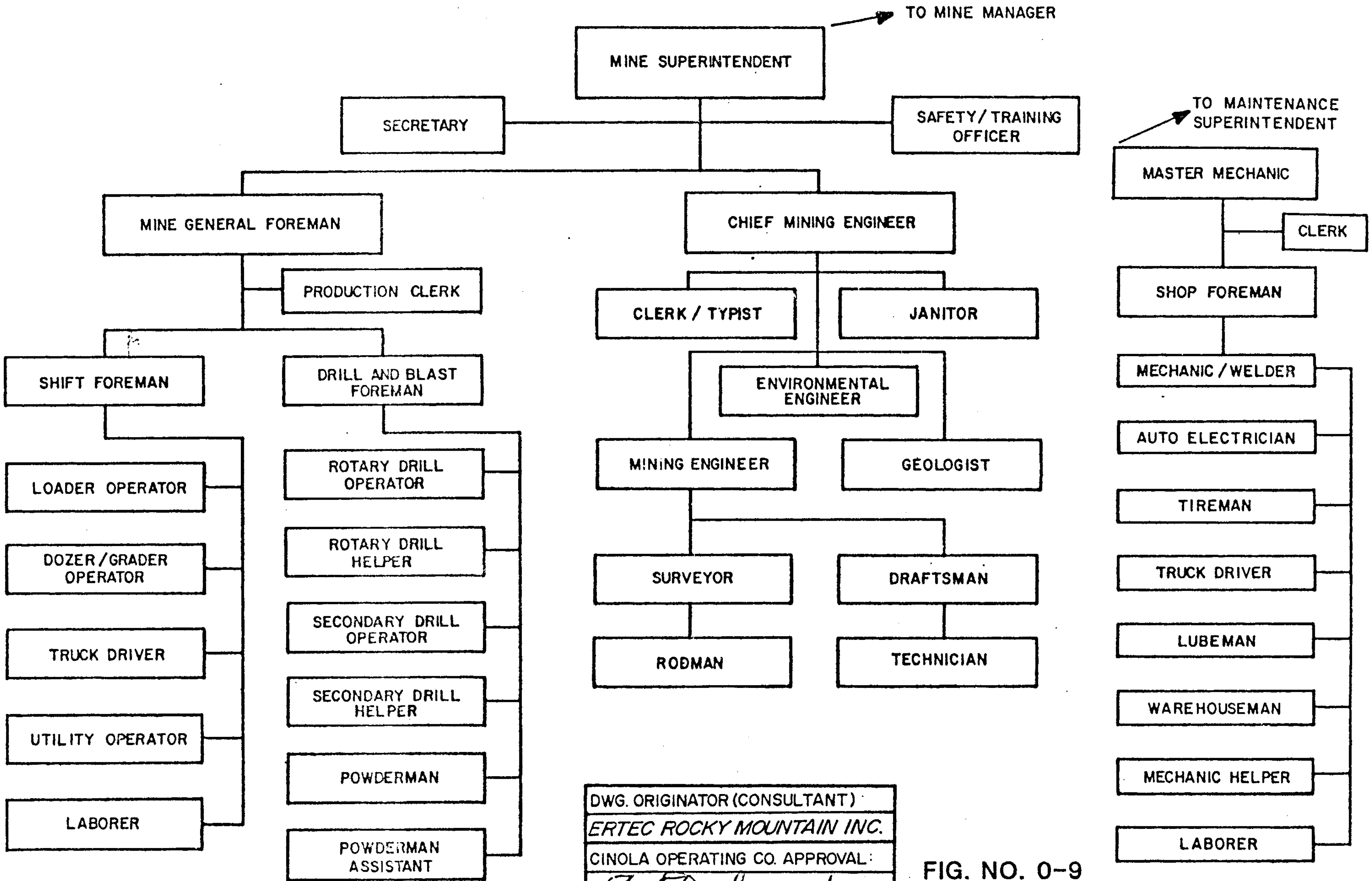
The manpower requirements are defined based on the necessity to operate and maintain the specified mine equipment, and to provide the necessary technical support. From year 1 to year 4 the mine production is planned on a three shifts (8-hour) per day, five day per week basis. Ore production is scheduled during that period to allow continuous, seven-day operation of the mill. For the remainder of the mine life, as designed, a two shift per day, five day a week operation is all that is required to maintain scheduled mill feed tonnage. If necessary to maintain equipment availability levels, preventive and corrective maintenance can also be scheduled during the down shifts on the weekend. An allowance of overtime payments has been included in the hourly labor costs. An allowance of eight percent has been made for vacation, sickness, and absenteeism (VSA) for the hourly paid personnel.

A summary of the manpower requirements for these personnel for year 1 is detailed in Table 0-6.

TABLE 0-5

MINE EQUIPMENT UNITS IN USE BY YEAR OF OPERATION

Description	-1	1	2	3	4	5	6	7	8
Rotary blasthole drill-229 mm	2	3	3	3	2	2	2	2	2
Secondary drill - 100 mm	1	1	1	1	1	1	1	1	1
ANFO truck - 5 tonne	1	1	1	1	1	1	1	1	1
Explosives truck - general purpose - 10,000 GVW	1	1	1	1	1	1	1	1	1
Blasthole dewatering truck	1	1	1	1	1	1	1	1	1
Wheeled loader - 10 m^3	2	4	4	4	4	4	3	3	2
Haul trucks - 77 tonne	5	16	16	15	15	16	11	11	7
Tracked dozer - 230 W	2	2	2	2	2	2	2	2	2
Wheeled dozer - 230 W	1	1	1	1	1	1	1	1	1
Motorgrader - 130 W	1	1	1	1	1	1	1	1	1
Water truck - 28,000 l	1	1	1	1	1	1	1	1	1
Vibrating roller - self propelled	1	1	1	1	1	1	1	1	1
Grid roller	1	1	1	1	1	1	1	1	1
Scraper - twin powered - 168/168/W	3	3	3	3	3	3	3	3	3
Fuel/lube truck	1	1	1	1	1	1	1	1	1
Tire truck	1	1	1	1	1	1	1	1	1
Mechanics field service truck - 10,000 GVW	1	1	1	1	1	1	1	1	1
Mechanics field service truck - 24,000 GVW	1	1	1	1	1	1	1	1	1
Lowboy - 60 tonne	1	1	1	1	1	1	1	1	1
Forklift - 5 tonne	1	1	1	1	1	1	1	1	1
Mobile light towers - 5kW	4	4	4	4	4	4	4	4	4
Crane - rough terrain - 22 tonne	1	1	1	1	1	1	1	1	1
Backhoe/seeder	1	1	1	1	1	1	1	1	1
Backhoe-tracked	1	1	1	1	1	1	1	1	1
Air compressor - portable 100 cfm	1	1	1	1	1	1	1	1	1
Welder - portable - 400 amp.	1	1	1	1	1	1	1	1	1
Pickups - 1/2 ton	9	9	9	9	9	8	8	8	8
Blazer	2	2	2	2	2	2	2	2	2
Ambulance	1	1	1	1	1	1	1	1	1
Pumps - pipe (lot)	1	1	1	1	1	1	1	1	1
Radios - mobile	26	28	28	28	28	28	27	27	26
Radios - fixed base	4	4	4	4	4	4	4	4	4



DWG. ORIGINATOR (CONSULTANT)
ERTEC ROCKY MOUNTAIN INC.
CINOLA OPERATING CO. APPROVAL:

FIG. NO. 0-9

MINE OPERATIONS ORGANIZATION CHART

CINOLA OPERATING COMPANY LTD.

DRAWN: P.D.C.	DATE 6/9/82
CHECKED I.S.	APPROVED
FIG. NO. 3-17	REV.

No VSA allowance is included for staff personnel. Sufficient flexibility is built into the organization to allow the duties of any staff member to be covered when absent under normal circumstances from the mine. Details of the requirements for salaried personnel for year 1 are shown in Table 0-6.

0.4.4 Minable Ore Reserves for Ultimate Pit Design

The three-dimensional block model derived by the inverse distance squared weighting technique (refer to Section 0.3) was used for the mining evaluation discussed in this section. For each of the 10 m x 20 m x 20 m blocks, gold grade and rock type were recorded. Four different rock types were identified - unconsolidated overburden, clay altered sediments, Haida shale, and silicified Skonun. The contacts of these rock types were derived from geologic level plans developed by the Cinola project geologist.

0.4.4.1 Ultimate Pit Generation

All preliminary computer planning used the MINEPAK system, a proprietary open pit computerized mine planning program. This program used the moving cone concept to "mine" the block model in accordance with economic and operations input criteria. Cones consisting of several 10 m x 20 m x 20 m blocks were superimposed within the block model. The net value (recoverable metal value minus the direct operating costs of mining, processing, and administration) was calculated for each cone. Cones with a positive net value were removed, while those with negative net values were left in place.

The computer-generated, 1:1000-scale topographical plot formed the basis for the manual refinement necessary to produce an operationally viable ultimate pit plan depicting the necessary haul roads and benches. Mine planning input was required to define haul road and primary crusher locations before final refinement could be completed. Every effort was made to minimize the additional stripping required to develop the haul road system for the mine. In the north, west, and south pit walls, consisting primarily of competent silicified Skonun or Haida shale, double

TABLE 0-6

HOURLY PAID MANPOWER REQUIREMENTS - YEAR 1

Section	Personnel per Shift A	B	C	Total Personnel
Operations	34	29	27	90
Maintenance	23	19	9	51
Training	5	0	0	5
VSA Allowance	5	4	3	12
Total	67	52	39	158

SALARIED MANPOWER REQUIREMENTS - YEAR 1

Position	Function	Number
Mine Superintendent	Operations Management	1
Safety/Training Officer	Operations Management	1
Secretary	Operations Management	1
General Mine Foreman	Production	1
Drill and Blast Foreman	Production	1
Shift Foreman	Production	3
Production Clerk	Production	1
Master Mechanic	Maintenance	1
Shop Foreman	Maintenance	2
Maintenance Clerk	Maintenance	1
Chief Mining Engineer	Engineering	1
Mining Engineer	Engineering	1
Geologist	Engineering	1
Environmental Engineer	Engineering	1
Surveyor	Engineering	1
Rodman	Engineering	1
Draftsman	Engineering	2
Technican	Engineering	1
Clerk/Typist	Engineering	1
Janitor	Engineering	1
Total Salaried		23

benching was possible at the pit limit. In all benches where the footwall fault was mined out, care was taken to ensure that the pit limit extended 20 metres beyond the fault to the west into the competent Haida shale.

In the less competent clay-altered sediment, comprising the northeast and east sides of the pit, double benching was not possible. The overall pit slope in this area averaged approximately 34.6 degrees.

Cross-sections were drawn through the north, south, east, and west walls of the as-designed ultimate pit. The refined pit was digitized and the final minable reserves calculated. These are summarized in Table 0-7. Detailed minable reserves by bench are included in Appendix B of the main report.

Table 0-8 details the reserves within the designed ultimate pit at varying cutoff grades.

A perspective view of the property at the end of mining is shown in Figure 0-10. The ultimate pit plan is shown on Drawing 30 1 07/0.

0.4.4.2 Grade Adjustment for Selective Mining Unit

As a check against the validity of the minable tonnes and grade reported against the $1/d^2$ model, a geostatistical model was produced. The digitized, manually refined ultimate pit plan was superimposed on the geostatistical model and a new estimate of minable reserves was reported. These are detailed below in Table 0-9.

This latter minable reserve shows an 11 percent increase in minable tonnes to 38.0 million and a 7 percent decrease in gold grade to 0.056 oz/tonne. Overall, the contained ounces of recoverable gold increased by 3 percent to 2.13 million ounces. Detailed bench by bench minable reserves, reported against the geostatistical model, are included in Appendix B.3.

TABLE 0-7

MINABLE RESERVES - REFINED ULTIMATE PIT - $1/d^2$ MODEL*

(Based on 10 m x 20 m x 20 m blocks at 0.033 ounce/tonne cutoff)

		Tonnes Waste x 10^6					
Tonnes Ore ($x10^6$)	Gold Grade oz/t	Silicified Skonun	Haida Shale	Clay Altered Sediments	Unconsol. Overburden	Total Waste	Waste/ Ore Ratio
34.32	0.060	22.49	7.91	13.72	3.03	47.15	1.37

*Note: This reserve definition is used for all project economic and financial analysis. However, 6.3 x 10^6 tonnes of low-grade ore are treated as waste for cost purposes, thus increasing total waste to 53.445 x 10^6 tonnes and waste/ore ratio to 1.56.

TABLE 0-8

MINABLE RESERVES REPORTED AT VARYING CUTOFF GRADES

Based on 10 m x 20 m x 20 m blocks and $1/d^2$ block model)

Cutoff Grade oz/tonne	Ore kt	Av. Grade oz/tonne
0.028	44,293	0.053
0.033	34,316	0.060
0.040	22,413	0.072
0.050	15,932	0.084
0.060	13,250	0.090
0.070	10,060	0.097

TABLE 0-9

MINABLE RESERVES - REFINED ULTIMATE PIT - GEOSTATISTICAL MODEL

(Based on 10 m x 20 m x 20 m blocks at 0.033 ounce/tonne cutoff)

		Tonnes Waste x 10^6					
Tonnes Ore ($x10^6$)	Gold Grade oz/t	Silicified Skonun	Haida Shale	Clay Altered Sediments	Unconsol. Overburden	Total Waste	Waste/ Ore Ratio
38.00	0.056	18.80	7.91	13.72	3.03	43.46	1.14

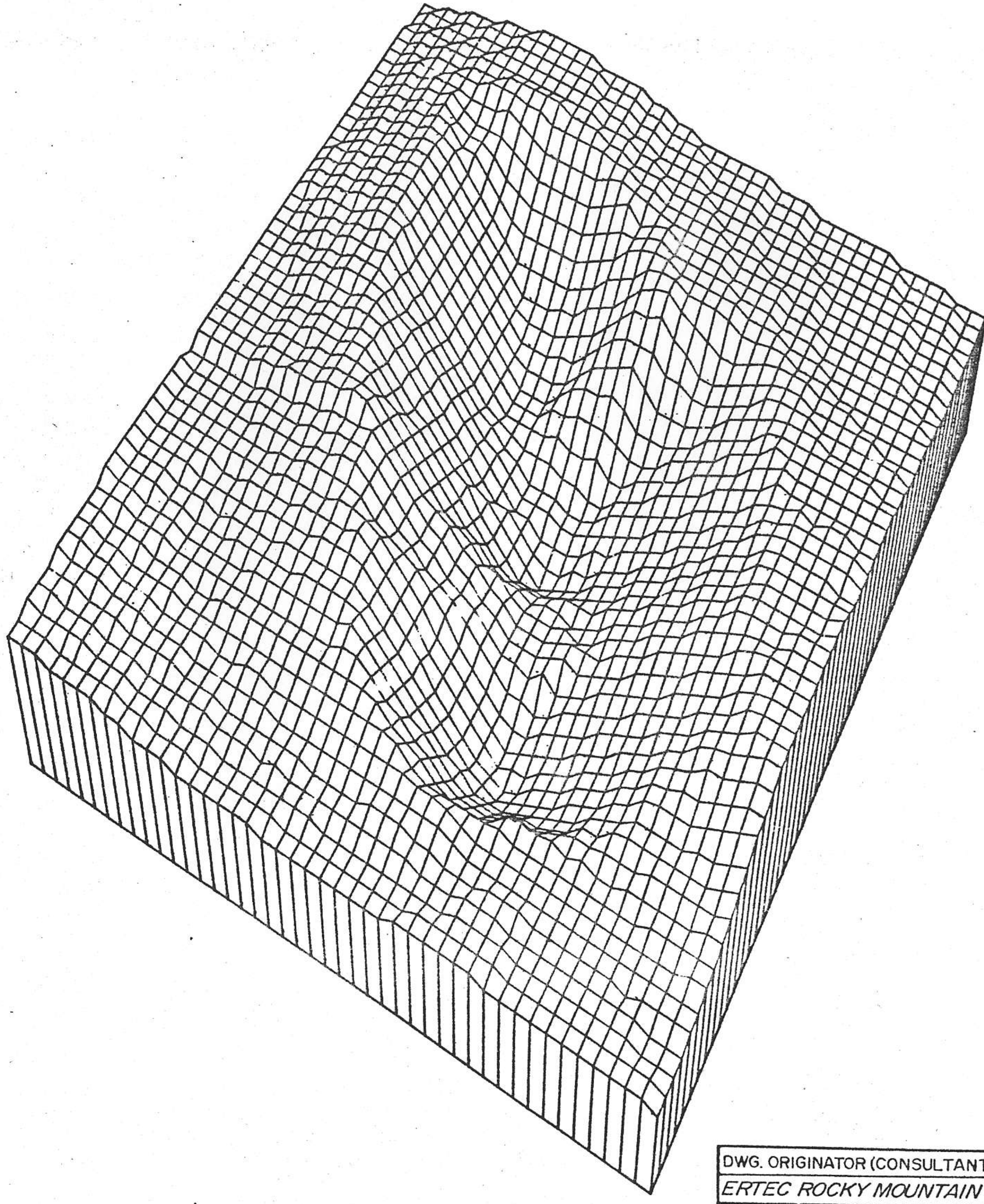


FIG. NO. 0-10

DWG. ORIGINATOR (CONSULTANT)	
ERTEC ROCKY MOUNTAIN INC.	
CINOLA OPERATING CO. APPROVAL:	
J. F. Delaney	

MESH PLOT — END OF MINING

CINOLA OPERATING COMPANY LTD.

DRAWN:	DATE: 6/15/82
CHECKED I.S.	APPROVED JFD
FIG. NO 3-6	REV.

On the basis of all work done to date, Cinola project engineers and geologists believe that the recoverable tonnes and grade reported in this study are conservative. Good operating procedures combined with a well-defined grade control program should ensure higher mill feed grades and an increase in the ore tonnes.

0.4.5 Mine Operations

In recommending the various requirements for operating equipment and outlining the method of operation, the following philosophy was adopted by the Cinola project engineers for this study:

1. Only equipment with a proven operating record and current availability on a competitive basis was considered.
2. Mining methods were considered which use the latest mining technology accepted by the industry that is capable of successful implementation on the Queen Charlotte Islands.
3. Wherever possible, the highest degree of operating flexibility was maintained. This influenced both equipment selection and the recommended mining method.
4. In recognition of the importance of training, Cinola project engineers selected an operating plan that facilitates the successful implementation of the project, while concurrently conducting the required, on-the-job equipment operator training programs.

Although the Cinola property can be classified as a large, low-grade gold deposit suitable for open pit mining, its daily material movement requirements are not large compared to many other open pit operations in British Columbia.

Nevertheless, due to the low grade of the gold mineralization, it is essential that the mine equipment be capable of maintaining the required productivity throughout its expected life while maintaining the required degree of operating flexibility and keeping initial capital and direct operating costs to a minimum. This equipment selection criteria must be balanced against the particular operations constraints peculiar to the Cinola operations, such as terrain, climatic conditions, location, and equipment operator training requirements. Given that the project is operated within accepted mining standards, the Cinola project engineers consider that the equipment selected is capable of meeting the target productivity requirements.

0.4.5.1 Preproduction Pit

In the design of the preproduction pit, several criteria were met:

1. At the end of preproduction, a minimum of three months of mill grade ore must be exposed.
2. During the last two months of the preproduction period, the equivalent of one month's ore at full production, about 400,000 tonnes, can be delivered to the mill.
3. About one million m^3 of silicified Skonun or other suitable material can be delivered for tailings dam construction.
4. The unconsolidated overburden in the area of the mine workings can be temporarily stockpiled until it can be reclaimed and laid as cover over the waste dumps.

In this period, 400,000 tonnes of ore at a grade of 0.089 oz/tonne are delivered to the primary crusher. A total of 598,000 tonnes of low-grade ore (0.036 oz/tonne) are temporarily stockpiled for processing at a later date. Silicified Skonun material totaling 730,000 m^3 is hauled to the tailings dam location. At the end of preproduction, approximately 1.47 million tonnes of ore at a grade of 0.076 oz/tonne are exposed in the pit.

0.4.5.2 Production Scheduling

The MINEPAK scheduling routine was used to provide a preliminary production schedule to conform with the specified mining period. This computer scheduling routine attempts to maximize the net value of ore mined and processed per scheduled period, in accordance with the input gold price and operating costs.

Every attempt was made to hold the grade as high as possible during the initial years of mining in order to maximize the rate of return on investment. MINEPAK's blending option was used in this case to force the mining of the higher grade blocks.

Computer topography for each mining period was produced as an aid in the manual refinement of these plans. This refinement was required for the following reasons:

1. To ensure that each plan is operationally viable.
2. To ensure that the waste rock requirements for the tailings dam construction are met.
3. To produce an acceptable estimate of the requirements per period for the mine equipment fleet.

Drawings 30 1 01/0 through 30 1 06/0 detail the mine plans from the end of preproduction through the fifth year of full production. The annual mine production material movement schedule is detailed in Table 0-10. In attempting to hold the grade as high as possible in the early years, low-grade ore above the ultimate design cutoff grade of 0.033 oz/tonne is assumed to be hauled to a temporary low-grade stockpile to the east of the pit limit. It is then assumed that this ore is reclaimed beginning in year 3 for processing in the mill. As a result of this double handling of low-grade material, which is classified as waste during the years when it is stockpiled, the life-of-mine stripping ratio increases from 1.37 to 1.56.

Note that all economic and financial analyses in this report use this schedule of mine production as the Base Case.

0.4.5.3 Tailings Dam Material

In an attempt to even out the haul truck requirements, more material is assumed to be hauled to the tailings dam site in the early years when the trucks do not have as great an adverse haul from the pit. Furthermore, in Year 5 the overall material movement requirements are dipped such that a two shift, five days per week operation is sufficient to meet the mill feed and waste production requirements.

TABLE 0-10

ANNUAL MINE PRODUCTION SCHEDULE

MATERIAL	YEAR -1	1	2	3	4	5	6	7	8	TOTAL
Mill Ore										
Mine, kt	400	4,725	4,725	4,224	3,077	3,197	3,384	3,764	523	28,019
Grade, oz/t	0.0889	0.0782	0.0717	0.0595	0.0605	0.0575	0.0539	0.0613	0.0569	0.0648
Stockpile, kt	0	0	0	501	1,648	1,528	1,341	961	318	6,297
Grade, oz/t	0	0	0	0.0387	0.0387	0.0387	0.0387	0.0387	0.0387	0.0387
Total mill feed, kt	400	4725	4,725	4,725	4,725	4,725	4,725	4,725	841	34,316
grade, oz/t	0.0889	0.0782	0.0717	0.0513	0.0529	0.0514	0.0496	0.0567	0.0500	0.060
Waste Material										
Low grade ore, kt (0.033 oz/t)	598	4,507	1,192	--	--	--	--	--	--	6,297
Grade, oz/t	0.0358	0.0389	0.0398	--	--	--	--	--	--	
Waste (0.033 oz/t)										
Overburden, kt	318	1,569	559	426	158	1	0	0	0	3,031
Clay altered sed., kt	0	1,429	4,302	2,169	2,930	2,372	350	162	4	13,718
Haida Shale, kt	216	1,337	1,052	1,751	1,123	781	606	958	87	7,911
Silicified Skonun, kt	3,289	4,665	3,875	4,154	1,963	2,094	1,263	1,086	97	22,487
Total waste, kt	4,421	13,508	10,980	8,500	6,174	5,248	2,219	2,206	189	53,445
Stripping ratio	11.05:1	2.86:1	2.32:1	1.80:1	1.31:1	1.11:1	0.47:1	0.47:1	0.22:1	1.56:1

Table 0-11 details the material hauled by year for the construction of the tailings dam.

TABLE 0-11

MINE WASTE ROCK HAULED FOR TAILINGS DAM CONSTRUCTION

Year	Cubic Metres (million)
-1	0.73
1	1.91
2	1.85
3	1.29
4	0.09
Total	5.87

0.4.5.4 Waste Dumps and Ore Stockpiles

Drawings 30 1 08/0 through 30 1 14/0 detail the waste dumps and the ore stockpiles from the end of preproduction through year 5 and the end of mining. Initially at the end of preproduction, unconsolidated overburden is stripped and temporarily stockpiled to the west of the pit. This material is fully reclaimed at the end of the mine life and covers the waste dump to provide a base for revegetation.

The main waste dump is located to the southwest of the pit. During the preproduction period, a ramp is initially opened to allow waste to be hauled directly to a high level dump at the 170 m elevation. In year 1, a new dump is developed below the existing 170 m dump at the 130 m level. At this time, waste is hauled from a southern pit exit. In year 3, this dump is raised to the 160 metre level, and remains at this elevation for the remaining mine life. Total material contained in this dump is approximately 31.75 million tonnes. The dump face at the end of mining is terraced to give a final slope of approximately 26 degrees, which provides an acceptable face for establishing vegetation.

Two ore stockpiles are developed east of the pit. The low grade stockpile mentioned previously contains 6.29 million tonnes at the end of year 2 before it is gradually reclaimed for processing. A smaller ore stockpile is established adjacent to the primary crusher. This is provided only to allow the ore trucks to dump their load if the crusher shuts down. This stockpile is be reclaimed as necessary by wheeled loaders and dumped directly into the primary.

Details of the waste material movement by classification are included in Table 0-10.

0.4.5.5 Mine Drainage

Estimates were made of the quantities of water that must be pumped from the mine to maintain it in a satisfactory operating condition. These estimates, which are detailed in Table 0-12 are for average conditions. Pumping systems are not designed to handle peak water inflows that might occur during a heavy rainstorm. It is most likely that during such periods, the mine would shut down until weather conditions allowed the resumption of normal mining operations. Operations would also temporarily cease on the lower bench until the pumping station was able to catch up with the sudden inflow of water and return to steady-state conditions.

The mine planning information was analyzed to develop locations for the pit water discharge lines. A permanently fixed discharge line is positioned near the ultimate pit rim to run southeast 600 m to the pit water storage pond. From this ultimate pit rim point, temporary lines extend into the pit bottom area. The existing adit into the ore zone is scheduled to be intersected by surface excavation at the end of year 2. The adit then serves as a subterranean passage way for the discharge line until year 6 of operation. As a result, some 30 m of elevation head are avoided during this period.

As can be seen from Table 0-12, pumping requirements vary from an initial 2,268 m^3 per day to 5,928 m^3 per day in the last year of mining.

TABLE 0-12

PIT DEWATERING STATISTICS

Year	Area Within Pit Perimeter (m^2)	Maximum Pit Depth (m)	Pumping Requirements Recharge	Leakage (m^3/day)	Total	(Gals/Min)
-1	54,531	20	2,268	0	2,268	416
1	115,750	60	2,826	566	3,392	622
2	150,600	90	3,113	524	3,637	667
3	161,800	130	3,202	650	3,852	707
4	205,800	140	3,542	393	3,935	722
5	248,700	170	3,860	792	4,652	853
6	274,100	170	4,044	161	4,205	771
7	332,900	220	4,458	1,470	5,928	1,088
8	332,900	220	4,458	1,470	5,928	1,088

Appendix C shows the year-end pumping characteristics based on the discharge line layout. The longest total length of dewatering pipe, nearly 1,200 metres, is required in year -1, whereas both the largest adverse pumping head and greatest expected flow rate are encountered in the last year of mining (115 metres and 5,928 m^3 per day, respectively).

As far as practical, surface runoff will be prevented from entering the pit by the use of surface drainage ditches. These will control the runoff and divert it around the active mine workings to the sediment dams.

0.5 MILL DESIGN

0.5 Mill Design

The combination of various unit processes selected for use in the metallurgical processing facility was developed to produce gold in the most economical, cost effective manner while giving full consideration to protection of the unique environment of the Queen Charlotte Islands. Numerous metallurgical processes were considered and tested during the course of development and the selected process scheme, presented herein, satisfies these objectives.

The pilot mill on the Queen Charlotte Islands processed over 5,000 tonnes of ore from the deposit. The test results from the pilot mill program were important in the development of appropriate flotation processes and the evaluation of the resultant flotation tailings and water quality characteristics. In addition to the pilot mill program, numerous bench scale tests were conducted on different oxidizing, leaching and gold recovery methods.

Resultant data from these tests were evaluated technically, economically and environmentally to determine the final process flowsheet and the general criteria of design for the metallurgical processing facility:

Nominal Process Capacity: The facility must process 4,725,000 tonnes per year of ore operating 350 days annually -- an average nominal feed rate of 13,500 tpd.

Design Capacity: A facility availability factor of 90 percent was assumed, thus requiring a maximum design feed rate of 15,000 tpd.

Crushing and Grinding: Ore is truck-hauled from the mine to the primary crushing station at a nominal top size of 107 cm.

Conventional crushing and grinding systems are utilized due to concerns associated with autogenous or semi-autogenous systems. Primary and secondary crushing are open circuit, and tertiary crushing is closed circuit. Rod and ball mill grinding utilize twin stream circuits.

Flotation: The twin stream flotation circuit utilizes rougher-scavenger flotation systems and produces a dewatered bulk sulphide concentrate. Thickened flotation tailings report to a tailings sump.

Oxidation: A single-stage, fluid-bed roaster is used to oxidize the concentrate.

Gas Treatment: Offgas from the roaster containing high concentrations of sulphur dioxide is treated in a contact-type acid plant. Commercial grade sulphuric acid is produced for sale. The neutralized weak acid pulp waste stream reports to the tailing sump to be mixed with flotation tailings.

Leaching and Residue Treatment: Gold is leached from the oxidized concentrate (calcine) using sodium cyanide solution. The gold-bearing solution (pregnant solution) is separated from the leached calcines using counter-current decantation (CCD) thickeners.

Precipitation and Smelting: The pregnant solution is clarified and deaerated using the Merrill-Crowe process. Gold is precipitated by the addition of zinc dust and the precipitate is dewatered utilizing a filter process. The gold-free solution (barren solution) from the filters is recycled back to the leach and CCD circuits. The gold-bearing precipitate is smelted and the gold cast into bars.

Cyanide Removal: The leached calcine pulp containing some barren solution is treated by the addition of sulphur dioxide and air to reduce the total cyanide concentration to acceptable levels. This treated pulp reports to the tailings sump and is mixed with the flotation tailings and the neutralized weak acid pulp.

Process Water Circuit: Major process water additions are required in the grinding, flotation, roaster, acid plant, leaching and CCD circuits and in the cyanide removal circuit. Water not lost to evaporation or consumed in the process (i.e. the production of sulphuric acid) ultimately reports with the flotation tailings, the neutralized weak-acid pulp, and the treated leached-calcine pulp. These three discharges are combined at the tailings sump and pumped to a tailings impoundment area in which the solids are retained. The quality of the tailings water meets or exceeds the Government of British Columbia Ministry of Environment (MOE) pollution control objectives. Tailings water is pumped from the impoundment area and is recycled to the metallurgical process facility.

<u>Facility operating Schedule:</u>

Primary crushing	3 shifts/d, 5d/week
Secondary/tertiary crushing	2 shifts/d, 7 d/week
Gold smelting	1 shift/d 5 d/week
Balance of facility	3 shifts/d, 7 d/week

0.5.1 Facilities Description

The general arrangement of the metallurgical processing complex is shown on Drawings 40 1 01 and 40 1 02 in the main report. This arrangement provides for an orderly flow of solids and liquids through the complex from the primary crusher station to the gold smelting area. Buildings are provided for the crushing, grinding, flotation, filtering, roasting, acid plant and gold recovery sections of the complex. Service roads are provided for access to all facilities in the complex.

The metallurgical processing complex is divided into 8 areas for discussion. These areas are:

1. Crushing and Storage
2. Grinding
3. Flotation, Dewatering and Tailings Disposal
4. Roasting
5. Acid Plant
6. Leaching and Decantation
7. Precipitation and Smelting
8. Cyanide Removal

Table 0-13, Process Equipment Unit List, lists the major equipment required for the facility. This table can be found at the end of Section 0.5. The following sections describing the 8 areas include unit numbers, i.e. (1-1) from the process equipment unit list (Table 0-13) for reference.

0.5.1.1 Crushing and Storage

Design parameters developed for the crushing circuit are as follows:

1.	Ore bulk density	1,600 kg/m^3
2.	Abrasion index	0.7
3.	Screening efficiency	95 percent
4.	Primary crusher product	15 percent minus 2 cm
5.	Secondary crusher product	30 percent minus 2 cm
6.	Tertiary crusher product	80 percent minus 2 cm
7.	Tertiary crusher recirculating load	30 percent

Figure 0-11 and Drawing 40 1 03 show the flowscheme for ore crushing and storage with nominal material and water flows. Drawings 40 1 07, 40 1 08, 40 1 09, and 40 1 10 show the general arrangement of the crushing and storage facilities.

The ore, nominally minus 107 cm, is trucked to the primary crusher station. The 42 x 65 gyratory crusher (1-1) reduces 787.5 tph of ore to a nominal minus 28 cm. A vibratory pan feeder (1-3) discharges the gyratory crusher product to the coarse-ore stockpile conveyor (1-4).

Coarse ore is reclaimed from the conical stockpile with 8 vibrating pan feeders (1-8) at a nominal rate of 844 tph. A coarse-ore surge bin ahead of the secondary/tertiary crushing section is provided as a means to more closely regulate the coarse-ore feed to the subsequent screening and crushing facilities. This is necessary to maintain good screening efficiencies.

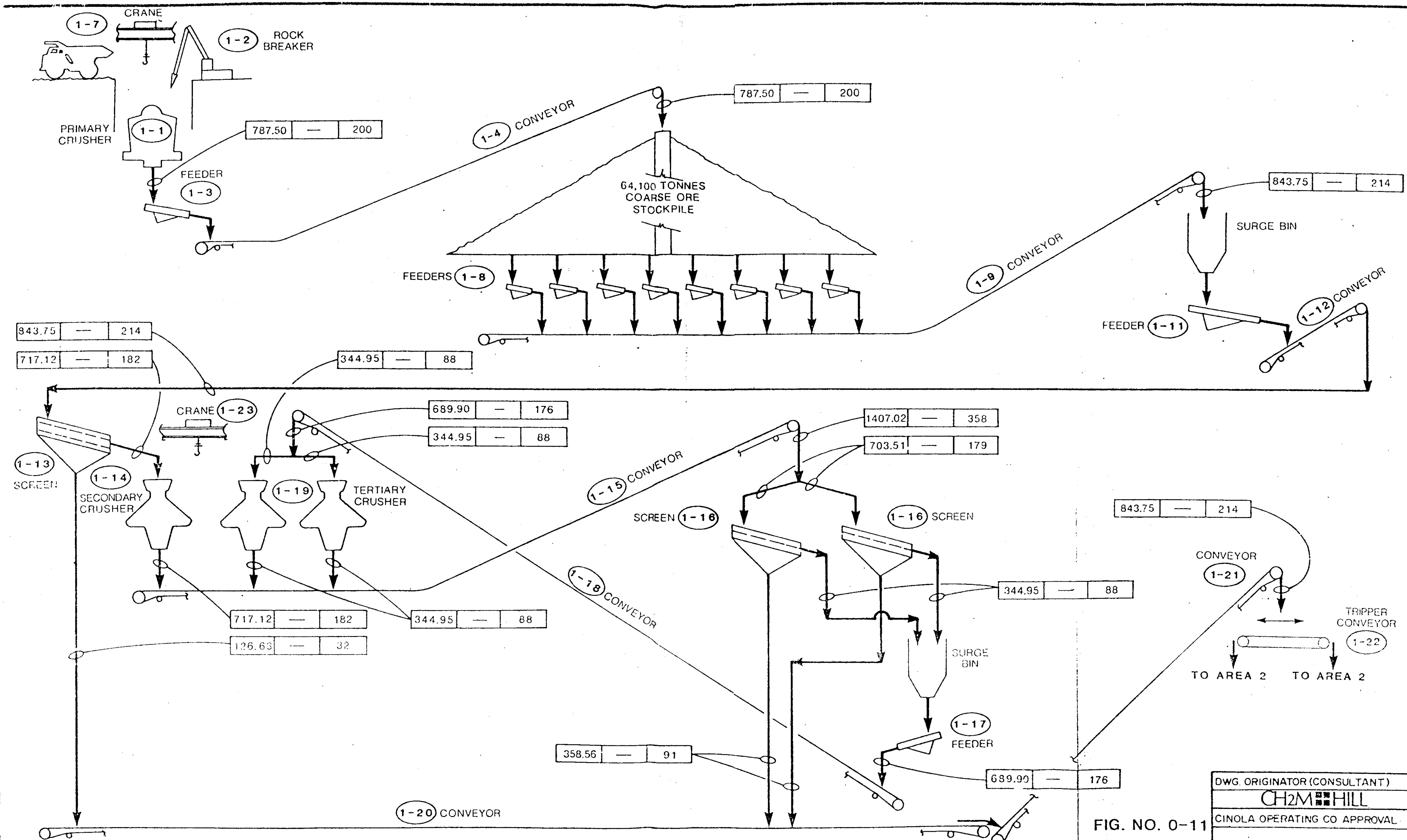


FIG. NO. 0-11

DWG. ORIGINATOR (CONSULTANT)
CH2M HILL
CINOLA OPERATING CO APPROVAL

LEGEND

TONNES PER HOUR	LITRES PER MINUTE PULP	LITRES PER MINUTE WATER

AREA 1
CRUSHING & STORAGE

CINOLA OPERATING COMPANY LTD.

DRAWN: RMS	DATE: 7-82
CHECKED	APPROVED
Fig. 4-1	REV.

The screening circuit is designed to discharge the plus 2 cm ore to a surge bin. The minus 2 cm ore from the screens is discharged to the fine ore collecting conveyor (1-20) which also receives the minus 2 cm ore from the double deck screen (1-13). The intermediate crushed ore vibrating feeder (1-17) discharges the plus 2 cm ore to the tertiary crusher feed conveyor (1-18).

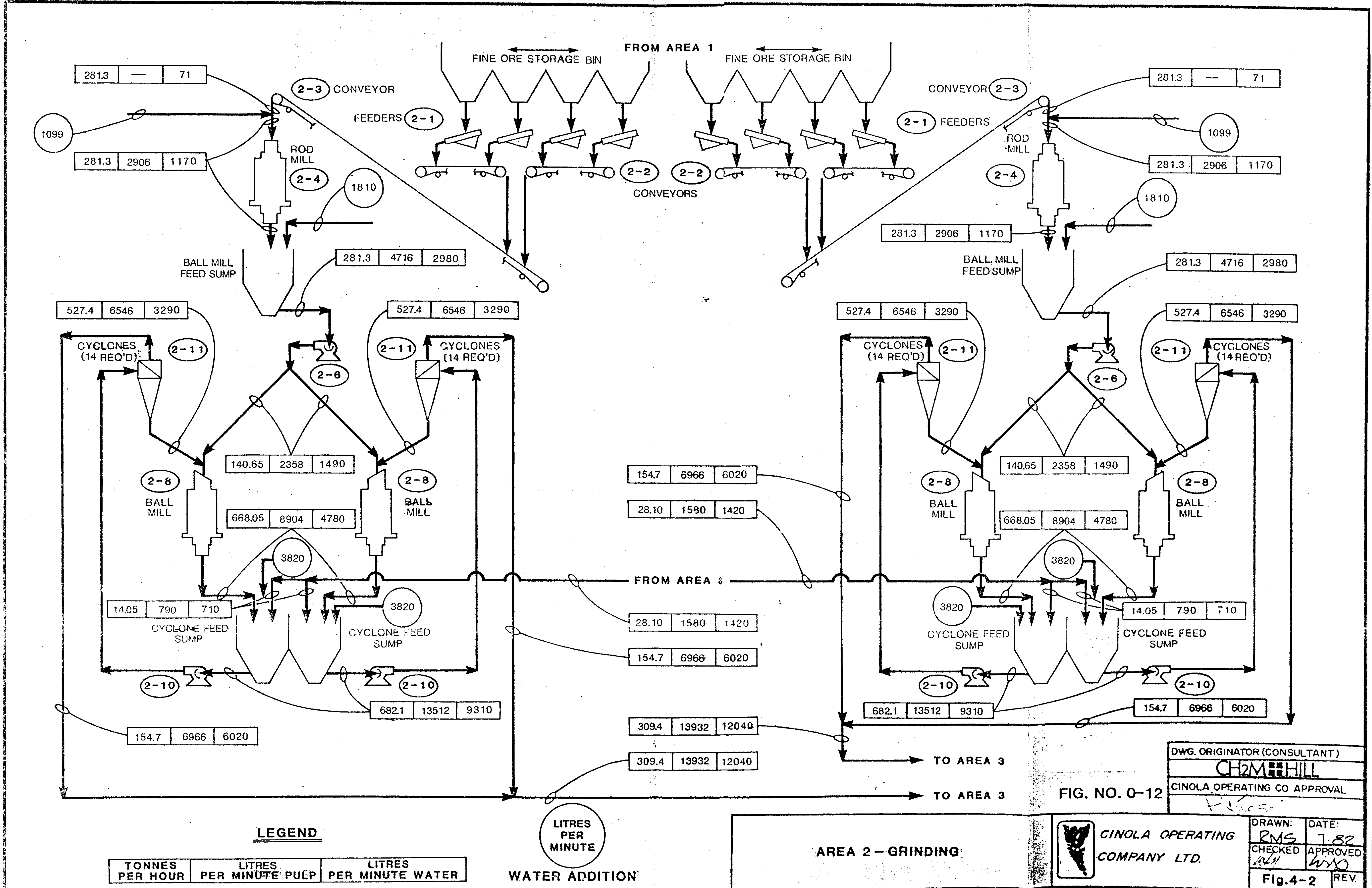
The fine ore collecting conveyor (1-20) discharges to the fine ore bin feed conveyor (1-21). The fine ore bin tripper conveyor (1-22) mounted on top of the fine ore bins receives the ore from the feed conveyor and distributes it the length of the bins.

0.5.1.2 Grinding

The design parameters for the grinding circuit are as follows:

1.	Ore, specific gravity	2.7
2.	Ore, moisture concentrate	1.5 percent
3.	Ore, Work Index	18.7 - 25.4
4.	Number of grinding circuits	2
5.	Rod mill, discharge size	80 percent minus 1,180 um
6.	Rod mill, pulp density	80 percent solids
7.	Ball mill, pulp density	70 percent solids
8.	Ball mill, circulating load	375 percent
9.	Cyclone, feed pulp density	55 percent solids
10.	Cyclone, overflow pulp density	30 percent solids
11.	Cyclone, overflow size	75 percent minus 45 um

The rod mill and ball mill grinding circuit is based on a twin stream concept. Consequently, half of the mill can be operated while the other half is shut down. For simplicity only one circuit is described. Figure 0-12 and Drawing 40 1 03 show the flowscheme for the grinding circuits with nominal material and water flows. Drawing 40 1 11 shows the general arrangement of the grinding circuits.



FROM AREA 1
FINE ORE STORAGE BIN
FINE ORE STORAGE BIN
2-3 CONVEYOR
FEEDERS 2-1
2-1 FEEDERS
CONVEYOR 2-3
2-2
2-2
CONVEYORS
ROD MILL
2-4
1099
1810
281.3 — 71
281.3 2906 1170
281.3 4716 2980
BALL MILL FEED SUMP
527.4 6546 3290
CYCLONES (14 REQ'D)
2-11
2-6
140.65 2358 1490
668.05 8904 4780
2-8
BALL MILL
3820
14.05 790 710
CYCLONE FEED SUMP
2-10
682.1 13512 9310
154.7 6966 6020
28.10 1580 1420
FROM AREA 3
309.4 13932 12040
TO AREA 3
TO AREA 3
LEGEND
TONNES PER HOUR
LITRES PER MINUTE PULP
LITRES PER MINUTE WATER
LITRES PER MINUTE
WATER ADDITION
AREA 2 — GRINDING
FIG. NO. 0-12
DWG. ORIGINATOR (CONSULTANT)
CH2M HILL
CINOLA OPERATING CO APPROVAL
CINOLA OPERATING COMPANY LTD.
DRAWN: RMS
DATE: 7-82
CHECKED
APPROVED
Fig.4-2
REV.

The rod mill feed conveyor discharges to the 4 m diameter by 6.5 m long rod mill at a rate of 281 tph. Soda ash is added to the rod mill to maintain a pH of 8. The rod mill reduces the fine ore to approximately 60 percent minus 1,180 microns. The rod mill product is collected in a sump and is pumped (2-6) to two ball mills. The 5 m diameter by 9.1 m long ball mills (2-8) operate with a 375 percent recirculating load, and further grind the ore to approximately 75 percent minus 45 microns.

The ball mill discharge is collected in a sump and is pumped (2-10) to classifying cyclone banks, one for each ball mill. The classifying cyclone banks (2-11) contain fourteen 25.4 cm diameter classifying cyclones. The cyclone underflow is directed back to the ball mills for additional grinding. The cyclone overflow, which contains 30 percent solids and 75 percent minus 45 microns material at a pulp density of 30 percent solids, is pumped (2-12) to the flotation conditioning tanks (3-1).

0.5.1.3 Flotation, Dewatering and Tailings Disposal

The design parameters for this area are:

1.	Feed, size	75 percent minus 45 um
2.	Feed, pulp density	30 percent solids
3.	Ore, specific gravity	2.7
4.	Number of flotation circuits	2
5.	Conditioning time	10 min
6.	Rougher flotation time	10 min
7.	Scavenger flotation time	10 min
8.	Gold recovery	82.64 percent (average)
9.	Ratio of concentration	10:1
10.	Concentrate thickener area	0.70 m^2/tpd
11.	Concentrate thickener, feed pulp density	25 percent
12.	Tailings thickener area	0.45 m^2/tpd
13.	Tailings thickener, feed pulp density	30 percent
14.	Filtering rate	742 kg/m^2/h

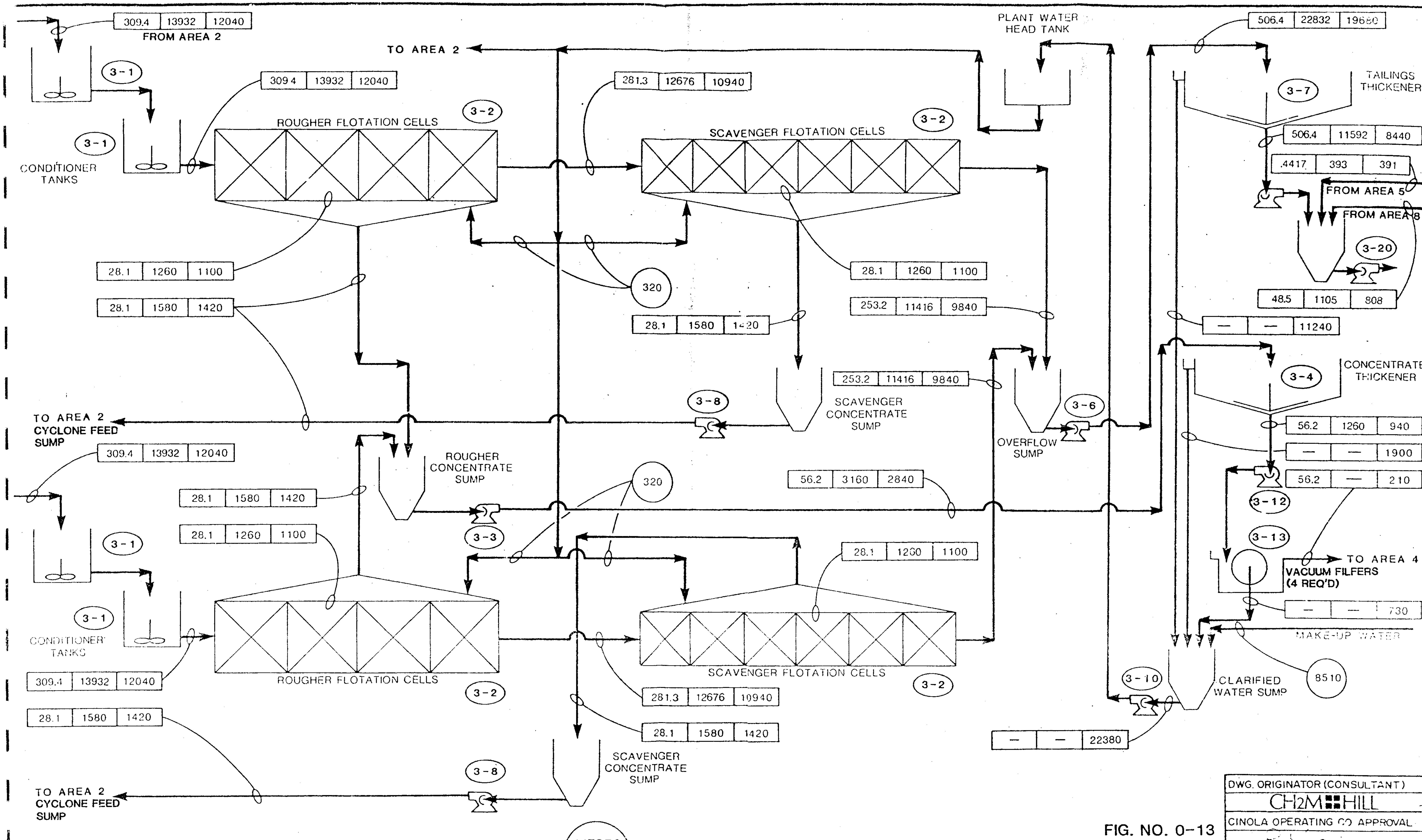


FIG. NO. 0-13

DWG. ORIGINATOR (CONSULTANT)
CH2M HILL
CINOLA OPERATING CO APPROVAL

AREA 3 – FLOTATION, DEWATERING AND TAILINGS DISPOSAL

CINOLA OPERATING COMPANY LTD.

DRAWN: RMS	DATE: 7-82
CHECKED	APPROVED
Fig.4-3	REV

The flotation circuit, like the grinding circuit, is based upon a twin stream concept. Figure 0-13 and Drawing 40 1 03 show the flowscheme for the flotation, dewatering and tailings disposal circuit with nominal material and water flows. Drawing 40 1 11 shows the general arrangement of the flotation and dewatering circuits. As with the grinding circuit, only one stream is described.

Ore from the grinding circuit reports to two 4.9 m diameter by 5.3 m high conditioner tanks (3-1) connected in series, where the flotation reagents are added. Conditioned pulp flows by gravity to the flotation cell bank (3-2). The bank consists of four 42.5 m^3 rougher cells and six 28.3 m^3 scavenger cells in series. The gold-bearing concentrate from the rougher cells is collected and pumped (3-3) to the 26.5 m diameter concentrate thickener (3-4). This thickener is common to both flotation circuits.

Tailings from the rougher cells flow by gravity to the scavenger cells. Scavenger cell concentrate is collected and pumped (3-8) to the cyclone feed sump (Area 2). Scavenger cell tailings are collected and pumped (3-6) to the 83.8 m diameter tailings thickener. This thickener is common on both flotation circuits.

Space will be left in the plant for a rougher concentrate cleaner circuit if, after production testing, it is found to be warranted.

The concentrate thickener underflow is pumped (3-12) to four 2.75 m diameter, 6-disc vacuum filters (3-13). The filter cake is collected on a conveyor (3-14) which discharges into the repulp tank (Area 4) for further processing.

Tailings thickener underflow is pumped (3-15) to the tailings sump. Neutralized weak acid pulp and treated calcine pulp is also pumped to this sump from Areas 5 and 7. The composited tailings is pumped (3-20) to the tailings impoundment area to be located approximately 6 km northwest of the metallurgical process facility.

The overflow from the concentrate and tailings thickeners is collected and pumped (3-10) to the facility process water head tank. The required make-up water for the processes is also pumped to this head tank.

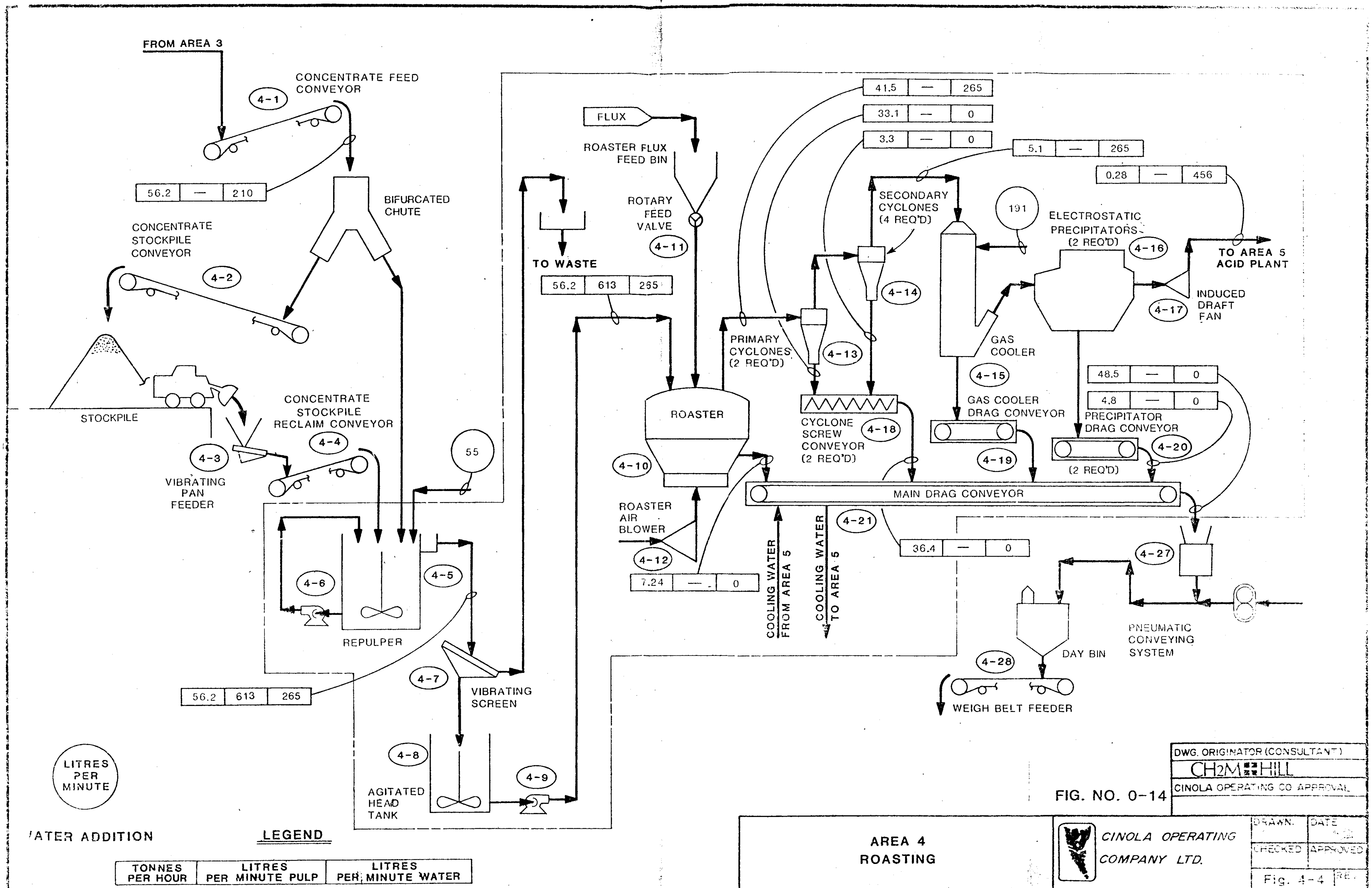
0.5.1.4 Roasting (Single Stage)

Design parameters used for development of the roasting circuit ate as follows:

1.	Concentrate moisture content	17.5 percent
2.	Roaster feed, pulp density	78 percent solids
3.	Concentrate, size	75 percent minus 45 um
4.	Nominal concentrate assays:	
	- Sulphur	14 percent
	- Iron	13 percent
	- Carbon	2 percent
	- Silica	62 percent
	- Alumina	4.6 percent
	- Arsenic	0.3 percent
	- Antimony	0.02 percent
	- Mercury	30 ug/g
5.	Roasting temperature	700°
6.	Sulphur oxidation in roaster	98 percent
7.	Arsenic volatization in roaster	50 percent
8.	Antimony volatization in roaster	67 percent
9.	Mercury volatization in roaster	75 percent
10.	Carbon oxidation in roaster	98 percent
11.	Roaster concentrate throughput	1,350 mtpd

Figure 0-14 and Drawings 40 1 05, and 40 9 03 show the flowscheme for the roasting area (Area 4) with nominal material and water flow rates.

Concentrate from the mill, with a moisture content of 17.5 percent, is transferred to a bifurcated chute that directs the concentrate either to an agitated 3.7 m diameter by 4.0 m high calcine repulp tank (4-5) or to a 925 m^2 storage pad outside the mill building via the concentrate stockpile conveyor (4-2).


FROM AREA 3
CONCENTRATE FEED CONVEYOR
4-1
56.2 — 210
BIFURCATED CHUTE
CONCENTRATE STOCKPILE CONVEYOR
4-2
STOCKPILE
CONCENTRATE STOCKPILE RECLAIM CONVEYOR
4-4
4-3
VIBRATING PAN FEEDER
55
4-6
4-5
REPULPER
56.2 613 265
4-7
VIBRATING SCREEN
4-8
AGITATED HEAD TANK
4-9
TO WASTE
56.2 613 265
FLUX
ROASTER FLUX FEED BIN
ROTARY FEED VALVE
4-11
ROASTER
4-10
ROASTER AIR BLOWER
4-12
7.24 — 0
41.5 — 265
33.1 — 0
3.3 — 0
5.1 — 265
0.28 — 456
PRIMARY CYCLONES (2 REQ'D)
4-13
SECONDARY CYCLONES (4 REQ'D)
4-14
CYCLONE SCREW CONVEYOR (2 REQ'D)
4-18
191
GAS COOLER
4-15
GAS COOLER DRAG CONVEYOR
4-19
ELECTROSTATIC PRECIPITATORS (2 REQ'D)
4-16
TO AREA 5 ACID PLANT
INDUCED DRAFT FAN
4-17
48.5 — 0
4.8 — 0
PRECIPITATOR DRAG CONVEYOR
4-20
(2 REQ'D)
MAIN DRAG CONVEYOR
4-21
COOLING WATER FROM AREA 5
COOLING WATER TO AREA 5
36.4 — 0
4-27
PNEUMATIC CONVEYING SYSTEM
DAY BIN
4-28
WEIGH BELT FEEDER
LITRES PER MINUTE
WATER ADDITION
LEGEND
TONNES PER HOUR
LITRES PER MINUTE PULP
LITRES PER MINUTE WATER
FIG. NO. 0-14
DWG. ORIGINATOR (CONSULTANT)
CH2M HILL
CINOLA OPERATING CO APPROVAL
AREA 4
ROASTING
CINOLA OPERATING COMPANY LTD.
DRAWN.
DATE
CHECKED
APPROVED
Fig. 4-4

Concentrate from surface storage is reclaimed by a front-end loader and transfered to the repulper tank (4-5).

In the agitated repulp tank, concentrate is formed into slurry of 78 percent solids. It is passed through a No. 3 mesh screen (4-7) to remove any foreign material before being pumped into the agitated head tank (4-8), and subsequently to the 8.2 m diameter fluid bed roaster (4-10). The slurry is fed through the roof of the roaster, and falls into the roaster bed. The amount of air added with the slurry is adjusted to provide a uniform dispersion over the bed.

Air is blown into the bottom of the roasters through tuyere pipes to maintain bed fluidity and to provide oxygen for the roaster reactions. The concentrate is roasted at about 700°C. The roaster bed consists of relatively coarse sand, sized to permit a high velocity of air through the roaster. The turbulence at this high space velocity promotes rapid dispersion of the slurry, and a corresponding high rate of water evaporation and sulphide oxidation. The sand stays in the bed and provides the fluid heat transfer medium. The majority of the calcine is much finer and is carried out of the bed and recovered in the gas cleaning section.

The calcine collection section of the roaster consists of two parallel sets of 2.4 m diameter primary (4-13) and 1.5 m diameter secondary cyclones (4-14), followed by a single 4.9 m diameter by 18.6 m high gas cooling tower (4-15), and then two parallel 6.1 m wide by 17.7 m high electrostatic precipitators (4-16). The gas, which leaves the roaster at 700°C is partially cooled and conditioned before going through the precipitators.

The gas is cooled in the cooling tower by spraying water into the tower. Temperature control on the gas cooler spray tower outlet regulates the flow of water to maintain a constant temperature to the precipitators. The cleaned gas, containing primarily nitrogen, sulphur dioxide, and carbon dioxide leaves the precipitators and is sent to the sulphuric acid plant. The gold-bearing calcine in the gas stream is captured in the cyclone and precipitator underflows, joins a small underflow stream of dust coming from the cooler, and travels to a 1,350-tonne bin.

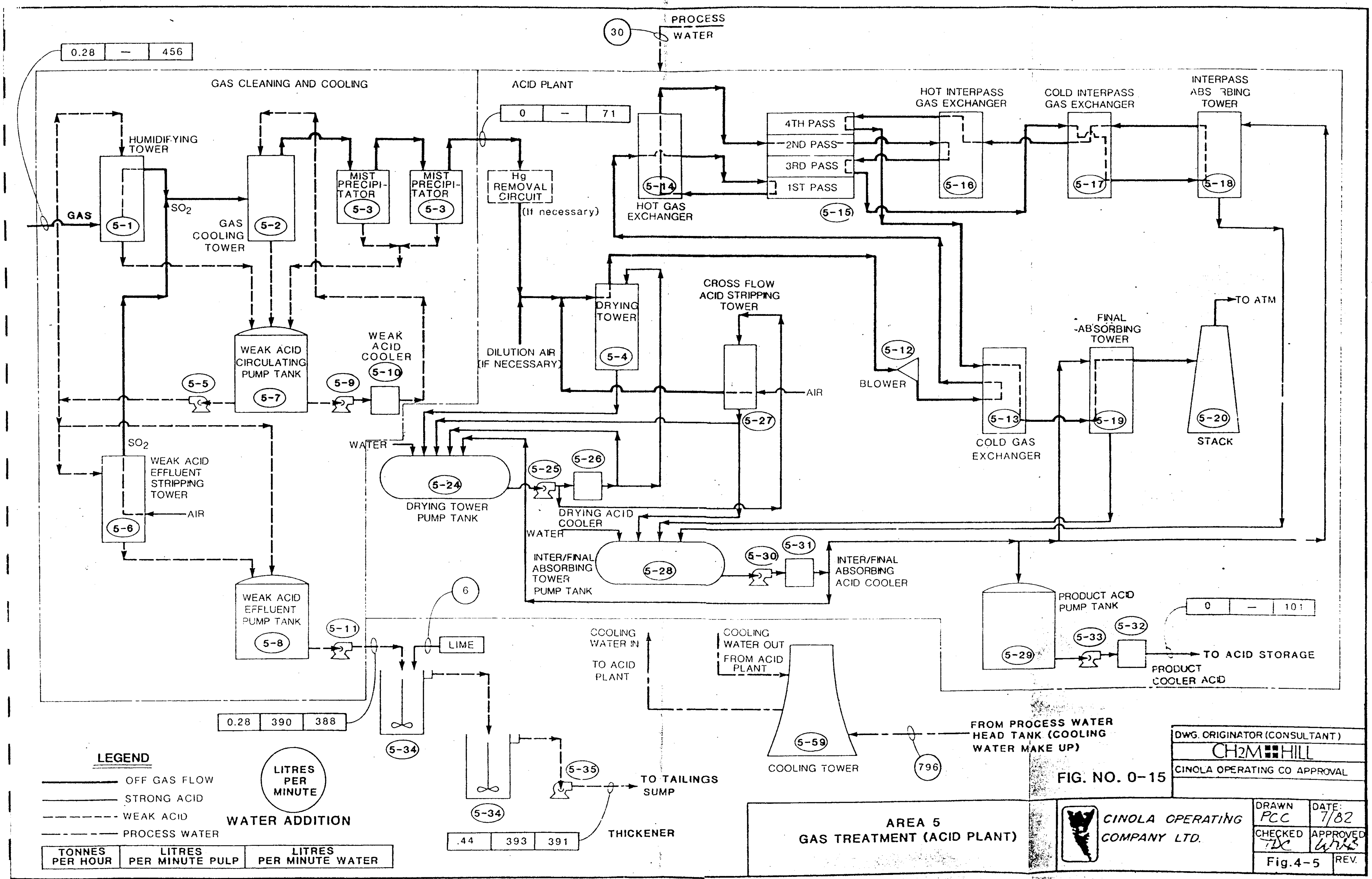
0.5.1.5 Acid Plant (Gas Treatment)

Design parameters used for development of the metallurgical type acid plant are as follows:

1.	Acid feed (roaster offgas)	1600 kg/min
2.	Nominal offgas analysis:	
	- Sulphur dioxide	14.0 percent
	- Sulphur trioxide	Trace
	- Oxygen	3.5 percent
	- Nitrogen	53.5 percent
	- Carbon dioxide	3.7 percent
	- Water vapor	25.0 percent
	- Dust	0.15 percent
3.	Offgas temperature to acid plant	350°C
4.	Product (acid) quality	
	- Color	Water white
	- Strength at 15°C	66° Baume
	- Sulphur dioxide content	50 ug/g
	- Nitrate content	50 ug/g
	- Arsenic content	1 ug/g
	- Lead content	2.3 ug/g
	- Mercury content	1 ug/g
5.	Tail gas analysis:	
	- Sulphur dioxide	less than 32 mg/mole
	- Sulphur trioxide	less than 0.114 g/m^3
6.	Autothermal operation at 5.5 percent sulphur dioxide or greater	

For ease of discussion, the gas treatment cicuit is divided into 6 systems. Figure 0-15 and Drawings 40 1 05, 40 9 04, 40 9 05, 40 9 06 and 40 9 07 show the flowscheme for the acid plant (Area 5) together with nominal material and water flow rates.

Because the acid plant facility is detail-designed specifically for the production of commercial grade sulphuric acid and is dependent upon the specific roaster offgas quality, it is impossible at this time to provide precise component sizes in the report.



PROCESS WATER
30
0.28 | — | 456
GAS CLEANING AND COOLING
ACID PLANT
0 | — | 71
HUMIDIFYING TOWER
GAS
5-1
SO2
GAS COOLING TOWER
5-2
MIST PRECIPITATOR
5-3
MIST PRECIPITATOR
5-3
Hg REMOVAL CIRCUIT
(If necessary)
HOT GAS EXCHANGER
5-14
4TH PASS
2ND PASS
3RD PASS
1ST PASS
5-15
HOT INTERPASS GAS EXCHANGER
5-16
COLD INTERPASS GAS EXCHANGER
5-17
INTERPASS ABSORBING TOWER
5-18
WEAK ACID CIRCULATING PUMP TANK
5-7
5-5
5-9
WEAK ACID COOLER
5-10
DILUTION AIR (IF NECESSARY)
DRYING TOWER
5-4
CROSS FLOW ACID STRIPPING TOWER
5-27
AIR
5-12
BLOWER
COLD GAS EXCHANGER
5-13
FINAL ABSORBING TOWER
5-19
TO ATM
STACK
5-20
SO2
WEAK ACID EFFLUENT STRIPPING TOWER
5-6
AIR
WATER
DRYING TOWER PUMP TANK
5-24
5-25
5-26
DRYING ACID COOLER
WATER
INTER/FINAL ABSORBING TOWER PUMP TANK
5-28
5-30
5-31
INTER/FINAL ABSORBING ACID COOLER
PRODUCT ACID PUMP TANK
5-29
5-33
5-32
0 | — | 101
TO ACID STORAGE
PRODUCT COOLER ACID
WEAK ACID EFFLUENT PUMP TANK
5-8
5-11
6
LIME
0.28 | 390 | 388
5-34
5-34
5-35
TO TAILINGS SUMP
.44 | 393 | 391
THICKENER
COOLING WATER IN
TO ACID PLANT
COOLING WATER OUT FROM ACID PLANT
5-59
COOLING TOWER
FROM PROCESS WATER HEAD TANK (COOLING WATER MAKE UP)
796
LEGEND
OFF GAS FLOW
STRONG ACID
WEAK ACID
PROCESS WATER
LITRES PER MINUTE
WATER ADDITION
TONNES PER HOUR | LITRES PER MINUTE PULP | LITRES PER MINUTE WATER
FIG. NO. 0-15
DWG. ORIGINATOR (CONSULTANT)
CH2M HILL
CINOLA OPERATING CO APPROVAL
AREA 5
GAS TREATMENT (ACID PLANT)
CINOLA OPERATING COMPANY LTD.
DRAWN PCC
DATE: 7/82
CHECKED
APPROVED
Fig.4-5
REV.

Gas Cleaning and Cooling System -- Gas containing sulphur dioxide exiting the electrostatic precipitators enters the sulphuric acid plant battery limits after having been partially cooled and cleaned. It is delivered to the bottom inlet of an open type humidifying tower (5-1), and passes up through the vessel counter-current to a flow of weak acid. Exiting from the top of the humidifying tower, the gas stream enters a packed gas cooling tower (5-2) and passes upwards through the packing contacting cooled, recirculated weak acid.

The cooled and partially-cleaned gas exiting the top of the gas cooling tower enters three parallel banks of electrostatic mist precipitators (5-3). Each bank consists of two units operating in series. Cleaned gas exiting the secondary mist precipitators is combined into a single stream prior to entering the drying tower (5-4).

The acid plant layout allows for the future addition of a mercury removal system between the exit of the mist precipitator and the inlet of the drying tower, should such a system be found to be required to meet product acid quality specifications. Mercury would be removed by scrubbing with a mercurous chloride solution producing mercuric chloride, for subsequent reduction to mercury metal.

Weak Acid Circulation System -- Weak acid is used to humidify and cool the offgas stream in the gas cleaning and cooling system. A sidestream of liquor from a centrifugal pump is directed to the weak acid effluent stripping tower (5-6) which serves to air strip the dissolved sulphur dioxide absorbed in the humidification tower from this liquid. Stripping air together with the sulphur dioxide is returned to the gas stream downstream of the humidifying tower. Stripped liquor flows by gravity to the weak acid effluent pump tank (5-8), and is then pumped to the neutralization facility. It is in this waste stream that the main portion of volatilized arsenic, antimony and mercury is removed.

Weak acid liquor is delivered to the gas cooling tower distributors. Prior to entering the gas cooling tower, the weak acid is cooled to the desired temperature needed to achieve a water balance in the acid plant process stream. The weak acid from both the humidifying tower and the gas cooling tower return by gravity to the weak acid circulating pump tank (5-7).

Drying, Conversion and Absorbing System -- Process gas from the mist precipitators enters the drying tower (5-4) and passes upward through a layer of packing counter-current to a flow of 93 percent sulphuric acid. The dried and essentially mist free gas leaving the drying tower passes in series through the shell sides of the cold gas-to-gas heat exchanger (5-13) and hot gas-to-gas heat exchanger (5-14) prior to entering the first pass of the converter (5-15). The incoming gas is raised to the proper temperature for conversion of sulphur dioxide to sulphur trioxide in the converter.

Four passes through the converter (5-15) are required to effect the final conversion of sulphur dioxide to sulphur trioxide. In the absorbing towers, the sulphur trioxide component in the gas is absorbed in the flow of 98 percent sulphuric acid as it passes up through the main packing in the tower. After the gas leaves the main packing, it passes through a mist eliminator prior to discharging to the atmosphere via the process gas exit stack (5-20).

Preheater System -- to obtain initial operating temperatures in the converter, an external source of heat must be supplied. This is provided by the preheater system (not shown in Figure 0-15).

In the event that the sulphur dioxide content in the process gas falls below the autothermal level, the extra heat required for conversion is supplied by this system. Under normal operation, however, the preheater system is not in use.

Strong Acid Circulation System -- Circulation acid for the drying tower (5-4) is provided from the drying tower pump tank (5-24). A circulation pump (5-25) delivers acid to the drying tower after it has passed through the drying acid cooler (5-26). Acid from the drying tower is returned by gravity to the drying tower pump tank.

Product acid, either as 93 or 98 percent sulphuric acid, is cooled in the product acid cooler (5-32) and delivered to the acid tank storage facilities by the product acid pump (5-33).

Effluent Treatment — The weak acid effluent contains solids and condensed fume scrubbed from the roaster offgas. This stream is neutralized and disposed of in an environmentally safe manner.

Estimates of acid plant effluent characteristics are as follows:

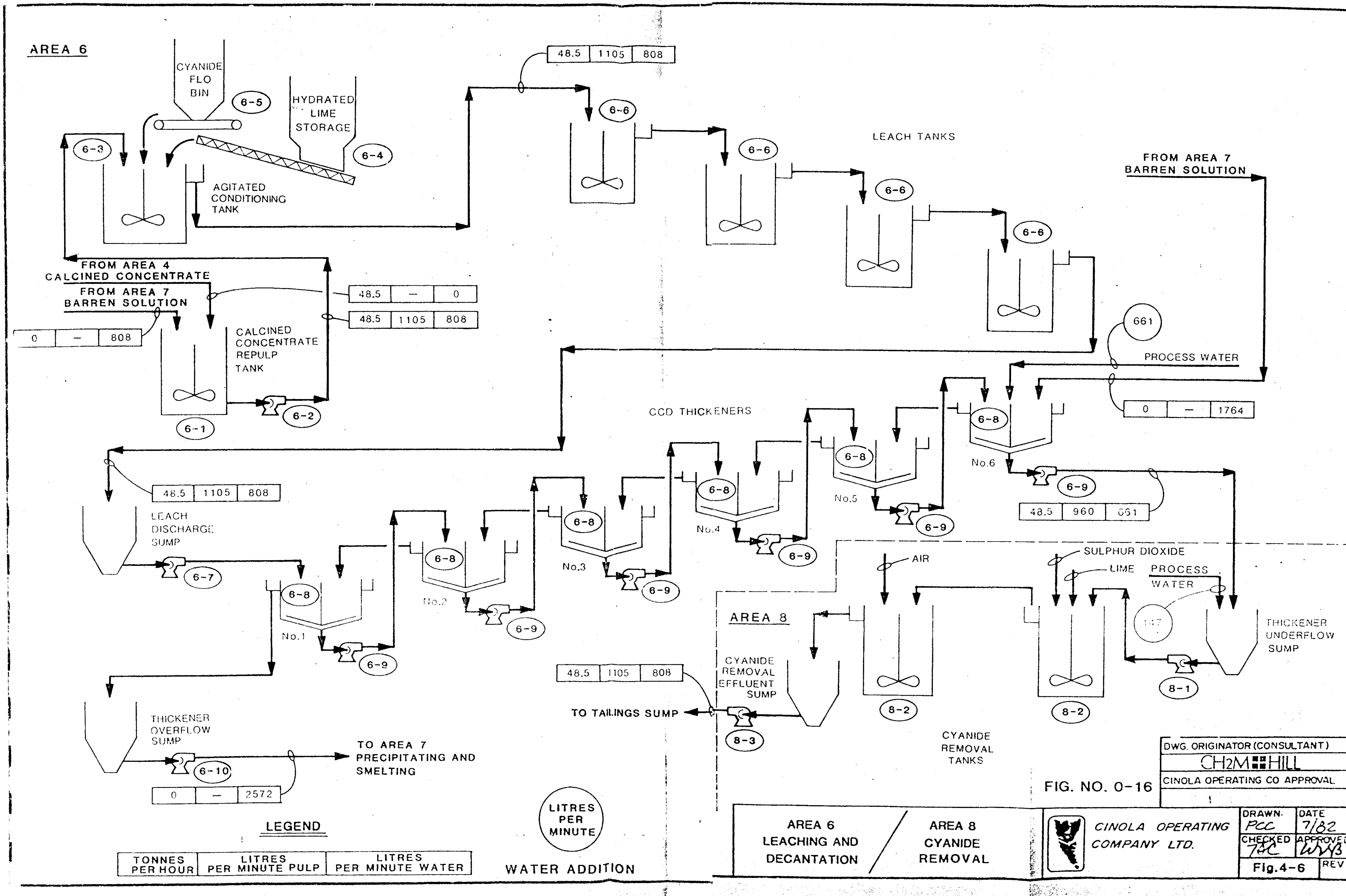
1.	Pulp	150 to 300 L/min
2.	Solids	3.5 to 7 kg/min
3.	Acid	3.5 to 7 kg/min
4.	Percent solids	1to 2 percent by wt

The weak acid pulp is pumped from the acid plant to two agitated effluent precipitation tanks (5-34) operated in series. A controlled amount of lime is added to the pulp to raise its pH to 10.5. The resulting neutralized weak acid pulp is then pumped (5-35) directly to the tailings sump.

0.5.1.6 Leaching and Decantation

Design parameters used for development of the leaching and counter-current decantation circuits are as follows:

1.	Calcine throughput	1,163 tpd
2.	Projected average gold recovery:	
	- Leaching	87.15 percent (average)
	- Decantation washing	99.5 percent (average)


AREA 6
CYANIDE FLO BIN
6-5
HYDRATED LIME STORAGE
6-4
6-3
AGITATED CONDITIONING TANK
48.5 | 1105 | 808
6-6
LEACH TANKS
6-6
6-6
6-6
FROM AREA 7 BARREN SOLUTION
FROM AREA 4 CALCINED CONCENTRATE
FROM AREA 7 BARREN SOLUTION
48.5 | — | 0
48.5 | 1105 | 808
0 | — | 808
CALCINED CONCENTRATE REPULP TANK
6-1
6-2
661
PROCESS WATER
0 | — | 1764
CCD THICKENERS
6-8
No.6
6-9
48.5 | 960 | 661
6-8
No.5
6-9
6-8
No.4
6-9
6-8
No.3
6-9
6-8
No.2
6-9
6-8
No.1
6-9
48.5 | 1105 | 808
LEACH DISCHARGE SUMP
6-7
THICKENER OVERFLOW SUMP
6-10
0 | — | 2572
TO AREA 7 PRECIPITATING AND SMELTING
SULPHUR DIOXIDE
AIR
LIME
PROCESS WATER
AREA 8
147
THICKENER UNDERFLOW SUMP
8-1
CYANIDE REMOVAL EFFLUENT SUMP
48.5 | 1105 | 808
TO TAILINGS SUMP
8-3
8-2
8-2
CYANIDE REMOVAL TANKS
LEGEND
TONNES PER HOUR | LITRES PER MINUTE PULP | LITRES PER MINUTE WATER
LITRES PER MINUTE
WATER ADDITION
FIG. NO. 0-16
DWG. ORIGINATOR (CONSULTANT)
CH2M HILL
CINOLA OPERATING CO APPROVAL
AREA 6 LEACHING AND DECANTATION / AREA 8 CYANIDE REMOVAL
CINOLA OPERATING COMPANY LTD.
DRAWN PCC
DATE 7/82
CHECKED
APPROVED
Fig.4-6
REV

3.	Leaching pulp density	50 percent solids
4.	Leaching retention time	20 hrs
5.	CCD thickener wash ratio	3 to 1
6.	CCD thickener underflow pulp density	55 percent solids
7.	Thickener area requirements	0.13 m^2/tpd

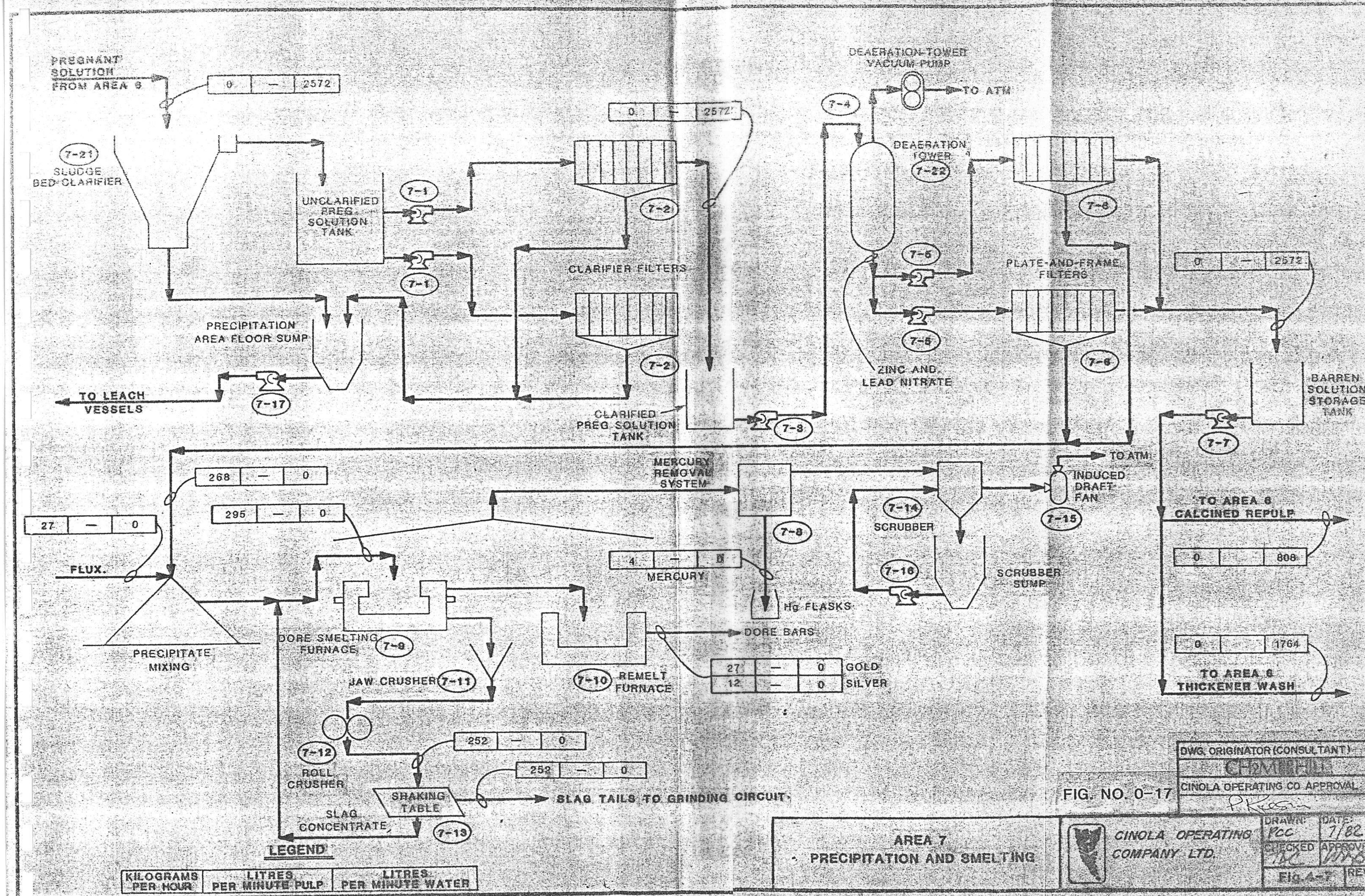
Figure 0-16 and Drawings 40 1 05 and 40 9 08 show the flowscheme for the leaching and decantation areas with nominal material and water flows.

Cooled calcine is delivered to the 3 m diameter by 3 m high calcined concentrate repulp tank (6-1) where barren solution is added to produce a pulp of 50 percent solids. The pulp is then delivered to the 6.1 m diameter by 6.1 m high agitated conditioning tank (6-3). Dry hydrated lime and sodium cyanide is fed to the conditioning tank. pH of the pulp is controlled at 10.5. The pulp overflows the conditioning tank into the first 7.6 m diameter by 7.6 m high tank of a 4 stage leach tank (6-6) circuit. The tanks are fitted with mechanical agitators and air spargers. The pulp overflows each stage of leaching and is gravity fed to the subsequent leach tank. Pulp retention time in the leach circuit is 20 hrs.

The pulp is gravity fed to the leach discharge sump, and is then pumped by the CCD thickener feed pump (6-7) to the number 1 thickener (6-8) in the CCD washing circuit. This circuit consists of six 17 m diameter conventional thickeners and is used to wash the leach discharge pulp. Barren solution and process water are added to number 6 thickener to wash the precious metal values from the incoming pulp. The design wash ratio is 3 to 1. Thickener underflows have an average pulp density of 55 percent solids and are pumped to subsequent thickeners in the circuit. Underflow from the final thickener is pumped to a cyanide removal circuit (Area 8). Thickener overflow flows by gravity, counter-current to the underflow slurry, starting at thickener number 6. The number 1 thickener overflow solution discharges to the thickener overflow sump where it is pumped to the precipitation circuit (Area 7).

0.5.1.7 Precipitation and Smelting

Design parameters used for development of the precipitation and smelting circuits are as follows:



PREGNANT SOLUTION FROM AREA 6
0 — 2572
7-21 SLUDGE BED CLARIFIER
UNCLARIFIED PREG. SOLUTION TANK
7-1
7-1
0 — 2572
7-2
CLARIFIER FILTERS
7-2
PRECIPITATION AREA FLOOR SUMP
TO LEACH VESSELS
7-17
CLARIFIED PREG SOLUTION TANK
7-3
DEAERATION TOWER VACUUM PUMP
TO ATM
7-4
DEAERATION TOWER
7-22
7-5
7-5
ZINC AND LEAD NITRATE
7-6
PLATE-AND-FRAME FILTERS
7-6
0 — 2572
BARREN SOLUTION STORAGE TANK
7-7
268 — 0
295 — 0
27 — 0
FLUX.
PRECIPITATE MIXING
MERCURY REMOVAL SYSTEM
7-8
4 — 0
MERCURY
Hg FLASKS
DORE SMELTING FURNACE
7-9
7-10 REMELT FURNACE
DORE BARS
27 — 0 GOLD
12 — 0 SILVER
JAW CRUSHER 7-11
7-12 ROLL CRUSHER
252 — 0
252 — 0
SHAKING TABLE
7-13
SLAG CONCENTRATE
SLAG TAILS TO GRINDING CIRCUIT
7-14 SCRUBBER
7-16
SCRUBBER SUMP
TO ATM
INDUCED DRAFT FAN
7-15
TO AREA 6 CALCINED REPULP
0 808
0 1764
TO AREA 6 THICKENER WASH
LEGEND
KILOGRAMS PER HOUR
LITRES PER MINUTE PULP
LITRES PER MINUTE WATER
FIG. NO. 0-17
DWG. ORIGINATOR (CONSULTANT)
CINOLA OPERATING CO. APPROVAL
AREA 7
PRECIPITATION AND SMELTING
CINOLA OPERATING COMPANY LTD.
DRAWN: PCC
DATE: 1/82
CHECKED
APPROVED
REV.
Fig. 4-7

1.	Pregnant solution flow	2,572 L/min
2.	Reagent consumption:	
	- Filter aid	94 kg/d
	- Lead nitrate	8 kg/d
	- Zinc	62 kg/d
3.	Precipitate production	268 kg/d
4.	Barren solution flow	2,572 L/min
5.	Daily average precious metals production	589.2 oz gold/day
		260.1 oz silver/day

Figure 0-17 and Drawings 40 1 05 and 40 9 08 show the flowscheme for precipitation and smelting areas (Area 7) with nominal material and water flows.

The pregnant solution (2,572 L/min) from the No. 1 thicknener is pumped from the pregnant solution feed sump to a 9.1 m diameter sludge bed clarifier (7-21), which separates most of the fine solids from the pregnant solution. The solution then overflows into the 6.1 m diameter by 6.1 m high unclarified pregnant solution tank. Unclarified pregnant solution is pumped to two 83.6 m^2 precoat clarifier filters (7-2) that remove the remaining slimes. Slimes from the sludge bed clarifier and clarifier filters pass to an area floor sump and are pumped to the leach circuit.

The clarified pregnant solution then flows to the 6.1 m diameter by 6.1 m high clarified pregnant solution storage tank. Clarified pregnant solution is pumped from the clarified pregnant solution tank through a 1.8 m diameter by 6.1 m high vacuum deaeration tower (7-22) to remove the dissolved oxygen in the clarified pregnant solution.

The vacuum is generated by the deaeration tower vacuum pump (7-4). The deaerated solution is treated with lead nitrate and zinc dust to precipitate the gold and silver from the solution. The solution is then delivered to the 83.6 m^2 53-chamber plate and frame filters by either one of the two plate and frame filter feed pumps (7-5) that operate in parallel, with only one pump and filter operating at a time. The precipitate is recovered in the plate and frame filters (7-6).

Barren solution from the precipitation circuit passes into the 6.1 m diameter by 6.1 m high barren solution storage tanks, and then is pumped to the repulp tank preceding the leach circuit and to the number 6 thickener in the CCD circuit wash solution.

Gold precipitate is transferred by hand from the plate and frame filter to the precipitate mixing area where fluxes are added. The mixed furnace feed is then charged by hand to the tilting, induction retorting/ smelting furnace. The precipitates are heated in the furnace (7-9) and a vacuum is drawn through the tight fitting hood that draws off all volatilized mercury from the precipitate. The mercury is collected in the mercury recovery system (7-8) and is prepared for sale. The precipitate is then smelted in the same furnace and cast as dore buttons. The cast dore buttons are remelted in the remelt furnace (7-10) and cast as dore bars. Slag from the smelting furnace is crushed in the laboratory jaw crusher (7-11) and the laboratory roll crusher (7-12). Any gold in the crushed slag is concentrated on the laboratory shaking table (7-13), and is combined with a new charge of precipitate and fluxes in the smelting furnace.

The offgas from the smelting area is collected and passed through the refining area exhaust scrubber (7-14). The scrubber solution is circulated to the scrubber through the scrubber recirculating pump (7-16). The bleed from this system reports to the weak acid neutralization system. The clean gas is then exhausted through the scrubber induced draft fan (7-15).

0.5.1.8 Cyanide Removal

Design parameters used for development of the cyanide removal circuit are as follows:

1.	Calcine pulp throughput:	
	- Solids	1,163 tpd
	- Liquid	1,163 tpd
2.	Calcine pulp density	50 percent solids
3.	Total cyanide content to be removed	42.5 kg/d

4. Reagent consumption

- Copper as $CuSO_4$	124.3 kg/d
- Sulphur dioxide (SO_2/air mixture)	174.5 kg/d
- Lime as CaO	195.3 kg/d

5. Reaction retention time — 1 hr required
   (2 hrs designated retention time provided)

Figure 0-16 and Drawings 40 1 05 and 40 9 08 show the flowscheme for the cyanide removal area (Area 8) with nominal material and water flows.

The leach residue from the last stage of the CCD circuit is diluted in the thickener underflow sump to 50 percent solids prior to being pumped to the cyanide removal circuit. At the same time, copper in the form of copper sulphate is added to raise the copper content of the liquid portion of the pulp to 50 mg/L.

The cyanide removal circuit consists of two 6.1 m diameter by 6.1 m high agitated cyanide removal tanks (8-2) similar to the leach tanks used for cyanidation. The feed pulp enters the first reactor where it is contacted with a sulphur dioxide/air mixture. A 3.5 volume percent sulphur dioxide mixture is used. About 4 kg of sulphur dioxide are required to remove 1 kg of cyanide. The reaction with sulphur dioxide, oxygen and water produces an acid that requires neutralization with slaked lime. Lime consumption is calculated to be 4.5 kg per kg cyanide in the feed.

The first reactor provides 60 minutes of retention time which is adequate to remove the majority of the cyanide. The second reactor provides an additional 60 min retention time in which only air is added. This vessel serves as a "polishing" reactor to remove the remaining quantities of cyanide. The effluent from the second tank is pumped by the cyanide removal effluent pump (8-3) to the flotation tailings sump for disposal (Area 3).

0.5.1.9 Electrical Distribution

Power generated at the power plant is received at the metallurgical processing complex at 13.8 kv. The mill complex is supplied from two parallel lines of 13.8 kv switchgear. Both 13.8 and 4.16 kv switchgear will be used to distribute the power to the various process areas of the complex.

A switchgear building is located in the crushing area and houses much of the electrical distribution equipment. A layout is shown on Drawing 40 1 18.

Key spare electrical equipment is provided due to the remote location of the project site. The time required to acquire and transport large electrical equipment to the plant site in the event of equipment failure justifies having this spare equipment installed.

The complex is provided with a distribution system that allows it to continue operation, at the 50 percent level, if failure of a main switchgear bus or main transformer occurs.

0.5.1.10 Instrumentation and Control

The metallurgical processing facility is designed with sufficient automatic instrumentation to measure and control solids and water flow for an efficient operation and to reduce labour requirements to a minimum.

0.5.1.11 Auxiliary Facilities

Potable Water -- Mill and concentrate treatment buildings are supplied from the main potable water source by an underground water main.

Sewage -- Waste from rest rooms located in the concentrator and concentrate treatment areas is delivered to the main sewage disposal facility by an underground sewer main.

Fire Protection -- Fire protection water mains, connected to the main fire protection system for the complex, are installed in all buildings. Fire hose stations are installed in suitable locations in all buildings to provide adequate coverage in the event of fire. Suitable outdoor hydrants and hose houses are provided in areas not serviced by the indoor fire hose stations.

0.5.2 Process Alternatives

During the development of the metallurgical feasibility study several process alternatives were considered but were not fully explored in detail. These alternatives should be considered for future application before the final design and construction phase of the project commences:

1. The use of an autogenous or semi-autogenous grinding circuit to replace the secondary/tertiary and rod mill/ball mill circuit used for this case.

2. The use of rubber or the newly developed magnetic rubber liners for rod and ball mills should be more thoroughly investigated to ascertain what savings in liner wear may be possible.

3. An installation of a regrind-cleaner flotation circuit should be more thoroughly investigated by means of bench-scale or laboratory pilot plant testing to improve the ratio of concentration significantly above that now predicted without reducing the flotation recovery. If higher concentration ratios can be obtained through a regrind and cleaner flotation circuit, this could reduce the size of the downstream processes and potentially improve the project economics.

4. Bench scale test results on calcine produced from two-stage fluid bed roasting at 475/540°C indicated increased gold extractions of 3 to 4 percent over the single stage 700°C tests. Offgas analysis indicated that an acid plant could still be used for gas treatment. This increased leach extraction at lower roasting temperatures should be confirmed with more tests and the cost effectiveness of two-stage roasting tensus single stage roasting should be established.

5. Biological oxidation of pyrite has indicated 82 to 93 percent of the gold and in excess of 98 percent of the silver can be extracted from flotation concentrates after 85 prcent pryite oxidation. The method has been demonstrated to work but has not been optimized. Additional test work should be performed to study the degree of pyrite oxidation required and the rates of oxidation with respect to pulp density and particle size. The method must also be optimized for cost effectiveness between gold and silver extraction and capital and operating costs.

0.5.3 Environmental Considerations

The mill process steps are designed to process and recover gold from a highly complex ore body. The gold can be associated with varying amounts of carbonaceous materials, various sulphide phases, and is also found as free gold. Further, the ore body contains arsenic and mercury with trace amounts of copper, lead, zinc and antimony.

The multiphase processing sequence to recover gold values from the ore is planned to properly treat all effluent streams to meet or exceed environmental requirements. Figure 0-18 shows the major process steps, with the various reagent addition points and discharges associated with the individual processes. Areas of potential environmental concern are denoted with an asterisk(*). The following comments indicate how engineering design deals with each area of concern.

0.5.3.1 Crushing and Storage (Area 1)

The principal environmental issue in this area is the generation of fugitive dust due to mechanical attrition and handling of the ore. Dust suppression water sprays are utilized to control dusting. Ore is transported to the coarse ore stockpile on conveyors that are covered with hoods, and all transfer points are totally enclosed. Ore at the stockpile is lowered by the use of stacking tubes to minimize the segregation of the coarse and finer ore sizes. Because the ore size is very coarse, wind-blown dust from the stockpile is not a problem, particularly due to wetting during the frequent periods of precipitation.

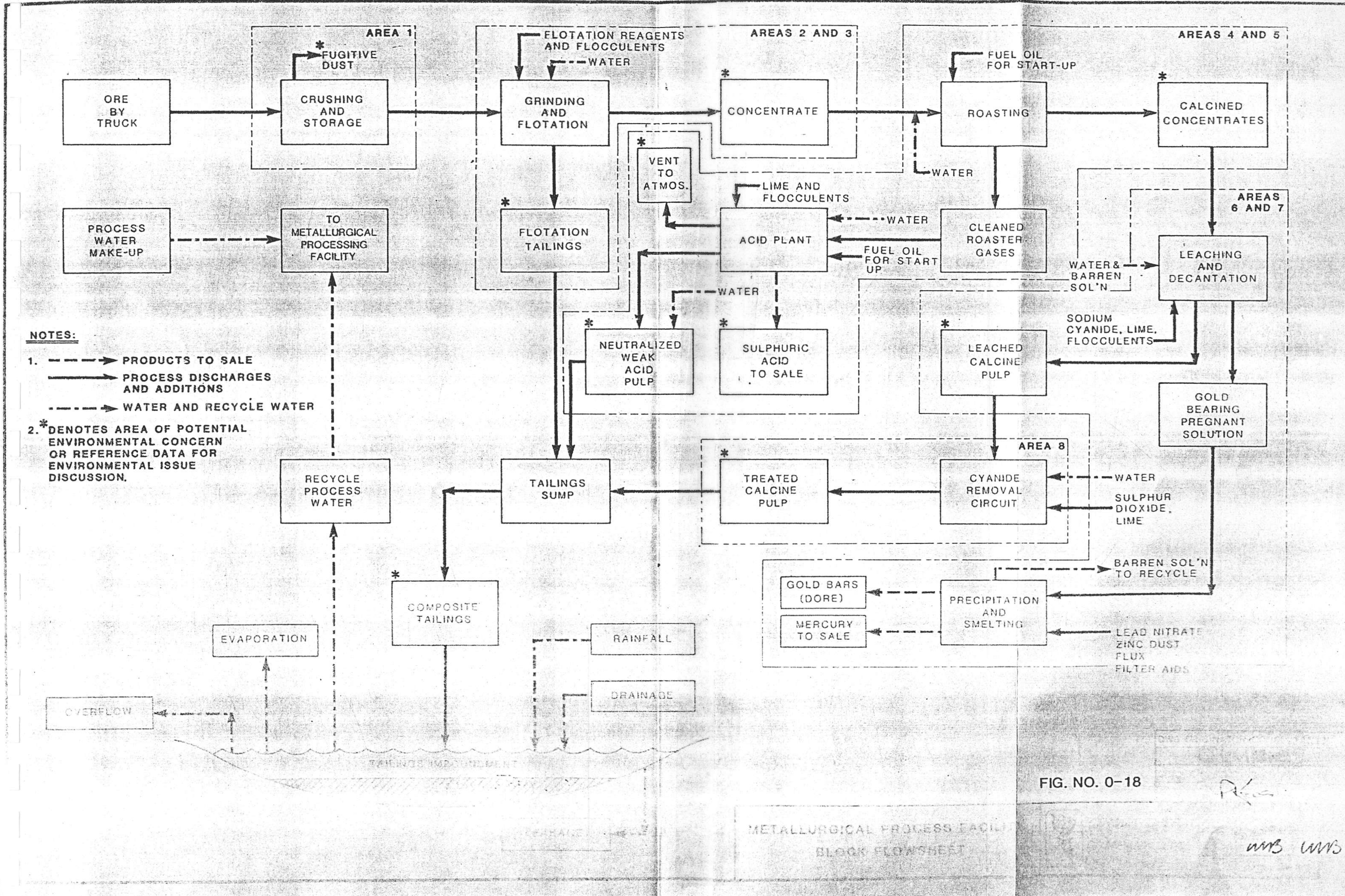

AREA 1
FUGITIVE DUST
ORE BY TRUCK
CRUSHING AND STORAGE
PROCESS WATER MAKE-UP
TO METALLURGICAL PROCESSING FACILITY
RECYCLE PROCESS WATER
EVAPORATION
OVERFLOW
AREAS 2 AND 3
FLOTATION REAGENTS AND FLOCCULENTS
WATER
GRINDING AND FLOTATION
CONCENTRATE
FLOTATION TAILINGS
VENT TO ATMOS.
LIME AND FLOCCULENTS
ACID PLANT
WATER
FUEL OIL FOR START UP
NEUTRALIZED WEAK ACID PULP
SULPHURIC ACID TO SALE
TAILINGS SUMP
COMPOSITE TAILINGS
RAINFALL
DRAINAGE
AREAS 4 AND 5
FUEL OIL FOR START-UP
ROASTING
WATER
CLEANED ROASTER GASES
CALCINED CONCENTRATES
AREAS 6 AND 7
WATER & BARREN SOL'N
LEACHING AND DECANTATION
SODIUM CYANIDE, LIME, FLOCCULENTS
LEACHED CALCINE PULP
GOLD BEARING PREGNANT SOLUTION
AREA 8
TREATED CALCINE PULP
CYANIDE REMOVAL CIRCUIT
WATER
SULPHUR DIOXIDE, LIME
BARREN SOL'N TO RECYCLE
GOLD BARS (DORE)
MERCURY TO SALE
PRECIPITATION AND SMELTING
LEAD NITRATE
ZINC DUST
FLUX
FILTER AIDS
NOTES:
1. PRODUCTS TO SALE
PROCESS DISCHARGES AND ADDITIONS
WATER AND RECYCLE WATER
2. * DENOTES AREA OF POTENTIAL ENVIRONMENTAL CONCERN OR REFERENCE DATA FOR ENVIRONMENTAL ISSUE DISCUSSION.
METALLURGICAL PROCESS FACIL
BLOCK FLOWSHEET
WWB WWB

FIG. NO. 0-18

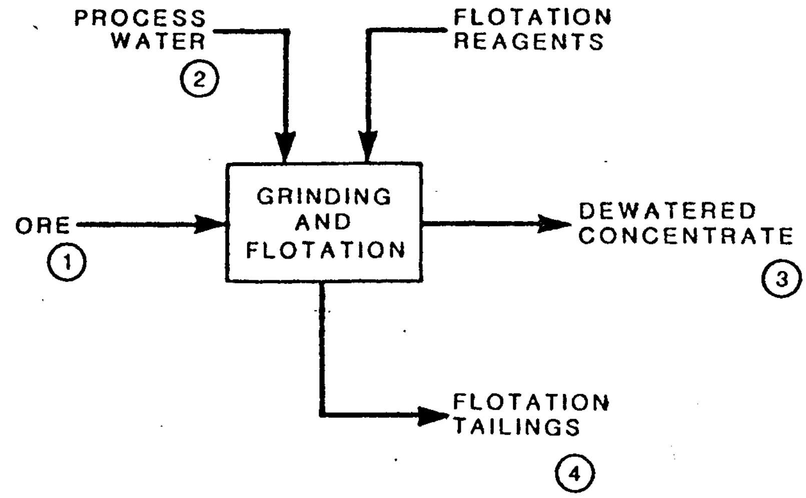
All screens and surge bins are equipped with properly sized baghouse dust collectors. After final sizing to 100 percent minus 2 cm, the ore is conveyed to the fine ore storage bins. Downstream from the fine ore bins, all processing is wet; consequently, fugitive dusts are not a problem.

0.5.3.2 Grinding and Flotation (Areas 2 and 3)

Conventional wet grinding, using rod and ball mills, is used to liberate the free gold and gold bearing minerals. Soda ash is added in the grinding circuit to maintain a basic pH in the ground pulp. The pulp is pumped to the flotation circuit for separation of the gold values. This separation is accomplished by the addition of reagents that activate the surfaces of the free gold and gold bearing minerals and permit them to selectively attach to air bubbles on which they float to the surface. Reagents added during this process include copper sulphate, pine oil, soda ash, potassium amyl xanthate and a di-thiophosphate compound (Aerofloat 31). However, only tenths of kilograms of reagents per tonne of dry ore processed are added, with virtually no impact on process water quality. Products that are produced from the flotation circuit are a dewatered gold-bearing concentrate, and a silicious flotation tailing. The flotation tailings are dewatered to nominally 50 percent solids, and are pumped at a pH of 8 to a sump for subsequent disposal in the tailings impoundment area. The concentrate is thickened and vacuum filtered to a moisture content of nominally 17.5 percent for subsequent processing.

The typical analysis from pilot mill test work conducted on the Queen Charlotte Islands from April through December, 1981, of ore, concentrates and tailings are summarized in Section 4.5.2 of the main report.

As noted from Figure 0-19 an average discharge of flotation tailings of 12,150 tpd of solids and 12,150 tpd of water results from operations. Comparing the tailings to the ore, arsenic is reduced by 86 percent, iron by 72 percent, sulphur by 84 percent, and mercury by 80 percent. The tailings solids contain elements and



	m t p d								
			critical constituents						
Flow No.	Total Solids	Water	C	Sb	As	Fe	S	SiO_2	kg/d Hg
1	13,500	206	29.7	0.41	4.69	243.0	225.8	11,070.0	50.5
2	0	12,232	-	-	-	-	-	-	-
3	1,350	288	28.4	0.27	4.05	175.5	189.0	850.5	40.5
4	12,150	12,150	1.3	0.14	0.64	67.5	36.8	10,219.5	10.0

DWG. ORIGINATOR (CONSULTANT)

CH2M HILL

CINOLA OPERATING CO. APPROVAL:

FIG. NO. 0-19

CONSTITUENT MATERIAL BALANCE
GRINDING AND FLOTATION

CINOLA OPERATING COMPANY LTD.

DRAWN: | DATE: 8-92

CHECKED: | APPROVED:

Fig. 4-12 | REV.

tests are shown in Tables 0-14 and 0-15. The test results indicated that a weak acid producing potential exists. Further, in an acid solution, iron, mercury and arsenic were solubilized up to 1 percent. In order to prevent these occurrences, a pH of 7.5 to 8 is required.

The flotation tailings slurry is discharged at a pH of 8 and the tailings impoundment water is also maintained at this level so dissolution of heavy metals from flotation tailings and/or the formation of acid producing compounds is prevented.

The water discharged at a nominal pH of 8.0 with the flotation tailings contains traces of flotation reagents that are added in extremely small quantities during the flotation process. In the tailings impoundment, the flotation reagents oxidize and have no effect upon water quality in the impoundment area.

As mentioned previously, the pilot mill processed in excess of 5,000 tonnes of ore. The flotation circuit was operated in closed loop with a lined tailings pond, and all flotation discharge water was recycled from the pond. Monthly pond water samples were collected and analyzed by two laboratories to measure water quality over the mill operating cycle. These results are shown in Table 0-16. LC50 static bioassays were performed on the pond water at various periods to show that the water was non-toxic. Results of all bioassays, shown in Table 0-17, indicated 100 percent survival at 100 percent concentration for all tests.

The following is summarized from Table 0-16 and shows the average tailings pond water quality measured during the operation of the pilot mill:

	mg/L
Dissolved solids	1,183
Aluminum	0.44
Antimony	0.39
Arsenic	0.022
Cadmium	0.006
Chromium	0.028
Cobalt	0.038

TABLE 0-14

ACID PRODUCTION POTENTIAL TESTS ON PILOT MILL ORE AND FLOTATION TAILINGS

Sample		Theoretical Acid ($kg\ H_2SO_4$/tonne)	Natural pH 10 g sample + 100 mL H_2O	Acid Consumption ($kg\ H_2SO_4$/tonne)	Potential Acid Producer
Ore	1.88	57.6	3.65	0.5	Yes
Ore	1.83	56.0	3.70	1.0	Yes
Tailings	0.24	7.4	5.10	2.9	Yes
Tailings	0.28	8.6	6.85	3.4	Yes

CONFIRMATION TEST FOR ACID PRODUCTION POTENTIAL

Sample	pH Before Addition of Extra Sample	pH After Addition of 0.5 x Original Weight	pH After Addition of 1.0 x Original Weight	Confirmed Acid Producer
Ore	1.30	1.35	1.40	Yes
Ore	1.40	1.45	1.50	Yes
Tailings	2.30	2.60	2.70	Yes-weak
Tailings	2.02	2.29	2.56	Yes

TABLE 0-15

IRON, ZINC, MERCURY AND ARSENIC LEACHING CHARACTERISTICS OF PILOT MILL TAILINGS SAMPLES[a]

		IRON			ZINC			MERCURY			ARSENIC		
Sample	Leachate pH	Head Solids %	Leachate[b] mg/L	% Extr.	Head Solids ug/g	Leachate mg/L	% Extr.	Head Solids ug/g	Leachate mg/L	% Extr	Head Solids ug/g	Leachate mg/L	% Extr.
Tailings	2.56	2.13	L0.3	L1	20.9	30.7	100	0.79	0.67	0.1	49.7	L0.2	L1

a Leach tests consisted of 30 or 60 g solids mixed with 75 ml solution.

b Solution iron values corrected for added iron in inoculum.

TABLE 0-16

PILOT MILL TAILINGS POND WATER ANALYSES

PARAMETERS (mg/L unless otherwise noted)		SAMPLE DATES: CINOLA LABORATORY May 15	Jun 6/15	Jul 12	Aug 10	Sep 16	Oct 20	Nov 21	Dec 15	B.C. RESEARCH LABORATORY Aug 10	Sep 16	Oct 20	Nov	Dec 18	Dec 19
Suspended Solids		L1.0	85.0	45.0	280.0	100.0	5.0	16.0	—	—	21.0	2	—	—	—
Dissolved Solids		370.0	660.0	1,165.0	1,290.0	1,200.0	1,410.0	1,680.0	—	—	1,022.0	1,280	—	—	—
pH		7.3	7.4	7.4	7.8	8.4	—	—	—	—	—	7.3	—	—	—
Aluminum	Al	—	—	1.0	L0.5	L0.5	L0.5	—	—	L0.5	—	—	—	—	—
Antimony	Sb	0.5	L0.5	—	L0.5	0.7	L0.5	—	—	—	—	—	—	—	—
Arsenic	As	0.02	0.11	4.6	0.019	L0.01	0.015	L0.01	—	L0.015	0.0014	0.0033	L.001	L0.02	L0.02
Cadmium	Cd	0.01	—	—	—	L0.03	—	L0.03	—	L0.05	L0.0002	0.00073	0.0005	0.0005	0.0074
Chromium	Cr	L0.02	0.06	—	—	L0.3	L0.3	—	—	L0.1	0.008	0.012			
Cobalt	Co	—	—	—	—	L0.2	L0.2	—	—	L0.1	0.021	0.043	—	—	—
Copper	Cu	—	—	0.09	—	L0.1	0.32	0.1	—	0.05	0.008	0.032	0.13	0.0039	0.009
Cyanide	CN_T	—	0.015	0.15	0.008	0.48	—	0.44 (.37[a])	—	0.002	—	—	0.235	0.028	0.006
Iron	Fe	0.13	0.12	0.86	0.50	L0.1	0.20	0.34	—	0.44	0.093	0.03	2.13	0.04	0.06
Lead,	Pb	—	—	—	—	L0.5	L0.5	L0.5	—	L0.2	L0.001	0.0011	0.033	L0.001	0.032
Manganese	Mn	0.49	0.3	0.76	0.56	0.31	0.39	—	—	0.59	—	—	—	—	—
Mercury	Hg (ug/l)	0.1	1.0	0.92	0.08	0.05	0.27	0.19	0.20	L0.1	0.13	0.08	0.13	0.16	0.05
Molybdenum	Mo	—	—	—	L0.02	L1.0	L1.0	—	—	L0.2	—	—	—	—	—
Nickel	Ni	—	L0.09	—	—	L0.10	L0.10	—	—	L0.1	0.036	0.076	—	—	—
Selenium	Se	—	—	0.08	—	—	—	—	—	L1.0	—	—	—	—	—
Zinc	Zn	0.02	0.16	0.16	—	0.06	0.66	0.10	—	L0.05	0.032	0.008	0.33	0.004	0.023
Silver	Ag	0.03	—	0.04	—	L0.03	L0.03	—	—	L0.05					

[a]CAN TEST analyses

NOTE: The Cinola laboratory only performed mercury analysis on the Dec 15 sample.
Cyanide and metal analysis during the month of December were performed by B.C. Research
All blanks indicate that analysis was not performed.
L = less than

TABLE 0-17

LC50 STATIC BIOASSAYS

PILOT MILL TAILINGS POND WATER

Sample Date	pH	Conductance mmho/cm	Test Concentration (% V/V)	Percent Survival 96-h	96-h LC50[2] (% V/V)
23/05/81	6.0	590	100	100	100
16/09/81	6.9	780	100	100	100
20/10/81	7.3	1200	100	100	100
14/11/81	6.5	1270	100	100	100
19/12/81	7.0	1100	100	100	100
12/01/82	7.0	950	100	100	100

[a] 96-h LC50 is the 96-h lethal concentration for 50% mortality

	mg/L
Copper	0.053
Cyanide (total)	0.084
Iron	0.428
Lead	0.030
Manganese	0.468
Mercury	0.314 ug/L
Nickel	0.077
Silver	0.025
Zinc	0.094
pH	8.0

All values are below the upper limit of the British Columbia Ministry of Environment pollution control objectives. The data represent the worst case flotation water quality to be discharged from the production facility, because treated effluent from the pilot mill alkaline chlorination circuit also reported to the tailings pond and biased the analysis, especially those of iron and cyanide, to higher concentrations than those which result from flotation tailings alone. Cyanide is not expected in the flotation tailings water, because it is not used in the flotation circuit. However, with the exception of cyanide, the data was used in this report as a basis for final water quality estimates to be discharged to the tailing impoundment area as shown in Section 4.5.6 of the main report.

0.5.3.3 Roasting and Acid Plant

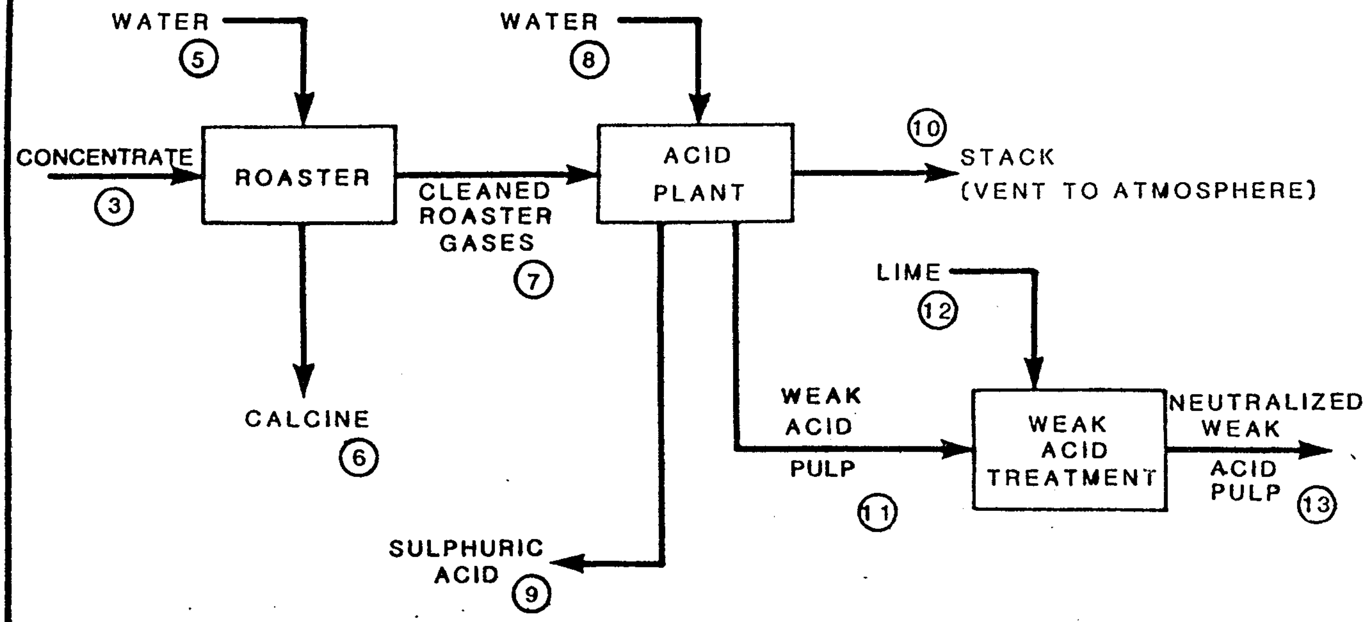
The concentrate is oxidized by using a single-stage, fluid-bed roaster. Because the concentrate contains approximately 2 percent carbon and 14 percent sulphur (primarily as pyrite), the roasting process is exothermic. Proper quantities of air and water are added to the roasting process to maintain a roasting temperature of 700°C. Quantities of arsenic, cadmium, antimony and mercury are volatilized during roasting. Exit gas from the roaster also contain entrained particulates and concentrations of oxygen, nitrogen, carbon dioxide, sulphur dioxide, and water vapor. Roaster gas is treated using cyclones and electrostatic precipitators to remove most of the particulates. The recovered particulates from the cyclone and precipitators are combined with the calcine for further processing. Because the roaster is of the fluid bed type, gas and dust leakage are easily controlled so that no fugitive emissions from this system are expected.

As seen in Figure 0-20, the 1,163 tpd of calcine that are produced from 1,350 tonnes of concentrate, contain over 99.5 percent of the iron and silica, about 50 percent and 34 percent respectively of arsenic and antimony (both as oxides), and about 25 percent of the mercury. Carbon and sulphur contents are reduced by over 98 percent with the balances reporting as oxides to the cleaned roaster gas, which report to the acid plant for further processing. The calcine is then leached to extract the gold.

Cleaned roaster gas enters the acid plant where commercial grade sulphuric acid is produced. Discharges from the acid plant include a gaseous discharge through a stack to the atmosphere, and a neutralized weak acid pulp that is mixed with flotation tailings prior to disposal.

Cleaned roaster gas from the electrostatic precipitators then report to a wet gas scrubbing system where additional gas cleaning and conditioning take place. The gas scrubbing system consists of a humidification tower, a gas cooling tower and a mist eliminator. The cleaned roaster gas that report to the wet gas scrubbing system contain small quantities of particulates, as well as volatilized antimony and arsenic oxides and mercury vapor.

The wet gas scrubbing system removes the particulates and serves to cool the gas stream to a point where gaseous antimony and arsenic precipitate and can be collected. Mercury also condenses due to the cooler gas temperatures. The treated cooled gas stream containing only trace amounts of antimony, arsenic, mercury, etcetera, is processed to commercial grade sulphuric acid. Approximately 99.2 percent of the sulphur entering the acid plant is converted to acid; consequently, the stack emission of sulphur dioxide to the atmosphere is approximately 0.9 tpd. The stack emission also contains 93.3, 1684.5 and 96.1 tpd respectively of oxygen, nitrogen and carbon dioxide.



	mtpd										
			critical constituents								
Flow No	Total Solids	Water	C	Sb	As	Fe	S	SiO_2	$Ca(OH)_2$	$CaSO_4$	kg/d Hg
3	1,350	288	28.4	0.27	4.05	175.5	189.0	850.5	-	-	40.5
5	-	368	-	-	-	-	-	-	-	-	-
6	1,163	0	0.5	0.092	2.16	174.9	3.4	847.6	-	-	10.0
7	4.0	656	27.9	0.178	1.89	0.6	185.6	2.9	-	-	30.5
8	-	43	-	-	-	-	-	-	-	-	-
9	-	145	-	0.00012	0.0014	-	184.2	-	-	-	0.5
10	–	–	27.9	-	-	-	0.4	-	-	-	-
11	6.8	554	-	0.178	1.88	0.6	1.0	2.9	-	-	30.0
12	2.1	8.6	-	-	-	-	-	-	2.8	-	-
13	10.6	562.6	-	0.178	1.88	0.6	1.0	2.9	0.5	4.25	30.0

DWG. ORIGINATOR (CONSULTANT)

CH2M HILL

CINOLA OPERATING CO. APPROVAL:

FIG. NO. 0-20

CONSTITUENT MATERIAL BALANCE ROASTING AND ACID PLANT

CINOLA OPERATING COMPANY LTD.

DRAWN:	DATE: 8-82
CHECKED WWS	APPROVED WWS
Fig. 4-13	REV.

The solids, containing some heavy metals collected from the wet gas scrubbing system, are mixed with other acid plant bleed streams and result in a weak acid pulp that is neutralized in the weak acid treatment system. Lime is added which reacts with the sulphates to produce calcium sulphate. In addition, any free heavy metals leached in the weak acid prior to neutralization are precipitated as hydroxides or arsenates. The neutralized weak acid pulp is discharged at a pH of 10.5 and reports to the tailings sump where the pulp is combined with flotation tailings.

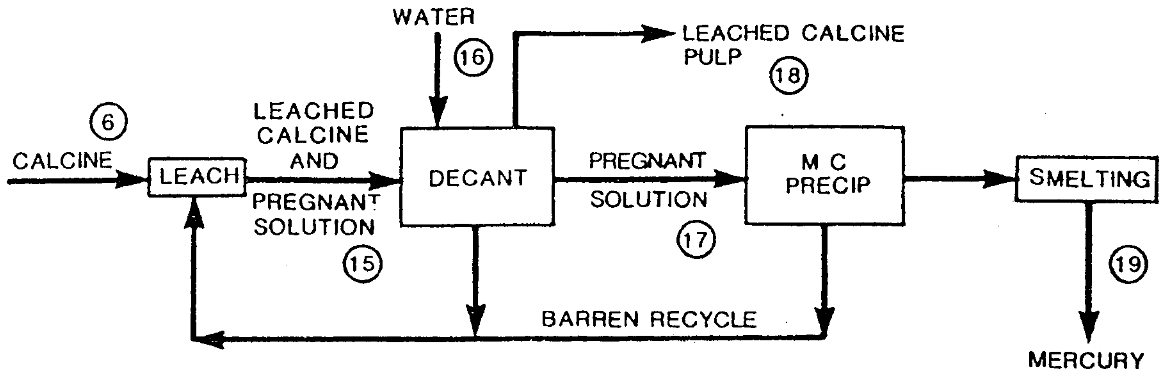
The stability of the heavy metals in the treated weak acid sludge is discussed in Section 4.5.6 of the main report. Figure 0-20 shows the material balances of the environmentally critical constituents for the acid plant.

0.5.3.4 Leaching, Precipitation and Smelting (Areas 6 and 7)

The calcine, produced from the roasting process, is leached to remove the contained gold values. The process utilizes a cyanide solution to dissolve the precious metals from the calcine in a series of agitated tanks. After the leaching process, a separation of the leached calcine from the pregnant solution takes place using a series of counter-current decantation thickeners. A mixture of process water and barren solution is added to the last thickeners as wash solution. The washed leached calcine pulp is then treated to remove the contained cyanide.

During the leach, 88 to 90 percent of the gold and 30 to 40 percent of the mercury are extracted from the calcine. As seen in Figure 0-21, the mass balance of other critical constituents after leaching, such as antimony, arsenic, iron, and sulphur, remain unchanged from the feed calcine. These constituents, as stabilizied oxides, are insoluable in the treated residue.

The pregnant solution containing the gold and mercury is clarified to remove any carry-over solids and the clarified solution is then deaereated to remove dissolved oxygen. The deaerated pregnant solution is treated with additions of small quantities of lead nitrate and zinc dust to precipitate the gold.



			m t p d						
			critical constituents						
								kg/d	
Flow No	Total Solids	Water	Sb	As	Fe	S	SiO_2	CN^-	Hg
6	1,163	–	0.092	2.16	174.9	3.4	847.6	–	10.0
15	1,163	1,163	0.092	2.16	174.9	3.4	847.6	117.2	10.3
16	-	952	-	-	-	-	-	-	-
17	-	3,732	-	-	0.06	0.09	-	247.9	5.08
18	1,163	952	0.092	2.16	174.9	3.4	847.6	43.7	5.94
19	-	-	-	-	-	-	-	-	4.06

FIG. NO. 0-21

DWG. ORIGINATOR (CONSULTANT)

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CINOLA OPERATING CO. APPROVAL:

CONSTITUENT MATERIAL BALANCE LEACHING, DECANTATION, PRECIPITATION AND SMELTING

CINOLA OPERATING COMPANY LTD.

DRAWN: PCCD	DATE: 8-82
CHECKED: WMB	APPROVED: WM
Fig. 4-14	REV.

Mercury also reports to the precipitate. The solution is then pumped through a filter press which separates the gold bearing precipitate from the barren solution. The resulting barren solution from the filter press is recycled back to the leaching and decantation circuits. The only barren solution bleed is that part of the solution that reports with the washed leached calcine in the underflow from the No. 6 CCD thickener.

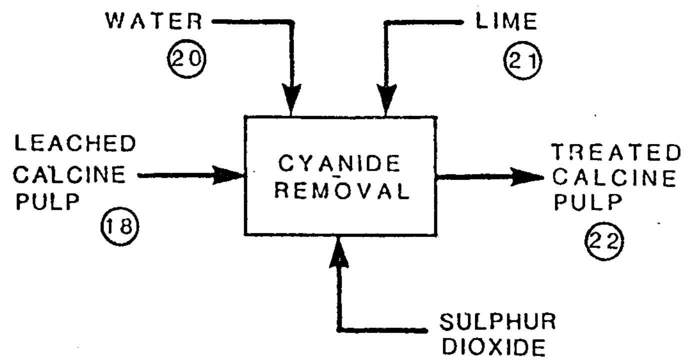
The precipitate is mixed with flux and is then smelted in an electric furnace. Since the precipitate contains mercury which volatilizes during the smelting process, a fume collection system collects the furnace offgas which is treated to remove the mercury. The mercury is collected in flasks and sold. The mercury free gas is further treated in a wet scrubber prior to discharge to the atmosphere. Slag from the furnace is crushed and classified. High gold bearing slag is recycled to the smelting furnace. Low gold bearing slag is returned to the grinding and flotation circuits for reprocessing. The gold (dore) that is produced from the smelting furnace is remelted and cast into bars for shipment and sale.

0.5.3.5 Cyanide Removal

The leached calcine pulp containing barren solution from the decantation circuit is diluted from the thickener underflow pulp density of 55 to 50 percent solids with process water. This dilution permits more efficient cyanide removal. Copper as copper sulphate is added at the dilution point prior to the reactor tanks in which sulphur dioxide, lime and air are added. The sulphur dioxide/air treatment removes the cyanide by oxidation to cyanate (see Figure 0-22).

This method of cyanide removal is preferred over the conventional alkaline chlorination method because it is more efficient.

The treated leached calcine pulp containing less than one mg/L cyanide (as CN^-) is pumped to the tailings sump and combined with flotation tailings and the neutralized weak acid pulp. The combined slurry is pumped to the tailings impoundment area.



	mtpd									
			critical constituents							
								kg/d		
Flow	Total Solids	Water	Sb	As	Fe	S	SiO_2	CN^-	CNO^-	Hg
18	1,163	952	0.092	2.16	174.9	3.4	847.6	43.7	13.3	5.94
20	-	210	-	-	-	-	-	-	-	-
21	0.2	0.8	-	-	-	-	-	-	-	-
22	1,163.2	1,162.8	0.092	2.16	174.9	3.4	847.6	1.2	76.8	5.94

FIG. NO. 0-22

DWG. ORIGINATOR (CONSULTANT)
CH2M HILL
CINOLA OPERATING CO. APPROVAL:

CONSTITUENT MATERIAL BALANCE
CYANIDE REMOVAL

CINOLA OPERATING COMPANY LTD.

DRAWN: | DATE:
CHECKED | APPROVED
Fig. 4-15 | REV.

During the process, a pH control system using slaked lime keeps the reaction pH at 9. In addition to controlling pH, the rate of sulphur dioxide addition is controlled through continuous measurement of the oxidation-reduction potential. The fine dispersion of sulphur dioxide/air in the pulp at pH 9 oxidizes the cyanide.

The sulphuric acid generated is neutralized with lime and metals are hydrolized out of the solution. Iron may form insoluble ferrocyanides.

In summary, the method converts cyanide to cyanate or combines it as insoluble metal complexes. Both products are environmentally acceptable. Since CNO is 1,000 to 1,500 times less toxic than cyanide, a level of 150 mg/L is acceptable. Should small levels of cyanide be dissolved from solids, they are destroyed through natural degradation.

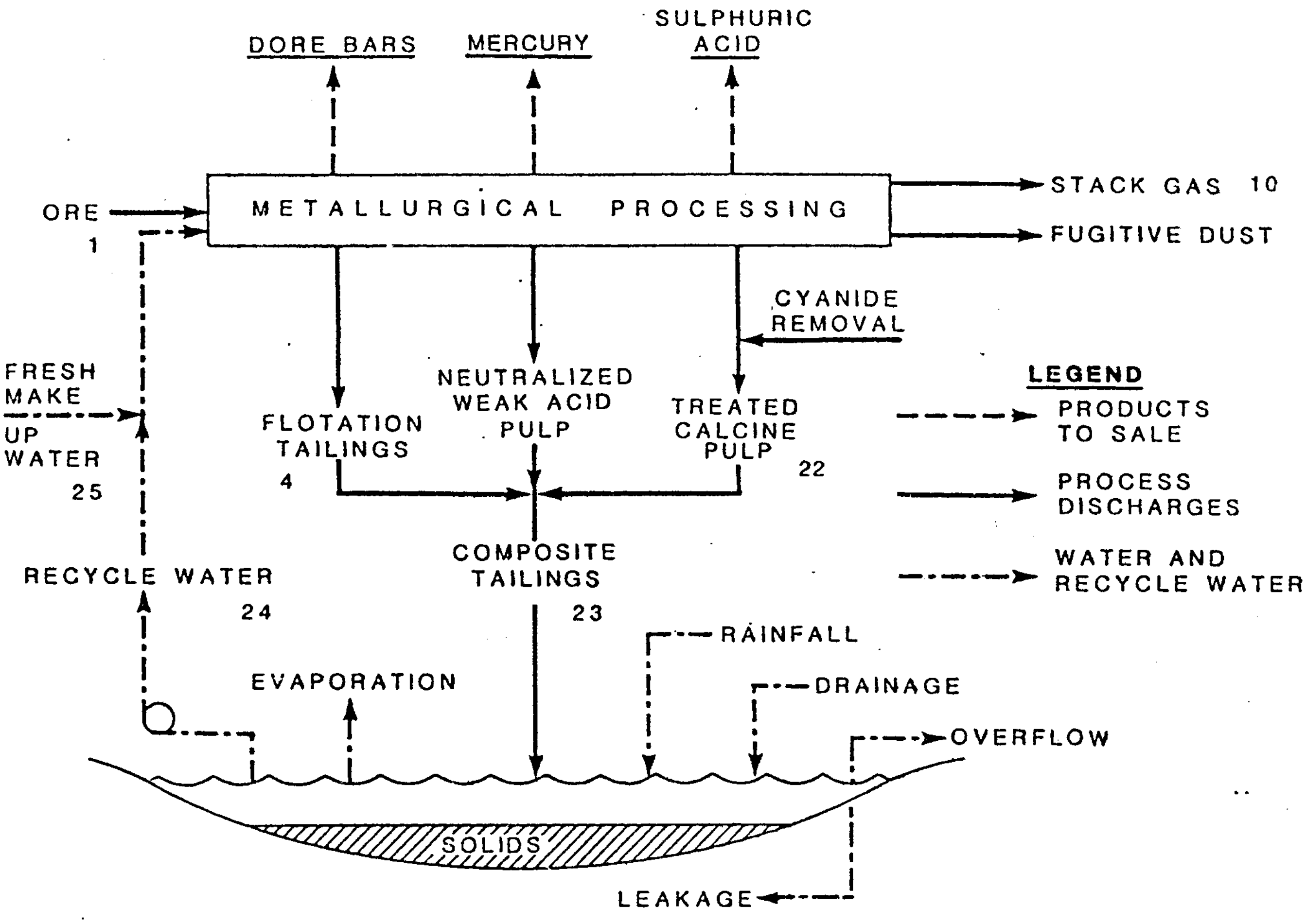
0.5.3.6 Summary of Process Emissions

A summary of environmentally important mass balances from the metallurgical processing facility is shown in Figure 0-23. Saleable products from the facility include dore bars, sulphuric acid and mercury. Atmospheric emissions consist of stack gas from the acid plant and fugitive dusts from the crushing and storage area. Emissions from the acid plant are calculated to be 0.4 tpd sulphur as sulphur dioxide. Total fugitive dust emissions per day are calculated to be approximately 118 kg.

Other major discharges consist of flotation tailings, neutralized weak acid pulp and treated calcine pulp, all of which are combined and pumped to the tailings impoundment area.

The solids in the composite tailings consist of 91.17 percent flotation tailings, 0.08 percent neutralized weak acid pulp and 8.75 percent treated calcine pulp.

Compared to the ore, the flotation tailing solids contain only 14 percent of the arsenic, 28 percent of the iron, 16 percent of the sulphur and 20 percent of the mercury.



mtpd

critical constituents

kg/d

Flow No	Total Solids	Water	C	Sb	As	Fe	S	SiO_2	$Ca(OH)_2$	$CaSO_4$	CN^-	Hg
1	13,500	206	29.7	0.41	4.69	243.0	225.8	11,070	-	-	-	50.5
4	12,150	12,150	1.3	0.14	0.64	67.5	36.8	10,219.5	-	-	-	10.0
10	-	-	27.9	-	-	-	0.4	-	-	-	-	-
13	10.6	562.6	-	0.178	1.88	0.6	1.0	2.9	0.5	4.25	-	30.0
22	1,163.2	1,162.8	-	0.092	2.16	174.9	3.4	847.6	-	-	1.2	5.94
23	13,323.8	13,875.4	1.3	0.41	4.68	243.0	41.2	11,070	0.5	4.25	1.2	45.94
24	-	10,570										
25	-	3,305.4										

FIG. NO. 0-23

DWG. ORIGINATOR (CONSULTANT)

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CINOLA OPERATING CO. APPROVAL:

SUMMARY
CONSTITUENT MATERIAL BALANCE

CINOLA OPERATING COMPANY LTD.

DRAWN:	DATE:
CHECKED	APPROVED
Fig.4-16	REV.

The solids in the neutralized weak acid pulp consist principally of gypsum and silica with minor amounts of iron, arsenic and antimony, all of which are primarily present as stable oxides.

The treated calcine pulp, which represents less than 9 percent of the total solids discharged, contains primarily silica, iron and sulphur as oxides and/or sulphates, and smaller amounts of arsenic and antimony, as oxides. A trace amount of mercury is also present. The liquids in the composite tailings are composed of 81.57 percent flotation tailings, 4.05 percent of neutralized weak acid pulp and 8.38 percent of the treated calcine pulp. The pH of each of these streams is 8.0, 10.5 and 9.0 prespectively, which results in a pH of 8.1 in the composite tailings water. The dissolved concentration of the composite liquor as compared to the British Columbia Ministry of Environment (BCMOE) Pollution Control Objectives is as follows:

Item or Element	Calc. Avg.	BCMOE Pollution Control Objectives Lower Limit	BCMOE Pollution Control Objectives Upper Limit
	mg/L	mg/L	mg/L
Dissolved solids	1,133	2,500	5,000
Aluminum	0.426	0.5	1.0
Antimony	0.383	0.25	1.0
Arsenic	0.102	0.1	1.0
Cadmium	0.046	0.01	0.1
Chromium	0.025	0.05	0.3
Cobalt	0.33	0.5	1.0
Copper	0.088	0.05	0.3
Cyanide	0.084	0.1	0.5
Iron	0.450	0.3	1.0
Lead	0.026	0.05	0.2
Manganese	0.410	0.1	1.0
Nickel	0.067	0.2	1.0
Mercury	0.495	-	5.0
Molybdenum	0.243	0.5	5.0
Silver	0.022	0.05	0.5
Zinc	0.124	0.2	1.0

As can be seen, the majority of elements are below the lower limit and all others, with the exception of iron, are less than one half the allowable range.

The concentrations of metallic pollutants in the tailings impoundment water will be controlled by any of three primary factors: (1) thermodynamic (solubility) limitations; (2) kinetic (rate of dissolution of leaching); (3) or the amount of the pollutant present in the tailings.

In the neutralized weak acid and treated calcine pulps, arsenic, antimony, lead and manganese remain relatively insoluble due to the stabilities (insolubilities) of the hydroxides. Aluminum remains relatively insoluble because it is present in complex alumino-silicate minerals. For all the thermodynamically-limited metals (Al, Sb, As, Cr, Pb, Mn), the final concentration in aqueous phase does not exceed some upper limit well below the pollution control objectives.

For the metals such as Co, Cu, Fe, Hg, Ni, and Zn, whose aqueous concentration is kinetically-limited, some leaching can take place if leach solution strength, temperature and agitation intensity are suitable. Because the tailings impoundment water is slightly basic, since there is no agitation and temperatures are ambient, the probability of leaching is very small.

From this analysis, it can be concluded that tailings water quality can be maintained well within or below the established pollution control objective concentrations.

0.5.4 Metallurgical Processing, Operational Requirements

0.5.4.1 Personnel

The staffing plan for the metallurgical processing facility is based upon the primary crusher station being operated 5 days per week 24 hours per day. The secondary/tertiary crusher station is operated 7 days per week, 16 hours per day, and the concentrate production and concentrate treatment sections are operated 7 days per week, 24 hours per day.

The total personnel projected for the metallurgical processing facility is 141, composed of 21 supervisory and administrative employees and 120 hourly operating and maintenance employees. The personnel requirements for the facility are indicated as follows:

1. Salary - Supervisory and Administrative

Mill Superintendent	1
General Mill Foreman	1
Shift Foreman	8
Maintenance Foreman	1
Instrument Foreman	1
Chief Metal Refiner	1
Mill Metallurgist	1
Metallurgical Engineers	3
Instrument Technicians	3
Mill Clerk	1
Total	21

2. Hourly-Operation

Primary Crusher Operator	3
Primary Cursher Operator Helper	3
Secondary/Tertiary Crusher Operator	3
Secondary/Tertiary Crusher Operator Helper	3
Grinding Operator	4
Grinding Operator Helper	4
Flotation Operator	4
Flotation Operator Helper	4
Filter Operator	4
Roaster Operator	4
Roaster Operator Helper	8
Acid Plant Operator	4
Acid Plant Operator Helper	4

Solution Operator	4
Filter/Clarifier Operator	4
Cyanide Removal Operator	4
Furnace Operator	1
Furnace Operator Helper	1
Labourer	5
Absenteeism and Turnover	8
Utility and Training	4
Janitor	1
Total	88

3. Hourly-Maintenance

Millwright	8
Electrician	8
Apprentice mechanic	8
Fitter	4
Welder	4
Total	32

0.5.4.2 Operating Supplies

The consumables and estimated rates of usage for the metallurgical processing facility are indicated as follows:

Item	Consumption (kg/tonne milled)	Annual Usage (tonnes)
Primary crusher liners	0.008	37.8
Secondary crusher liners	0.013	61.4
Tertiary crusher liners	0.025	118.1
Rod mill liners	0.058	274.1
Ball mill liners	0.207	978.1
Rods	0.500	2,362.5

Balls	2.000	9,450.0
Copper sulphate	0.190	897.8
Potassium amyl xanthate	0.175	826.9
Aerofloat 31	0.175	826.9
Pine oil	0.100	472.5
Flocculant	0.025	118.1
Soda ash	0.20	945.0
Sodium cyanide	0.022	104.0
Lime	0.316	1,493.1
Filter aid	0.016	75.6
Fuel oil (#2 diesel)	0.042	198.5
Silica sand	0.030	141.8
Zinc powder	0.0054	25.5
Lead nitrate	0.0006	2.8
Borax	0.0010	4.7
Niter	0.0005	2.4

TABLE 0-13

PROCESS EQUIPMENT UNIT LIST

Unit Number	Description
1-1	Primary gyratory crusher, 42 x 65
1-2	Hydraulic rock breaker, pedestal mountined, heavy-duty
1-3	Vibrating pan feeder, 183 cm wide x 265 cm long
1-4	Coarse ore stockpile conveyor, 152.4 cm wide, 147 m long
1-5	Not used
1-6	Not used
1-7	Primary crusher station bridge crane, 72.5 tonne capacity
1-8	Coarse ore stockpile reclaim feeder, 127 m wide, 244 cm long
1-9	Stockpile reclaim conveyor, 152.4 cm wide, 152.4 m long
1-10	Not used
1-11	Coarse ore surge bin vibrating feeder, 183 cm wide
1-12	Coarse ore transfer conveyor, 152 cm wide, 54.2 m long
1-13	Coarse ore vibrating screen, 244 cm wide, 610 cm long, double deck
1-14	Secondary cone crusher, 17 x 84
1-15	Intermediate crusher ore conveyor, 106.6 cm wide, 83.8 m long
1-16	Intermediate crushed ore vibrating screen, 300 cm wide, 850 cm long, single deck
1-17	Intermdeiate crushed ore vibrating feeder, 127 cm wide, 244 cm long
1-18	Tertiary crusher feed conveyor, 91.4 cm wide, 72.5 m long
1-19	Teritary cone crusher, 5 x 84
1-20	Fine ore collecting conveyor, 91.4 cm wide, 85.6 m long
1-21	Fine ore bin feed conveyor, 91.4 cm wide, 63.1 m long
1-22	Fine ore bin tripper conveyor, 91.4 cm wide, 67.1 m long
1-23	Secondary/tertiary crushing station bridge crane, 36.5 tonne capacity
1-24	Not used
1-25	Not used
1-26	Primary crusher station sump pump, 3.8 cm vertical
1-27	Secondary/tertiary crusher station sump pump, 3.8 cm vertical
2-1	Fine ore bin reclaim feeders, 61 cm wide, 183 cm long
2-2	Fine ore collecting conveyors
2-3	Rod mill feed conveyors, 60.9 cm wide, 13.7 m long
2-4	Rod mills, 411.5 cm diameter, 640 cm long
2-5	Not used
2-6	Rod mill discharge pump, 6 x 4
2-7	Not used
2-8	Ball mill, 503 cm diameter, 914.5 cm long
2-9	Mill bridge crane, 36.5 tonne capacity

TABLE 0-13 (continued)

Unit Number	Description
2-10	Classifying cyclone feed pump, 12 x 10
2-11	Classifying cyclone, 14 - 25.4 cm diameter, per bank inlcuding feed manifold and over flow/underflow launders
2-12	Classifying cyclone over flow pump, 10 x 8
3-1	Conditioner tank, 2- 488 cm diameter 533 cm high tanks per bank with agitator
3-2	Flotation cell bank, 4- 1500 cu ft cells, 6- 1000 cubic feet cells per bank
3-3	Concentrate thickener feed pump, 8 x 6
3-4	Concentrate thickener, 36.6 m diameter
3-5	Not used
3-6	Tailings thickener feed pump, 12 x 10
3-7	Tailing thickener, 83.8 m diameter
3-8	Scavenger concentrate pump, 6 x 4
3-9	Not used
3-10	Water tower feed pump, 10 x 8
3-11	Not used
3-12	Concentrate thickener underflow pump, 6 x 4
3-13	Vacuum filter, 275 cm diameter, 6 disc
3-14	Concentrate filter cake collection conveyor, 45.7 cm wide, 30.8 m
3-15	Tailings thickener underflow pump, 12 x 10
3-16	Mill air compressor, 10 cu m per minute, 8.5 atmospheres pressure
3-17	Reagent feeders - dry
3-18	Reagent feeders - wet
3-19	Mill sump pump, 3.8 cm vertical
4-1	Concentrate transfer conveyor, 45.7 cm wide, 36.6 m long
4-2	Concentrate stockpile feed conveyor, 45.7 cm wide, 31.1 m long
4-3	Concentrate stockpile reclaim feeder, 45.7 cm wide, 183 cm long
4-4	Concentrate stockpile reclaim conveyor, 45.7 cm wide, 49.5 m long
	Roaster, installed with following equipment unit no's. 4-5 through 4-26
4-5	Repulp tank, with agitator, 3.7 m diameter by 4.0 m high
4-6	Repulper recirculation pump
4-7	Vibrating screen
4-8	Head tank, with agitator
4-9	Roaster slurry feed pump
4-10	Fluid bed roaster
4-11	Roaster flux feed bin and rotary valve
4-12	Roaster air blower
4-13	Primary cyclones, 2.4 m diameter
4-14	Secondary cyclones, 1.5 m diameter

TABLE 0-13 (continued)

Unit Number	Description
4-15	Gas cooling tower, 4.9 m diameter by 18.6 m high
4-16	Electrostatic precipitator, 6.1 m wide by 17.7 m high
4-17	Roaster offgas fan
4-18	Cyclone screw conveyor
4-19	Gas cooler drag conveyor
4-20	Precipitator drag conveyor
4-21	Main drag conveyor
4-22	Gas cooler water pump
4-23	Cooling water circulating pump
4-24	Roster blower lube oil pump
4-25	Roaster burners
4-26	Roaster preheat fan
4-27	Pneumatic conveyor system, 1292 tonne per day capacity
4-28	Weigh belt feeders, 46 cm wide, 200 cm long, 0.5 percent accuracy, variable drive
	Acid plant, installed with unit numbers 5-1 through 5-59
5-1	Humidifying tower
5-2	Cas cooling tower
5-3	Electrostatic mist precipitator
5-4	Drying tower
5-5	Humidifying tower circulation pump
5-6	Weak acid effluent stripping tower
5-7	Weak acid circulating pump tank
5-8	Weak acid effluent pump tank
5-9	Gas cooling tower circulating pump
5-10	Weak acid cooler
5-11	Weak acid effluent pump
5-12	SO_2 Blower
5-13	Cold gas exhanger
5-14	Hot gas exchanger
5-15	Converter
5-16	Hot interpass gas exchanger
5-17	Cold interpass gas exchanger
5-18	Interpass abosorbing tower
5-19	Final absorbing tower
5-20	Process gas exit stack
5-21	Preheater furnace combustion air fan
5-22	Preheater furnace
5-23	Preheater exchanger
5-24	Drying tower pump tank
5-25	Drying tower acid pump
5-26	Drying tower acid cooler
5-27	Cross flow acid stripping tower
5-28	Interpass/final absorbing tower pump tank
5-29	Product acid pump tank

TABLE 0-13 (continued)

Unit Number	Description
5-30	Interpass/final absorbing tower acid pump
5-31	Interpass/final absorbing tower acid cooler
5-32	Product acid cooler
5-33	Product acid pump
5-34	Effluent precipitation tank
5-35	Effluent pump
5-36	SO_2 blower turbine
5-37	Strong acid sampling bay
5-38	Emergency safety showers
5-39	Blower hoist
5-40	Strong acid pump hoist
5-41	SO_2 Blower lube oil pump
5-42	SO_2 Blower auxiliary lube oil pump
5-43	SO_2 Blower lube oil cooler
5-44	SO_2 Blower lube oil heater
5-45	Weak acid sump pump
5-46	Mist precipitator flushing pump
5-47	Mist precipitator insulator purge air heater
5-48	Mist precipitator insulator purge air fan
5-49	Buidling ventilators
5-50	Preheater furnace temporing air fan
5-51	Weak acid effluent stripping tower blower
5-52	Dilution air filter
5-53	SO_2 blower lube oil filter
5-54	Mist precipitator insulator dry air filter
5-55	Vacuum breaker tank
5-56	Mist precipitator flush tank
5-57	Hot water storage tank
5-58	SO_2 blower lube oil console
5-59	Cooling tower
6-1	Calcine repulp tank, 3 m diameter, 3 m high, with 3 hp agitator
6-2	Repulp conditioning tank feed pump
6-3	Conditioning tank, 6.1 m diameter, 6.1 m high, with 15 hp agitator
6-4	Hydrated lime screw feeder, 11.6 m long, variable speed drive, 340 kg/h
6-5	Cyanide flow bin and feeder, with variable speed belt feeder, 15 cm wide
6-6	Leach tank, 7.6 m diameter, 7.6 m high, with 60 hp agitator
6-7	CCD thickener feed pump, 900 L/min
6-8	CCD thickener, 16.8 m diameter
6-9	CCD thickener underflow pump, 1,000 L/min
6-10	Pregnant solution feed pump, 2,700 L/min
6-11	Flocculant screw feeder
6-12	Flocculant mixing tank 2.44 m diamter, 2.44 m high, with 1 hp agitator
6-13	Flocculant feed pump

TABLE 0-13 (continued)

Unit Number	Description
7-1	Clarifier filter feed pump
7-2	Clarifier filter, pressure leaf type, 25 leaves, 83.6 m^2 filter area
7-3	Deaeration tower feed pump, 2,700 L/min
7-4	Deaeration tower vacuum pump, rotary vane, 4.3 mm^3/m at 610 mm Hg vacuum
7-5	Plate and frame filter feed pump, 2,700 L/min
7-6	Plate and frame filter 83.61 square m filtering area, 83.6 m^2 filter area, 53-chamber
7-7	Barren solution return pump, 2,700 L/min
7-8	Mercury recovery system
7-9	Tilting smelting furnace, induction type, 0.07 cubic m (295 kg) capacity
7-10	Remelt furnace, induction type, 68 kg capacity
7-11	Laboratory jaw crusher
7-12	Laboratory roll crusher
7-13	Laboratory shaking table
7-14	Refining exhaust scrubber system
7-15	Scrubber fan
7-16	Scrubber recirculating pump
7-17	Floor sump pump
7-18	Lead nitrate feeder, vibrating hopper, screw feeder
7-19	Precipitation reagent mix tank, 0.9 m diameter, 0.9 m high, with 1/4 hp agitator
7-20	zinc dust feeder, screw type
7-21	Sludge bed clarifier, 9.1 m diameter, 2,700 L/min
7-22	Deaeration tower, 1.8 m diameter by 6.1 m high
8-1	Cyanide removal feed pump, 1,200 L/min
8-2	Cyanide removal tank, 6.1 m diameter, 6.1 m high, with 15 hp agitator
8-3	Tailings transfer pump, 1,200 L/min

0.6 POWER PLANT
DESIGN

0.6 Power Plant Design

The selection of a suitable method of power generation for the Cinola Project involved study of several alternatives. A general capacity of 34 megawatts is required for the mining and milling operation and for domestic demand associated with the project.

Diesel generated power is preferred for the project on the basis of its cost and reliability. Alternatives are hydro-electric, wind power and coal-fired steam boiler systems. These are discussed below.

B.C. Hydro prepared a pre-feasibility overview of the potential for hydro-electric power development on the Queen Charlotte Islands about ten years ago. An up-date seven years later identified potential sites which included Ian Lake and the Ain River, Eden Lake, and Yakoun Lake. Since the salmon fishery of Graham Island is of major local provincial importance, the potential detrimental effects of hydro-electric development on the fishery and the costs associated with pre-development fisheries studies were considered by B.C. Hydro to render the economics of all of the schemes unattractive.

The use of a 35 MW coal-fired stream boiler was also considered. Coal occurs on Graham Island in the Brent Creek - Yakoun Lake area; however, the relatively low grade and the environmental sensitivity of the area rules out this fuel source at the present time. Waste wood is available on Graham Island from logging and mill operations, and present production is estimated to be sufficient to support a 5 MW power plant. As an alternative, coal could be imported to the Island from the Mainland at a very competitive cost compared to fuel oil of the same heat value.

A single unit plant with no standby capability would have lower operating costs than a diesel power plant, but having no standby capability it is unacceptable from an operating standpoint. Furthermore, the capital cost of a coal-fired plant with standby capacity is too high for a project of this size.

The remaining alternative for power generation is a diesel-driven facility. The power plant consists of six 7.5 MW four-stroke diesel generators with five units operating and one on standby. Power output is at 13.8 kv, 3 phase and 60 Hertz with a power factor of 0.8.

The diesel engines run at 600 revolutions per minute using marine diesel oil fuel. Exhaust is connected to a system of waste heat boilers; cooling water passes through heat exhangers for waste heat recovery, which provides all space and process heat requirements.

Air intake pressure for the engines is boosted by exhaust-driven turbochargers. Silencers and lagging for noise attenuation are provided for the air intake and exhaust systems. Combustion air is cooled.

The generator windings are of the brushless self-excitation type in open drip-proof enclosures. Generator voltage output is automatically regulated. Equipment for transformation of the 13.8 kv output to lower voltages is located in the metallurgical processing plant motor control center. The plant operates 7 days per week, 24 hours per day. A planned maintenance schedule ensures regular rotation of units for standby duty.

The diesel generators are housed in a pre-engineered building fitted with an 18-tonne capacity crane. The building is equipped with overhead sliding doors, lighting, and insulation , and includes a control room, office and washrooms.

0.7 SUPPORT FACILITIES

0.7 Support Facilities

0.7.1 Access Road and Site Preparation

The mine site is presently served by a route which follows a system of logging haul roads, mostly along the Queen Charlotte main line road, which provides access to tree farm licenses held by MacMillan Bloedel Ltd. Because this road would not adequately service the mine, a study of alternative routes (along with upgrading portions of the existing road) was undertaken.

The route selected to serve the mine is 23.3 km in length and offers the following advantages:

1. Provides access to the tailings disposal area
2. Is the safest route
3. Affords lower capital and operating costs
4. Ensures the shortest construction period
5. Ensures minimal disturbance to the environment
6. Provides considerable forestry benefit and opportunity for cost sharing with MacMillan Bloedel

Plans call for new construction of two sections of access road with the following parameters:

1. Plant site to tailings pond -- Length 6.5 km, width 13.7 m (to conform to the Department of Mines regulation requiring a haulage road to be three times the width of the haulage vehicle).

2. Tailings pond to main road -- Length 5.3 km, width 7.6 m and routed to join the Port Clements-Juskatla main road at the Ferguson Bay turn-off, thereby ensuring that mine traffic does not conflict with logging traffic into and out of Ferguson Bay.

   The route to Port Clements from the mine continues along the main road for another 11.5 km, making the total journey 23.3 km in length.

Pipelines for conveying tailings to the pond, reclaimed water from the pond, fuel oil from erguson Bay and acid to Ferguson Bay follow a right-of-way along the route.

The surveyed right-of-way does not exceed 8 percent in grade. Total ballast requirements for the 11.8 km of new road are 255,500 m^3. Bridge crossings are single-lane and constructed of local timber to support 100 tonne loads. The surface of the ballast road is dressed with minus 2 cm crushed stone. On the advice of local logging operators, these roads are unpaved to ensure safer winter driving conditions.

The plant site is located immediately north-west of the open pit. This area was recently logged, and is covered with slash left by the logging operations.

The near surface soils are formed of fluvio-glacial materials consisting mainly of dense silty gravel and hard gravely clay. No special designs are required for near surface foundation loads up to 40 kilopascals on these undisturbed materials at their in-situ density.

Machine cutting of slopes to create a level site produces a poor quality of material for compaction, primarily because of the high rainfall in the area. All heavy buildings or structures sensitive to settlement are sited where no filling is to take place.

Clearing of one-half of the total areas of the open pit, waste dump and tailings pond occurs in the preproduction phase of the operation (Years -2 and -1). The remainder is cleared in the initial years of production as required.

0.7.2 Construction Camp and Housing

Construction of the project occurs over a period of 24 months. The peak complement of the construction workforce is 300 persons in the year immediately preceding commissioning of the project. A camp is constructed as a permanent feature to accommodate this workforce. It also serves as accommodation for permanent single-status employees during the operating phase, even though there is presently some uncertainty as to the eventual demand for this type of housing.

The construction camp is made up of seven units. Each unit contains 42 single occupancy rooms and is intended to meet the requirements of the British Columbia and Yukon Building Trades Council in all respects.

The kitchen and dining facilities have a minimum seating capacity of 225 persons. The recreational area has a minimum area of 223 m^2 or 0.75 m^2 per person. All facilities in the complex are connected by covered walkways.

The total construction and operating workforce peaks at 450 persons in the 4th quarter of Year -1. At this point in time there are 248 permanent operating employees, 150 of which are housed in newly constructed housing which complements the 300-person construction camp.

The permanent labour force peaks at 350 persons in the 2nd quarter of Year 1. It is assumed that 40 percent of this labour force is single status or living in existing housing on the island. The housing construction program provides 210 houses for employees, and construction is planned to keep pace with the increase in permanent workers.

0.7.3 Buildings and Facilities

Buildings and support facilities consist of the maintenance shops, warehouse, administration and engineering office building, changehouse and dry, assay laboratory, the explosive and ammonium nitrate storage building, and facilities for plant site sanitary waste disposal.

A 3,000 m^2 pre-engineered building serves as a combined tool storage/maintenance shop and as an office building. It includes eleven bays, each 9.1 m wide, 18 m long and 12 m high, capable of accommodating the mine haulage trucks. Six bays are dedicated for the repair and maintenance of mining equipment, two bays for heavy welding repair, two bays for tire replacement and lubrication, and one bay for small vehicle repair.

Space is allocated for a machine shop, electrical repair shop, tool crib, lunch room, and offices for supervisory and maintenance records staff.

The warehouse is located adjacent to the maintenance shops. The building is also pre-engineered and includes an office and service counter, heating space and washrooms. Satellite warehousing, located in the maintenance shops, metallurgical processing plant and the assay laboratory, is supplied from the main warehouse.

A single storey, 720 m^2 pre-engineered structure is used to house project staff. The building is located at the entrance to the plant site. Mine management, mine supervision, mine engineering, geological, environmental, accounting, purchasing, personnel, first-aid and training fuctions are located in this office building.

Locker space, shower and changing facilities for 130 people are housed in an insulated pre-engineered structure having a floor area of 720 m^2. Locker space for street clothes is separated from the work clothes drying area by a shower and toilet enclosure. A roof height of 6.1 m accommodates the suspended clothes hooks required for drying clothes. The building is located next to the office.

The preparation of samples from the mine and metallurgical processing plant and assaying of samples by atomic absorption analysis and fire assay methods occurs in the assay laboratory. A wet chemical laboratory is also provided for support analysis. Together, these facilities are capable of generating complete analyses for all mine and environmental control samples. A pre-engineered structure is used having an area of 544 m^2.

A packaged water gel slurry, classified as a high explosive, is used as a blasting agent in the mining operation. A magazine with a floor area of 260 m^2, and a minimum ceiling height of 2.5 m is required to store two month's supply of this explosive. The magazine is located, subject to approval by the regulatory authorities, in an isolated area northwest of the open pit. A separate magazine is planned for storing detonators, detonating relays and primadets. This facility has a floor area of 2 m^2 and a height of 1 m.

Ammonium nitrate is not an explosive and need not be stored in a magazine. It becomes an explosive when mixed with fuel oil during the charging operation. A building with a floor space of 435 m^2 and a minimum ceiling height of 3 m houses this material.

Sanitary sewage is collected through a system of sanitary sewers and directed to a septic tank. The septic tank and field is located in an area to the east of the plant site which provides good drainage. Effluent from the assay laboratory is directed through a lime neutralizing pit prior to its discharge into the septic field. Solid waste, such as domestic garbage, is disposed of by burial in the mine waste dump area.

0.8 PROJECT COSTS

0.8 Project Costs

The following discussion of costs is summarized from Section 12 of the main report.

Data and discussion regarding project economics is complete within this report. No further information on this subject is included in the main report.

0.8.1 Capital Costs

The capital costs schedule for the project, which is based on 1982 Canadian dollars, is shown in Table 0.18. During the initial two-year preproduction development period, \$218 MM of capital is invested into creation of the mine and mill. (See note.) Replacement and additional capital investment throughout the operating years amount to nearly \$9 MM. About \$15 MM is returned at the close of the project through salvage and retention of working capital. Thus, total net capital investment amounts to \$212.173 MM.

Approximately 65 percent of total net capital investment is in the mill, power plant, tailings pond and engineering and construction management. Mining equipment accounts for 11 percent. Access road and site preparation, preproduction mine development, mill start-up, and initial project administration add up to 12 percent, but this cost includes all expenses necessary to process an initial 400,000 tonnes of 0.0889 oz/tonne ore in Year -1. All other capital costs amount to 12 percent, one-quarter of which is British Columbia sales tax. (Other sales taxes or duties are included, but are not isolated.)

Note that the cost of treating 400,000 tonnes of ore in preproduction Year -1 is treated as initial capital in the following economic analysis. The net cash flow obtained from this preproduction ore, however, amounts to \$17.5 MM, and directly offsets a portion of initial capital cost.

Note: MM = million

TABLE 0-18

CINOLA OPERATING COMPANY LTD. QUEEN CHARLOTTE GOLD PROJECT
CAPITAL COST EXPENDITURE SCHEDULE
January 1982 Canadian Dollars (x000)

Item	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Exploration, Project Feasibility											
1. Engineering and Construction Management	6,010	8,894									$14,904
2. Access Road and Site Preparation	1,862	621									2,483
3. Construction Camp and Employee Housing	4,592	4,592									9,184
4. Mining Equipment	10,209	8,910		186	1,702	1,195	1,163				23,365
5. Metallurgical Processing	32,140	49,140									81,280
6. Power Plant	10,920	16,380									27,300
7. Ancillary Building, Pipelines, Oil/Diesel Storage	7,225	1,838									9,063
8. Preproduction Development	1,027	9,709									10,736
9. Tailings and Water Management		13,765	5	529	4	33	4	86			14,426
10. Project Administration	2,500	4,014									6,514
11. Start-up Expenditures		5,526									5,526
12. Inventory and Working Capital; Salvage; Reclamation		11,226								(11,124)	102
13. Taxes, British Columbia Sales Tax	2,365	4,538	17	30	119	90	87	23	17	4	7,290
TOTAL: ANNUAL	78,850	139,153	22	745	1,825	1,318	1,254	109	17	(11,120)	
TOTAL: CUMULATIVE		218,003									212,173

0.8.2 Operating Costs

Operating costs for the project are given in summary form in Table 0-19. Mining, milling, general and administrative charges are expressed in 1982 Canadian dollars.

Average unit production costs over the eight operating years (33,916,000 tonnes milled) are analyzed below:

Function	Cost/tonne	%
Mining	$2.78	21
Milling	9.11	70
General Administrative	1.08	9
	$12.97	100

On the basis of input factors, the average cost can be characterized as follows:

Factor	Cost/tonne	%
Labor	$2.78	21
Material & supplies	6.78	52
Power	2.92	23
B.C. sales tax	0.49	4
	$12.97	100

TABLE 0-19

CINOLA OPERATING COMPANY, LTD.

QUEEN CHARLOTTE GOLD PROJECT

ANNUAL OPERATING COSTS

January 1982 Canadian Dollars (X000)

Function	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mine Production:									
Labor	6,494	5,965	5,847	5,112	4,635	4,048	4,014	577	36,692
Material & Supplies	11,690	10,130	9,445	7,670	6,904	5,413	5,584	873	57,709
Total	18,184	16,095	14,292	12,782	11,539	9,461	9,598	1,450	94,401
Metallurgical Processing:									
Labor	5,658	5,658	5,658	5,658	5,658	5,658	5,658	1,007	40,613
Material & Supplies Power, Taxes	37,402	37,402	37,402	37,402	37,402	37,402	37,402	6,657	268,471
Total	43,060	43,060	43,060	43,060	43,060	43,060	43,060	7,664	309,084
General & Administrative:									
Labor	2,832	2,732	2,722	1,989	1,926	1,910	1,909	898	16,918
Material & Supplies, Taxes	3,473	3,274	3,222	2,338	2,226	2,120	2,129	772	19,554
Total	6,305	6,006	5,944	4,327	4,152	4,030	4,038	1,670	36,472
Total Operating Costs:									
$/Year	67,549	65,161	64,296	60,169	58,751	56,551	56,696	10,784	439,957
$/Tonne	14.30	13.79	13.61	12.73	12.43	11.97	12.00	12.82	12.97
Taxes: British Columbia Sales Tax Included in G & A and Metallurgical Processing	2,586	2,487	2,445	2,294	2,245	2,155	2,165	400	16,777

0.9 ECONOMIC ANALYSIS

0.9 Economic Analysis

The June 1980 prefeasibility study prepared by Wright Engineers Ltd. examined project profitability by using a computer program that was capable of calculating Federal and Provincial taxes, determining and internal rate of return, and calculating net present value at various discount rates. With the aid of this program, Wright provided values for a Base Case and nine variations.

The assumptions used to construct Wrights' Base Case are listed on page 12-3 of the June 1980 study, and are included here in Table 0-20 for ease of reference.

In 1981, a copyrighted computer program known as the Interactive Financial Planning System was used to create a computer routine that produces results almost identical to Wright Engineers'. During the months of Final Feasibility Study evolution (June 1981 to September 1982), the program was used to make several profitability analyses and various sensitivity studies to help guide selection of mine production rate, define mineable reserves and to select the Base Case metallurgical flowsheet.

0.9.1 Comparison of Prefeasibility and Final Feasibility Values

As a result of thousands of additional manhours of investigation, a new September 1982 Base Case, far more documented than the June 1980 version, is presented and discussed in this section. In order to provide an immediate and readily available link to the earlier Wright work, the new Base Case parameters also are shown in Table 0-20.

The data on Table 0-20 offer the following comparisons between the Prefeasibility and Final Feasibility studies of June 1980 and September 1982:

1. Mineable reserves are about 6 percent higher. However, cut-off grade also is increased from 0.026 oz/tonne (1980) to 0.033 oz/tonne (1982).

TABLE 0-20

COMPARISON OF BASE CASE STUDIES

	June 1980	September 1982
Study Name	Wright Prefeasibility	C.O.C.L. Final Feasibility
Case Designation	Base Case 1.0	Base Case
Reserves (MM tonnes)	32.450	34.316
Grade (Tr oz/tonne)		
Gold	0.0606	0.060
Silver	0.0606	0.060
Recovery (percent)		
Gold	87	71
Silver	53	50
Mineable Gold (Tr oz)	1,967,000	2,059,000
Recoverable Gold (Tr oz)	1,711,000	1,462,000
Milling Rate (MM tonnes/yr)	3.175	4.725
(tonnes/day)	9,072	13,500
Capital Investment ($MM)		
Initial Capital	165.6 (1980)	218 (1982)
Preproduction NCF Credit	(0.0)	(17.3)
Net Deferred Capital	(3.0)	(6)
Total	162.6	194.7
Operating Costs ($/tonne milled)		
Mining	2.16	2.78
Milling	6.89	9.11
Gen & Admin	1.49	1.08
Total	10.54	12.97
Prices		
Gold ($US/Tr oz)	500	500
Silver ($US/Tr oz)	12	10
Sulfuric Acid ($/tonne)		0
Exchange Rate ($CDN/$US)	1.15	1.20
Escalation	None	None
Equity Financing	100%	100%
AFTER-TAX PROFITABILITY		
Project IRR	23.35%	19.6%
Project NPV (10%)	$85.6 MM	$51.0 MM

(Note that cut-off grade is based on preliminary operating costs and prices different from those established for the Base Case. However, it is not considered useful to revise cut-off grade and to make a new determination of mineable reserve values at this time. See section 3.1.2.3, p. 3-9.)

2. The average grade of mineable reserves is about one percent lower.

3. Average mill recovery is reduced from 87 to 71 percent -- an 18.4 percent decline. Thus, although total mineable gold is about 5 percent greater, recoverable gold is reduced by 14.6 percent.

4. Initial capital increased by $52.4 MM; almost 32 percent. (See note.)

5. Operating costs increased by $2.34/tonne, or 22.2 percent.

6. Gold price is projected at the same level after experiencing a recent reversal in a long down cycle. Silver price is projected at $2/oz less.

7. After-tax Internal-Rate-of-Return (IRR) is about 3.7 percentage points lower and Net Present Value (10%) is diminished, primarily by the increase in early capital investment.

One might immediately wonder how the adverse effects of substantially lower recovery and higher capital/operating costs are mitigated to maintain a large measure of the profitability projected in the preliminary study. The answer resides primarily in two factors:

1. The annual ore production rate is nearly 50 percent higher in the final study. The rate is increased from 3.175 MM tonnes/year to 4.725 MM tonnes/year, and mine life is decreased from 10+ to 7+ years.

Note: MM = million

2. The ore reserve definition in the Final Study identifies higher-grade lenses which can be mined in the earlier years of the project; in fact, 0.4 MM tonnes of 0.0889 oz/tonne mill feed is scheduled for Year -1 prior to the full production period.

The cumulative result of these two factors is that far more gold is produced during the period Year -1 through Year 2 of project life (574,500 Tr oz) in the Final Study than in the Prefeasibility study (313,900 Tr oz).

Furthermore, all of the recoverable gold, 1,462,000 oz is produced in 7+ project years, in contrast to a production of about 1,181,000 oz during the same period in the June 1980 scheme.

Sensitivity analyses conducted during the course of the Final Study indicate clearly that capital and operating costs at this project do not increase in proportion to gold production. Therefore, increased gold revenue during the first few project years, coupled with a reduction in cost per oz of gold produced -- due to both economy of scale and production of higher-grade lenses -- combine to recover all initial capital (excluding interest charges) and yield a positive after-tax cash flow a short time after the end of Year 2. In the earlier study, an equivalent return is not realized until nearly one full year later. Thus, the advantage of earlier net cash flow offsets a large portion of the adverse effect of the lower recovery mentioned above, and maintains most of the originally forecast level of profitability.

Additionally, an increase in the Canadian/U.S. dollar exchange rate increases project revenue in terms of Canadian dolllars for any given U.S. gold price. For the Final Study, a rate of 1.20 is considered to be reasonable in view of trends established during the past several months. Increasing the exchange rate from 1.15 to 1.20 increases Base Case after-tax IRR by about 2.9 percent.

The foregoing analysis immediately characterizes the economic nature of this project. Quite likely, a production rate of 4.725 MM tonnes/year is the maximum appropriate for this property. This is true not so much relative to access or space, but rather in terms of limited project life. Therefore, if the price of gold is below that forecast in the Base Case, either recovery, costs, ore grade or exchange rate must be improved to maintain profitability at the level of the Base Case.

However, if gold price increases above the Base Case levels, options are created to either (1) increase profitability or (2) reduce the size of the operation -- thus decreasing capital requirements and extending mine life. The latter course of action would not be the most profitable in terms of net present value, but it could offer other advantages while maintaining an adequate IRR.

0.9.2 Discussion of the Base Case

The Final Feasibility Study Base Case is defined in detail by several computer-generated tables.

Table 0-21 (two pages) contains all pertinent operating and cost data; Table 0-22 (two pages) contains those values needed to derive net cash flow for each project year. Federal and Provincial tax calculations which are summarized on Table 0-22. However, the individual tax schedules are omitted from this summary.

The following items are worth consideration on Table 0-21:

1. Waste mined -- includes both waste and any low-grade ore that is moved to a waste dump, or to a stockpile. Annual quantities are obtained from Table 0-10, Annual Mine Production Schedule, (page 0-42).

2. Waste-to-ore ratio -- is derived by adding mill feed, waste and low-grade ore and dividing by mill feed.

3. Ore mined -- equals mill feed. The value is obtained from Table 0-10.

4. Gold grades -- are determined by the specific mine plan used for the Base Case. Annual values come from Table 0-10.

TABLE 0-21 BASE CASE

PRODUCTION STATISTICS AND OPERATING PROFIT (M$)

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
WASTE MINED (M TONNES)	0	0	4421	13509	10981	8500	6176	5249	2221	2207
WASTE TO ORE RATIO	0	0	12.05	3.859	3.324	2.799	2.307	2.111	1.470	1.467
ORE MINED (M TONNES)	0	0	400	4725	4725	4725	4725	4725	4725	4725
GOLD GRADE (OZ PER TONNE)	0	0	.0889	.0782	.0717	.0513	.0529	.0514	.0496	.0567
SILVER GRADE (OZ PER TONNE)	0	0	.0889	.0782	.0717	.0513	.0529	.0514	.0496	.0567
GOLD RECOVERY (PERCENT)	0	0	.8100	.7800	.7600	.6900	.7000	.6900	.6800	.7100
SILVER RECOVERY (PERCENT)	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000
GOLD PRICE ($ US PER OZ)	0	500	500	500	500	500	500	500	500	500
SILVER PRICE ($ US PER OZ)	0	10	10	10	10	10	10	10	10	10
ACID PRICE (NET CA$ PER TONNE)	0	0	0	0	0	0	0	0	0	0
EXCHANGE RATE ($CAND VS $US)	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200
GOLD PRODUCTION (M OZS)	0	0	28.80	288.2	257.5	167.3	175.0	167.6	159.4	190.2
SILVER PRODUCTION (M OZS)	0	0	17.78	184.7	169.4	121.2	125.0	121.4	117.2	134.0
ACID PRODUCTION (M TONNES)	0	0	0	212	212	212	212	212	212	212
NET GOLD REVENUE	0	0	17282	172924	154485	100350	104980	100546	95619	114129
NET SILVER REVENUE	0	0	213.4	2217	2033	1454	1500	1457	1406	1607
ACID REVENUE	0	0	0	0	0	0	0	0	0	0
TOTAL REVENUE	0	0	17496	175141	156518	101805	106480	102003	97025	115736
-MINING COSTS	0	0	0	18185	16095	15292	12784	11549	9464	9599
-MILLING COSTS	0	0	0	43060	43060	43060	43060	43060	43060	43060
-GENERAL & ADMININSTRATION	0	0	0	6303	6005	5944	4328	4153	4030	4040
TOTAL OPERATING COSTS	0	0	0	67548	65161	64296	60172	58754	56554	56699
PRE-TAX OPERATING PROFIT	0	0	17496	107593	91357	37509	46308	43250	40471	59037

INITIAL AND REPLACEMENT CAPITAL COSTS

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
ENGINEERING AND CONST MGT	0	6010	3894	0	0	0	0	0	0	0
ACCESS ROAD AND SITE PREP	0	1862	621	0	0	0	0	0	0	0
CONST CAMP AND EMPL HOUSING	0	4592	4592	0	0	0	0	0	0	0
MINING EQUIPMENT	0	10209	3910	0	186	1702	1195	1163	0	0
MILL, POWER PLANT AND TAILINGS	0	43060	79235	5	529	4	33	4	96	0
ANC. BLDGS, PIPELINES, FUEL STORAGE	0	7225	1838	0	0	0	0	0	0	0
PREPROD DEVELOPMENT	0	1027	9709	0	0	0	0	0	0	0
PROJECT ADMINISTRATION	0	2500	4014	0	0	0	0	0	0	0
START-UP EXPENSES	0	0	5526	0	0	0	0	0	0	0
INVENTORY, WORK CAP, SALV, RECLAM	0	0	11226	0	0	0	0	0	0	0
BRIT COLUMBIA SALES TAX	0	2365	4538	17	30	119	90	87	23	17
TOTAL PROJECT CAPITAL COSTS	0	78850	139153	22	745	1825	1318	1254	109	17
BANK LOAN DRAW	0	0	0	0	0	0	0	0	0	0
+ PRE-PRODUCTION INTEREST	0	0	0	0	0	0	0	0	0	0
+ PRODUCTION INTEREST	0	0	0	0	0	0	0	0	0	0
- REPAYMENTS	0	0	0	0	0	0	0	0	0	0
YEAR-END DEBT	0	0	0	0	0	0	0	0	0	0

TABLE 0-21 BASE CASE

PRODUCTION STATISTICS AND OPERATING PROFIT (M$)

	1992	1993	1994	1995	TOTAL
WASTE MINED (M TONNES)	199.2	0	0	0	53453
WASTE TO ORE RATIO	1.225	0	0	0	
ORE MINED (M TONNES)	841	0	0	0	34316
GOLD GRADE (OZ PER TONNE)	.0500	0	0	0	
SILVER GRADE (OZ PER TONNE)	.0500	0	0	0	
GOLD RECOVERY (PERCENT)	.6800	0	0	0	
SILVER RECOVERY (PERCENT)	.5000	.5000	.5000	.5000	
GOLD PRICE ($ US PER OZ)	500	500	500	500	
SILVER PRICE ($ US PER OZ)	10	10	10	10	
ACID PRICE (NET CA$ PER TONNE)	0	0	0	0	
EXCHANGE RATE ($CAND VS $US)	1.200	1.200	1.200	1.200	
GOLD PRODUCTION (M OZS)	28.59	0	0	0	1462
SILVER PRODUCTION (M OZS)	21.03	0	0	0	1012
ACID PRODUCTION (M TONNES)	0	0	0	0	
NET GOLD REVENUE	17156	0	0	0	877471
NET SILVER REVENUE	252.3	0	0	0	12140
ACID REVENUE	0	0	0	0	
TOTAL REVENUE	17409	0	0	0	889611
-MINING COSTS	1450	0	0	0	94410
-MILLING COSTS	7664	0	0	0	309083
-GENERAL & ADMININSTRATION	1670	0	0	0	36475
TOTAL OPERATING COSTS	10785	0	0	0	439968
PRE-TAX OPERATING PROFIT	6624	0	0	0	449644

INITIAL AND REPLACEMENT CAPITAL COSTS

	1992	1993	1994	1995	TOTAL
ENGINEERING AND CONST MGT	0	0	0	0	14904
ACCESS ROAD AND SITE PREP	0	0	0	0	2483
CONST CAMP AND EMPL HOUSING	0	0	0	0	9134
MINING EQUIPMENT	0	0	0	0	23365
MILL, POWER PLANT AND TAILINGS	102	0	0	0	123108
ANC. BLDGS, PIPELINES, FUEL STORAGE	0	0	0	0	9063
PREPROD DEVELOPMENT	0	0	0	0	10736
PROJECT ADMINISTRATION	0	0	0	0	6514
START-UP EXPENSES	0	0	0	0	5526
INVENTORY, WORK CAP, SALV, RECLAM	-11226	0	0	0	0
BRIT COLUMBIA SALES TAX	4	0	0	0	7290
TOTAL PROJECT CAPITAL COSTS	-11120	0	0	0	212173
BANK LOAN DRAW	0	0	0	0	0
+ PRE-PRODUCTION INTEREST	0	0	0	0	0
+ PRODUCTION INTEREST	0	0	0	0	0
- REPAYMENTS	0	0	0	0	0
YEAR-END DEBT	0	0	0	0	0

5. Silver grades -- are assumed to equal gold grades. Although this correlation is not verified by detailed analysis, it is considered to be a very conservative assessment of the ore body's silver content. Section 2.1.3.4 states: "The ratio of gold to silver varies widely. Above grades of 0.07 oz/tonne, the average ratio of gold to silver is about 1:1, decreasing to as low as 1:5 at lower gold grades". In other words, silver content might be two or more times that of gold.

6. Gold recoveries -- are directly related to specific mill design and to mill feed grade. The latter correlation is illustrated in Figure 4-10, "Total Gold Recovery As A Function of Ore Feed Grade". Recovery increases with increasing mill feed grade. For the range of annual mill feed grades defined in the Base Case (0.0500 to 0.0889 oz/tonne), average net gold recovery ranges from 68 to 81 percent. A weighted average is 71 percent.

7. Silver recovery -- is assumed to be 50 percent. In reality it might be as high as gold recovery. However, silver recovery is not verified in detailed analysis, and this reduction is believed to offer an adequate contingency to compensate for lack of data.

8. Gold price -- is established by noting that two years is required to construct the mine and mill. During that period, gold is assumed to increase in value to $US 500/oz (in 1982 dollars) from its current market level. Many complex projections of gold price trends could be prepared, but would be characterized by a considerable measure of uncertainty. It is believed that a $US 500/oz gold price is reasonable for a constant-dollar Base Case, especially in view of gold's renewed market strength.

9. Silver price -- is assumed to be $US 10/oz (in 1982 dollars). The only substantiation for this value is the conviction that silver is significantly underpriced in current recessionary markets.

10. Acid price -- is a net price; that is, revenue less cost of transportation. Acid production charges are included in milling cost. The minimal level acid price projected for the Base Case is assumed to be just high enough to pay for transportation costs. In reality, a differential of $10 to $20/tonne might be realized with market price improvements.

11. Exchange rate -- is the ratio of Canadian dollars received in exchange for a U.S. dollar. A ratio of 1.20 is used for the Base Case. Total Canadian-dollar revenue from gold and silver is directly proportional to the exchange rate.

12. Gold and silver production -- equals ore mined x grade x recovery.

13. Acid production -- of 212,000 tpy is based on the metallurgical balance of the specific flowsheet used for the Base Case. This flowsheet includes crushing, grinding, flotation, roasting and leaching. Sulfuric acid is produced from roaster offgas.

14. Net gold and net silver revenue -- equals production x price.

15. Acid revenue -- equals production x net price. For the Base Case, this quantity is zero.

16. Mining costs -- are the annual total charges for mining, transportation and final disposition of ore, low-grade ore and waste. The average cost/tonne of ore milled is shown in Table 0-20.

17. Milling costs -- include all expenses for ore processing, power generation, tailings disposal and process waste treatment.

18. General and administration -- covers all expenses for overhead salaries, services and materials, including those of an off-property head office. The average cost/tonne of ore milled is shown in Table 0-20.

19. Initial and replacement capital costs -- are based on data obtained from each appropriate section of the feasibility report which are summarized in Section 0.8, "Project Costs".

20. Bank loan draw -- includes lines that are available to examine the effects of debt financing. However, the Base Case assumes 100 percent equity financing; no debt funds are used in the derivation of IRR or NPV figures.

The following comments apply to Table 0-22:

1. Gross revenue and pre-tax operating profit -- are transferred from Table 0-21.

2. Federal income, B.C. income and Provincial mining taxes -- result from calculations made by the IFPS computer program. These calculations are based upon careful examination of Canadian and British Columbia tax laws. Furthermore, at various periods during evolution of the program, the results were compared to those obtained by Wright Engineers, given the same input values, and in each case the two were virtually identical. However, different assumptions regarding the tax characterization of various costs can result in minor changes in project IRR and NPV values.

3. Net equity cash flow (NECF) -- in the Base Case this value equals cash flow before financing, because no debt funds are used to finance the project.

   NECF is shown for each year of the project life. These values are used to determine two key indicators of project profitability. They are:

   (a) DCF Rate of Return (After-Tax) or IRR --
   is calculated by an iterative procedure that discounts each annual after-tax NECF by a selected trial percent, determines the algebraic sum of the discounted values, and ultimately reports that percent which results in a sum of zero.

TABLE 0-22 BASE CASE

CASH FLOW SUMMARY (THOUSANDS $CANADIAN)

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
GROSS REVENUE	0	0	17496	175141	156518	101805	106480	102003	97025	115736
- OPERATING COSTS	0	0	0	67548	65161	64296	60172	58754	56554	56699
PRE-TAX OPERATING PROFIT	0	0	17496	107593	91357	37509	46308	43250	40471	59037
FEDERAL INCOME TAXES PAID	0	0	0	0	0	1868	6782	6044	4343	10635
BC INCOME TAXES PAID	0	0	0	0	0	3532	7398	7172	5155	4164
PROVINCIAL MINING TAXES PAID	0	0	0	0	0	542	2783	4055	3700	3506
- TOTAL TAXES PAID	0	0	0	0	0	5942	16963	17271	13198	18305
CASHFLOW BEFORE CAPITAL COSTS	0	0	17496	107593	91357	31567	29345	25979	27273	40732
- PROJECT CAPITAL COSTS	0	78850	139153	22	745	1825	1318	1254	109	17
+ SALVAGE	0	0	0	0	0	0	0	0	0	0
CASHFLOW BEFORE FINANCING	0	-78850	-121657	107571	90612	29742	28027	24725	27164	40715
+ BANK LOAN DRAWDOWNS	0	0	0	0	0	0	0	0	0	0
+ FINANCED INTEREST CHARGES	0	0	0	0	0	0	0	0	0	0
- BANK LOAN REDUCTIONS(EXCL INT PAID)	0	0	0	0	0	0	0	0	0	0
- CAPITALIZED INTEREST	0	0	0	0	0	0	0	0	0	0
- EXPENSED INTEREST	0	0	0	0	0	0	0	0	0	0
NET EQUITY CASHFLOW	0	-78850	-121657	107571	90612	29742	28027	24725	27164	40715
NET PRESENT VALUE (8 PCT)	0	-75873	-108393	88743	69215	21036	13355	14993	15252	21167
NET PRESENT VALUE (10 PCT)	0	-75181	-105450	84764	64910	19369	16593	13307	13291	18110
NET PRESENT VALUE (12 PCT)	0	-74506	-102638	81031	60943	17860	15027	11836	11611	15538
NET PRESENT VALUE (15 PCT)	0	-73528	-98648	75849	55558	15857	12994	9968	9523	12411
NET PRESENT VALUE (20 PCT)	0	-71980	-92548	68193	47869	13093	10282	7559	6920	8644
NET PRESENT VALUE (30 PCT)	0	-69156	-82077	55826	36173	9133	6620	4493	3797	4378
NET PRESENT VALUE (40 PCT)	0	-66640	-73442	46385	27909	6543	4404	2775	2178	2332
NET PRESENT VALUE (50 PCT)	0	-64331	-66222	39036	21921	4797	3014	1772	1298	1297

AFTER TAX PAYBACK PERIOD - YRS	3.078
PERCENT EQUITY RETURNED (PRE-TAX)	218.4
PERCENT EQUITY RETURNED (AFTER-TAX)	175.5
DCF RATE OF RETURN (PRE-TAX)	.2504
DCF RATE OF RETURN (AFTER-TAX)	.1964 Project IRR

ESCALATION FACTORS
****

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
CAPITAL COSTS	0	1	1	1	1	1	1	1	1	1
OPERATING COSTS	0	1	1	1	1	1	1	1	1	1
GOLD PRICES	0	1	1	1	1	1	1	1	1	1
SILVER PRICES	0	1	1	1	1	1	1	1	1	1

TABLE 0-22 BASE CASE

CASH FLOW SUMMARY (THOUSANDS $CANADIAN)

	1992	1993	1994	1995	TOTAL
GROSS REVENUE	17409	0	0	0	889611
- OPERATING COSTS	10785	0	0	0	439968
PRE-TAX OPERATING PROFIT	6624	0	0	0	449644
FEDERAL INCOME TAXES PAID	5566	-202.4	0	0	35036
BC INCOME TAXES PAID	2989	-113.8	0	0	30296
PROVINCIAL MINING TAXES PAID	5975	221	0	0	20782
- TOTAL TAXES PAID	14530	-95.13	0	0	86114
CASHFLOW BEFORE CAPITAL COSTS	-7906	95.13	0	0	363530
- PROJECT CAPITAL COSTS	11120	0	0	0	212173
+ SALVAGE	0	0	0	0	0
CASHFLOW BEFORE FINANCING	3214	95.13	0	0	151357
+ BANK LOAN DRAWDOWNS	0	0	0	0	0
+ FINANCED INTEREST CHARGES	0	0	0	0	0
- BANK LOAN REDUCTIONS(EXCL INT PAID)	0	0	0	0	0
- CAPITALIZED INTEREST	0	0	0	0	0
- EXPENSED INTEREST	0	0	0	0	0
NET EQUITY CASHFLOW	3214	95	0	0	151358
NET PRESENT VALUE (8 PCT)	1547	42.34	0	0	66083
NET PRESENT VALUE (10 PCT)	1300	34.92	0	0	51047
NET PRESENT VALUE (12 PCT)	1095	28.90	0	0	37826
NET PRESENT VALUE (15 PCT)	852.0	21.90	0	0	20857
NET PRESENT VALUE (20 PCT)	568.6	14.01	0	0	-1384
NET PRESENT VALUE (30 PCT)	265.8	6.044	0	0	-30542
NET PRESENT VALUE (40 PCT)	131.5	2.776	0	0	-47423
NET PRESENT VALUE (50 PCT)	68.26	1.345	0	0	-57397

NPV (10 PCT)

ESCALATION FACTORS

	1992	1993	1994	1995
CAPITAL COSTS	1	1	1	1
OPERATING COSTS	1	1	1	1
GOLD PRICES	1	1	1	1
SILVER PRICES	1	1	1	1

For the Base Case, the after-tax DCF ROR (or IRR) is 19.6 percent. This means that if capital is invested as indicated in Table 0-21, it can be expected to yield an after-tax return of 19.6 percent -- assuming that other Base Case values do not change during the life of the project.

(b) Net Present Value (10 PCT) --
is calculated by discounting each annual NECF by 10 percent and determining the algebraic sum of the values.

For the Base Case, the NPV (10 PCT) amounts to $51 MM. This means that if capital is invested as indicated in Table 0-21, it can be expected to yield an after-tax rate of 10 percent, plus after-tax amounts equivalent to a Present Value of $51 MM.

0.9.3 Sensitivity Analyses

The Base case represents that constant-dollar scenario which is considered to be the project geologists' and engineers' best estimate of future production and attendant capital and expense costs. It provides a close estimate of Federal and Provincial taxes for the amounts and timing of revenue, and thereby provides an accurate assessment of net equity cash flow (NECF) that can be derived on the basis of extensive data accumulated from over $12 MM of pilot plant and feasibility study work to date.

This is a highly-documented feasibility study; millions of dollars were spent to minimize unknowns and to reduce the potential range of variability associated with key input values. With this foremost in mind, a carefully-selected range of values is examined in this section in order to determine how certain changes could change project profitability in relation to the Base Case. Additionally, two cases of price/cost escalation are outlined to illustrate how current-dollar scenarios affect profitability values.

The variables which produce the greatest effects on profitability (IRR and NPV 10 PCT) are those which directly change annual revenue:

1. Gold price
2. Mill feed grade
3. Net gold recovery
4. Exchange rate

Any given percent change of any one of these variables changes project profitability by nearly the same degree. For example, a 5 percent increase in mill feed grade is about equal to a 5 percent increase in exchange rate.

Other variables change profitability to a lesser degree (given an equal percent change); nevertheless, they are very important:

5. Mining, milling and G & A unit production costs
6. Mill, power plant and tailings pond capital
7. Mining equipment capital
8. Access, housing and other plant capital
9. Engineering construction management and project administration capital
10. Inventory and working capital
11. All other capital
12. Acid price
13. Silver price

When current-dollar scenarios are constructed, the following variables are superimposed upon the foregoing factors:

14. Gold, silver and sulphuric acid annual price escalation rates
15. Annual operating cost escalation rate
16. Annual capital cost escalation rate

Current-dollar scenarios have a significant effect on IRR and NPV. However, no one can say much about the purchasing power of current (inflated) dollars, and inflated price and cost levels might be difficult to rationalize. However, in a dynamic, inflating economy subject to high interest rates, current-dollar cases may provide a better assessment potential profitability than more traditional constant-dollar cases.

Debt financing is not examined in this study. However, it can be readily examined using the IFPS program, and when doing so, three other variables must be considered. They are:

17. Equity/debt ratio
18. Interest rate
19. Proportion of net cash flow committed to debt repayment

The effects of changes in these variables can be computed with the IFPS program whenever it is useful.

0.9.3.1 Revenue Variables

Figure 0-24 illustrates the change in project IRR, and the change in NPV (10 PCT), caused by plus/minus percentage changes in the four important revenue-affecting variables listed above (1 through 4).

For example, an increase of 10 percent in any of these variables increases after-tax IRR by about 6 percent and NPV (10 PCT) of about $33 MM.

However, each of the four variables are not likely to vary within the same range. Therefore, it is not appropriate to apply broad ranges on an indiscriminate basis. Rather, judgement must be applied to select an appropriate range for each variable. Such selected ranges are shown in the following tabulation, along with corresponding changes in IRR and NPV (10 PCT).

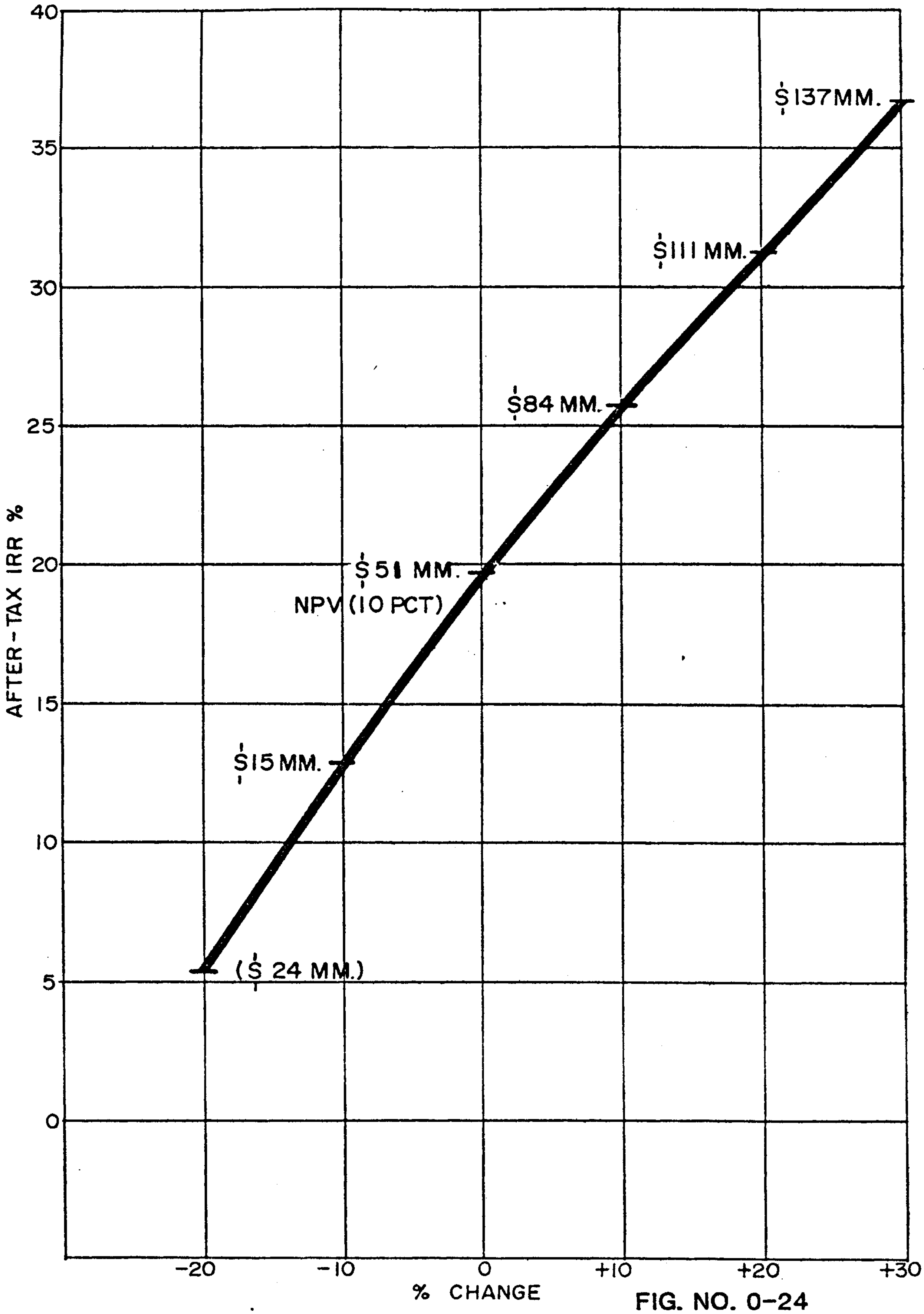

FIG. NO. 0-24

IRR & NPV (10 PCT) VERSUS

CHANGE IN REVENUE VARIABLES

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	FIG. NO. 17-1	REV.

EFFECTS OF CHANGES IN INDEPENDENT
REVENUE VARIABLES ON PROJECT PROFITABILITY

	Variable	Range	Variation	IRR	NPV (10 PCT)
1.	Gold price ($U.S.)			(%)	($MM)
	Low	400	-20%	5.5	(24)
	Base Case	500		19.6	51
	High	600	+20%	31.3	111
2.	Mill feed grade (Tr oz/tonne)				
	Low	0.057	-5%	16.3	33
	Base Case	0.060		19.6	51
	High	0.063	+5%	23.0	69
3.	Net gold recovery (%)				
	Low	68	-4.23%	16.9	36
	Base Case	71		19.6	51
	High	75	+5.63%	23.3	71
4.	Exchange rate (CDN./U.S.)				
	Low	1.15	-4.17%	16.9	36
	Base Case	1.20		19.6	51
	High	1.25	+4.17%	22.5	67

0.9.3.2 Unit Production Cost Variables

Mining, milling and G & A unit costs are considered to be as accurately estimated as is useful for this type of study. In view of more difficult-to-pinpoint variables, in fact, production cost estimates are likely to have a relatively high order of reliability.

An appropriate range of variation of unit production costs is judged to be -$0.00 to +$1.00/tonne, or -0% to +7.7%.

Figure 0-25 illustrates the changes in IRR and NPV (10 PCT) caused by changes within the range of variability discussed above. They are summarized below:

	Variable	Range	Variation	IRR	NPV (10 PCT)
5.	Unit production costs ($/tonne)			(%)	($MM)
	Low	12.88	-0%	19.6	51
	Base Case	12.88		19.6	51
	High	13.88	+7.7%	17.2	38

0.9.3.3 Capital Variables

Ranges in capital costs are important in project profitability. Two possible types of changes can be envisioned. One is a change caused by modifying the design or identifying some element previously overlooked. This is not likely to affect every category of capital to the same degree. In order to examine the potential impact of this type of change, several categories are examined on an independent basis in the tabulation below.

The second type of change is caused by well-established inflationary trends. Such trends also stand to affect unit production costs and product prices. However, it is appropriate to examine the effects of escalating prices, unit costs and capital in current-dollar scenarios, and this is done later in this section.

For the moment, therefore, only capital changes due to estimating errors and/or revised project definition are considered in the tabulation below.

Figure 0-26 shows the change in IRR and NPV (10 PCT) caused by an increase in mill, power plant and tailings pond capital ranging from zero to 10 percent. Changes in other capital variables are not plotted because their potential range of variation results in only a small change in profitability values.

0.9.3.4 By-Product Prices

Comments regarding silver and acid prices were made earlier. The Base Case assumes $10/oz silver, and a net acid value of $0/tonne.

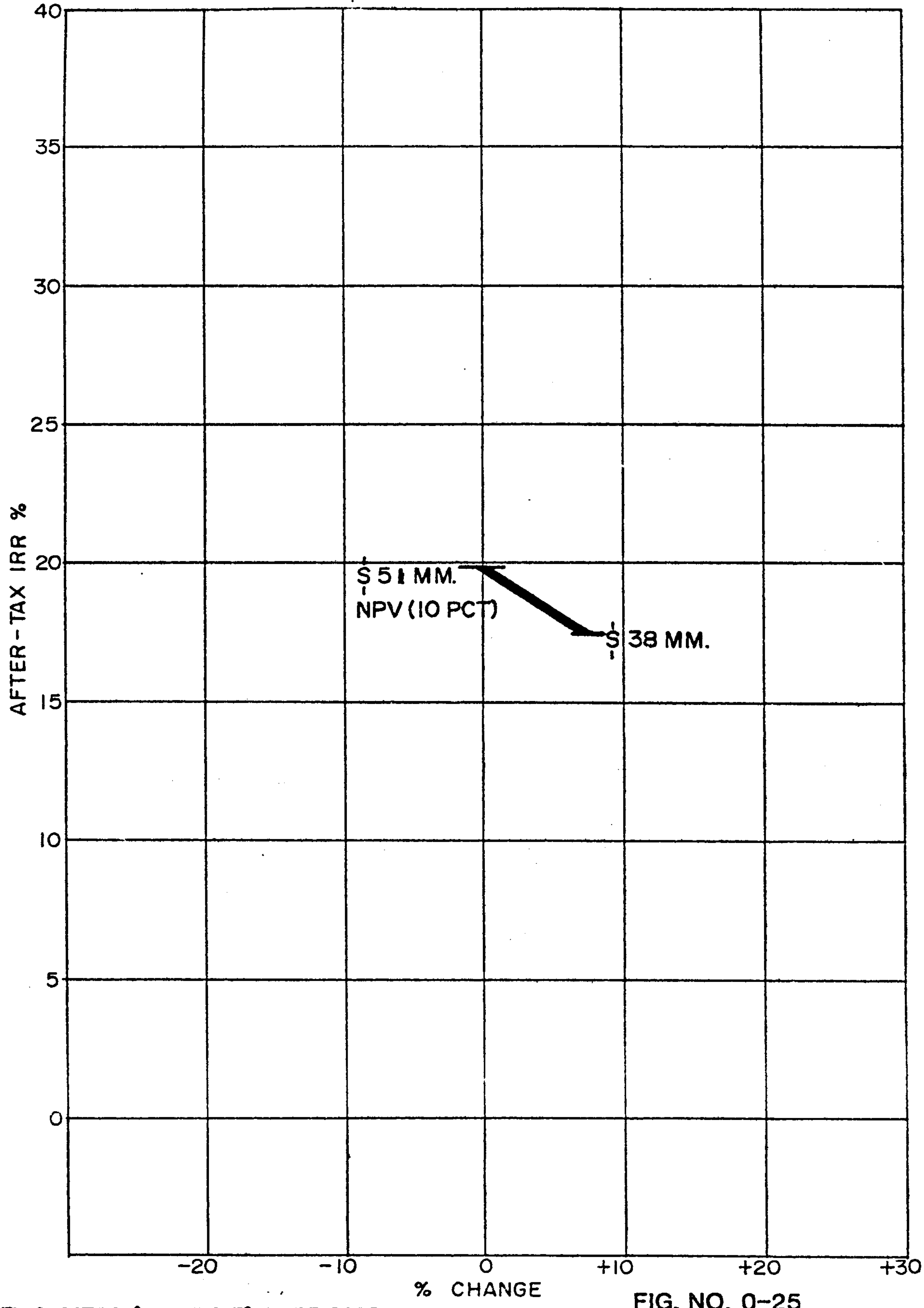

FIG. NO. 0-25

IRR & NPV (10 PCT) VERSUS CHANGES IN UNIT PRODUCTION COST VARIABLES

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CHECKED	APPROVED
FIG. NO. 17-2	REV.

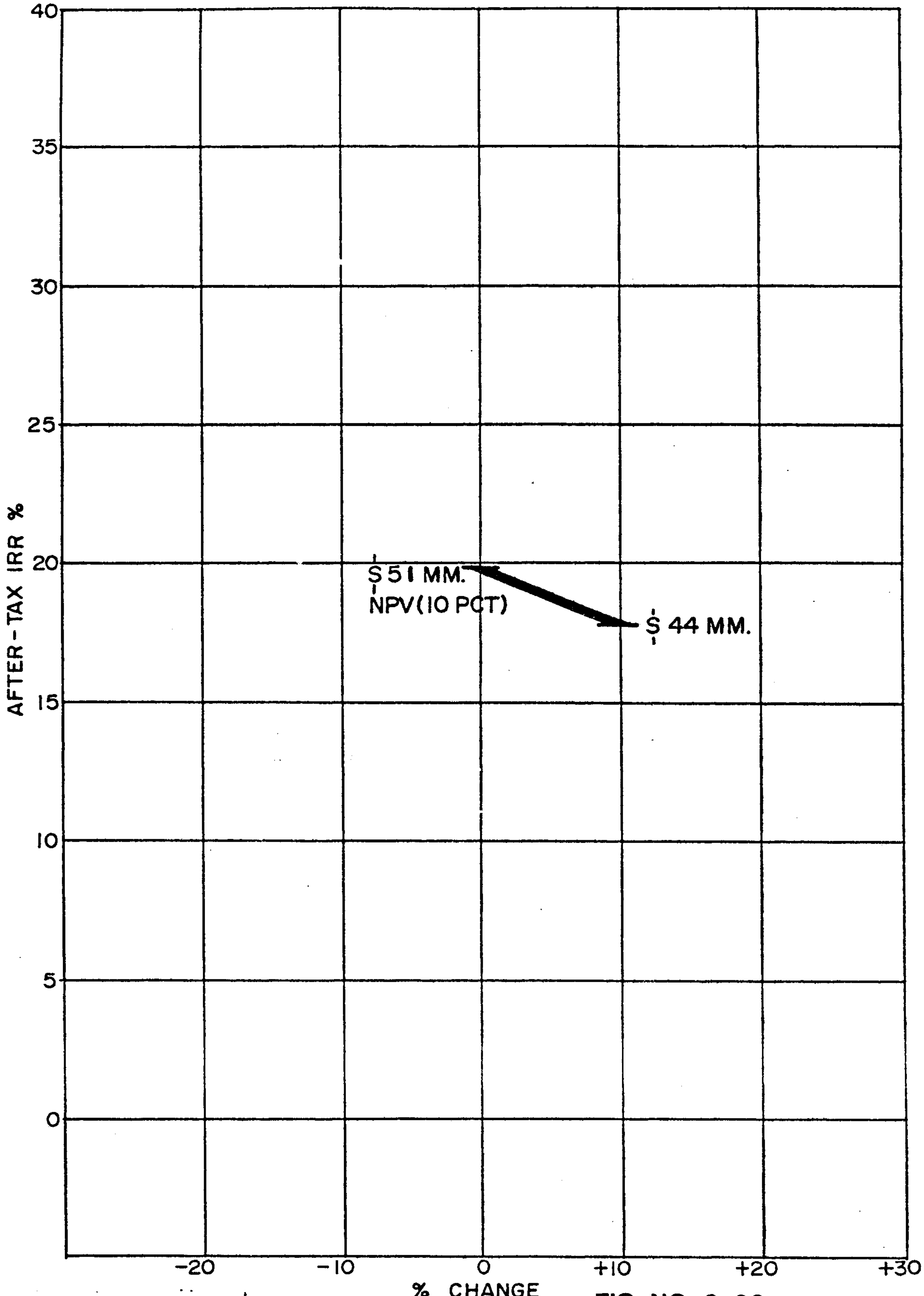

FIG. NO. 0-26

IRR & NPV (10 PCT) VERSUS CHANGES IN MILL, POWER PLANT & TAILINGS POND CAPITAL

CINOLA OPERATING COMPANY LTD.

DRAWN: S.S.	DATE: SEPT. 82
CHECKED	APPROVED
FIG. NO. 17-3	REV.

EFFECTS OF CHANGES IN INDEPENDENT CAPITAL VARIABLES ON PROJECT PROFITABILITY

	Variable	Range	Variation	IRR	NPV (10 PCT)
6.	Mill, power plant and pond			(%)	($MM)
	Low	122.3	-0%	19.6	51
	Base Case	122.3		19.6	51
	High	134.5	+10%	17.6	44
7.	Mining equipment				
	Low	18.2	-5%	19.8	52
	Base Case	19.1		19.6	51
	High	20.1	+5%	19.4	50
8.	Access, housing and other plant				
	Low	19.7	-5%	19.8	52
	Base Case	20.7		19.6	51
	High	23.8	+15%	19.0	48
9.	Engineering/construction management and project administration				
	Low	19.3	-10%	20.0	52
	Base Case	21.4		19.6	51
	High	23.6	+10%	19.2	49
10.	Inventory and working capital				
	Low	10.1	-10%	19.8	52
	Base Case	11.2		19.6	51
	High	12.3	+10%	19.4	50
11.	All other capital				
	Low	23.3	0%	19.6	51
	Base Case	23.3		19.6	51
	High	25.5	+10%	19.2	49

The effect of increasing or decreasing silver price, within a limited range, is not important so far as project profitability is concerned. for example, if silver price increases 30 percent to a level of $13/tonne milled, project IRR increases from 19.6 to 19.8 percent, and NPV (10 PCT) from $51 MM to $52.5 MM.

Increases in net acid price have the following impacts:

Net acid price ($/tonne)	After-tax IRR (%)	NPV (10 PCT) ($MM)
0	19.6	51
10	20.6	57
20	21.6	62
30	22.5	68
40	23.4	72

Therefore, an appreciable increase in net acid price (sales price less transportation cost) provides a significant increase in profitability -- about one percentage point in after-tax IRR per $10/tonne increase. This fact is based on an annual production rate of 212,000 tonnes of 93 percent sulphuric acid.

0.9.3.5 Current-Dollar Scenarios

Tables 0-23 (2 pages) and 0-24 (2 pages) illustrate the effects on revenue, costs, net equity cash flow and profitability from the assumption that gold/silver prices and capital/operating costs increase at the rate of 10 percent per year. Escalation starts at the beginning of project year -2, and average yearly prices and costs are used to determine operating profit and cash flow.

Table 0-24 indicates that in an inflating economy which experiences price/cost inflation of the magnitude projected, after-tax IRR increases to 29.1 percent, and NPV (10 PCT) increases to $124 MM.

Tables 0-25 (2 pages) and 0-26 (2 pages) provide a variation on this theme. In this case, prices are escalated at 8 percent per year, and costs are increased at 10 percent per year in order to examine a more restrictive inflationary trend. Table 0-26 indicates that under such an assumption after-tax IRR is 23.7 percent, and NPV (10 PCT) is $84 MM.

This case concludes the sensitivity analyses conducted for the Final Feasibility Study. It does not, however, exhaust the range of reasonable possibilities that are worth attention. Other cases can be calculated readily with the IFPS program at any time.

TABLE 0-23 10%-10%

PRODUCTION STATISTICS AND OPERATING PROFIT (M$)

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
WASTE MINED (M TONNES)	0	0	4421	13509	10981	8500	6176	5249	2221	2207
WASTE TO ORE RATIO	0	0	12.05	3.959	3.324	2.799	2.307	2.111	1.470	1.467
ORE MINED (M TONNES)	0	0	400	4725	4725	4725	4725	4725	4725	4725
GOLD GRADE (OZ PER TONNE)	0	0	.0889	.0782	.0717	.0513	.0529	.0514	.0496	.0567
SILVER GRADE (OZ PER TONNE)	0	0	.0889	.0782	.0717	.0513	.0529	.0514	.0496	.0567
GOLD RECOVERY (PERCENT)	0	0	.3100	.7800	.7600	.6900	.7000	.6900	.6800	.7100
SILVER RECOVERY (PERCENT)	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000
GOLD PRICE ($ US PER OZ)	0	525	577.5	635.3	698.8	768.7	845.5	930.1	1023	1125
SILVER PRICE ($ US PER OZ)	0	10.50	11.55	12.71	13.98	15.37	16.91	18.60	20.46	22.51
ACID PRICE (NET CA$ PER TONNE)	0	0	0	0	0	0	0	0	0	0
EXCHANGE RATE ($CAND VS $US)	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200
GOLD PRODUCTION (M OZS)	0	0	28.80	288.2	257.5	167.3	175.0	167.6	159.4	190.2
SILVER PRODUCTION (M OZS)	0	0	17.78	184.7	169.4	121.2	125.0	121.4	117.2	134.0
ACID PRODUCTION (M TONNES)	0	0	0	212	212	212	212	212	212	212
NET GOLD REVENUE	0	0	19961	219700	215900	154269	177525	187030	195651	256877
NET SILVER REVENUE	0	0	246.4	2817	2841	2236	2536	2711	2877	3618
ACID REVENUE	0	0	0	0	0	0	0	0	0	0
TOTAL REVENUE	0	0	20207	222516	218741	156505	180061	189740	198528	260495
-MINING COSTS	0	0	0	23103	22494	23509	21619	21467	19364	21605
-MILLING COSTS	0	0	0	54708	60178	66196	72816	80097	88107	96918
-GENERAL & ADMININSTRATION	0	0	0	8008	8393	9138	7319	7726	8247	9093
TOTAL OPERATING COSTS	0	0	0	85819	91065	98843	101753	109290	115718	127615
PRE-TAX OPERATING PROFIT	0	0	20207	136697	127676	57662	78308	80450	82810	132880

INITIAL AND REPLACEMENT CAPITAL COSTS

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
ENGINEERING AND CONST MGT	0	6311	10273	0	0	0	0	0	0	0
ACCESS ROAD AND SITE PREP	0	1955	717.3	0	0	0	0	0	0	0
CONST CAMP AND EMPL HOUSING	0	4822	5304	0	0	0	0	0	0	0
MINING EQUIPMENT	0	10719	10291	0	259.9	2616	2021	2163	0	0
MILL, POWER PLANT AND TAILINGS	0	45213	91574	6.353	739.3	6.149	55.80	7.441	176.0	0
ANC. BLDGS, PIPELINES, FUEL STORAGE	C	7586	2123	0	0	0	0	0	0	0
PREPROD DEVELOPMENT	0	1078	11214	0	0	0	0	0	0	0
PROJECT ADMINISTRATION	0	2625	4636	0	0	0	0	0	0	0
START-UP EXPENSES	0	0	6383	0	0	0	0	0	0	0
INVENTORY, WORK CAP, SALV, RECLAM	0	0	12966	0	0	0	0	0	0	0
BRIT COLUMBIA SALES TAX	0	2483	5241	21.60	41.93	182.9	152.2	161.8	47.06	38.26
TOTAL PROJECT CAPITAL COSTS	0	82793	160722	27.95	1041	2805	2229	2333	223.0	38.26
BANK LOAN DRAW	0	0	0	0	0	0	0	0	0	0
+ PRE-PRODUCTION INTEREST	0	0	0	0	0	0	0	0	0	0
+ PRODUCTION INTEREST	0	0	0	0	0	0	0	0	0	0
- REPAYMENTS	0	0	0	0	0	0	0	0	0	0
YEAR-END DEBT	0	0	0	0	0	0	0	0	0	0

TABLE 0-23 10%-10%

PRODUCTION STATISTICS AND OPERATING PROFIT (M$)

	1992	1993	1994	1995	TOTAL
WASTE MINED (M TONNES)	189.2	0	0	0	53453
WASTE TO ORE RATIO	1.225	0	0	0	
ORE MINED (M TONNES)	841	0	0	0	34316
GOLD GRADE (OZ PER TONNE)	.0500	0	0	0	
SILVER GRADE (OZ PER TONNE)	.0500	0	0	0	
GOLD RECOVERY (PERCENT)	.6800	0	0	0	
SILVER RECOVERY (PERCENT)	.5000	.5000	.5000	.5000	
GOLD PRICE ($ US PER OZ)	1238	1362	1498	1648	
SILVER PRICE ($ US PER OZ)	24.76	27.23	29.96	32.95	
ACID PRICE (NET CA$ PER TONNE)	0	0	0	0	
EXCHANGE RATE ($CAND VS $US)	1.200	1.200	1.200	1.200	
GOLD PRODUCTION (M OZS)	28.59	0	0	0	1462
SILVER PRODUCTION (M OZS)	21.03	0	0	0	1012
ACID PRODUCTION (M TONNES)	0	0	0	0	
NET GOLD REVENUE	42477	0	0	0	1469389
NET SILVER REVENUE	624.7	0	0	0	20506
ACID REVENUE	0	0	0	0	
TOTAL REVENUE	43101	0	0	0	1489895
-MINING COSTS	3591	0	0	0	156752
-MILLING COSTS	18975	0	0	0	537995
-GENERAL & ADMININSTRATION	4135	0	0	0	62058
TOTAL OPERATING COSTS	26701	0	0	0	756805
PRE-TAX OPERATING PROFIT	16400	0	0	0	733090

INITIAL AND REPLACEMENT CAPITAL COSTS

	1992	1993	1994	1995	TOTAL
ENGINEERING AND CONST MGT	0	0	0	0	16583
ACCESS ROAD AND SITE PREP	0	0	0	0	2672
CONST CAMP AND EMPL HOUSING	0	0	0	0	10125
MINING EQUIPMENT	0	0	0	0	28071
MILL, POWER PLANT AND TAILINGS	252.5	0	0	0	138031
ANC. BLDGS, PIPELINES, FUEL STORAGE	0	0	0	0	9709
PREPROD DEVELOPMENT	0	0	0	0	12292
PROJECT ADMINISTRATION	0	0	0	0	7261
START-UP EXPENSES	0	0	0	0	6383
INVENTORY, WORK CAP, SALV, RECLAM	-27794	0	0	0	-14828
BRIT COLUMBIA SALES TAX	9.903	0	0	0	8380
TOTAL PROJECT CAPITAL COSTS	-12704	0	0	0	239508
BANK LOAN DRAW	0	0	0	0	0
+ PRE-PRODUCTION INTEREST	0	0	0	0	0
+ PRODUCTION INTEREST	0	0	0	0	0
- REPAYMENTS	0	0	0	0	0
YEAR-END DEBT	0	0	0	0	0

TABLE 0-24 10%-10%

CASH FLOW SUMMARY (THOUSANDS $CANADIAN)

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
GROSS REVENUE	0	0	20207	222516	218741	156505	180061	189740	198528	260495
- OPERATING COSTS	0	0	0	85819	91065	98843	101753	109290	115718	127615
PRE-TAX OPERATING PROFIT	0	0	20207	136697	127676	57662	78308	80450	82810	132880
FEDERAL INCOME TAXES PAID	0	0	0	0	0	11746	6135	10882	20465	18762
BC INCOME TAXES PAID	0	0	0	0	0	15432	6532	12906	8797	15088
PROVINCIAL MINING TAXES PAID	0	0	0	0	0	5741	5077	7918	8228	10507
- TOTAL TAXES PAID	0	0	0	0	0	32919	17744	31706	37490	44357
CASHFLOW BEFORE CAPITAL COSTS	0	0	20207	136697	127676	24743	60564	48744	45320	88523
- PROJECT CAPITAL COSTS	0	82793	160722	27.95	1041	2806	2229	2333	223.0	38.26
+ SALVAGE	0	0	0	0	0	0	0	0	0	0
CASHFLOW BEFORE FINANCING	0	-82793	-140514	136669	126634	21937	58335	46412	45097	88484
+ BANK LOAN DRAWDOWNS	0	0	0	0	0	0	0	0	0	0
+ FINANCED INTEREST CHARGES	0	0	0	0	0	0	0	0	0	0
- BANK LOAN REDUCTIONS(EXCL INT PAID)	0	0	0	0	0	0	0	0	0	0
- CAPITALIZED INTEREST	0	0	0	0	0	0	0	0	0	0
- EXPENSED INTEREST	0	0	0	0	0	0	0	0	0	0
NET EQUITY CASHFLOW	0	-82793	-140514	136669	126634	21937	58335	46412	45097	88484
NET PRESENT VALUE (8 PCT)	0	-79668	-125194	112748	96731	15516	38203	28143	25320	46001
NET PRESENT VALUE (10 PCT)	0	-78940	-121795	107693	90714	14286	34536	24979	22065	39357
NET PRESENT VALUE (12 PCT)	0	-78232	-118548	102950	85170	13173	31277	22218	19276	33768
NET PRESENT VALUE (15 PCT)	0	-77205	-113939	96366	77644	11696	27045	18711	15809	26973
NET PRESENT VALUE (20 PCT)	0	-75579	-106893	86640	66898	9657	21401	14189	11489	18786
NET PRESENT VALUE (30 PCT)	0	-72614	-94799	70927	50553	6736	13730	8433	6303	9514
NET PRESENT VALUE (40 PCT)	0	-69973	-84826	58932	39003	4326	9167	5210	3616	5067
NET PRESENT VALUE (50 PCT)	0	-67600	-76486	49595	30636	3538	6272	3327	2155	2819

AFTER TAX PAYBACK PERIOD - YRS	2.684
PERCENT EQUITY RETURNED (PRE-TAX)	321.0
PERCENT EQUITY RETURNED (AFTER-TAX)	224.3
DCF RATE OF RETURN (PRE-TAX)	.3734
DCF RATE OF RETURN (AFTER-TAX)	.2914

ESCALATION FACTORS
****

CAPITAL COSTS	0	1.050	1.155	1.271	1.398	1.537	1.691	1.860	2.046	2.251
OPERATING COSTS	0	1.050	1.155	1.271	1.398	1.537	1.691	1.860	2.046	2.251
GOLD PRICES	0	1.050	1.155	1.271	1.398	1.537	1.691	1.860	2.046	2.251
SILVER PRICES	0	1.050	1.155	1.271	1.398	1.537	1.691	1.860	2.046	2.251

TABLE 0-24 10%-10%

CASH FLOW SUMMARY (THOUSANDS $CANADIAN)

	1992	1993	1994	1995	TOTAL
GROSS REVENUE	43101	0	0	0	1489895
- OPERATING COSTS	26701	0	0	0	756805
PRE-TAX OPERATING PROFIT	16400	0	0	0	733090
FEDERAL INCOME TAXES PAID	19860	62.59	0	0	87913
BC INCOME TAXES PAID	11569	-185.5	0	0	70139
PROVINCIAL MINING TAXES PAID	19696	786	0	0	57953
- TOTAL TAXES PAID	51126	663.1	0	0	216005
CASHFLOW BEFORE CAPITAL COSTS	-34726	-663.1	0	0	517085
- PROJECT CAPITAL COSTS	-12704	0	0	0	239508
+ SALVAGE	0	0	0	0	0
CASHFLOW BEFORE FINANCING	-22022	-663.1	0	0	277577
+ BANK LOAN DRAWDOWNS	0	0	0	0	0
+ FINANCED INTEREST CHARGES	0	0	0	0	0
- BANK LOAN REDUCTIONS(EXCL INT PAID)	0	0	0	0	0
- CAPITALIZED INTEREST	0	0	0	0	0
- EXPENSED INTEREST	0	0	0	0	0
NET EQUITY CASHFLOW	-22022	-663	0	0	277576
NET PRESENT VALUE (8 PCT)	-10601	-295.5	0	0	146905
NET PRESENT VALUE (10 PCT)	-8905	-243.7	0	0	123747
NET PRESENT VALUE (12 PCT)	-7504	-201.7	0	0	103348
NET PRESENT VALUE (15 PCT)	-5838	-152.8	0	0	77111
NET PRESENT VALUE (20 PCT)	-3896	-97.75	0	0	42594
NET PRESENT VALUE (30 PCT)	-1821	-42.18	0	0	-3030
NET PRESENT VALUE (40 PCT)	-900.8	-19.37	0	0	-29898
NET PRESENT VALUE (50 PCT)	-467.7	-9.388	0	0	-46221

ESCALATION FACTORS

	1992	1993	1994	1995
CAPITAL COSTS	2.476	2.723	2.996	3.295
OPERATING COSTS	2.476	2.723	2.996	3.295
GOLD PRICES	2.476	2.723	2.996	3.295
SILVER PRICES	2.476	2.723	2.996	3.295

TABLE 0-25 8%-10%

PRODUCTION STATISTICS AND OPERATING PROFIT (M$)

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
WASTE MINED (M TONNES)	0	0	4421	13509	10981	8500	6176	5249	2221	2207
WASTE TO ORE RATIO	0	0	12.05	3.859	3.324	2.799	2.307	2.111	1.470	1.467
ORE MINED (M TONNES)	0	0	400	4725	4725	4725	4725	4725	4725	4725
GOLD GRADE (OZ PER TONNE)	0	0	.0889	.0782	.0717	.0513	.0529	.0514	.0496	.0567
SILVER GRADE (OZ PER TONNE)	0	0	.0889	.0782	.0717	.0513	.0529	.0514	.0496	.0567
GOLD RECOVERY (PERCENT)	0	0	.8100	.7800	.7600	.6900	.7000	.6900	.6800	.7100
SILVER RECOVERY (PERCENT)	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000	.5000
GOLD PRICE ($ US PER OZ)	0	520	561.6	606.5	655.1	707.5	764.1	825.2	891.2	962.5
SILVER PRICE ($ US PER OZ)	0	10.40	11.23	12.13	13.10	14.15	15.28	16.50	17.82	19.25
ACID PRICE (NET CA$ PER TONNE)	0	0	0	0	0	0	0	0	0	0
EXCHANGE RATE ($CAND VS $US)	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200	1.200
GOLD PRODUCTION (M OZS)	0	0	28.80	288.2	257.5	167.3	175.0	167.6	159.4	190.2
SILVER PRODUCTION (M OZS)	0	0	17.78	184.7	169.4	121.2	125.0	121.4	117.2	134.0
ACID PRODUCTION (M TONNES)	0	0	0	212	212	212	212	212	212	212
NET GOLD REVENUE	0	0	19411	209766	202391	141987	160420	165936	170429	219694
NET SILVER REVENUE	0	0	239.6	2689	2663	2058	2292	2405	2506	3094
ACID REVENUE	0	0	0	0	0	0	0	0	0	0
TOTAL REVENUE	0	0	19651	212455	205054	144045	162712	168341	172935	222788
-MINING COSTS	0	0	0	23103	22494	23509	21619	21467	19364	21605
-MILLING COSTS	0	0	0	54708	60178	66196	72816	80097	88107	96918
-GENERAL & ADMININSTRATION	0	0	0	8008	8393	9138	7319	7726	8247	9093
TOTAL OPERATING COSTS	0	0	0	85819	91065	98843	101753	109290	115718	127615
PRE-TAX OPERATING PROFIT	0	0	19651	126636	113988	45202	60959	59051	57217	95173

INITIAL AND REPLACEMENT CAPITAL COSTS

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
ENGINEERING AND CONST MGT	0	6311	10273	0	0	0	0	0	0	0
ACCESS ROAD AND SITE PREP	0	1955	717.3	0	0	0	0	0	0	0
CONST CAMP AND EMPL HOUSING	0	4822	5304	0	0	0	0	0	0	0
MINING EQUIPMENT	0	10719	10291	0	259.9	2616	2021	2163	0	0
MILL, POWER PLANT AND TAILINGS	0	45213	91574	6.353	739.3	6.149	55.30	7.441	176.0	0
ANC. BLDGS, PIPELINES, FUEL STORAGE	0	7586	2123	0	0	0	0	0	0	0
PREPROD DEVELOPMENT	0	1078	11214	0	0	0	0	0	0	0
PROJECT ADMINISTRATION	0	2625	4636	0	0	0	0	0	0	0
START-UP EXPENSES	0	0	6383	0	0	0	0	0	0	0
INVENTORY, WORK CAP, SALV, RECLAM	0	0	12966	0	0	0	0	0	0	0
BRIT COLUMBIA SALES TAX	0	2483	5241	21.60	41.93	182.9	152.2	161.8	47.06	38.26
TOTAL PROJECT CAPITAL COSTS	0	82793	160722	27.95	1041	2806	2229	2333	223.0	38.26
BANK LOAN DRAW	0	0	0	0	0	0	0	0	0	0
+ PRE-PRODUCTION INTEREST	0	0	0	0	0	0	0	0	0	0
+ PRODUCTION INTEREST	0	0	0	0	0	0	0	0	0	0
- REPAYMENTS	0	0	0	0	0	0	0	0	0	0
YEAR-END DEBT	0	0	0	0	0	0	0	0	0	0

TABLE 0-25 8%-10%

PRODUCTION STATISTICS AND OPERATING PROFIT (M$)

	1992	1993	1994	1995	TOTAL
WASTE MINED (M TONNES)	189.2	0	0	0	53453
WASTE TO ORE RATIO	1.225	0	0	0	
ORE MINED (M TONNES)	841	0	0	0	34316
GOLD GRADE (OZ PER TONNE)	.0500	0	0	0	
SILVER GRADE (OZ PER TONNE)	.0500	0	0	0	
GOLD RECOVERY (PERCENT)	.6800	0	0	0	
SILVER RECOVERY (PERCENT)	.5000	.5000	.5000	.5000	
GOLD PRICE ($ US PER OZ)	1039	1123	1212	1309	
SILVER PRICE ($ US PER OZ)	20.79	22.45	24.25	26.19	
ACID PRICE (NET CA$ PER TONNE)	0	0	0	0	
EXCHANGE RATE ($CAND VS $US)	1.200	1.200	1.200	1.200	
GOLD PRODUCTION (M OZS)	28.59	0	0	0	1462
SILVER PRODUCTION (M OZS)	21.03	0	0	0	1012
ACID PRODUCTION (M TONNES)	0	0	0	0	
NET GOLD REVENUE	35668	0	0	0	1325701
NET SILVER REVENUE	524.5	0	0	0	18471
ACID REVENUE	0	0	0	0	
TOTAL REVENUE	36192	0	0	0	1344173
-MINING COSTS	3591	0	0	0	156752
-MILLING COSTS	18975	0	0	0	537995
-GENERAL & ADMININSTRATION	4135	0	0	0	62058
TOTAL OPERATING COSTS	26701	0	0	0	756805
PRE-TAX OPERATING PROFIT	9491	0	0	0	587367

INITIAL AND REPLACEMENT CAPITAL COSTS

	1992	1993	1994	1995	TOTAL
ENGINEERING AND CONST MGT	0	0	0	0	16583
ACCESS ROAD AND SITE PREP	0	0	0	0	2672
CONST CAMP AND EMPL HOUSING	0	0	0	0	10125
MINING EQUIPMENT	0	0	0	0	28071
MILL, POWER PLANT AND TAILINGS	252.5	0	0	0	138031
ANC. BLDGS, PIPELINES, FUEL STORAGE	0	0	0	0	9709
PREPROD DEVELOPMENT	0	0	0	0	12292
PROJECT ADMINISTRATION	0	0	0	0	7261
START-UP EXPENSES	0	0	0	0	6383
INVENTORY, WORK CAP, SALV, RECLAM	-27774	0	0	0	-14828
BRIT COLUMBIA SALES TAX	9.903	0	0	0	8380
TOTAL PROJECT CAPITAL COSTS	-12704	0	0	0	239508
BANK LOAN DRAW	0	0	0	0	0
+ PRE-PRODUCTION INTEREST	0	0	0	0	0
+ PRODUCTION INTEREST	0	0	0	0	0
- REPAYMENTS	0	0	0	0	0
YEAR-END DEBT	0	0	0	0	0

TABLE 0-26 8%-10%

CASH FLOW SUMMARY (THOUSANDS $CANADIAN)

	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
GROSS REVENUE	0	0	19651	212455	205054	144045	162712	168341	172935	222788
- OPERATING COSTS	0	0	0	85819	91065	98843	101753	109290	115718	127615
PRE-TAX OPERATING PROFIT	0	0	19651	126636	113988	45202	60959	59051	57217	95173
FEDERAL INCOME TAXES PAID	0	0	0	0	0	6270	6065	8378	6109	15407
BC INCOME TAXES PAID	0	0	0	0	0	8942	6449	9938	7249	6218
PROVINCIAL MINING TAXES PAID	0	0	0	0	0	2551	3441	5641	5420	5472
- TOTAL TAXES PAID	0	0	0	0	0	17763	15955	23957	18778	27097
CASHFLOW BEFORE CAPITAL COSTS	0	0	19651	126636	113988	27439	45004	35094	38439	68076
- PROJECT CAPITAL COSTS	0	82793	160722	27.95	1041	2806	2229	2333	223.0	38.26
+ SALVAGE	0	0	0	0	0	0	0	0	0	0
CASHFLOW BEFORE FINANCING	0	-82793	-141071	126608	112947	24633	42775	32762	38216	68037
+ BANK LOAN DRAWDOWNS	0	0	0	0	0	0	0	0	0	0
+ FINANCED INTEREST CHARGES	0	0	0	0	0	0	0	0	0	0
- BANK LOAN REDUCTIONS(EXCL INT PAID)	0	0	0	0	0	0	0	0	0	0
- CAPITALIZED INTEREST	0	0	0	0	0	0	0	0	0	0
- EXPENSED INTEREST	0	0	0	0	0	0	0	0	0	0
NET EQUITY CASHFLOW	0	-82793	-141071	126608	112947	24633	42775	32762	38216	68037
NET PRESENT VALUE (8 PCT)	0	-79668	-125690	104448	86276	17422	28013	19866	21457	35371
NET PRESENT VALUE (10 PCT)	0	-78940	-122278	99765	80910	16042	25324	17633	18698	30263
NET PRESENT VALUE (12 PCT)	0	-78232	-119017	95371	75965	14792	22935	15684	16335	25965
NET PRESENT VALUE (15 PCT)	0	-77205	-114391	89272	69252	13133	19831	13208	13397	20740
NET PRESENT VALUE (20 PCT)	0	-75579	-107316	80262	59668	10844	15693	10016	9736	14445
NET PRESENT VALUE (30 PCT)	0	-72614	-95175	65706	45089	7564	10104	5953	5342	7315
NET PRESENT VALUE (40 PCT)	0	-69973	-85162	54594	34788	5419	6722	3677	3064	3896
NET PRESENT VALUE (50 PCT)	0	-67600	-76789	45944	27325	3973	4599	2348	1826	2168

AFTER TAX PAYBACK PERIOD - YRS	2.861
PERCENT EQUITY RETURNED (PRE-TAX)	255.4
PERCENT EQUITY RETURNED (AFTER-TAX)	195.7
DCF RATE OF RETURN (PRE-TAX)	.3006
DCF RATE OF RETURN (AFTER-TAX)	.2367

ESCALATION FACTORS
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	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991
CAPITAL COSTS	0	1.050	1.155	1.271	1.398	1.537	1.691	1.860	2.046	2.251
OPERATING COSTS	0	1.050	1.155	1.271	1.398	1.537	1.691	1.860	2.046	2.251
GOLD PRICES	0	1.040	1.123	1.213	1.310	1.415	1.528	1.650	1.782	1.925
SILVER PRICES	0	1.040	1.123	1.213	1.310	1.415	1.528	1.650	1.782	1.925

TABLE 0-26 8%-10%

CASH FLOW SUMMARY (THOUSANDS $CANADIAN)

	1992	1993	1994	1995	TOTAL
GROSS REVENUE	36192	0	0	0	1344173
- OPERATING COSTS	26701	0	0	0	756805
PRE-TAX OPERATING PROFIT	9491	0	0	0	587367
FEDERAL INCOME TAXES PAID	10236	-278.3	0	0	52187
BC INCOME TAXES PAID	7991	-47.42	0	0	46740
PROVINCIAL MINING TAXES PAID	11751	423	0	0	34699
- TOTAL TAXES PAID	29979	97.24	0	0	133626
CASHFLOW BEFORE CAPITAL COSTS	-20488	-97.24	0	0	453741
- PROJECT CAPITAL COSTS	-12704	0	0	0	239508
+ SALVAGE	0	0	0	0	0
CASHFLOW BEFORE FINANCING	-7785	-97.24	0	0	214233
+ BANK LOAN DRAWDOWNS	0	0	0	0	0
+ FINANCED INTEREST CHARGES	0	0	0	0	0
- BANK LOAN REDUCTIONS(EXCL INT PAID)	0	0	0	0	0
- CAPITALIZED INTEREST	0	0	0	0	0
- EXPENSED INTEREST	0	0	0	0	0
NET EQUITY CASHFLOW	-7785	-97	0	0	214232
NET PRESENT VALUE (8 PCT)	-3747	-43.23	0	0	103705
NET PRESENT VALUE (10 PCT)	-3148	-35.66	0	0	84232
NET PRESENT VALUE (12 PCT)	-2653	-29.51	0	0	67115
NET PRESENT VALUE (15 PCT)	-2064	-22.36	0	0	45153
NET PRESENT VALUE (20 PCT)	-1377	-14.30	0	0	16376
NET PRESENT VALUE (30 PCT)	-643.9	-6.171	0	0	-21366
NET PRESENT VALUE (40 PCT)	-318.5	-2.834	0	0	-43296
NET PRESENT VALUE (50 PCT)	-155.3	-1.373	0	0	-56373

ESCALATION FACTORS
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	1992	1993	1994	1995
CAPITAL COSTS	2.476	2.723	2.996	3.295
OPERATING COSTS	2.476	2.723	2.996	3.295
GOLD PRICES	2.079	2.245	2.425	2.619
SILVER PRICES	2.079	2.245	2.425	2.619