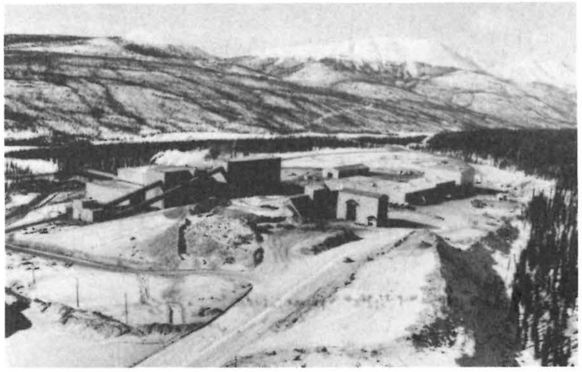
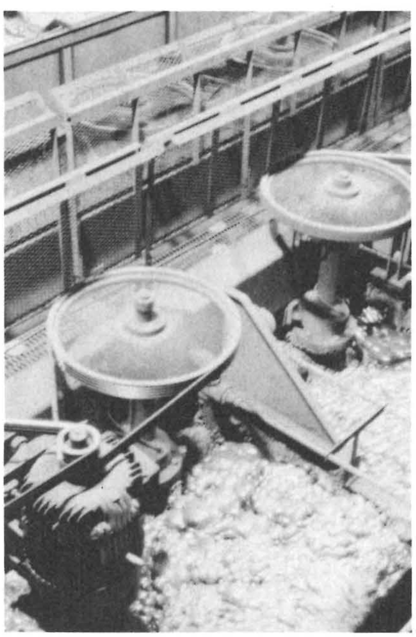
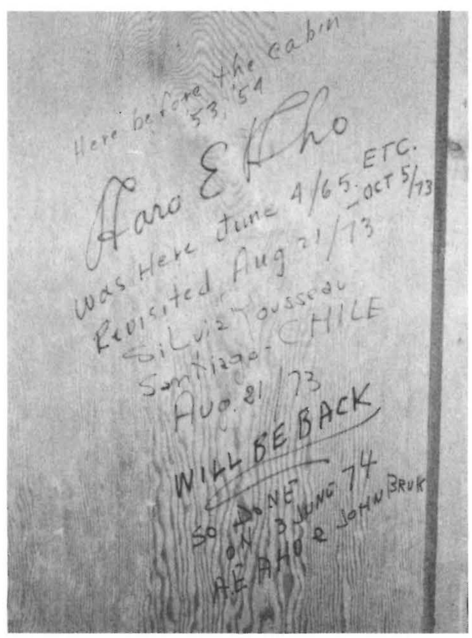
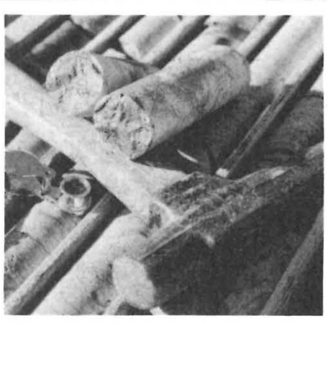


The Development of Vangorda Plateau Ore Deposits







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1.
PROGRAM

THE DEVELOPMENT OF THE VANGORDA PLATEAU ORE DEPOSITS

A continuing objective of Cyprus Anvil Mining Corporation is to increase the mineable ore reserves in the Anvil District to extend the life of the Faro Operation. To this end, an agreement was consummated between Cyprus Anvil Mining Corporation and Kerr Addison Mines Limited and Canadian Natural Resources Limited on May 15th, 1979, whereby Cyprus Anvil acquired all of the mineral properties of the respective companies in the Anvil District. These properties cover approximately 31 square kilometres and include the Grum and Vangorda deposits, located on the Vangorda Plateau. These two lead, zinc and silver orebodies contain significant amounts of ore mineable by open pit methods. This development program includes only the proven open pit portions of the Vangorda Plateau deposits.

Subsequent tc the acquisition, studies were conducted to determine the most financially attractive schedule for the development of the Vangorda Plateau deposits. The recommended program is based upon the sequential development of the Vangorda and Grum open pit reserves and the concurrent milling of these ores with mill feed from the Faro pit, which otherwise would have been exhausted in 1989. In the economic analysis of this development, the present mining plan is used as the base case and identified as Case 1.

The program, identified as Case 4B-V provides for modifications to the existing concentrator to be completed by January 1st, 1982 and the subsequent construction of ancillary facilities to treat Vangorda Plateau ores. Pre-production stripping of the Vangorda deposit will commence in 1984 to permit ore production prior to the end of that year. Vangorda and Faro ores will be milled in equal proportions at the current capacity of 3.4 million tonnes per year until 1988, at which time the Vangorda reserves will be exhausted. To ensure continuity of ore supply, the pre-production stripping of Grum will commence in 1985 to permit the phased development of Vangorda ore with that from Grum in 1988.

The Vangorda deposit is structurally simple and well understood based on the results of recent intensive drilling. The early development of this deposit will extend the life of the Faro pit providing both increased flexibility and reliability in ore supply and will allow the time necessary to develop an efficient mining plan for the geologically more complex Grum deposit. Since less pre-production stripping is required, the Vangorda deposit can be developed sooner than the Grum, thereby facilitating the early inclusion of the Vangorda Plateau ore to economic advantage into the overall mining plan. The ore from the Vangorda Plateau will be transported approximately 14 kilometres to the mill by one of three haulage systems. Currently under study are off-highway trucks, railway and a cable belt conveyor. Supplemental power generating facilities and new tailings impoundment ponds are provided for in the program.

The total capital cost of this development program during the next nine years will be $155.9 million in January 1980 dollars. Escalated for inflation, the capital budget is estimated to be $239.3 million.

The appreciable reduction in the cost per pound of marketable metal produced realized through the implementation of the 4B-V program reflects the significant increase in metal production, particularly that of gold and silver.

The project will be financed by internal cash flow generation and borrowing through available lines of credit. When compared to the cash flow generated by milling only the Faro deposits, the program will yield a return on investment based on discounted cash flows (DCF/ROI) of 36.6%. This DCF/ROI results from the increased production of lead, zinc and precious metals, improved metallurgical performance and the extension from 1989 to 1997 of the open pit mining operations. Using an after tax discount rate of 12 percent the comparative net present value per share on currently issued shares of the cash flows available for distribution at equilibrium prices are; Case 1 $27.28, Case 4B-V $39.75 for an increase of 45.5 percent. Discounting at 15 percent the development case increased the net present value per share from $24.05 to $32.83 or 36.5 percent.

FIGURE I-1 VANGORDA PLATEAU DEVELOPMENT
PAYABLE METAL PRODUCTION

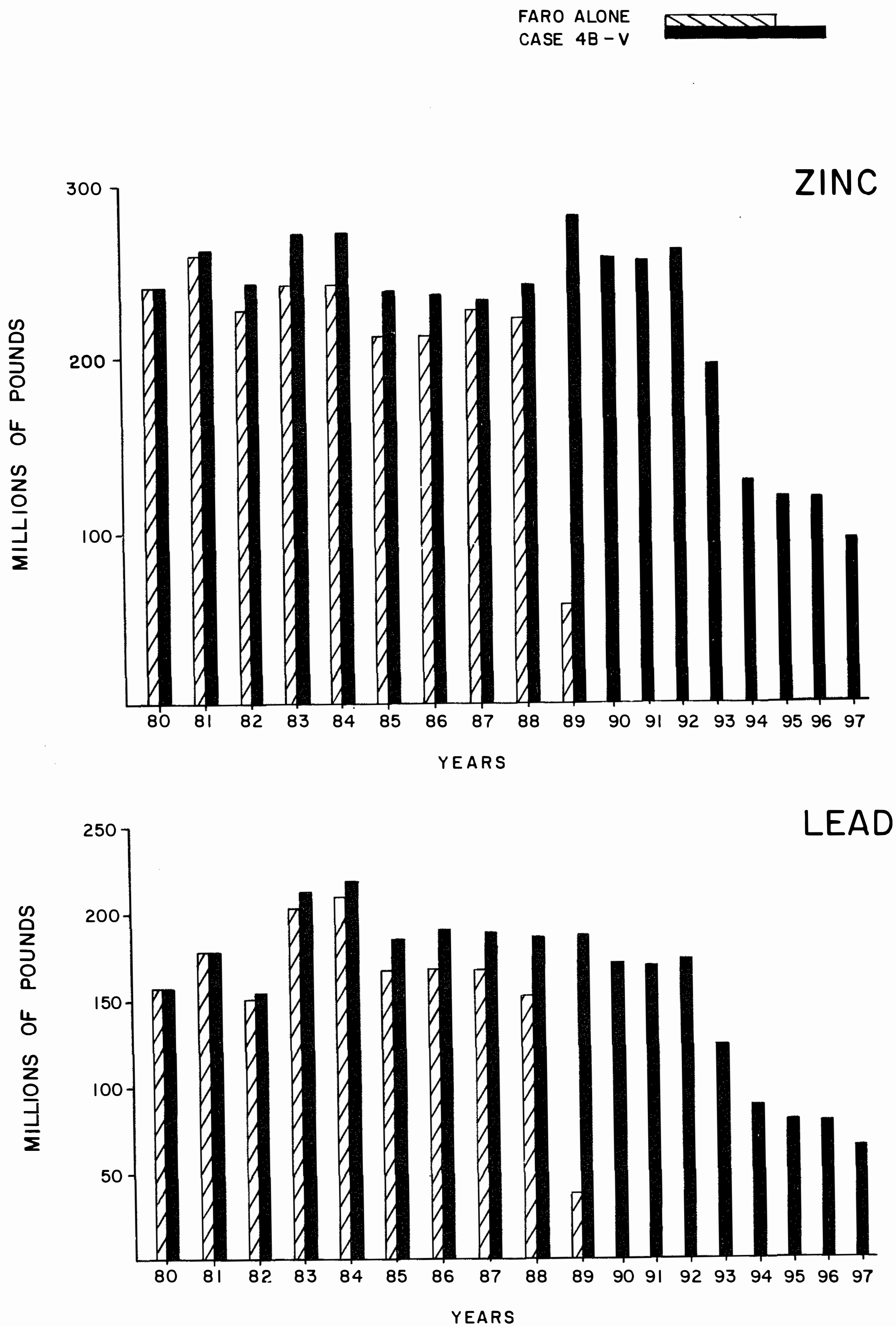


FIGURE 1-2 VANGORDA PLATEAU DEVELOPMENT
PAYABLE METAL PRODUCTION

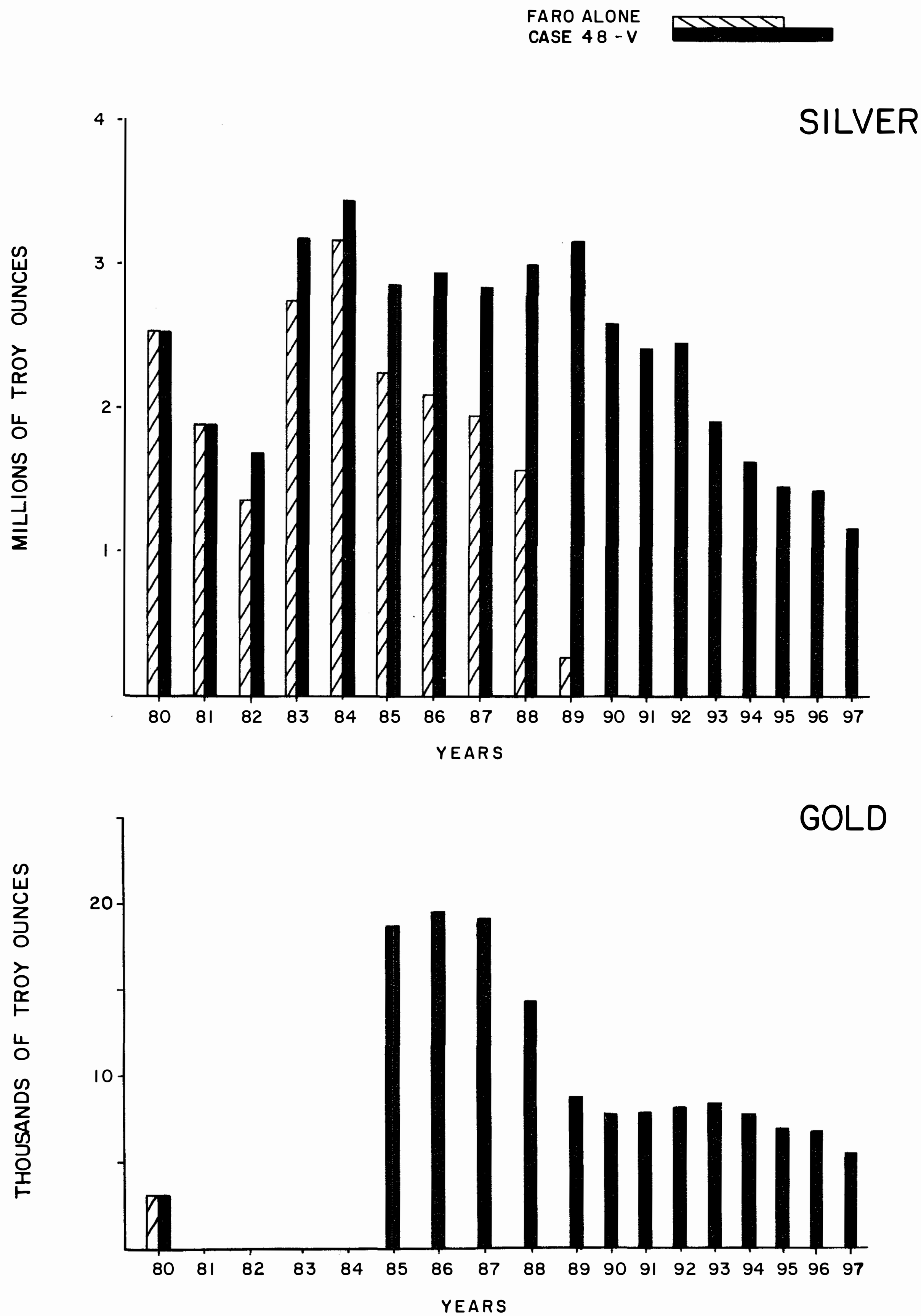


FIGURE I-3 VANGORDA PLATEAU DEVELOPMENT

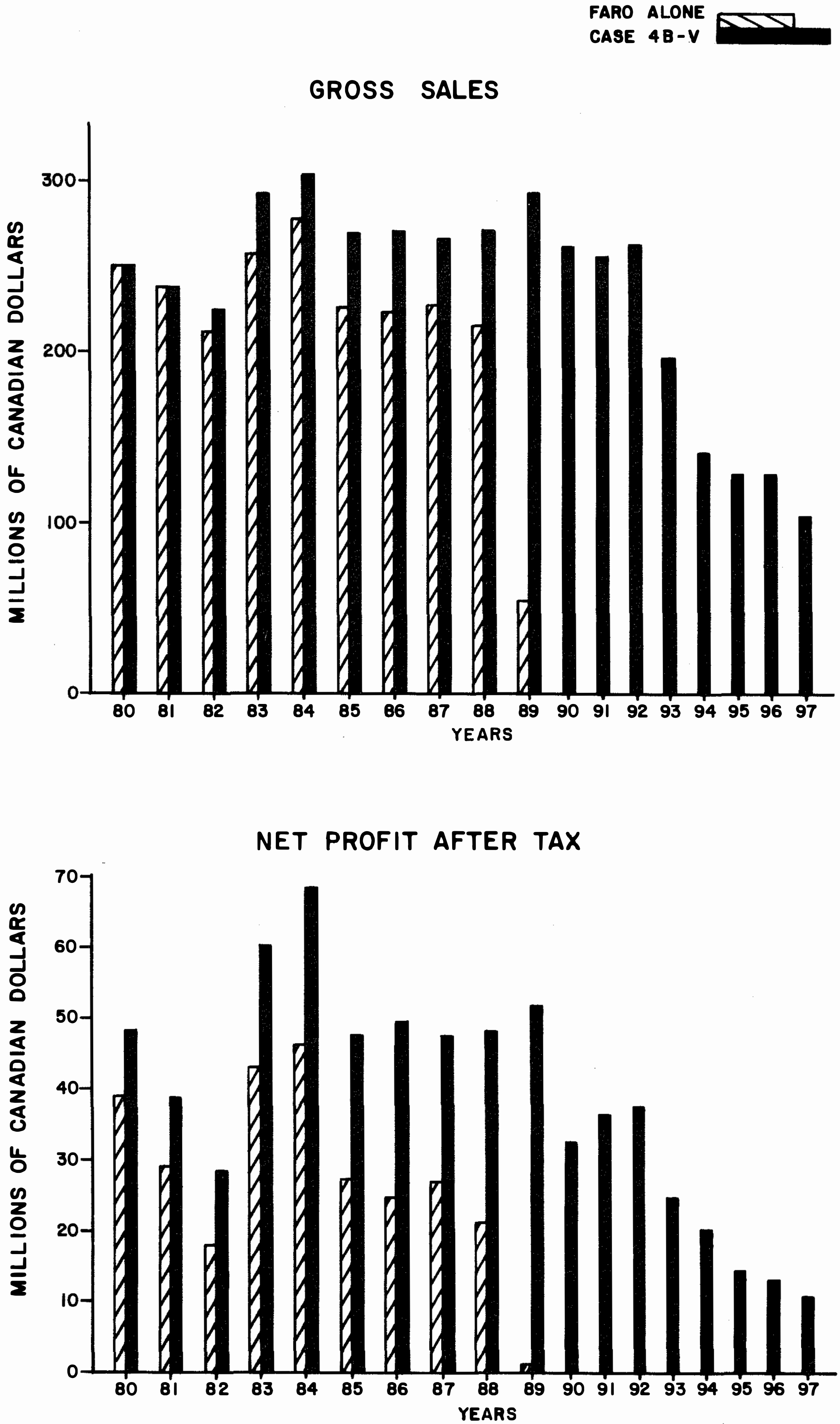


FIGURE I-4 VANGORDA PLATEAU DEVELOPMENT

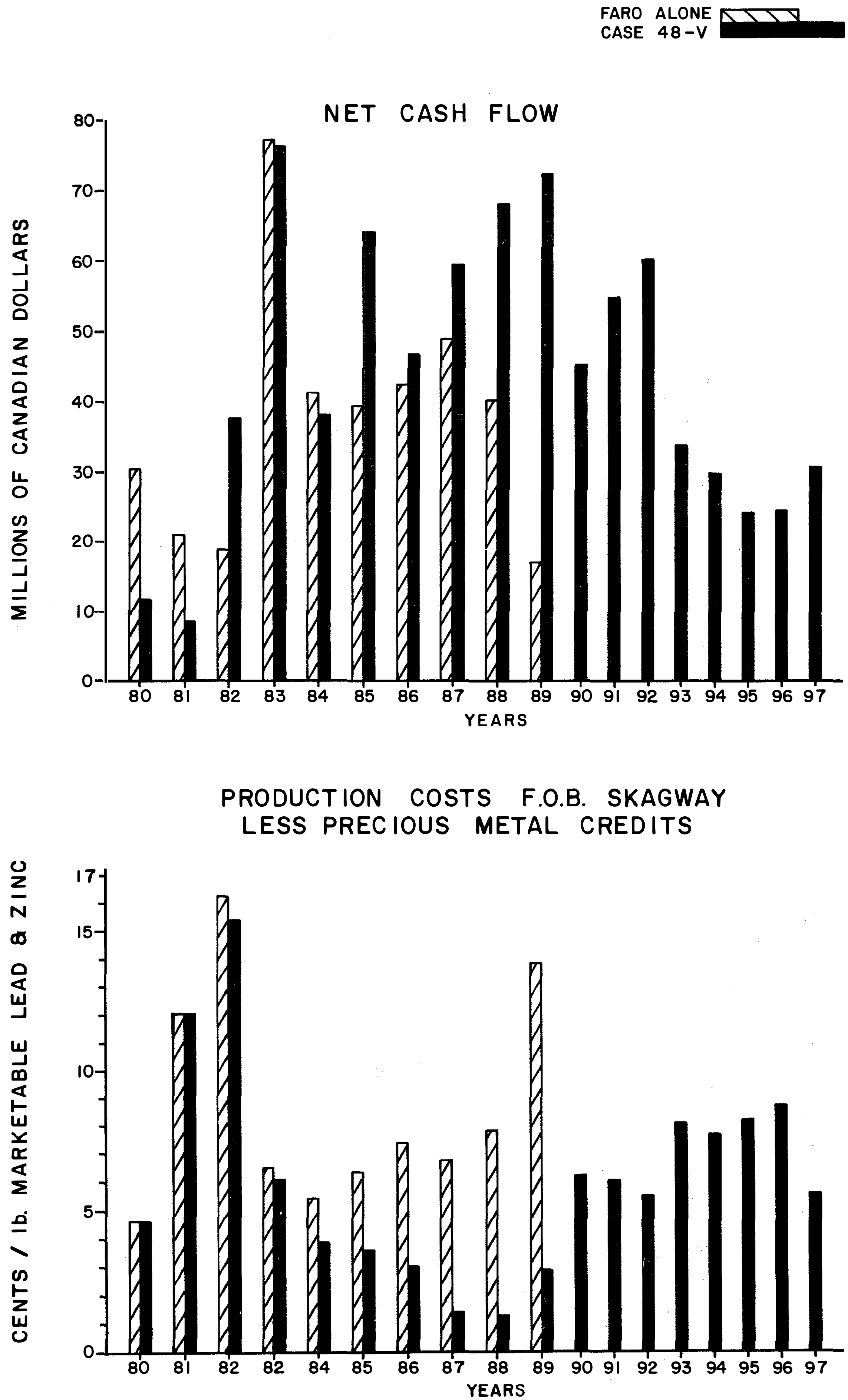


FIGURE 1-5 VANGORDA PLATEAU DEVELOPMENT SENSITIVITY ANALYSIS

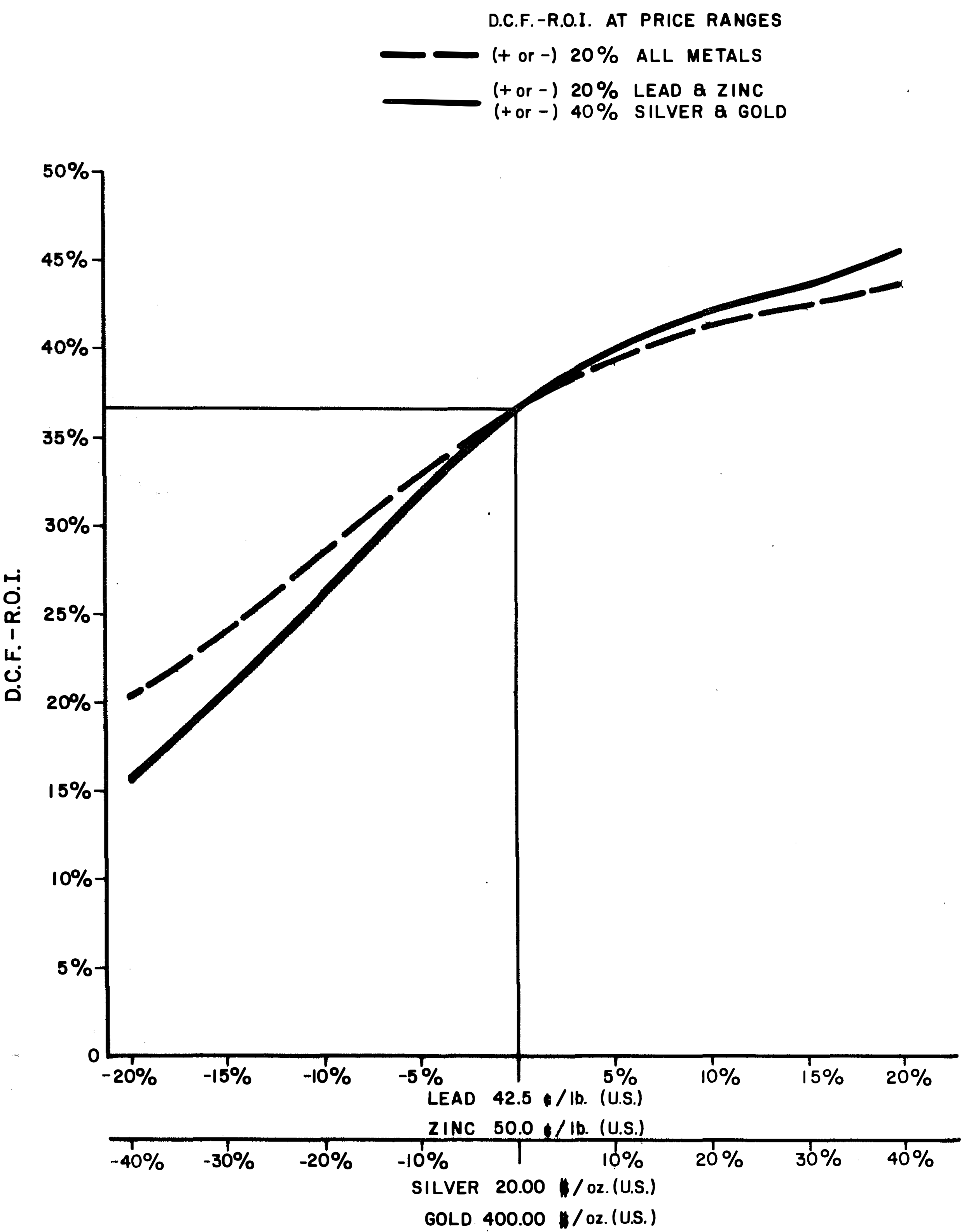
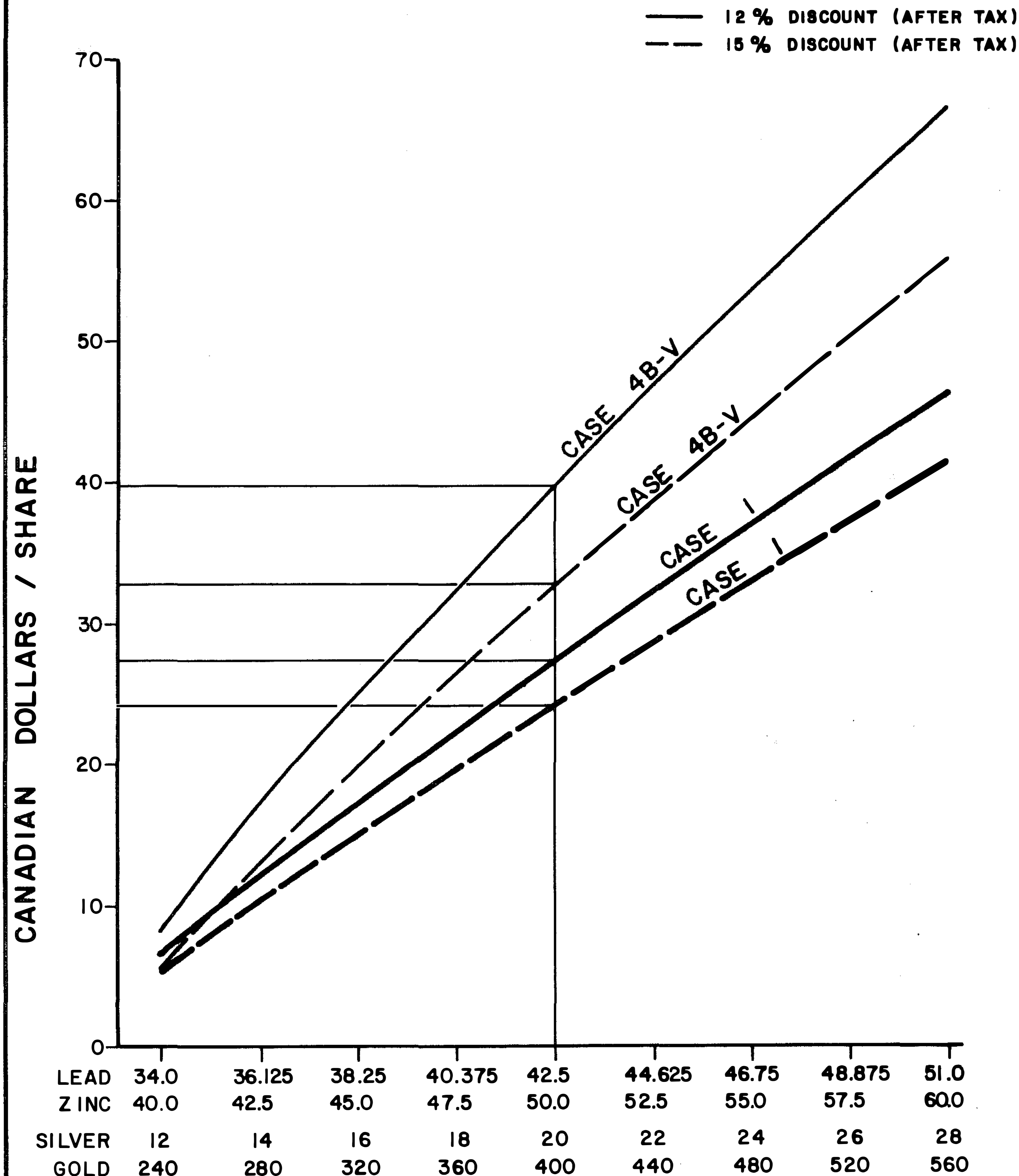


FIGURE I-6 VANGORDA PLATEAU DEVELOPMENT NET PRESENT VALUE PER SHARE OF FUTURE CASH FLOW



NOTE: LEAD and ZINC in U.S. CENTS / POUND
SILVER and GOLD in U.S. DOLLARS / TROY OUNCE

FIGURE 1-7

VANGORDA PLATEAU DEVELOPMENT

CAPITAL EXPENDITURES

CDN. $ MILLION

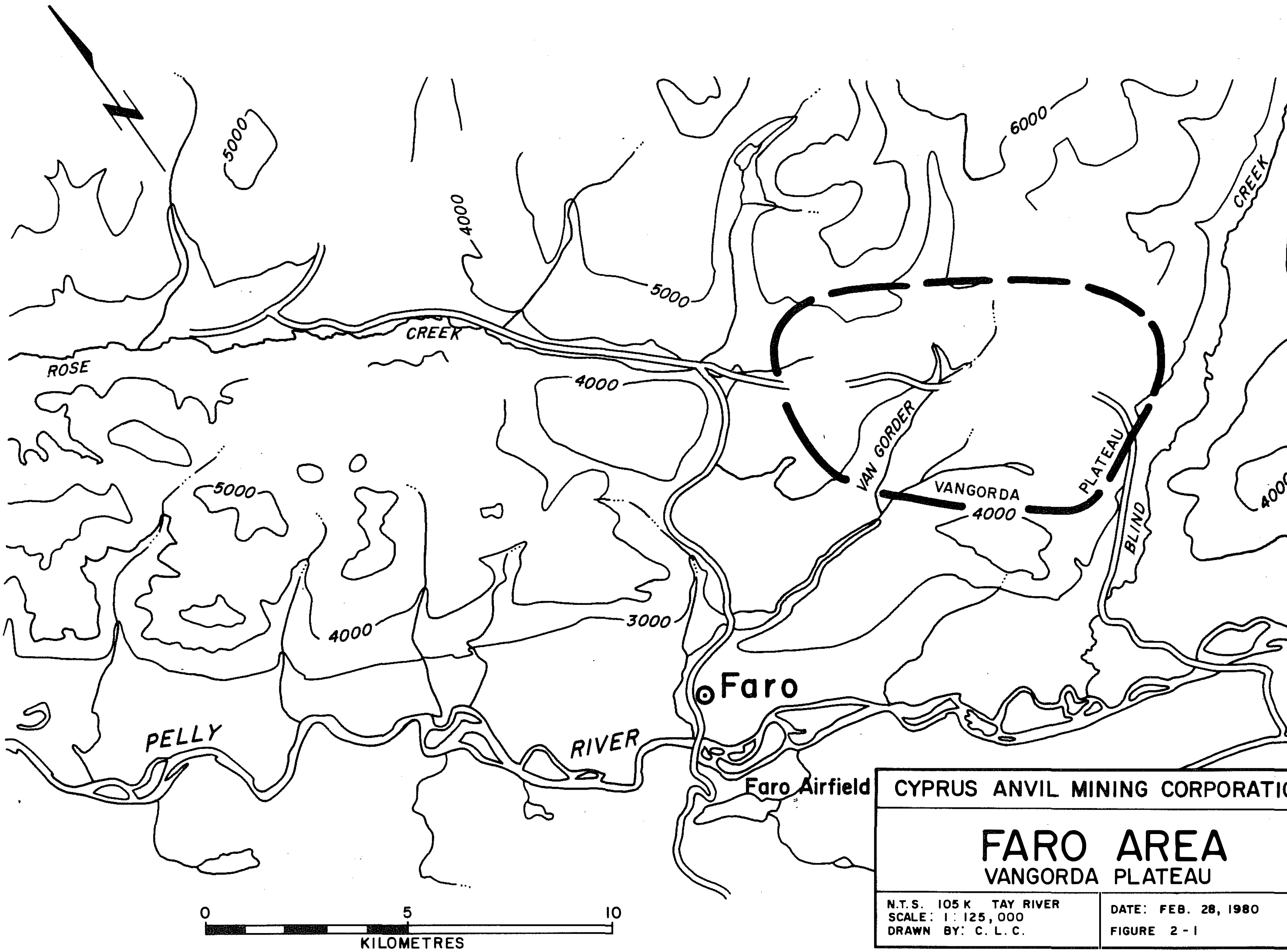
	1980	1981	1982	1983	1984	1985	1986	1987	1988	TOTAL
CAPITAL EXPENDITURES JANUARY 1980 DOLLARS										
Mill Modifications	18.970	14.411	-	-	-	-	-	-	-	33.381
Power Plant	4.610	6.390	-	-	1.000	-	-	-	-	12.000
Tailings Dam	4.420	7.580	-	-	-	-	-	-	-	12.000
Ore Haulage	-	-	-	17.150	17.150	-	-	-	-	34.300
Mine Equipment	-	-	-	-	10.585	5.120	-	-	-	15.705
Mine Facilities	-	-	-	-	6.515	-	-	-	-	6.515
Townsite	-	5.000	-	-	2.000	-	-	-	-	7.000
Pre-Production Stripping	-	-	-	-	4.000	2.520	7.111	12.050	7.500	33.181
Other	1.861	-	-	-	-	-	-	-	-	1.861
TOTALS										
Annual	29.861	33.381	-	17.150	41.250	7.641	7.111	12.050	7.500	155.943
Cumulative	29.861	63.242	63.242	80.392	121.642	129.283	136.394	148.444	155.944	
ESCALATION FACTOR										
Annual	7.5	15.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	
Cumulative		1.236	1.36	1.496	1.645	1.81	1.99	2.19	2.41	
CAPITAL EXPENDITURES ESCALATED FOR INFLATION										
Annual	32.100	41.267	-	25.656	67.856	13.830	14.151	26.390	18.075	
Cumulative	32.100	73.367	73.367	99.023	166.879	180.709	194.860	221.250	239.325	

2.
SUMMARY

2.1 INTRODUCTION

An agreement was consummated on May 15th, 1979 between Cyprus Anvil Mining Corporation and Kerr Addison Mines Ltd. and Canadian Natural Resources Ltd., whereby Cyprus Anvil acquired all of the mineral property interests of the respective companies in the Anvil District which include those of the Kerr Addison subsidiary, Vangorda Mines Ltd. The principal properties involved cover approximately 31 square kilometers in the heart of the known mineralized section of the District and include the Grum and Vangorda deposits, (Figure 2-1) both of which contain significant amounts of ore mineable by conventional open pit methods.

Engineering studies have been conducted to establish the most financially attractive sequential development program by which ores may be mined from the Faro, Grum and Vangorda deposits to feed the modified mineral processing facilities. These investigations were conducted using the mineable reserves shown below, all of which are economically available by surface mining.

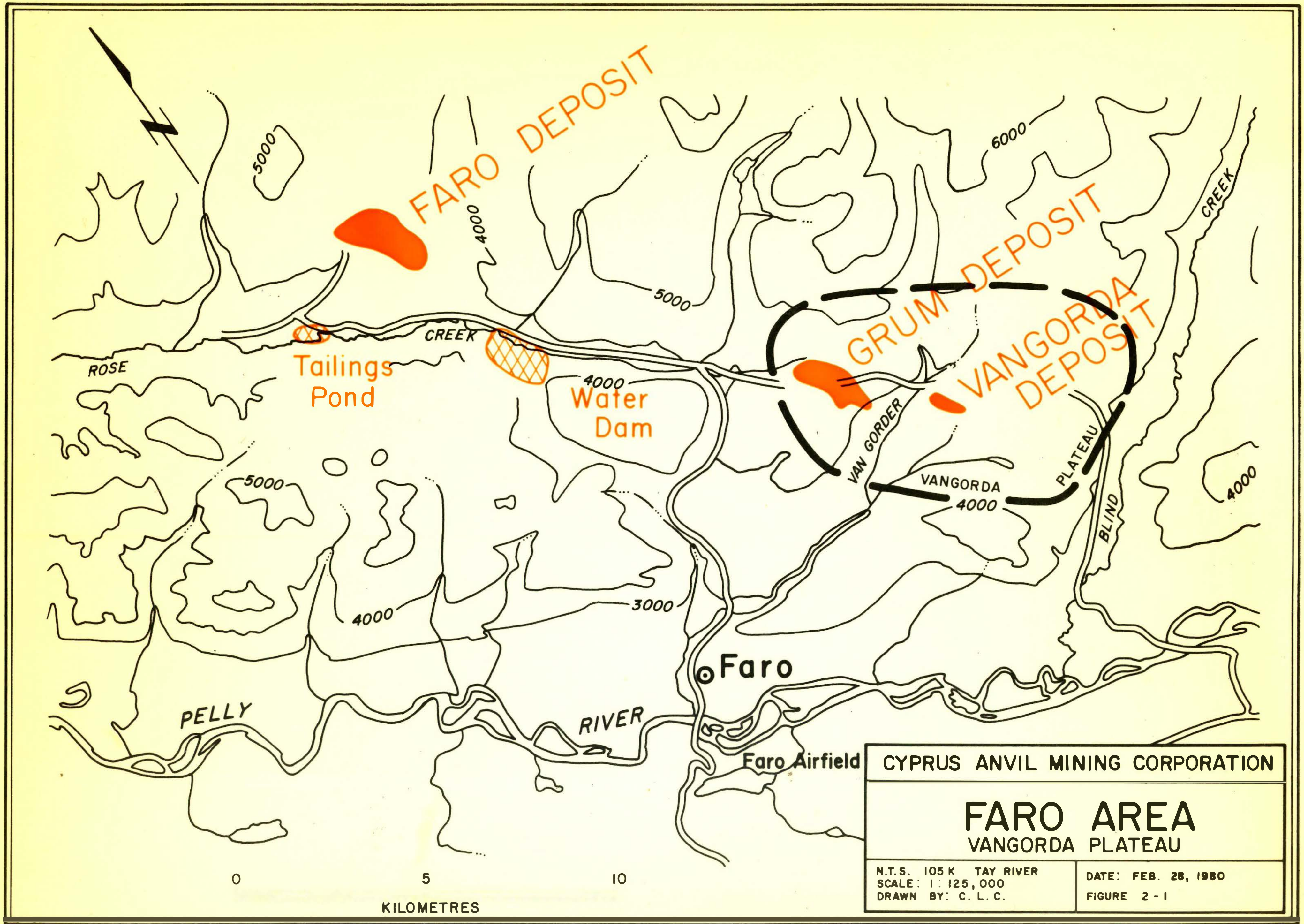


CYPRUS ANVIL MINING CORPORATION

FARO AREA
VANGORDA PLATEAU

N.T.S. 105 K TAY RIVER
SCALE: 1 : 125,000
DRAWN BY: C. L. C.

DATE: FEB. 28, 1980
FIGURE 2 - 1



FARO DEPOSIT
GRUM DEPOSIT
VANGORDA DEPOSIT
Tailings Pond
Water Dam
ROSE
CREEK
VAN GORDER
VANGORDA
PLATEAU
BLIND
CREEK
PELLY
RIVER
Faro
Faro Airfield
5000
4000
6000
3000
0
5
10
KILOMETRES
CYPRUS ANVIL MINING CORPORATION
FARO AREA
VANGORDA PLATEAU
N.T.S. 105 K TAY RIVER
SCALE: 1 : 125,000
DRAWN BY: C. L. C.
DATE: FEB. 28, 1980
FIGURE 2 - 1

FIGURE 2-2

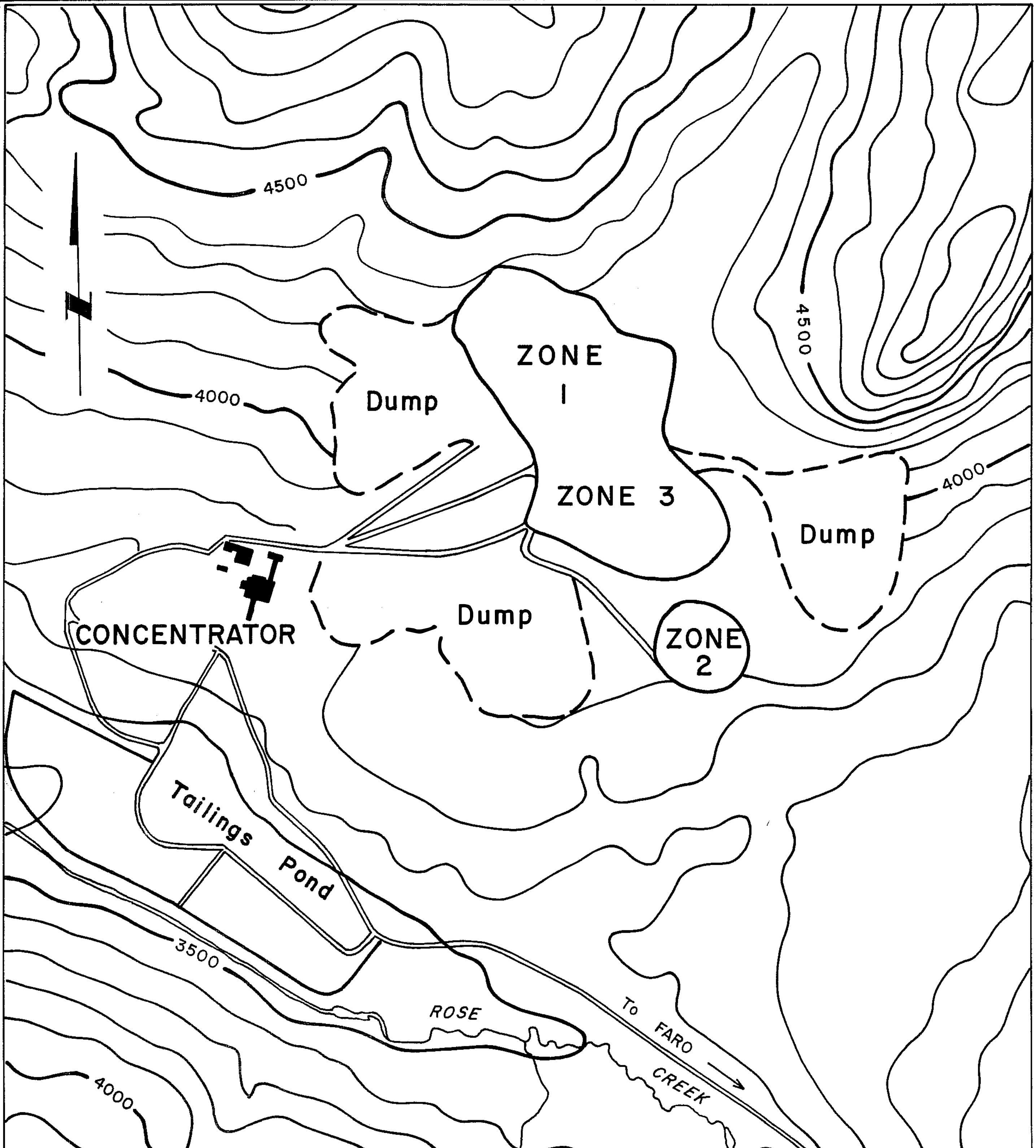
FARO, GRUM & VANGORDA DEPOSITS

MINEABLE OPEN PIT ORE RESERVES

JANUARY 1st, 1980

DEPOSIT	MINEABLE RESERVES (tonnes)	ORE GRADE LEAD Pb %	ORE GRADE ZINC Zn %	ORE GRADE SILVER Ag g/tonne
Faro - Zone I & III	26,385,000	2.9	4.6	34.8
- Zone II	3,724,000	2.7	4.3	42.5
Grum	15,583,000	3.1	5.0	47.0
Vangorda	6,134,000	3.5	4.6	50.2
Total	51,826,000	3.0	4.7	40.8

The above data is adjusted for waste dilution.



1000 0 1000 2000

SCALE 1" = 2000'

CYPRUS ANVIL MINING CORPORATION

FARO DEPOSIT

YUKON TERRITORY

FARO MINESITE

NTS 105 K – 6
SURVEY BY:
DRAWN BY: C. L. C.

DATE: MARCH 10, 1980

FIG. 2 - 3

2.2 THE FARO DEPOSIT

The Faro deposit consists of three discrete zones (Figure 2-3). Zone I has been the source of ore since the commencement of operations in late 1969. Since this ore is virtually exhausted, the 1980 ore supply will be taken almost exclusively from the Zone II which exists as a small, physically detached deposit. Thereafter ore will be mined from Zone III until the presently defined mineable reserves are exhausted in 1989, based on the present mining schedule which does not provide for the Vangorda Plateau ores.

FIGURE 2-4

FARO DEPOSITS - ANNUAL PRODUCTION SCHEDULE

YEAR	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	TOTAL
Waste (000'm^3)	9342	10872	10022	8268	7560	2678	1828	1530	382	-	52,482
Ore (000's tonnes)	3317	3404	3404	3404	3404	3404	3404	3404	3404	912	31,461
Lead (% Pb)	2.8	3.0	2.6	3.4	3.5	2.8	2.8	2.8	2.7	2.0	2.9
Zinc (% Zn)	4.4	5.2	4.7	4.8	4.8	4.2	4.2	4.5	4.4	3.9	4.6
Silver (Ag g/tonne)	41.6	31.3	28.7	40.0	44.3	39.1	36.8	34.2	28.5	22.0	35.6
Gold (Au g/tonne)											N/A

N/A: Not available at this time.

Gold assays performed on representative samples of ore from Zone III indicated that gold distribution was erratic. For this purpose it was assumed that no payable gold would be produced when milling only Zone III ore.

In addition to these in-situ ore reserves, there are approximately 1.5 million tonnes of oxidized ore assaying 2.8% lead and 4.8% Zn that are stockpiled of which 1.3 million tonnes will be milled in 1982.

The Zones I and III ore reserves quoted in Figure 2-2 are based upon a 4.0% combined lead plus zinc cut-off grade, while a 4.5% cut-off has been applied to Zone II to compensate to some degree for the generally inferior metallurgical response exhibited by this material. Ore grades and tonnages predicted using the computerized model have been adjusted to compensate for waste dilution. The "cut-off grade", for this purpose is defined as the sum of the average contained lead and zinc values below which ore will not be mined.

2.3 THE GRUM DEPOSIT.

The Grum discovery hole was drilled in the Fall of 1973. Thereafter exploration programs were conducted that included a total of 41,000 metres of surface diamond drilling, 15,000 metres of underground diamond drilling and 2,900 metres of underground development. The information obtained from these extensive programs was interpreted to develop an understanding of the complex structural geology of the Grum deposit.

The original drill hole log data was compiled and stored on magnetic tape by Noranda Mines Ltd. This information was transformed by Cyprus Anvil to a format compatible with the computerized geologic model programs currently in use and applied to calculate approximate geological reserves. The application of associated programs and basic operating principles established the phase development of the open pit, generating volumes of material to be moved and ore grades by year as shown in Figure 2-5.

FIGURE 2-5

GRUM DEPOSIT - ANNUAL PRODUCTION SCHEDULE

YEAR	1	2	3	4	5	6	7	8	9	10	TOTAL
Waste (000's m^3)	4010	4010	4010	4010	4010	4010	4010	4010	2001	0	34,081
Ore (000's tonnes)	1701	1701	1701	1701	1701	1701	1701	1701	1701	274	15,583
Lead (% Pb)	3.9	2.9	2.9	3.1	3.3	3.2	2.8	2.7	2.8	2.9	3.1
Zinc (% Zn)	6.4	4.9	4.8	5.0	5.4	5.0	4.5	4.4	4.5	4.7	5.0
Silver (Ag g/tonne)	58.5	45.3	44.8	46.5	50.4	48.5	42.6	42.1	44.1	46.3	47.0
Gold (Au g/tonne)											N/A

N/A: Not available at this time.

Since gold determinations were not performed initially on samples of the Grum core, gold values were not integrated into the preliminary computer model. Pilot plant and laboratory flotation test results indicated probable gold contents of 3.5 grams per tonne in the lead concentrate.

This schedule was based on a 4.0% cut-off grade and assumes an average mine ore production rate of 4,660 tonnes per calendar day. The ore grades and tonnages shown were adjusted to compensate for waste dilution.

2.4 THE VANGORDA DEPOSIT

Mineralized showings in the Anvil District were first located in a small outcrop on Vangorda Creek which were subsequently proven to be the surface manifestation of the Vangorda deposit.

Extensive drilling carried out by Prospectors Airways, a subsidiary of Kerr Addison Mines Ltd. in the mid 1950's, clearly established a significant mineralized tonnage but failed to produce precise information relating to ore reserves and grades due to difficulties encountered in achieving satisfactory core recoveries. Accordingly in 1979, an extensive diamond drilling program was implemented to better define the tonnage, grade and morphology of the deposit between section lines 2 West and 12 East.

Using only data generated by the 1979 diamond drill program, total approximate geological reserves were calculated for this portion of the deposit using the same basic techniques that were applied in the calculations of the Grum reserves. Mine development and production plans were then produced to yield the values shown below (Figure 2-6).

FIGURE 2-6

VANGORDA DEPOSIT - ANNUAL PRODUCTION SCHEDULE

YEAR	1	2	3	4	TOTAL
Waste (000's m^3)	2920	2920	659	-	6,499
Ore (000's tonnes)	1701	1701	1701	1031	6,134
Lead (% Pb)	3.5	3.6	3.6	3.2	3.5
Zinc (% Zn)	4.9	4.8	4.7	3.6	4.6
Silver (Ag g/tonne)	47.3	52.6	53.0	46.6	50.2
Gold (Au g/tonne)					N/A

N/A: Not available at this time.

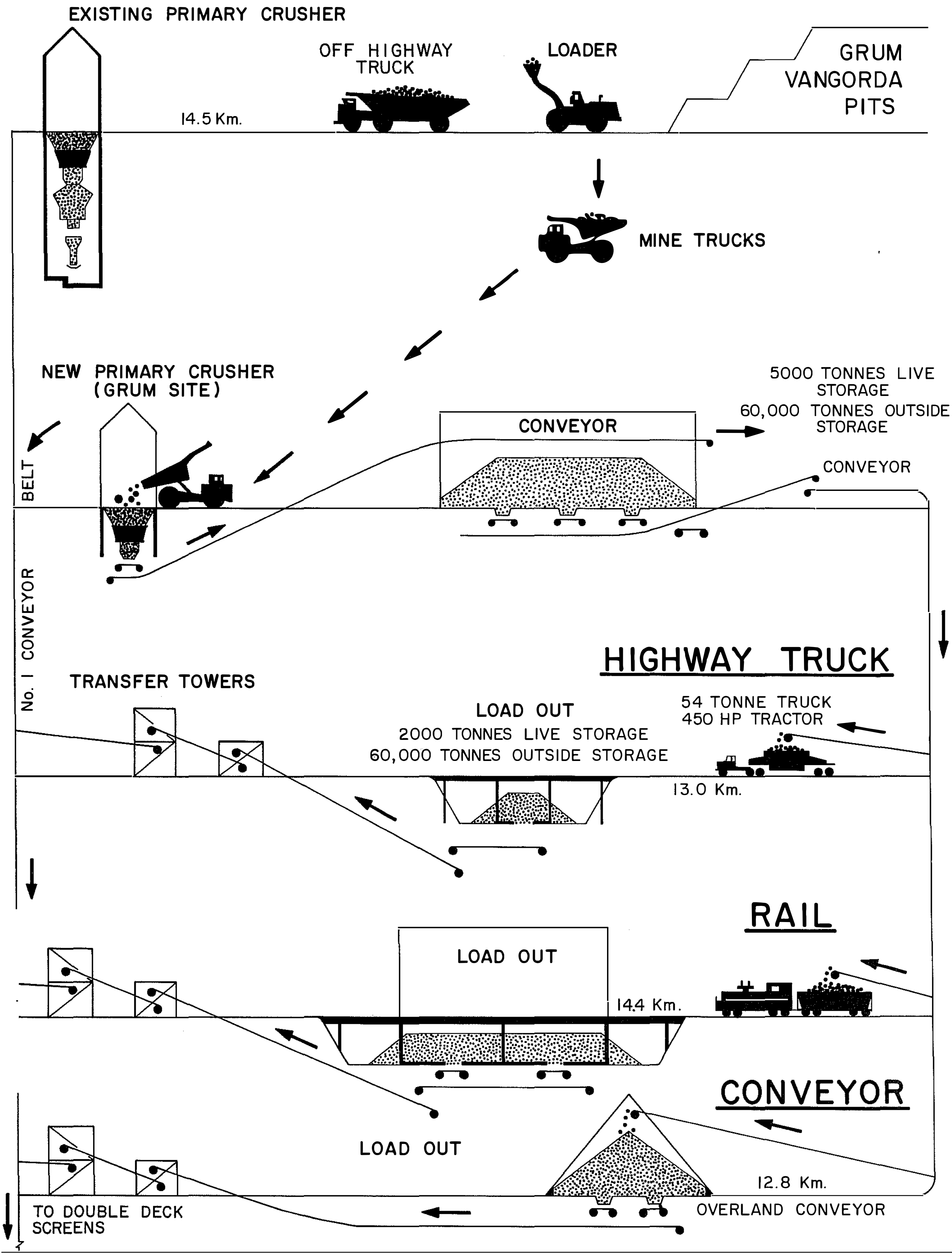
While gold determinations were carried out on the majority of sulphide core samples, the gold distribution data is incomplete at the time of writing this report. Gold contents of lead concentrate were predicted to be 6.5 grams/tonne based upon the results of laboratory testwork.

The schedule was based upon a 4.0% cut-off grade and an average mine ore production rate of 4,660 tonnes per calendar day. Appropriate factors were applied to compensate for dilution.

2.5 HAULAGE SYSTEMS

A number of systems were considered to haul ore from the Vangorda Plateau to the concentrator, including the use of off-highway trucks, highway trucks, conveyor systems and railroad. The rail, conveyor and highway truck alternatives included crushing and ore stockpiling facilities at the Vangorda Plateau. The off-highway system assumed that the haulage units would be loaded at the mine face and subsequently travel directly to the existing primary crusher at the mill site as shown in Figure 2-7.

Based upon these preliminary studies, approximate capital and operating costs were developed for each haulage system. The direct operating costs shown in Figure 2-8, do not include amortization or interest on capital but are based upon the total anticipated operating costs expressed in 1980 dollars divided by the total tonnage of ore to be transported.



ORE HAULAGE ALTERNATIVES

March 1980 P.T. /c.l.c.

FIGURE 2 - 7

FIGURE 2-8

RELATIVE COSTS OF THE HAULAGE SYSTEMS

SYSTEM	*INITIAL CAPITAL COST $ million	*AVERAGE OPERATING COST $/tonne	NUMBER OF EMPLOYEES REQUIRED
Off-Highway Trucks	32.5	1.37	37
Highway Trucks	42.4	2.12	68
Rail	38.6	1.39	33
Conventional Conveyor	44.1	1.34	30
Cable Belt Conveyor	38.1	1.65	30

* includes $100,000 per employee capital housing provision and $5,575/employee for housing maintenance per year.

Operating and Capital Costs cover transportation of ore from the pit face to the existing crushing plant.

The conventional conveyor belt alternative was rejected since the overall system design was not conducive to efficient operation in cold weather. The best highway trucking alternative utilized 450 HP tractor/bottom dump-trailer units capable of hauling a 54 tonne pay load. This labour intensive approach was rejected due to the high operating costs.

Extensive detailed engineering studies are now in progress to thoroughly evaluate the rail, conveyor and off-highway trucking alternatives. In view of the above, and for reasons given later, operating and capital costs have been applied that will provide sufficient funds for any one of the rail, conveyor or the off-highway transportation alternatives. The final selection of the transportation system to be used will be made later this year based upon the results of these studies now in progress.

2.6 CONCENTRATOR MODIFICATIONS

2.61 METALLURGY

a) Grum Ore:

Extensive metallurgical testwork was conducted on samples of Grum ores over recent years, under the direction of the Noranda Milling Committee. Results from laboratory and pilot plant testwork clearly demonstrated that optimum metallurgical performance could only be achieved by fine grinding this material to maximize the liberation of the finely disseminated minerals. These findings were confirmed by the results of detailed microscopic analyses of the various ore types within the Grum deposit.

Although the Grum metallurgical forecast used in this report depends heavily on the results from these programs, the metallurgical data inferred by Noranda were adjusted downward in this analysis to reflect the effect of mill feed grades lower than those experienced during the pilot plant work. Thus the results shown below are based on the average diluted feed grade for the Grum deposit.

FIGURE 2-9

PREDICTED PLANT METALLURGICAL PERFORMANCE - GRUM ORE

CONCENTRATE	ASSAYS LEAD Pb %	ZINC Zn %	GOLD Au g/tonne	SILVER Ag g/tonne	ARSENIC As %	MERCURY Hg g/tonne	DISTRIBUTION LEAD Pb %	ZINC Zn %	GOLD Au %	SILVER Ag %
Lead	60	11	3.5	750	0.10	90	80	*	33	65
Zinc	2.5	55	*	*	0.05	650	*	83	*	*

b) Vangorda Ore:

The earlier lack of reproducibility in metallurgical test results generated by a number of competent authorities since the discovery of the Vangorda deposit, reflects the difficulties that were experienced in acquiring adequate quantities of representative ore samples.

The 1979 Vangorda diamond drill program was based upon a closely spaced drill pattern using larger diameter drill bits to generate sufficient quantities of representative ore samples for metallurgical testwork. A program was implemented in 1979 to develop metallurgical response profiles for each of the known principal ore species within the Vangorda deposit. The forecast plant metallurgical performance was based upon the results of this testwork and weighted to compensate for the relative occurrence of these ore types within the Vangorda orebody. The metallurgical parameters shown below are predicated upon the average feed grade for the deposit.

FIGURE 2-10

PREDICTED PLANT METALLURGICAL PERFORMANCE - VANGORDA ORE

CONCENTRATE	ASSAYS LEAD Pb %	ZINC Zn %	GOLD Au g/tonne	SILVER Ag g/tonne	ARSENIC As %	MERCURY Hg g/tonne	DISTRIBUTION LEAD Pb %	ZINC Zn %	GOLD Au %	SILVER Ag %
Lead	50	*	6.5	575	0.25	60	80	*	35	55
Zinc	*	52.8	*	*	0.10	300	*	77	*	*

During the course of this comprehensive metallurgical investigation, it became evident that the best metallurgical performance could only be achieved by finely grinding the ores prior to the flotation process.

c) Faro Ore:

Testwork has been conducted to evaluate the effects of finer grinding Faro ores. The acquisition of the Vangorda Plateau deposits stimulated the implementation of comprehensive test programs to study the metallurgical responses of various species of Faro ore types at pre-determined levels of grind. Technical investigations were conducted by Cyprus Anvil, Kamloops Research & Assay Laboratory, Sachtleben Bergbau and the Mitsui Mining & Smelting Company Limited. Results from each of these independently conducted studies confirmed that the improved mineral liberation achieved through finer grinding significantly enhanced metallurgical performance.

The Figure 2-11 indicates the optimum grind levels as determined from laboratory scale flotation tests. The unit "P_{80}" represents the size (microns) of a hypothetical screen mesh through which 80% of the material would pass.

FIGURE 2-11

PRIMARY GRIND LEVEL FOR OPTIMUM METALLURGY - FARO ORE

DATA SOURCE	TYPE OF TESTS	APPROXIMATE NO. OF TESTS	OPTIMUM GRIND P_{80}
Cyprus Anvil Test Laboratory	Rougher & Cleaner Tests	100	40-45
Kamloops Research Laboratory	Rougher & Cleaner Tests	120	45-50
Sachtleben	Cleaning Testing	30	52
Mitsui Mining & Smelting	Rougher & Cleaner Tests	50	37-70

Pilot plant testwork conducted in 1979 on two samples of Faro ores, were carried out at selected levels of grind to generate results that could be used to determine the economic optimum metallurgical flowsheets. In this case it was determined that the flowsheet which produced the best metallurgical performance also generated the most favourable return on investment.

The results of the pilot plant testwork obtained when treating the sample considered most typical of Zone III ores, were adjusted to compensate for atypical levels of sample oxidation. This data was then used as a basis to predict plant metallurgical performance.

FIGURE 2-12

PREDICTED PLANT METALLURGICAL PERFORMANCE - FARO ORE

ORE SOURCE	CONC.	ASSAYS LEAD Pb %	ZINC Zn %	GOLD Au g/tonne	SILVER Ag g/tonne	ARSENIC As g/tonne	MERCURY Hg %	DISTRIBUTION LEAD Pb %	ZINC Zn %	GOLD Au %	SILVER Ag %
Faro Zone III	Lead	67	*	0.6	600	0.03	40	87.5	*	33	65
	Zinc	*	53.5	*	*	0.01	300	*	88.5	*	*

The present concentrator design is predicated upon the production of three concentrates; namely selective lead, selective zinc and a bulk concentrate. The modified plant flowsheet makes no such provision for the production of bulk concentrate, initially intended to provide an economic outlet for those flotation middlings that were incapable of producing acceptable selective concentrate grades without finer grinding. It is clear from results of recent work that the higher degree of mineral liberation achieved through finer grinding obviates the need to produce bulk concentrate, which is a less profitable product.

d) Ore Type Compatibility:

Testwork has been carried out by Lakefield Research to investigate the metallurgical compatibility of the subject ores. It has been determined that the flowsheet selected provides the optimum flotation environment for all the ores under investigation. Further it is noted that while each ore type within its respective deposit exhibits differing metallurgical response, concurrent milling of two discrete ore species produces no adverse effect on either of the two constituent metallurgical results.

2.62 MODIFIED CONCENTRATOR DESIGN

The forecast metallurgical performance of each ore will only be achieved by grinding finer and providing a controlled flotation environment which depends upon longer flotation times and lower pulp densities than those currently in use.

The finer grind will be achieved by incorporating an additional grinding circuit into the modified mill design. The required flotation residence times and pulp densities will be attained within the confines of the existing flotation section by replacing existing flotation machines with larger and more efficient units. The elimination of bulk concentrate production will permit the re-allocation of thickener capacities to accommodate the finer concentrates produced. Existing disc filters will be replaced with larger units to provide the additional filtering capacity required to effectively dewater the finer concentrates. The filter performance will be further enhanced by the application of heat to the filter feed pulps.

The Anvil district ores are metallurgically complex, consisting of multiple finely disseminated minerals contained in generally massive sulphide matrices. It is known that each deposit consists of several readily identifiable ore types that exhibit differing metallurgical characteristics. To maximize plant metallurgical efficiencies certain sophisticated, well-proven instrumentation systems will be installed that will monitor, and in some cases control critical operating parameters. Though basic initially, this instrumentation will be capable of organized development in accordance with long term process control strategies.

Figure 2-13 schematically represents the modified plant flow-sheet and Figure 2-14 provides an appreciation of the new physical layout of the Anvil concentrator.

The capital cost of the modifications to the concentrator will be $33.4 million. The operating costs were increased to reflect the higher consumptions of power, grinding liners and media.

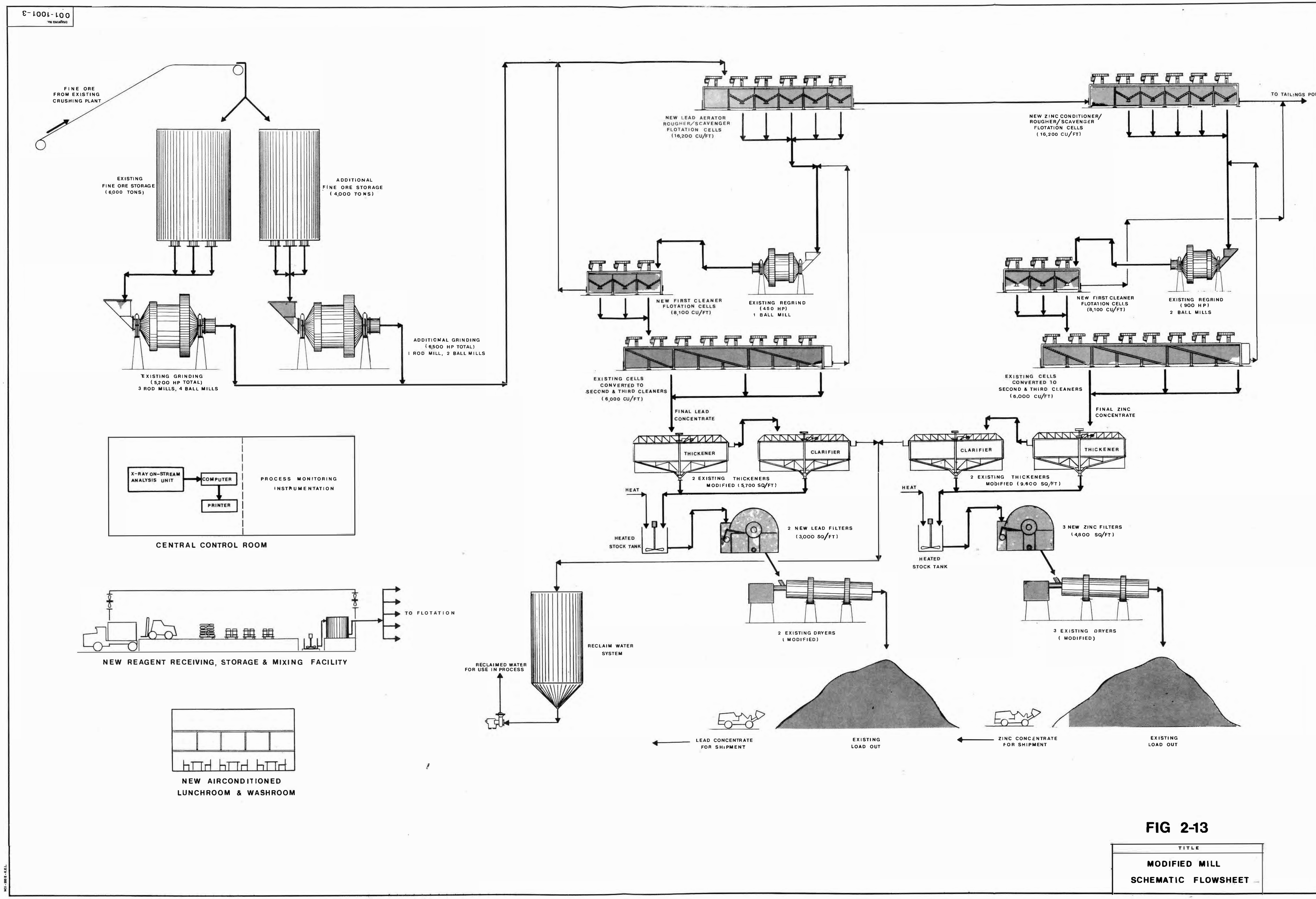


FIG 2-13

TITLE

MODIFIED MILL SCHEMATIC FLOWSHEET

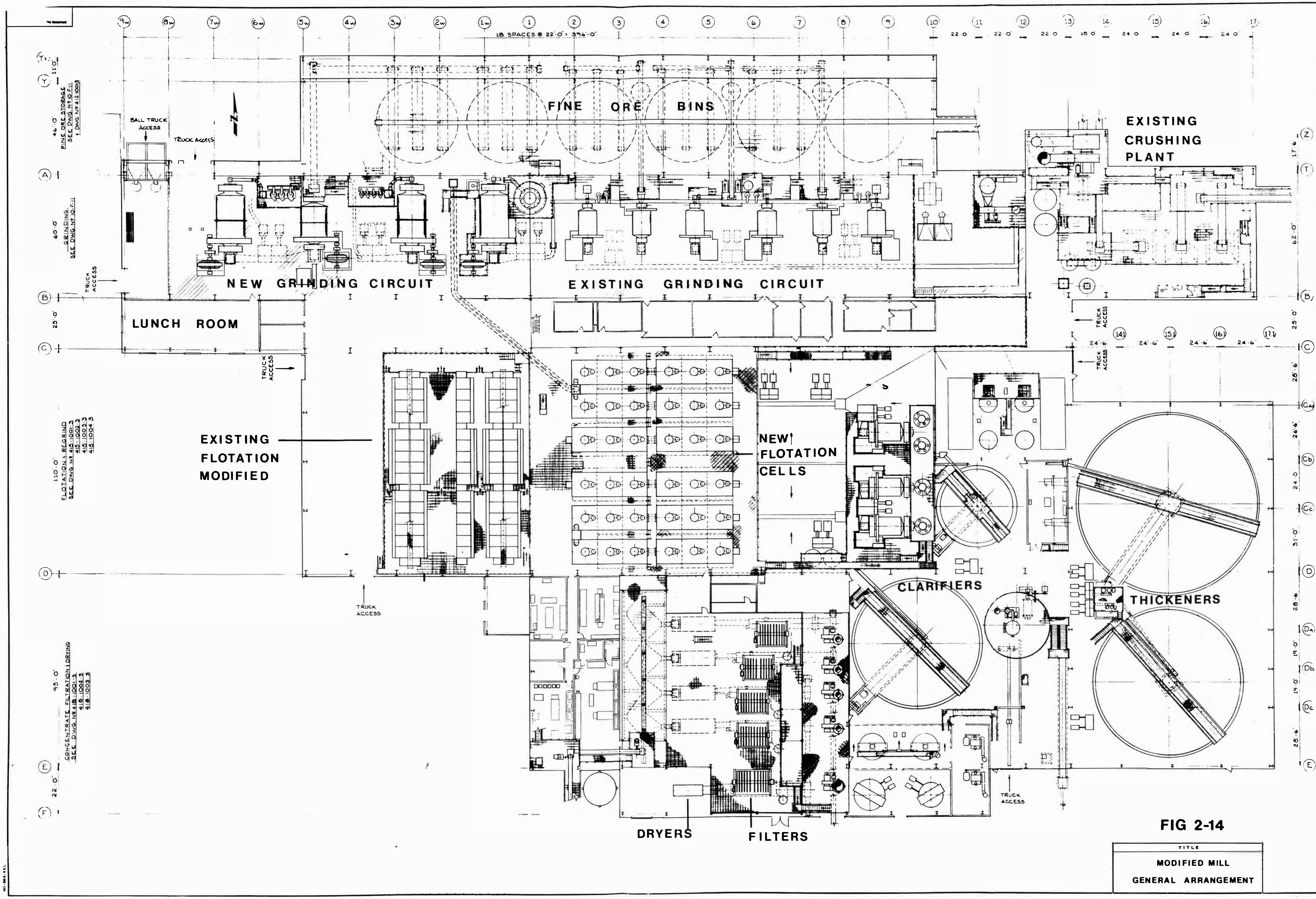


FIG 2-14

TITLE
MODIFIED MILL
GENERAL ARRANGEMENT

2.7 POWER SUPPLY

The Northern Canada Power Commission (N.C.P.C.) presently supplies hydroelectric and diesel generated power to service the minesite and the town of Faro by means of its operating facilities at Whitehorse and Aishihik. The power is transmitted to Faro via a 138 kV overhead line.

The current minesite electrical load, has a maximum demand of 16 megawatts (MW) and an annual energy consumption of approximately 100 million kilowatt hours (100 KW hrs).

The development of the Vangorda deposits and attendant mill modifications will impose an additional maximum 9 MW load on the system, most of which is scheduled to be on line by January 1st, 1982.

FIGURE 2-15

MINESITE POWER STATISTICS

	TOTAL CONNECTED HP	MAXIMUM DEMAND (MW)	ENERGY CONSUMPTION (GW hr/year)
Proposed	46,470	25	155
Present	30,006	16	100
Increase	16,464	9	55

While N.C.P.C. will provide some of this additional energy, it will be necessary to install supplementary electric generating facilities interconnected with the utility grid to ensure the maintenance of adequate system reserve capacity. Failure to provide such a plant would increase the risk of power shortages to Cyprus Anvil thereby jeopardizing the continuity of production at planned levels.

For this purpose, it was assumed that one generating unit would be located in close proximity to the concentrator to provide normal system reserve and long term emergency power supply. It was further assumed that two generating units would be installed adjacent to the existing N.C.P.C. power plant in Faro. The total cost of these facilities, the transmission line and transformers will be $12.0 million.

The N.C.P.C. propose to install an additional hydroelectric turbine in the Whitehorse plant to be on line by mid 1983. Further the Commission has initiated studies to evaluate potential sites for small hydroelectric developments in the Yukon, some of which are within a 150 kilometre radius of Faro. As these studies proceed and system load forecasts are refined, the required power generating capabilities will be constantly reviewed to examine the possibility of reducing the size of the proposed plant at Faro and the premium to be paid for diesel electric energy.

2.8 ENVIRONMENTAL CONSIDERATIONS

New tailings impoundment facilities will be required to accommodate the increased total tonnages of finer tailings, and to provide additional time for water clarification. Dykes, creek diversion canals and related structures will be designed to ensure compliance with all regulatory requirements. Capital cost of the new facilities will be $12.0 million.

The modified mill flowsheet depends upon the internal recirculation of certain process waters to maintain fresh water consumption at current levels while operating the flotation circuit at lower pulp densities. The use of this recycled water represents one of the first steps in a long term strategy designed to minimize the use of fresh water.

Base line studies have commenced to monitor the indigenous flora, fauna, fish populations and water qualities in those areas most likely to be affected by future mine developments. Programs will be established to reduce to an absolute minimum any detrimental impact future mining activities may impart to the natural environment.

Building design, mechanical layouts and equipment selections were conducted with a view to improving the working environment within the modified mill by reducing spillage and minimizing air and noise pollution. By so doing, the Company has the opportunity to demonstrate its sincere interest in the health and welfare of its employees which will be reflected in improved morale.

2.9 OUTLOOK FOR LEAD AND ZINC

2.91 LEAD

After four consecutive years of declining statistical balances for lead metal, recent statistics indicate the world lead market will move into a small surplus position in 1980. The primary explanation for this lies in the major decline to date of the North American automobile market which affects the demand for storage batteries, the most important end-use of lead metal. Prices in 1979 were dramatically improved but mine production increased only slightly since long lead times are required before production can actually respond to improved market conditions.

In 1980 the first major lead mine in ten years has come on stream - the Aggeneys Mine in South Africa producing about 90,000 tonnes of lead in concentrate annually. The ability of the lead smelting industry in Europe to cover its concentrate needs over the next six months has been somewhat relieved by the arrival of these concentrates some four or five months ahead of the original schedule. Production and milling problems continue to restrict mine output at Aznacollar (Spain) and Woodlawn (Australia). However, a concern over long term lead supply arises from the fact that less than one third of all lead is mined from lead ores while projected developments indicate a continued preponderance of zinc in combined orebodies.

Secondary recovery is more important in lead than for any other metal, contributing about 38% of feed to refineries. However, secondary smelters in industrialized nations face strict environmental regulations which raise the break-even prices needed to sustain production; moreover, the supply of scrap lead is quite unpredictable. The current scrap shortage in Europe illustrates how quickly inventories in one region can be depleted by high prices, how quickly scrap metal can flow to the shortage area, (in this case from the U.S.), and how at the same time weakening battery demand can create conditions for further market imbalance.

Forecasts of lead consumption vary but even a conservative growth trend rate of 2.0% from 1979 levels with adjustments for the slowdown in 1980 maintain the supply/demand balance in a reasonably balanced state. Major mine additions as occur in 1980 and 1983 are quickly absorbed. The long run supply curve for lead can be expected to parallel the costs of bringing new production on stream given:

a) Declining grades in existing orebodies.

b) The decreasing ratio of lead to zinc in anticipated combined orebodies, which are likely to be developed in the future.

c) The costs of complying with strict environmental regulations affecting both primary and secondary producers of metal.

2.92 ZINC

In the last two years some major adjustments have occurred in the zinc industry which, barring a severe and prolonged recession in 1980, should provide the foundations for a more profitable future for producers of both concentrates and metal.

1978: Mine production declined by 2.0% from the historical high of 1977; metal production remained virtually constant. Since consumption rose dramatically by about 7.7% from the previous year, an extremely large adjustment to metal inventories took place, close to a 400,000 tonnes decline.

1979: Mine production declined slightly while metal production jumped by 10.3%. Consumption of metal still managed a 2.4% increase over 1978. Accordingly, there occurred a substantial drawdown on zinc metal in concentrate inventories of about 330,000 tonnes.

Not surprisingly, a zinc concentrate shortage has emerged early in 1980. Production of both metal and concentrates so far this year is lower than many industry observers had originally estimated. Demand for metal should remain sluggish as the United States moves into recession.

The full impact of the concentrate shortage will begin to be widely felt in 1981 as the western economies begin to recover and the demand for zinc assumes a 2.0% growth trend. As smelters try to increase capacity utilization by increasing production they will be severely constrained by the shortage of zinc concentrates. The price of zinc will have to rise in response to these forces and such movement will be encouraged by the cost of energy and currency exchange pressures experienced by existing smelters in Japan and Europe.

Rising concentrate and metal prices, security of concentrate supplies and relatively lower energy costs will benefit integrated Canadian zinc producers over the next five years.

2.10 FINANCIAL ANALYSIS

The financial analyses are based on a comparison of cash flows generated by processing the reserves outlined in the current operating plan with those realized by milling the Vangorda Plateau ores concurrently with those from the Faro pit. These cash flows exclude dividends, exploration expenditures and interest other than 1980 commitments and continuing interest on housing mortgages. The cash flows do provide for those capital costs required for the existing mining programs and the cost of routine equipment replacements.

A number of development cases were examined, varying the time and sequence of pit operations (Section 5). The most attractive case, identified as 4B-V requires that the mill modifications be completed by 1982. Vangorda ore would be supplied to the mill from 1985 to late 1988 with ore from the Faro pit in equal proportions. When exhausted the Vangorda ore, would be replaced by material from the Grum in 1988.

The basic assumptions used in the construction of these cases are:

a) Open pit reserves of the Vangorda Plateau deposits are as shown in Figure 2-2.

b) The capital expenditures shown in Figure 2-18 are based on January 1980 dollars.

c) Ores from all sources are compatible for milling in any given ratio without adverse effects on the metallurgical performance.

d) Mining costs are based on experience with the Faro operation. Contract pre-production stripping costs for the Vangorda deposit are based on the results of an independent study. Pre-production stripping unit costs for the Grum deposit are based upon experience.

e) Milling costs will increase by about ten percent due principally to increased steel consumption. Power costs have been increased to reflect the high price of diesel generated power.

f) Transportation to tidewater and terminal costs are at present levels.

g) Ocean freight costs will be as follows:

FIGURE 2-16

OCEAN FREIGHT RATES U.S. $/tonne

	1980	1981	1982 AND ONWARD
To Japan	20.50	22.00	22.00
To Europe	39.50	40.00	40.00

h) All concentrate will be sold at treatment charges at current levels under Japanese and European concentrate sales contracts.

i) The price and exchange assumptions shown below are expressed in 1980 dollars.

FIGURE 2-17

METAL PRICE AND EXCHANGE RATE ASSUMPTIONS

	1980	1981	1982 AND ONWARD
Zinc U.S. ¢/lb	39.23	44.0	50.0
Lead U.S. ¢/lb	43.0	42.5	42.5
Silver U.S. $/oz	24.00	20.00	20.00
Gold U.S. $/oz	456.00	400.00	400.00
U.S. 1$ = CDN. $	1.124	1.050	1.025

j) No residual values for either plant or equipment have been included in these calculations.

k) In the opinion of the company's tax counsel, each of the Grum and Vangorda deposits will constitute a "new mine" and all expenditures being considered, with the possible exception of the tailings dam, will be subject to the fast write-off provisions for new mines (100%).

With the exception of railway track and related property, and possibly the tailings dam, all expenditures will also qualify for earned depletion (33.3%). The company is seeking an advance tax ruling from the Department of National Revenue confirming the opinion of counsel.

As in any analysis of this nature the most subjective area and also the area of greatest impact is that of metal prices. Sensitivities have been calculated and the results tabulated in section 10 of this report and illustrated graphically in the Program section. These indicate that should equilibrium prices be 20% lower for lead and zinc and 40% lower for silver and gold than those expected, the DCF/ROI is 15.5% as opposed to 36.6% DCF/ROI in the base price comparison. If prices are 20% and 40% respectively, higher than those expected the DCF/ROI is 45.6%.

Other sensitivities were run using equilibrium prices as follows:

Feed grades + 10%	- DCF/ROI	40.4%
- 10%	- DCF/ROI	31.0%

Operating costs + 10%	-	32.7%
Capital costs + 10%	-	33.2%

FINANCIAL COMPARISONS

(Values Reported as $MM)

	CASE 1	CASE 4B-V	DIFFERENCE
DCF/ROI	-	36.6	-
N.P.V. at 12 percent	209.4	305.1	95.7
at 15 percent	184.6	252.0	67.4
Value per Share			
12 percent	27.28	39.75	12.47
15 percent	24.05	32.83	8.78

The 10 year cash flow included in section 10 indicates that assuming dividends and exploration are continued at current levels, the only year in which annual cash generation is less than expenditure is 1981. This deficiency is covered by cash on hand at the end of 1980. Under the same assumption, should metal prices for lead and zinc be 10% lower than expected and silver and gold 20% lower than expected, maximum year end debt (1981) would be in the order of $35,000,000, well within the company's borrowing potential.

FIGURE 2-18

CAPITAL EXPENDITURES

VANGORDA PLATEAU DEVELOPMENT: CASE 48-V

(IN CONSTANT JANUARY 1980 $000's)

	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	TOTAL
VANGORDA PLATEAU DEVELOPMENT																	
Mill Modifications	18.970	14.411	-	-	-	-	-	-	-	-	-	-	-	-	-	-	33.381
Power Plant	4.610	6.390	-	-	1.000	-	-	-	-	-	-	-	-	-	-	-	12.000
Tailings Dam	4.420	7.580	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12.000
Ore Haulage	-	-	-	17.150	17.150	-	-	-	-	-	-	-	-	-	-	-	34.300
Mine Equipment	-	-	-	-	10.585	5.120	-	-	-	-	-	-	-	-	-	-	15.705
Mine Facilities	-	-	-	-	6.515	-	-	-	-	-	-	-	-	-	-	-	6.515
Townsite	-	5.000	-	-	2.000	-	-	-	-	-	-	-	-	-	-	-	7.000
Pre-Production Stripping	-	-	-	-	4.000	2.520	7.111	12.050	7.500	-	-	-	-	-	-	-	33.181
Other	1.861	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.861
ANNUAL TOTALS	29.861	33.381	-	17.150	41.250	7.641	7.111	12.050	7.500	-	-	-	-	-	-	-	155.943
CUMULATIVE TOTALS	29.861	63.242	63.242	80.392	121.642	129.283	136.394	248.444	155.944	-	-	-	-	-	-	-	-
CAPITAL BUDGET ESCALATED FOR INFLATION																	
Annual	32.100	41.267	-	25.656	67.856	13.830	14.151	26.390	1B.075	-	-	-	-	-	-	-	-
Cumulative	32.100	73.367	73.367	99.023	166.879	180.709	194.860	221.250	239.325	-	-	-	-	-	-	-	-
ESCALATION FACTOR																	
Annual	7.5	15.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	-	-	-	-	-	-	-	-
Cumulative	-	1.236	1.36	1.496	1.645	1.81	1.99	2.19	2.41	-	-	-	-	-	-	-	-
CONTINUING CAPITAL																	
Present Mining Plan	17.265	6.475	2.100	2.000	2.000	2.000	2.000	1.000	-	-	-	-	-	-	-	-	34.840
Development Plan	-	-	-	1.000	1.000	1.000	1.000	2.000	3.000	3.000	3.000	3.000	2.000	1.000	1.000	1.000	23.000

FIGURE 2-19

VANGORDA PLATEAU DEVELOPMENT

CASE 4B-V PRODUCTION STATISTICS

YEAR	TOTAL MINE PRODUCTION WASTE (M^3 000)	ORE (000 tonnes)	FEED GRADE LEAD %	ZINC %	LEAD CONC. GRADE % Pb	LEAD RECOVERY %	ZINC CONC. GRADE % Zn	ZINC RECOVERY %	CONCENTRATE PRODUCTION LEAD (DMT)	ZINC (DMT)	PAYABLE METAL PRODUCTION LEAD (000/lbs)	ZINC (000/lbs)	SILVER (000/oz)	GOLD (oz.)
1980	9,102	3,322	2.83	4.41	58.23	77.70	50.53	80.39	114,713	213,968	157,155	239,693	2,546	3,044
1981	10,872	3,404	3.00	5.18	61.00	84.00	50.40	80.00	130,296	260,440	178,566	260,060	1,907	-
1982	10,022	3,404	2.60	4.65	61.64	83.39	51.60	81.89	119,733	251,202	154,572	241,457	1,700	-
1983	8,268	3,404	3.40	4.80	67.00	87.50	53.50	88.50	151,148	270,284	212,095	270,971	3,030	-
1984	7,560	3,404	3.50	4.80	67.00	87.50	53.50	88.50	155,593	270,284	218,333	270,971	3,459	-
1985	4,400	3,402	3.13	4.56	56.77	83.36	53.14	82.30	156,096	240,226	185,426	239,086	2,862	18,685
1986	4,300	3,402	3.22	4.48	56.57	83.27	53.15	82.40	160,995	236,009	190,526	234,921	2,948	19,539
1987	1,333	3,402	3.18	4.43	56.66	83.31	53.15	82.46	158,818	233,529	188,259	232,471	2,835	19,160
1988	2,400	3,402	3.14	4.46	59.39	83.35	53.75	84.11	149,752	237,416	186,250	239,115	3,067	14,303
1989	4,800	3,402	3.16	5.15	63.07	85.32	54.31	85.40	142,034	275,505	187,602	280,390	3,169	8,756
1990	4,700	3,402	2.87	4.69	63.38	83.66	54.25	85.64	128,653	251,889	170,787	256,050	2,591	7,699
1991	4,383	3,402	2.85	4.65	63.29	83.56	54.26	85.60	127,792	249,332	169,380	254,394	2,406	7,830
1992	4,000	3.402	2.92	4.78	63.20	83.47	54.28	85.53	131,194	256,263	173,655	260,637	2,485	8,099
1993	4,000	2,613	2.77	4.72	61.74	81.89	54.54	84.59	95,943	191,292	124,057	195,501	1,916	8,276
1994	4,000	1,701	3.15	4.84	60.00	80.00	55.00	83.00	71,442	124,241	89,776	128,049	1,636	7,741
1995	2,500	1,701	2.81	4.51	60.00	80.00	55.00	83.00	63,731	115,770	80,086	119,318	1,459	6,906
1996	2,500	1,701	2.78	4.50	60.00	80.00	55.00	83.00	63,050	115,513	79,231	119,054	1,444	6,832
1997	581	1,306	2.75	4.50	60.00	80.00	55.00	83.00	47,887	88,689	64,228	96,095	1,170	5,447

N.B. Bulk production will be 34,724 DMT and 35,000 DMT in 1980 & 1981 respectively.
Thereafter no bulk concentrate will be produced.

3.
THE GRUM DEPOSIT

3.1 REVIEW OF EXPLORATION AND DEVELOPMENT 1973-1976.

The Grum deposit was discovered in 1973 by the late Dr. Aaro Aho of the A.E.X. Syndicate while drilling a broad area of high Turam response and a coincident gravity anomaly in the general area of the Champ Deposit.

A.E.X. Syndicate drilled 16 holes totalling 4,000 metres to July 1974 when Kerr Addison Mines Limited assumed control of the project and drilled a further 13,000 metres of surface diamond drill holes spaced 60 metres apart on section lines 120 metres apart. This work indicated a deposit 1,680 metres long and 366 metres wide between sections 62W and 88W.

Upon completion of this initial drilling phase, an extensive underground development program was initiated. The deposit was explored by means of an 800 metre decline on a 16 percent grade. Two ramps, 120 metres apart, were driven along the plunge of the mineralization with connecting cross cuts at 120 metre intervals. Ring drilling in the plane of the geologic sections was conducted from the ramps at 60 metre intervals. The program included 2,900 metres of openings and 15,000 metres of diamond drilling.

Concurrent with the underground development, a further 24,000 metres of surface diamond drilling was carried out to complete the grid at 60 metre spacing on lines 60 metres apart from section 60W to 88W. Fill-in holes at 30 metre spacing were also drilled on sections 64W and 68W (Figure 3-7).

3.2 GEOLOGY

Both the Grum and Vangorda deposits occupy similar positions within the regional stratigraphy of the Anvil Range. The section is generally dominated by calcareous chlorite and chlorite-sericite phyllites in the hanging wall, and non-calcareous biotite-sericite phyllites and biotite-garnet-staurolite schists in the footwall. It appears that the amphibolite facies footwall sequence is separated from the hanging wall green schist facies by a low angle thrust fault at the base of the sulphide horizon. The sulphide horizons are closely associated with graphitic phyllite.

The sulphide horizons and enclosing rock units have been subjected to at least four periods of plastic deformation. The main structure controlling the distribution of rock type is a North-Westerly plunging series of more or less concentric "S" shaped recumbent folds. This folding was accompanied by a pervasive axial plane foliation which in turn has been kink folded and gently warped into broad synforms and antiforms. The deposit has been further complicated by North East striking high angle faults which cut all structures except the footwall thrust fault.

Though the sulphide horizon stratigraphy is complicated by overturned folds, two basic horizons generally predominate. The upper horizon consists of massive pyritic sulphides, variably base metal bearing and variably baritic. The lower horizon consists of pyritic quartzites, banded or foliated and generally deficient in base metals.

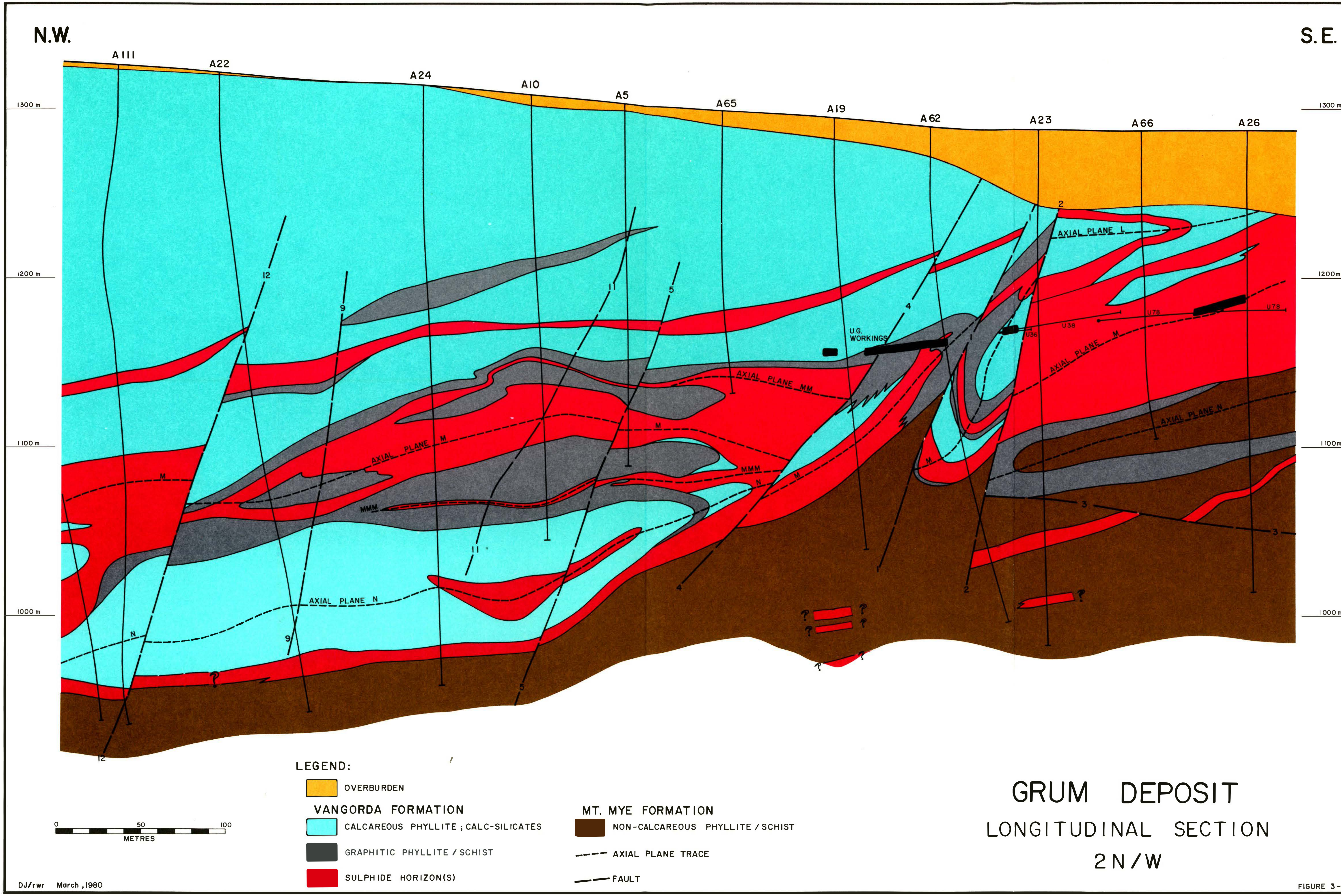
3.3 THE DEVELOPMENT OF GEOLOGICAL RESERVES

3.31 THE CONSTRUCTION OF GEOLOGICAL SECTIONS

Geological sections and plans were constructed on a 1:500 scale by plotting data obtained from the underground development and the diamond drilling results. Since the deposit is both strataform and stratabound it was possible to treat the mineralized zones as rock types when constructing geological sections. Geological plans were constructed from the sections at 9 metre intervals corresponding to the mine levels. Simplified geological longitudinal and cross sections are shown in Figures 3-1 and 3-2 respectively. Equivalent sections of the Faro deposit are shown in Figure 3-3 and 3-4 to demonstrate the relative structural complexity of the Grum deposit.

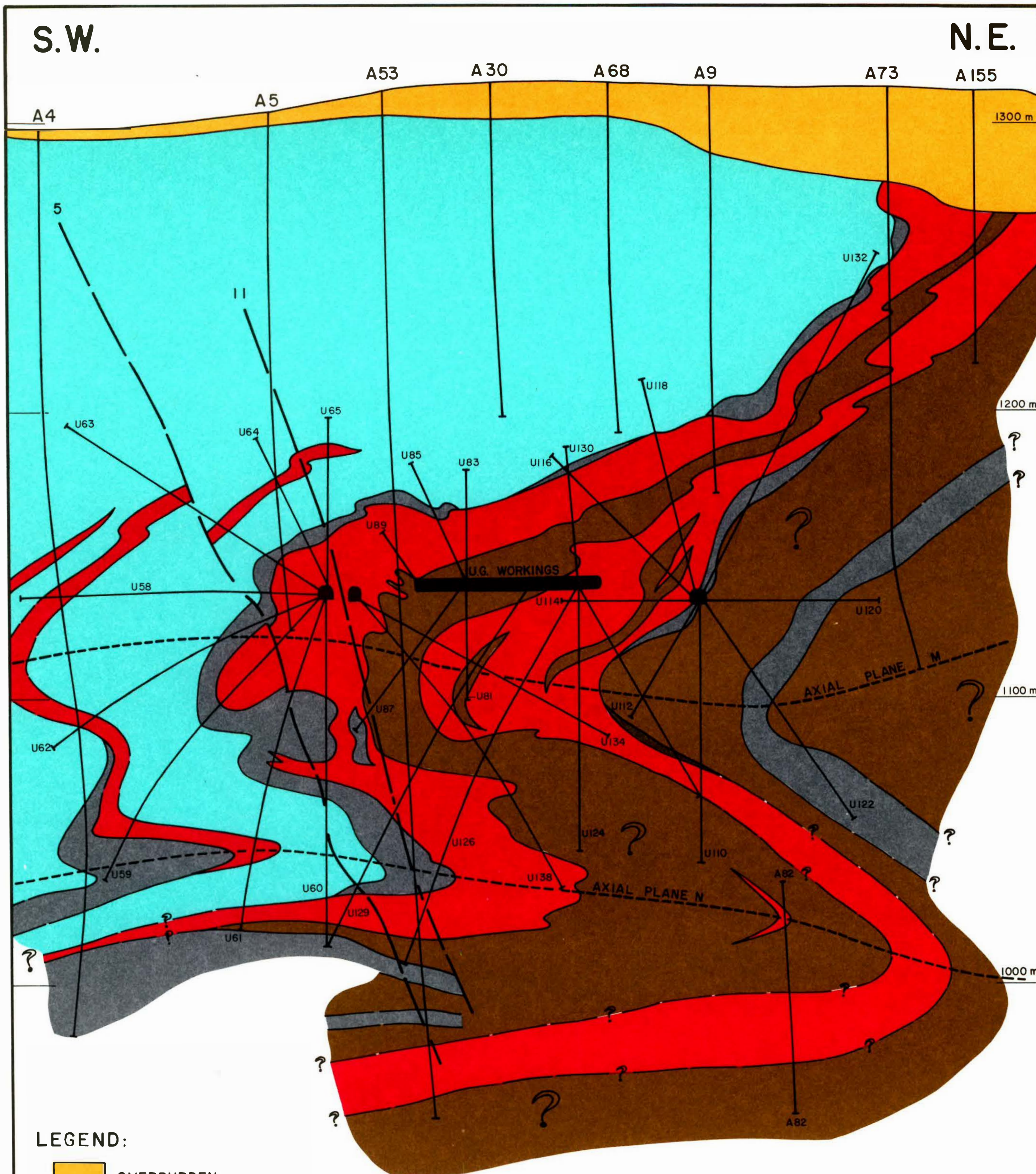
3.32 THE DEVELOPMENT OF THE KERR ADDISON TONNAGE AND GRADE MODEL

Cyprus Anvil has begun to re-log the original diamond drill core and extend the assay and lithological data to allow the construction of an independent tonnage and grade model. In view of the time required to complete this project however, the Kerr Addison model was used as a basis for this evaluation.



GRUM DEPOSIT
LONGITUDINAL SECTION
2N/W

FIGURE 3-1



LEGEND:

OVERBURDEN

VANGORDA FORMATION

CALCAREOUS PHYLLITE ; CALC-SILICATES

GRAPHITIC PHYLLITE / SCHIST

SULPHIDE HORIZON(S)

MT. MYE FORMATION

NON-CALCAREOUS PHYLLITE / SCHIST

AXIAL PLANE TRACE

FAULT

0 50 100
METRES

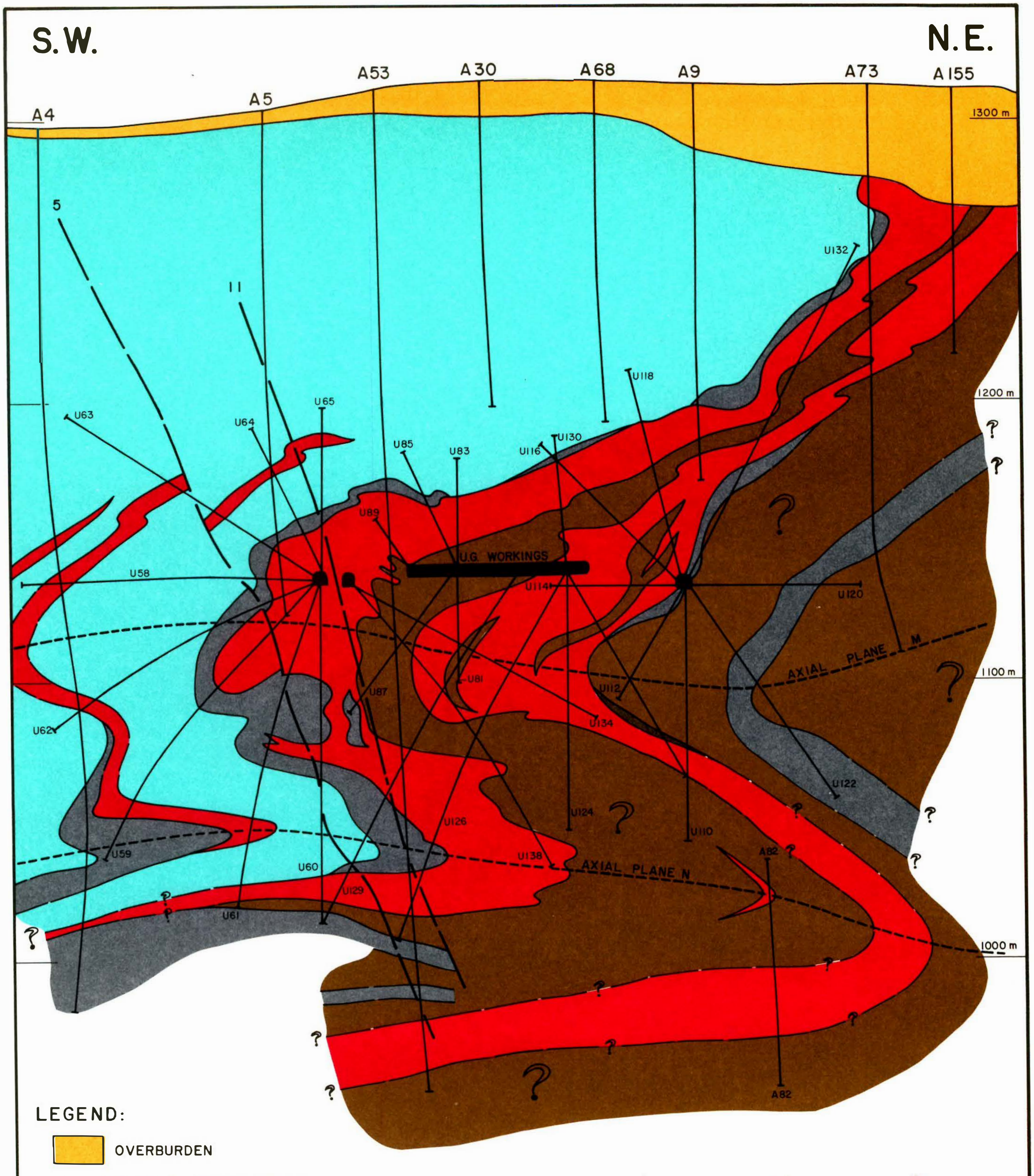
GRUM DEPOSIT

CROSS SECTION

76 W

DJ/rwr March, 1980

FIGURE 3-2



LEGEND:

OVERBURDEN

VANGORDA FORMATION

CALCAREOUS PHYLLITE ; CALC-SILICATES

GRAPHITIC PHYLLITE / SCHIST

SULPHIDE HORIZON(S)

MT. MYE FORMATION

NON-CALCAREOUS PHYLLITE / SCHIST

AXIAL PLANE TRACE

FAULT

0 50 100
METRES

GRUM DEPOSIT
CROSS SECTION
76 W

DJ/rwr March,1980

FIGURE 3-2

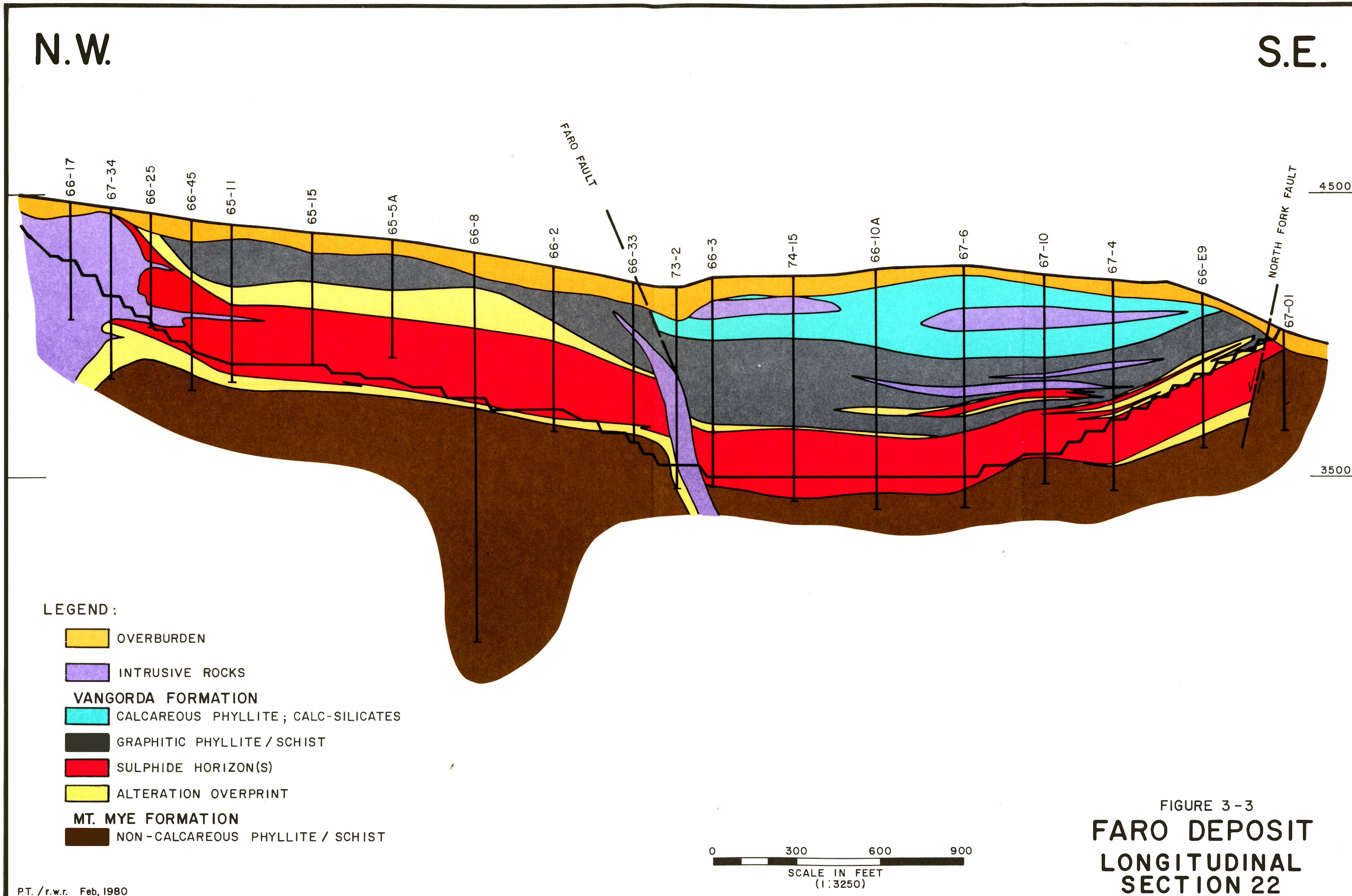


FIGURE 3-3

FARO DEPOSIT

LONGITUDINAL SECTION 22

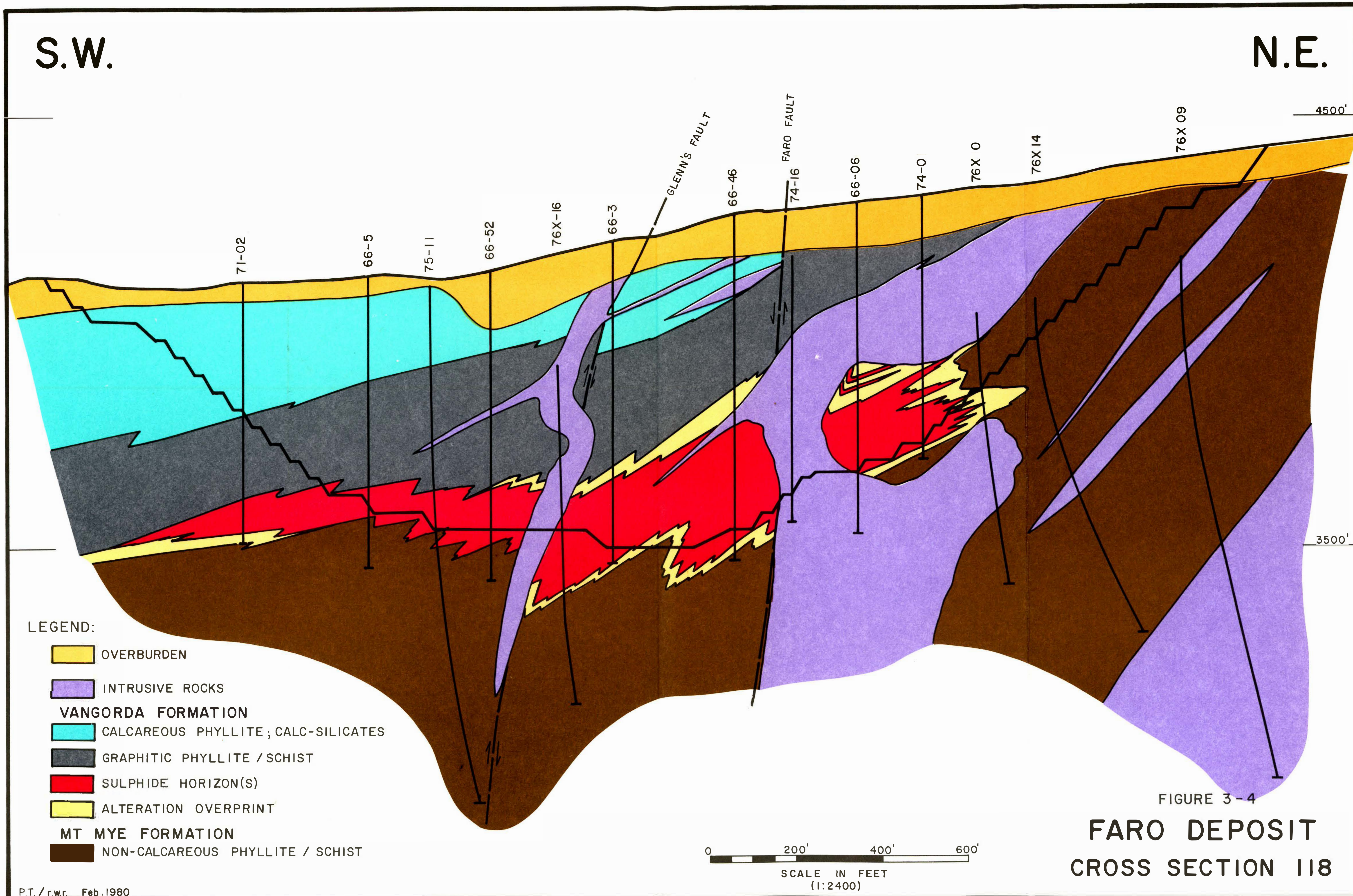


FIGURE 3-4
FARO DEPOSIT
CROSS SECTION 118

The original assaying program was limited to the determination of lead, zinc and silver values contained only in samples obtained from sulphide horizon lithologies. All other intervals which included many thin bands of less than 3 metres in thickness and consisting of internal phyllite waste were designated "Not Assayed".

A three dimensional block grid was established based upon unit block dimensions, 9.0 metres bench height by 15.0 metres width on easting by 15.0 metres width on northing. Those blocks that were intersected by drill holes were designated actual value blocks and were assigned mineral values based upon the weighted average assays of the samples within the block. "Not Assayed" intersections were ignored when actual block values were determined, on the assumption that selective mining practice and effective ore control would minimize mining dilution. A minimum sample length of 0.4 metres was applied to all calculations. Blocks containing less than the minimum sample length of sulphide mineralization were ignored.

The geology of the Grum Deposit was represented by three "rock" types in the computer model; massive sulphides including baritic facies, quartz sulphides, and overburden. The complex geometry of the deposit required a 5 metre by 5 metre by 9 metre block size to define the geology. Geologic bench plans were used to enter the "rock" codes.

Having established assay data and designated rock types for the smaller blocks, interpolative techniques were applied to determine the ore grades and tonnage factors to be applied to the larger unit block. This information was then transferred to the MINTEC computerized systems currently in use by Cyprus Anvil. The following values were assigned for each unit block:

1. Lead content (%)
2. Zinc content (%)
3. Silver content (g/tonne)
4. The number of 5 metre massive sulphide blocks
5. The number of 5 metre quartzitic sulphide blocks.
6. Combined lead and zinc content (%)
7. Proportion of unit block beneath topography (%)
8. Unit block weight (tonnes)

Lead, zinc and silver values were derived by averaging the values of the smaller 5 metre blocks within the 15 metre unit block. Since the specific gravities of the massive sulphides and quartzitic sulphides were significantly different (4.10 and 3.18 respectively) the block tonnage was calculated on a weighted average basis, dependent upon the relative occurrence of each rock type within the block.

3.33 GEOLOGICAL RESERVES

The following geological reserves were calculated for the Grum Deposit between sections 62W and 86W:

FIGURE 3-5

GRUM GEOLOGICAL RESERVES

CUT-OFF GRADE (Pb + Zn)%	4.0%	4.5%	5.0%
Tonnage (000's tonnes)	27,650	25,233	22,945
Lead (% Pb)	3.1	3.3	3.4
Zinc (% Zn)	4.9	5.1	5.3
Silver (Ag g/tonne)	47.9	50.0	52.0

3.4 THE PLANNING OF THE GRUM PIT

An "ultimate economic pit" was constructed based upon anticipated metal prices, metallurgical performance, operating costs and the pit slope angles. Using these values a three phase ultimate pit was designed by the application of the conventional "moving cone" technique. This inter-active approach calculates the net value of a cone originating at a specified base block and expanding upward to the surface at a given pit slope. Those cones that generated a positive net value were designated "mineable". The "ultimate pit" therefore represents the sum of all those cones that yield a positive net value and are "mineable". Through the judicious selection of the base blocks, the pit mining program was developed into a series of nine sequential mining phases.The application of this computerized program at pre-determined cut-off grades produced the undiluted mining reserves for each of the nine phases.

3.41 ANNUAL PRODUCTION SCHEDULES

Based upon the required production rate of 4,660 tonnes of ore per day, the annual production schedule was derived using a nine phase, mine development program and computing phase inventories by bench. Total pre-production volumes were estimated to be 12 million cubic metres.

Due to the complex geological ore/waste contacts the values of lead, zinc and silver were adjusted downward by 6 percent throughout the life of the open pit to reflect mining dilution. The figure 3-6 below tabulates the forecast annual production statistics by year at two discrete cut-off grades.

FIGURE 3-6

GRUM ANNUAL PRODUCTION SCHEDULE

A. Cut-off grade 4.0%

YEAR	1	2	3	4	5	6	7	8	9	10	TOTAL
Waste (000's m^3)	4010	4010	4010	4010	4010	4010	4010	4010	2001	0	34,081
Ore (000's tonnes)	1701	1701	1701	1701	1701	1701	1701	1701	1701	274	15,583
Lead (% Pb)	3.9	2.9	2.9	3.1	3.3	3.2	2.8	2.7	2.8	2.9	3.1
Zinc (% Zn)	6.4	4.9	4.8	5.0	5.4	5.0	4.5	4.4	4.5	4.7	5.0
Silver (Ag g/tonne)	58.5	45.3	44.8	46.5	50.4	48.5	42.6	42.1	44.1	46.3	47.0
Gold (Au g/tonne)											N/A

N/A: Not available at this time.

B. Cut-off grade 4.5%

YEAR	1	2	3	4	5	6	7	8	9	TOTAL
Waste (000's m^3)	4300	4300	4300	4300	4300	4300	4300	4300	0	34,400
Ore (000's tonnes)	1701	1701	1701	1701	1701	1701	1701	1701	1701	14,679
Lead (% Pb)	4.0	3.0	3.1	3.2	3.3	3.1	2.8	2.8	2.9	3.1
Zinc (% Zn)	6.5	5.1	5.0	5.3	5.4	5.0	4.6	4.5	4.8	5.1
Silver (Ag g/tonne)	59.3	46.7	46.4	48.8	50.9	47.9	43.9	43.8	47.4	48.4
Gold (Au g/tonne)										N/A

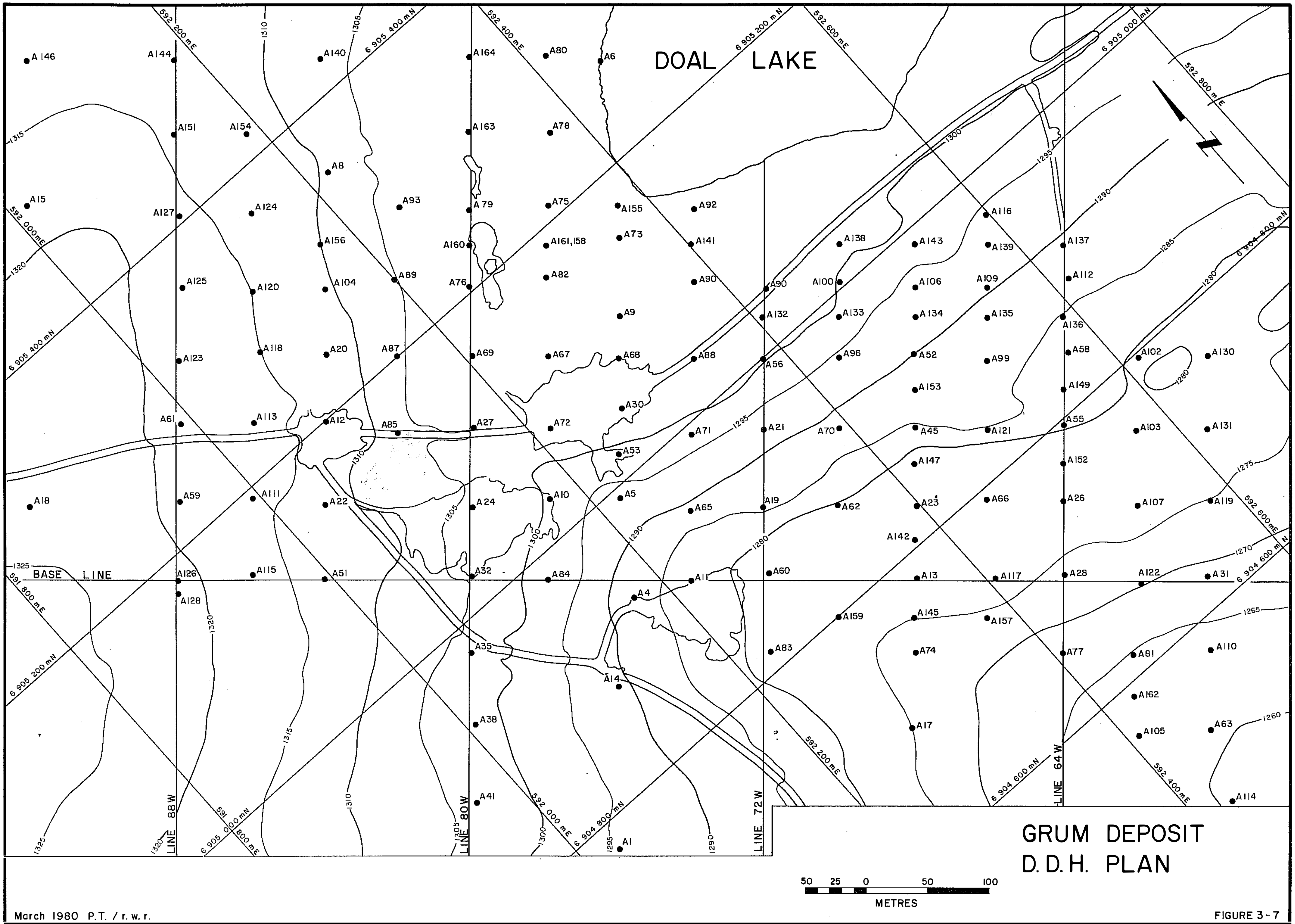
N/A: Not available at this time.

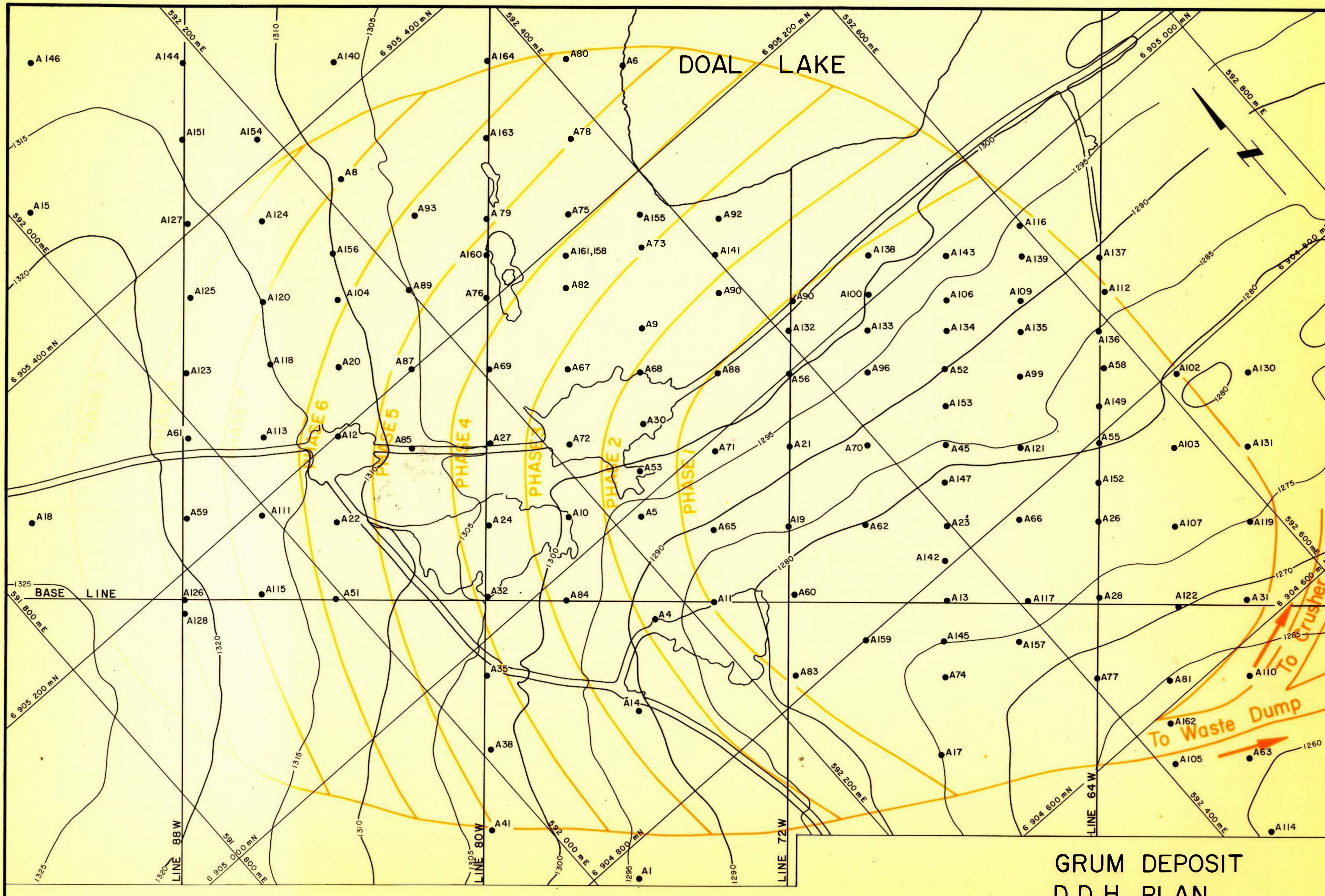
3.5 THE OPERATION OF THE GRUM PIT

The mine operating strategies and budgets were based, partially on the computerized mine models. In addition the many similarities between the Faro mine and the proposed Grum pit, permitted valuable experience to be integrated into the analysis. The first phase of an extensive geotechnical study of the deposit and host rocks was completed to predict pit wall angles and explore the sub-surface water conditions. A limited amount of seismic work was also conducted to more clearly define the overburden/waste rock contact and determine the extent and continuity of permafrost in the overburden.

Based on this information, and the results of the many engineering studies performed by Kerr Addison, the production plans, capital requirements and operating budgets were developed. This information is sufficiently accurate to permit a meaningful evaluation of the preliminary development plans. These plans will be subject to change as on-going detailed engineering studies proceed. The present phase development design of the Grum pit is shown in Figure 3-7 in relation to the original diamond drill pattern.

In the present plan the 12 million cubic metres of pre-production overburden stripping will be removed by a contractor to minimize


DOAL LAKE
BASE LINE
LINE 88W
LINE 80W
LINE 72W
LINE 64W
592 200 mE
592 400 mE
592 600 mE
592 800 mE
592 000 mE
591 800 mE
6 905 400 mN
6 905 200 mN
6 905 000 mN
6 904 800 mN
6 904 600 mN
GRUM DEPOSIT
D.D.H. PLAN
50 25 0 50 100
METRES
March 1980 P.T. / r. w. r.
FIGURE 3-7



GRUM DEPOSIT
D.D.H. PLAN
OPEN PIT OUTLINE

March 1980 P.T. / r. w. r.

FIGURE 3 - 7

short term manpower peaks and the attendant problems encountered in accommodating employees. The stripping contractor will provide camp accommodation for his workers.

The costs of this overburden stripping were estimated by Wright Engineers Limited, December 1979. Mining equipment selection was based upon the production criteria as shown and was chosen when practicable to duplicate equipment that is presently providing satisfactory service at the Faro pit.

FIGURE 3-8

EQUIPMENT PERFORMANCE CRITERIA

ITEM	PERFORMANCE
Shovel P & H 2100 or equivalent	3,000,000 m^3/year
Drill Electric M4 or equivalent	10,000 m^3/shift
Loader L800 in ore or equivalent	148 m^3/hour
in waste	281 m^3/hour

Haulage of the ore and waste will be accomplished using electric wheel trucks similar to those currently in service.

Suitable waste dump sites were selected following deep hole drilling and some geotechnical evaluations of the potential sites.

The local mine infrastructure was predicated upon the number of employees and types of equipment scheduled to work at the Vangorda Plateau open pit minesites. The capital provides for the shops, offices, warehouse and ancillary facilities deemed necessary to support an efficient mining operation, while reflecting the availability of existing support services at the Faro minesite.

Operating costs in general were based upon current experience in the Faro pit.

The study assumed that adequate supplies of fresh water could only be supplied by pumping from a well located near Vangorda Creek. Power will be supplied by a "branch" high tension line taken off the main transmission line servicing the existing facilities. A sub-station will be located at the Grum minesite.

4.
THE VANGORDA DEPOSIT

4.1 HISTORICAL REVIEW

The Vangorda deposit, discovered in 1953 by the late Mr. Al Kulan, was the first mineral deposit discovered in the Anvil District. The property was subsequently optioned by Prospector's Airways Co. Ltd., a subsidiary of Kerr Addison Mines Ltd. During 1954 and 1955 the deposit was diamond drilled on a 60 metre center grid pattern with fill-in holes at 30 metre spacing along alternate cross sections.

A preliminary mining and cost study was conducted by General Engineering Company Ltd. in 1964. The results of this study indicated a probable mineable ore reserve of 5.6 million tonnes with grades of 3.0% lead, 4.9% zinc, 50.9 gms/tonne silver, and 0.75 gms/tonne gold. This data was based upon a 10 percent mining dilution factor.

4.2 THE 1979 EXPLORATION & DEVELOPMENT PROGRAM

Analysis of the earlier Vangorda diamond drill data indicated serious deficiencies in the identification of those geologic units now recognized as critical to the understanding of deposits in the Anvil District. Poor core recoveries, inadequate logging techniques, assaying inconsistencies and sample deterioration further complicated the task.

Accordingly a re-drilling program was initiated along sections 2E, 6E, 10E, 18E and 26E (Figure 4-4) to assess the reliability of original assay values, evaluate original geological interpretation and provide samples of various ore types for metallurgical bench testwork.

The results of this re-drilling program, when compared with original data clearly indicated the need to extend the re-drilling program to obtain dependable information to construct a geologically based grade and tonnage model.

The extended program provided for the systematic drilling of all section 2W to 12E inclusive at 30 metre centres along sections 60 metres apart. Fifty three holes were drilled in an area approximately 800 metres long (NW-SE) by 200 metres wide (NE-SW).

All holes were logged for structure and lithology, and the sulphide bearing lithologies were routinely sampled at 1.0 to 2.0 metre long assay intervals selected to correspond with lithologic breaks where possible. Units less than one metre thick were combined with other units above or below and the samples assayed for lead, zinc, silver and copper. Gold, barium, mercury, manganese, pyrite, pyrrhotite, and specific gravity determinations were added for intervals of potential mining width grading more than 3% combined lead-zinc.

The data generated by the 1979 diamond drill program was used as the basis for the development of the tonnage and grade model, and the preliminary mine operating plans.Work performed by others determined that mineralized zones between sections 14E and 30E offered a relatively small economic tonnage potential. In 1979 some test drilling in this area yielded poor core recoveries when using conventional equipment. This zone containing sulphide sub-outcrops will be further examined in 1980.

4.3 GEOLOGY

The Vangorda deposit lies within the Anvil Range which is located along the southwestern margin of the Selwyn Fold Basin, immediately northeast of the Tintina Trench. Regionally stratabound between lower Paleozoic units, the deposit is spatially related to a graphitic phyllite that occurs at a facies change between calcareous phyllites of the Vangorda formation above and non-calcareous phyllites of the Mt. Mye formation below. Meta volcanic rocks are rare in the immediate vicinity of the deposit but are regionally common throughout the Vangorda formation.The deposit has been deformed to produce large scale "Z" symmetry folds. The principal foliation as observed in outcrop and core is axial and planar to these folds.

The sulphide horizon consists of the following principal lithologies from top to bottom:

a) Sulphide bearing graphitic quartizite, generally pyrite rich and locally base metal bearing.

b) Barite and massive pyritic sulphides, which are base metal bearing.

4.4 THE DEVELOPMENT OF GEOLOGICAL RESERVES

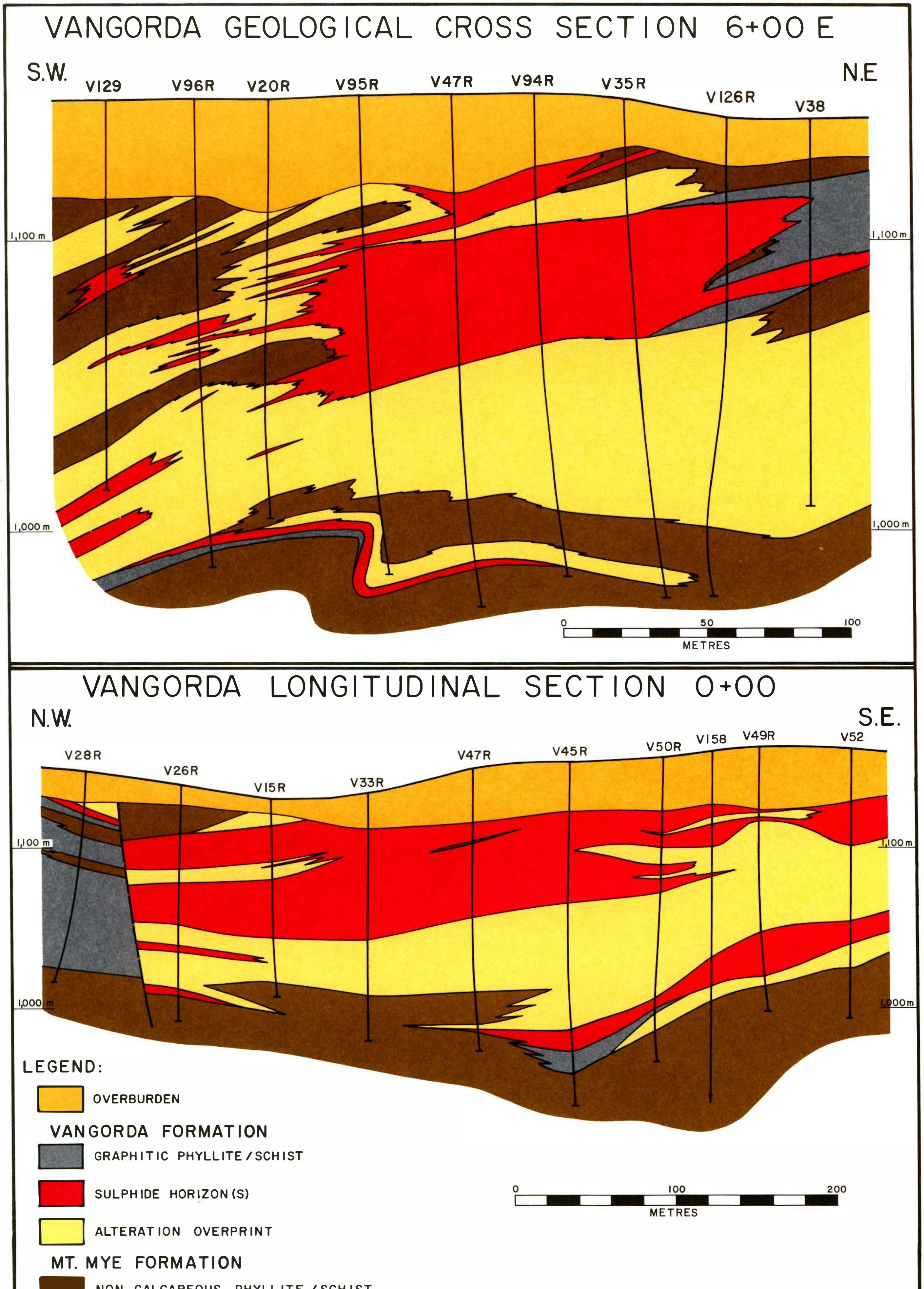
4.41 THE CONSTRUCTION OF GEOLOGICAL SECTIONS.

Geological cross-sections, oriented at 050° azimuth from true North and looking toward the North West were constructed at 1:500 scale (Figure 4-1). Drill holes were computer plotted in their deviated position onto the plan of the section. Structural data was plotted with reference to an average S2 strike of 130°azimuth with dips to the South West.

Using the strataform and stratabound nature of the deposit as a guide, a generalized outline of the horizon was interpreted on each section. At the margins of the deposit, the sulphide limit was extrapolated by one-half the drill hole spacing.

4.42 DEVELOPMENT OF THE TONNAGE AND GRADE MODEL

A three dimensional block model, parallel to the section grid, was developed for the Vangorda deposit. The unit block dimensions used were 6.3 metres bench height by 19.0 metres width on Easting by 61.0 metres width on Northing.



March 1980 P.T. / r.w.r.

FIGURE 4 - 1

The sulphide horizon outline on the geological sections was converted into block data. Blocks were designated as "sulphide" when fifty percent or more of the block was located within the sulphide horizon. All other blocks were coded "waste".

The following values were assigned to each unit block:

1. Lead content (%)
2. Zinc content (%)
3. Copper content (%)
4. Silver content (gms/tonne)
5. Combined lead and zinc content
6. Rock Code
7. Proportion of unit block beneath topography (%)
8. Unit block weight (tonnes)

Composite lead, zinc, copper and silver values over a bench height were calculated from each drill hole. For all external contacts, and for internal contacts where the waste thickness was greater than one-half the bench height, "not assayed" intervals were excluded from the composite calculation. In these cases, a dip projection method was simulated by calculating bench height composites up or down from the contact. When this failed to give a composite length greater than 3.2 metres, no value was calculated for that bench.

The composite values were used to interpolate grades into the blocks. Interpolation was achieved by level, and then by section using a search distance along a section of 30 metres. Where two or more composites were used, an inverse squared weighting was applied to the values.

Tonnage for "sulphide" blocks was calculated at 4.2 tonnes per cubic metre, based solely on a geological estimate of rock type distribution and past experience.

The topographic surface used in the model was interpolated and extrapolated from drill hole collar elevations

4.43 GEOLOGICAL RESERVES

The following geological reserves were calculated at varying combined lead-zinc cut-off grades:

FIGURE 4-2

VANGORDA GEOLOGICAL RESERVES

CUT-OFF GRADE (Pb + Zn)	3.5%	4.0%	4.5%	5.0%
Tonnage(tonnes 000's)	7,090	6,751	6,473	6,134
Lead (% Pb)	3.4	3.5	3.6	3.7
Zinc (% Zn)	4.5	4.6	4.7	4.8
Silver (Ag g/tonne)	49.6	50.7	51.7	52.8

4.5 THE PLANNING OF THE VANGORDA PIT

The program used to develop the Vangorda "ultimate economic pit" was the same as that used for the Grum pit analysis. The metallurgical performance, operating costs and pit slope angle criteria were adjusted to reflect the differing operating and geotechnical parameters between the two deposits.

4.51 ANNUAL PRODUCTION SCHEDULES

Using the phase inventories by bench, a production rate of 4660 tonnes per day, and 934,000 cubic meters of pre-production alluvial stripping, ore delivery schedules were determined and annual feed grades were calculated. Since the model block grades do not include any dilution at geological ore-waste contacts, these grades were subsequently adjusted downward to allow for mining dilution.

Using historical data from the Faro Deposit as a guide, a review of the waste/ore contacts to be mined yielded the following grade dilution schedule for the Vangorda Deposit.

FIGURE 4-3

VANGORDA PIT DILUTION FACTORS

YEAR	1	2	3	4
Lead	-5%	-3%	-2%	-2%
Zinc	-3%	-3%	-2%	-2%
Silver	-5%	-3%	-2%	-2%

FIGURE 4-4

VANGORDA ANNUAL PRODUCTION SCHEDULE

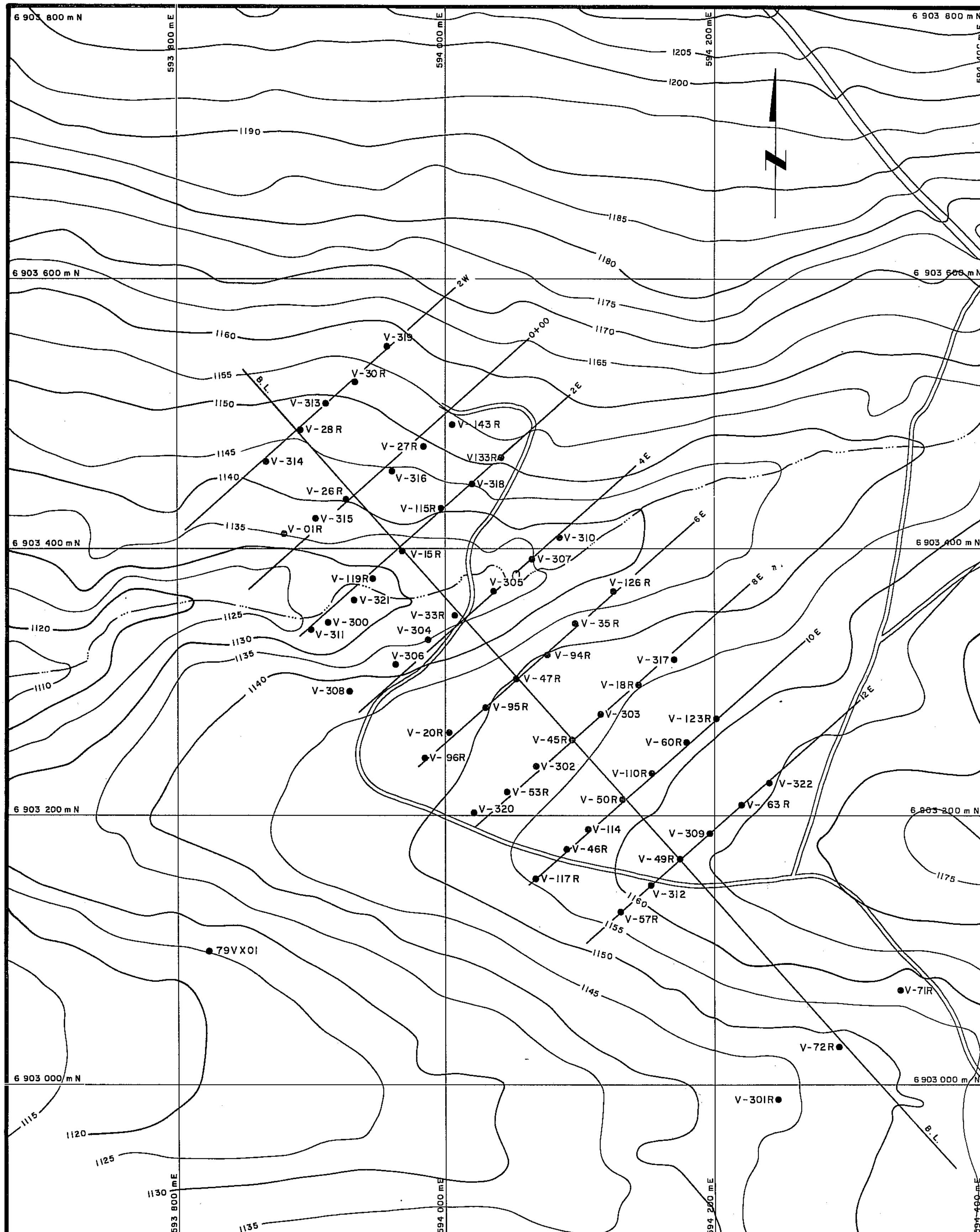
A. Cut-off grade = 4.0%

YEAR	1	2	3	4	TOTAL
Waste (000's m^3)	2920	2920	659	0	6499
Ore (000's tonnes)	1701	1701	1701	1031	6134
Lead (% Pb)	3.5	3.6	3.6	3.2	3.5
Zinc (% Zn)	4.9	4.8	4.7	3.6	4.6
Silver (Ag g/tonne)	47.3	52.6	53.0	46.6	50.2
Gold (Au g/tonne)					N/A

N/A: Not available at this time.

B. Cut-off grade = 4.5%

YEAR	1	2	3	4	TOTAL
Waste (000's m^3)	2920	2920	696	0	6536
Ore (000's tonnes)	1701	1701	1701	877	5980
Lead (% Pb)	3.5	3.6	3.6	3.2	3.5
Zinc (% Zn)	5.0	4.8	4.7	3.7	4.6
Silver (Ag g/tonne)	47.5	52.7	53.9	47.0	50.7



VANGORDA DEPOSIT

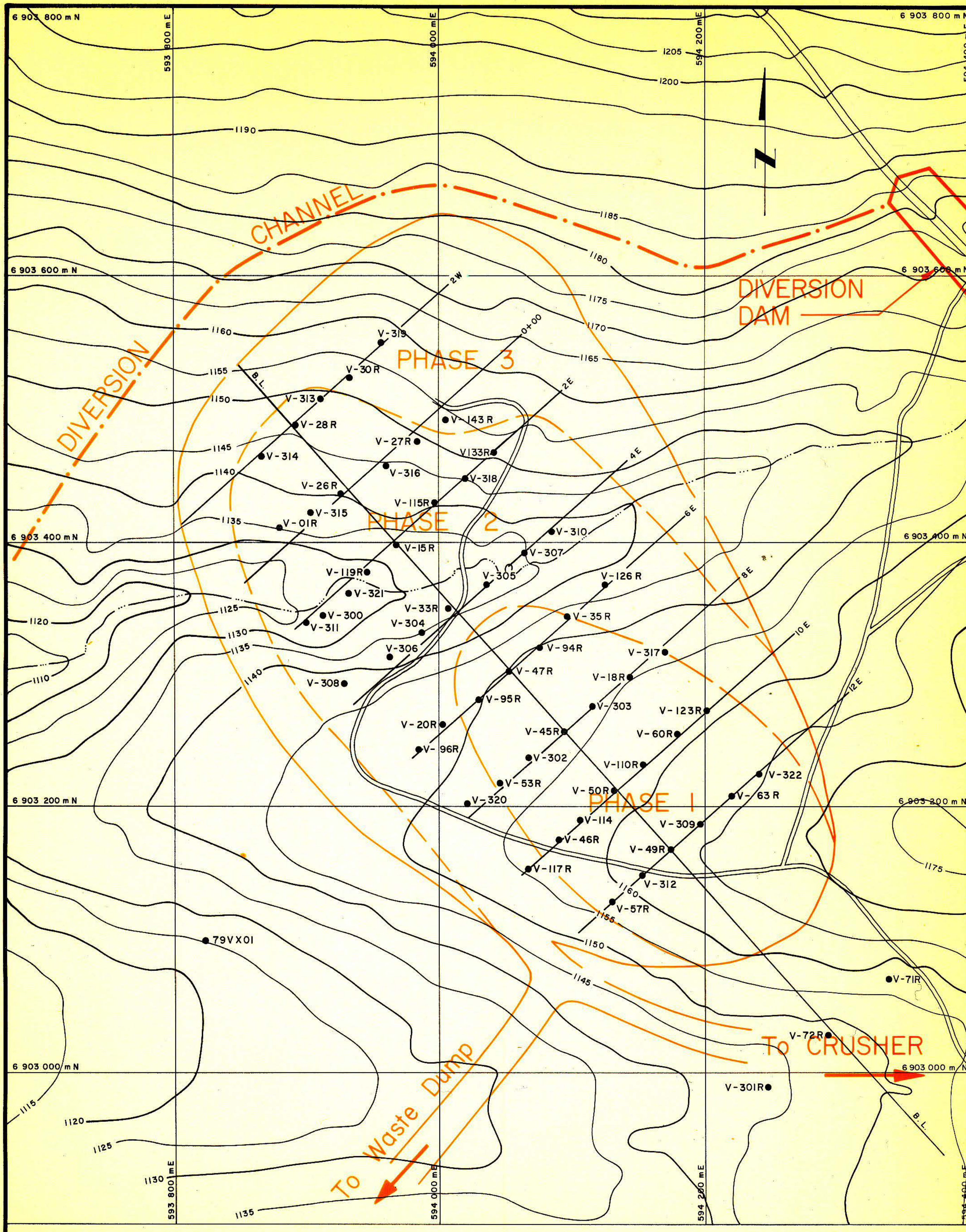
D.D.H. PLAN

50 25 0 50 100

METRES

March 1980 P.T. / c.l.c.

FIGURE 4-5



VANGORDA DEPOSIT

D.D.H. PLAN

OPEN PIT OUTLINE

March 1980 P.T. / c.l.c.

FIGURE 4-5

4.6 THE OPERATION OF THE VANGORDA PIT

The preliminary mine plan formed the basis for predicting operating criteria, capital expenditures and operating costs. Though detailed studies of this deposit commenced in the Spring of 1979, the high quality of the data generated by the closely spaced drilling program provided a sound basis for this analysis - and for the necessary continuing detailed engineering studies. As shown in Figure 4-5 the Westerly portion of the deposit to be mined is situated in the Vangorda Creek Valley and directly beneath the creek. Preliminary engineering studies were conducted in 1979 to design an environmentally acceptable diversion ditch to direct the creek flow to the west of the ultimate pit wall. A suitable routing was located by which the flow will be returned to the original water course to the South of the open pit.

All mine production equipment will be similar to units presently in service. Waste will be mined primarily by an electric shovel, supplemented as required by a loader. A suitable waste dump site was located and tested by deep hole drilling immediately to the south of the proposed open pit. Ore will be mined with a front end loader and transported to the Grum crushing plant by large electric wheel trucks. The complete cycle time for an ore truck to be loaded at the bottom of the ultimate pit, to deliver ore to the crusher and to return to the ore face was predicted to be approximately 40 minutes, assuming a maximum ramp grade of 8 percent. Blast hole drilling will be carried out by one electric drill.

The mechanical shops, offices, warehouse facilities and other services will be located adjacent to the Grum open pit in deference to the larger size of the Grum deposit.

Power to the pit will be supplied by a 4160 volt overhead line from the main substation located at the Grum minesite.

5.
MINE DEVELOPMENT PLANS

5.1 INTRODUCTION

In order to maximize the benefits derived from the acquisition of the Grum and Vangorda deposits a number of mine development plans were evaluated to select the one which offered the best security of ore supply and the most attractive return on investment.

The financial benefits attributed to each development plan were calculated by comparing the cash flow generated by the particular plan with that which would result from the implementation of the present mining schedule, which excludes Vangorda Plateau ores.

Any development program to integrate the ore reserves in the Vangorda Plateau orebodies into the Faro operation and treat these ores in the existing mill would consist of the following activities:

- A modification of the existing mill in order to provide additional grinding and flotation capacity to treat the Vangorda Plateau ores. Although mineralogically similar to the Faro ore, these ores are structurally different and require a finer grind in order to achieve satisfactory metallurgical recoveries. No increase in total throughput is envisaged at this time.

- Additional tailings disposal capacity.

- The provision of additional electric power required by the increased mill horsepower requirements.

- Construction of additional married housing accommodation in the Town of Faro to improve the ratio of married to single employees in order to promote long term stability of the work force.

- Provision of an ore haulage system to transport the ore from the Vangorda Plateau orebodies to the existing mill, a distance of approximately 14 kilometers.

- Pre-production stripping of the orebodies.

- Construction of ancillary facilities such as a shop, change house, warehouse, and offices.

- Construction of required housing facilities in the Town of Faro for any additional personnel required.

These modifications, made for the purpose of milling the Vangorda Plateau ores, will enhance the metallurgical performance of Faro ores, thus offering a wide variety of development options.

Having identified those cases which offered the most favourable returns on investment, practical considerations were integrated into the final analysis to select the recommended mine development plan.

5.2 MINE DEVELOPMENT CASES

Eight mine development cases were examined and are schematically depicted in Figure 5-1. These cases were divided into four series and are briefly described below.

SERIES 1:

This series considered only Faro pit ores and made no provision for the mining of the recently acquired Vangorda Plateau orebodies. Accordingly mining operations cease in 1990.

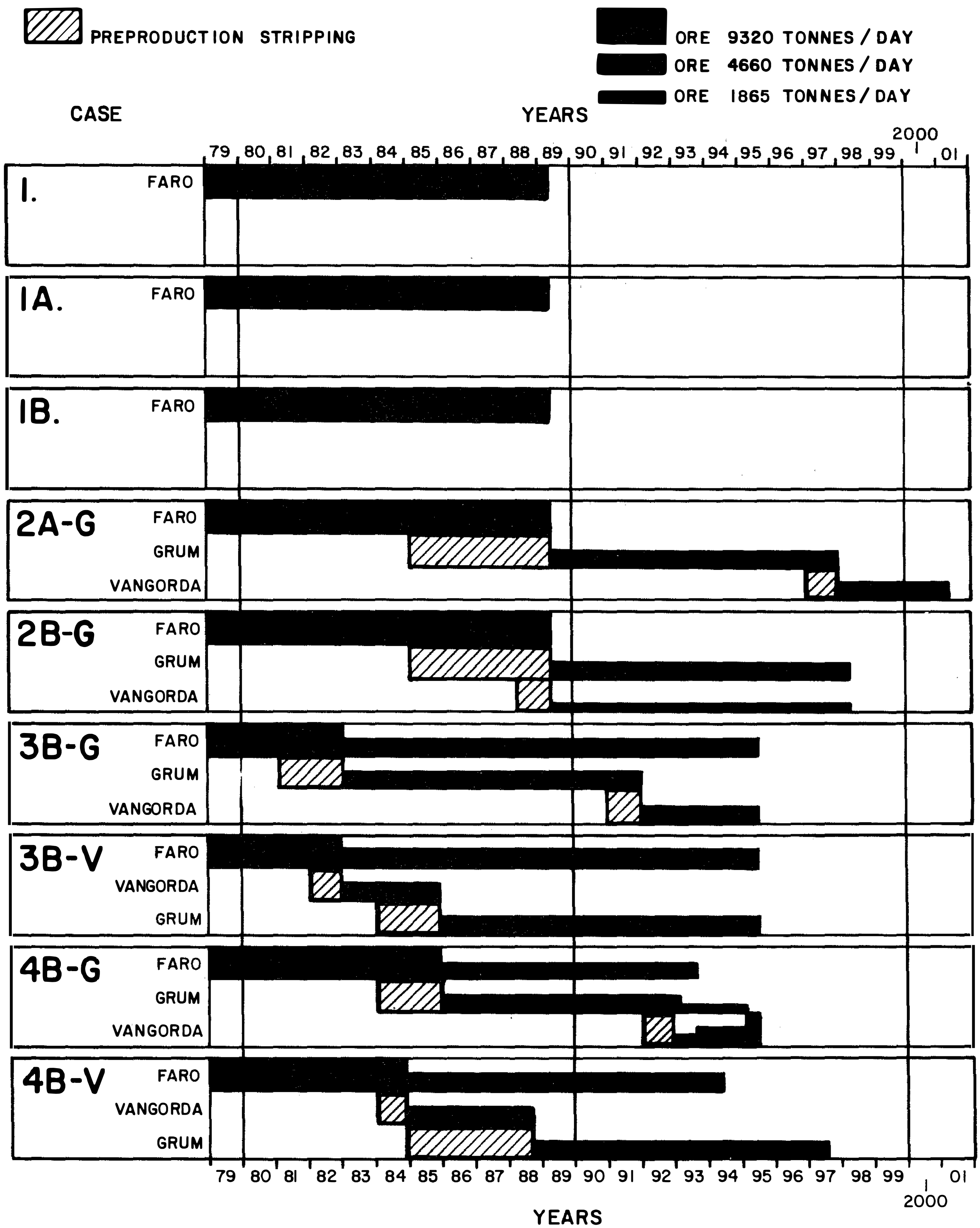
CASE 1:

This case is the present Faro mining plan and represents the base case against which all development plans are compared. The case provides for no major capital expenditures during the rest of the life of the Faro pit, and the metallurgical performance equal to that achieved in 1978 and 1979.

CASE 1A:

This case differs from case 1 in that $11 million would be spent in 1980 and 1981 to upgrade the existing flotation circuit and install basic instrumentation systems to improve metallurgical control. Metallurgical performance was improved to reflect the changes in the flotation circuit.

FIGURE 5-1 VANGORDA PLATEAU DEVELOPMENT
MINING ALTERNATIVES



CASE 1B:

This case was based on the implementation of the recommended mill modifications to be complete by year end 1981. The modifications, which included the additional grinding circuit, expanded flotation and dewatering sections and the supplemental power generating facility would cost approximately $44 million. The plant metallurgical performance was adjusted to recognize the benefits derived from finer grinding.

In addition to the mill modifications, the new tailings impoundment facilities costing $12 million were required to provide adequate capacity and settling times for the finer tailings.

SERIES 2:

These cases assume that Vangorda Plateau ores would be mined upon the exhaustion of the Faro pit ore reserves in 1989.

CASE 2A-G:

This case is identical to case 1A through 1988. Thereafter only Grum and Vangorda ores would be milled in turn at the rate of 4,660 tonnes per day until the exhaustion of these deposits in

2001. The finer grind necessary for the effective selective flotation of these minerals would be obtained by treating the reduced throughput in the existing grinding circuit.

CASE 2B-G:

This case is identical to case 1B through 1988. Thereafter, 6,525 tonnes of ore per day would be mined from the Grum and Vangorda deposits concurrently in the proportion of 4660 tonnes and 1825 tonnes respectively until the exhaustion of the Vangorda Plateau ores in 1998.

SERIES 3:

These cases assumed that Vangorda Plateau ore would be mined concurrently with that from the Faro pit in equal proportions commencing in 1983 to provide a total daily mill feed tonnage of 9,320 tonnes.

CASE 3B-G:

This case is identical to case 1B through 1982. Grum ore deliveries would commence in 1983 at the rate of 4,660 tonnes per day and continue until the exhaustion of the Grum reserves in 1992. Grum ore would be replaced with that from Vangorda until the exhaustion of this deposit in 1995.

CASE 3B-V:

Case 3B-V assumes that ore from the Vangorda pit is mined in 1983 followed by Grum ore in 1986. Otherwise this case is the same as 3B-G.

SERIES 4:

This series assume that the Vangorda Plateau ore would be mined concurrently with that from the Faro pit and in equal proportions commencing in 1985 to sustain a mill throughput rate of 9,320 tonnes per day.

CASE 4B-G:

This case is identical to case 1B through 1985. Grum ore deliveries would commence in 1985 at the rate of 4,660 tonnes per day and continue until the exhaustion of these reserves in 1993. Grum ore would be replaced with that from the Vangorda until the exhaustion of this deposit in 1995.

CASE 4B-V:

Case 4B-V assumes that ore from the Vangorda pit is mined in 1985 followed by Grum ore in 1988. Otherwise this case is the same as 4B-G.

5.3 THE ANALYSIS OF THE MINE DEVELOPMENT CASES

In a preliminary financial analysis, Cases 1A and 2A-G were immediately precluded from further consideration because of the very poor relative economic performance. For practical reasons, in that Grum mining plans could not be developed and implemented in the time schedule, Case 3B-G was rejected.

Complete financial analyses were carried out for each of the five remaining development cases. Of these five, the most attractive return on investment was exhibited by case 4B-V. In addition to being the most financially attractive, this case also is desireable from a practical standpoint. Adequate time is available to develop the necessary planning and construction of an ore haulage system as well as to develop the necessary mine planning to commence ore production from the Vangorda Deposit in 1985 and from the Grum Deposit in 1988.

Case 2B-G demonstrated the next best DCF/ROI of the five cases. This case is not recommended because of the fact that after 1988, the full capacity of the mill is not utilized, thereby reducing concentrate output and sales. This under-utilization results from the fact that in the present planning the Grum Deposit is limited to an annual production rate of 1.7 million tonnes due to the complexity of the orebody, and the Vangorda Deposit reserves are not sufficient to sustain full mill capacity throughout the life of the Grum Deposit. Further, this case depends both upon an earlier total supply of ore from the Vangorda Plateau and attendant dependency upon the ore haulage system.

Case 4B-G shows no advantage, either financial or practical, over Case 4B-V. The same applies to Cases 1B and 3B-V. Case 1B is not truly a development case, because it does not include any ore development from the Vangorda Plateau. Case 3B-V has a practical limitation in that the time allowed for planning and construction of the ore haulage system is limited.

Based on both financial and physical considerations, Case 4B-V is the recommended development case.

FIGURE 5-2

VANGORDA PLATEAU DEVELOPMENT

NET CASH FLOWS OF ALTERNATIVE DEVELOPMENT CASES

YEAR	CASE 1	CASE 1B	CASE 2B-G	CASE 3B-V	CASE 4B-G	CASE 4B-V
1980	30.538	11.764	10.540	11.685	11.686	11.686
1981	21.258	8.523	5.772	(9.884)	8.025	8.063
1982	18.837	34.917	35.718	9.518	30.691	37.794
1983	77.366	91.381	94.041	100.118	56.508	76.833
1984	41.374	57.173	42.987	48.137	38.138	38.208
1985	39.751	47.652	48.765	63.421	113.155	64.206
1986	42.283	55.946	53.514	101.832	27.436	46.447
1987	49.023	59.453	44.965	38.236	59.133	59.761
1988	40.253	50.977	33.896	55.430	59.260	68.625
1989	17.341	17.251	106.253	58.733	64.207	72.823
1990	-	-	2.194	58.896	56.179	45.527
1991	-	-	46.513	59.710	45.444	54.991
1992	-	-	43.652	53.041	50.631	60.625
1993	-	-	49.570	48.181	51.246	33.927
1994	-	-	43.938	51.885	76.180	29.807
1995	-	-	41.590	25.806	43.345	24.234
1996	-	-	36.624	-	-	24.551
1997	-	-	42.541	-	-	30.647
1998	-	-	10.261	-	-	-
TOTAL	78.023	435.036	793.336	774.746	791.263	788.746
15%	184.654	209.483	244.626	246.980	248.995	252.021
12%	209.428	230.065	296.686	300.766	302.784	305.140
DCF/ROI	-	31.0%	34.7%	30.2%	33.0%	36.6%

6.
ORE HAULAGE

6.1 INTRODUCTION

Various ore haulage systems were examined to determine the most dependable and cost efficient means of transporting 1.7 million tonnes of ore per year from the Vangorda Plateau to the existing mill site; a distance of approximately 14 kilometres. In consideration of possible mill expansions, and the requirement to produce the entire mill ore supply from the Vangorda Plateau in the future, each of the systems investigated would be capable of providing increased haulage rates to meet projected requirements.

A preliminary engineering study was commenced in the fall of 1979 to evaluate a number of potentially feasible ore haulage systems which included:

a) Trucking utilizing off-highway equipment
b) Rail transportation
c) Conveyor by means of a conventional belt system
d) Trucking using smaller highway trucks
e) Conveying ore on a cable belt

The capital expenditures, operating costs and manpower requirements shown in Figure 6-1 were derived from preliminary data generated during the feasibility study of the haulage alternatives conducted by Swan Wooster Engineering Co. Ltd., February 1980.

FIGURE 6-1

ORE HAULAGE SYSTEMS - PRELIMINARY COST DATA

ITEM	OFF-HIGHWAY TRUCKS	RAIL	CABLE BELT CONVEYOR	HIGHWAY TRUCKS	CONVENTIONAL BELT CONVEYOR
Initial Capital Cost ($million)	32.5	38.6	38.1	42.4	44.1
Average Direct Operating Costs ($/tonne)	1.37	1.39	1.65	2.12	1.34
Total Number of Employees	37	33	30	68	30

The capital and operating expenses include the cost of housing, and provide for the transportation of ore from the pit face to the existing crusher.

6.2 TRANSPORTATION ALTERNATIVES

6.21 OFF-HIGHWAY TRUCKING

Off-highway trucking utilizes equipment which, by virtue of size and weight of the individual units, may not travel upon conventional highways. Conceptually this alternative required that off-highway trucks and or trailers be loaded at the respective ore face by a front end loader. The truck would then proceed directly to the existing primary crusher at the millsite, over a 30.5 metre wide haulage road to be built between the Vangorda Plateau and the millsite. Figure 6-2.

The 136 tonne Caterpillar tractor/rear dump trailer was found to be the most cost efficient off-highway unit, when compared to various rear dump, side dump and train configurations. This unit which is currently successfully used by Pine Point Mines Ltd. in the Northwest Territories offered the most attractive net payload/gross weight ratio and was demonstrably more fuel efficient than the other units considered. An analysis of the cycle times between the Grum deposit and the mill based on a maximum speed of 48 km/hr indicated that 7 trucks operating and 1 truck spare would be required to haul the planned tonnage on a one shift per day, 7 days per week basis.

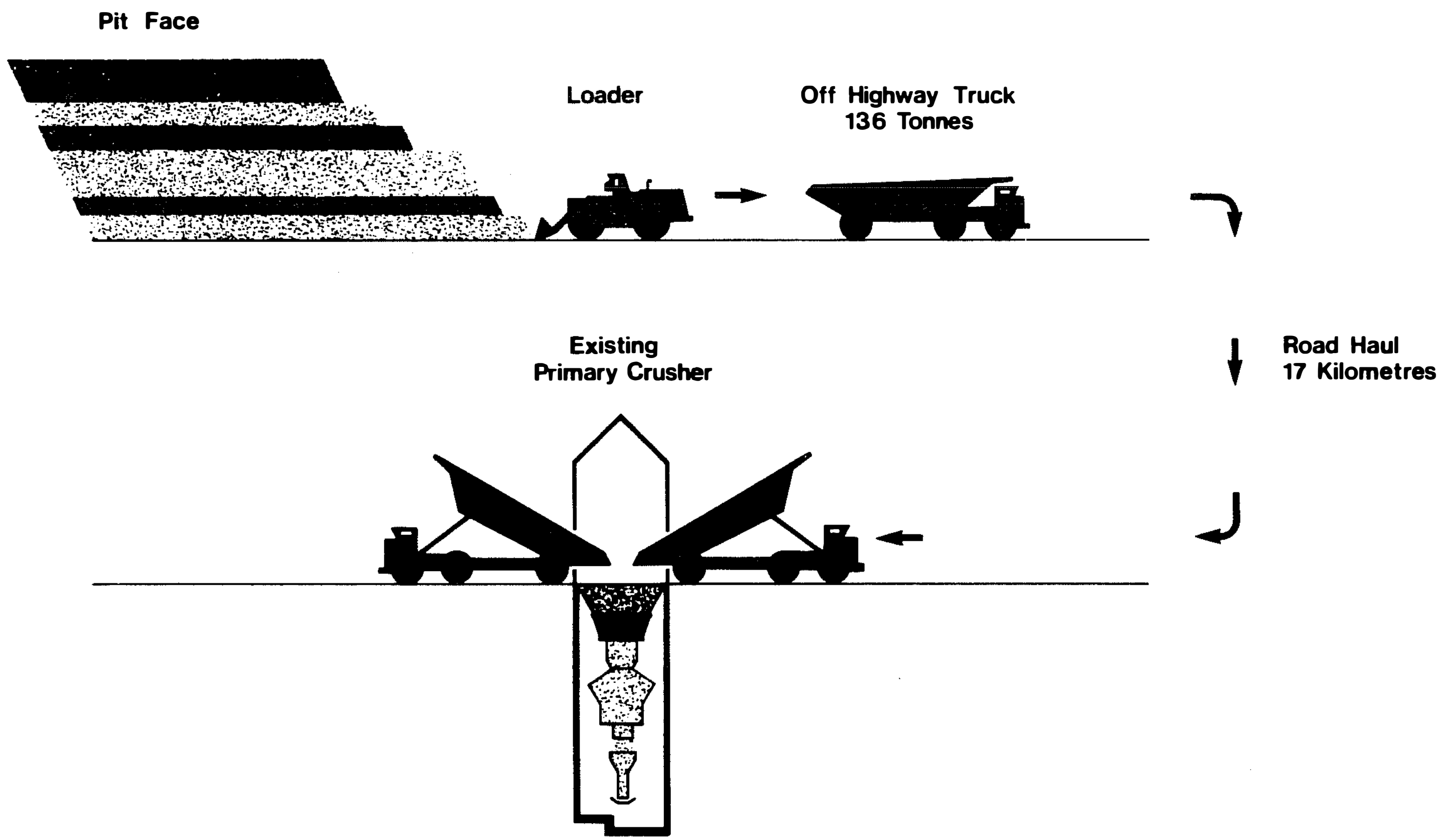


FIGURE 6-2 OFF HIGHWAY TRUCKING SYSTEM
THE DEVELOPMENT OF THE VANGORDA PLATEAU ORE DEPOSITS

That identical units were working successfully at similar latitudes was considered to be an advantage of this system. Scheduled daily ore commitments could be maintained despite temporary reductions in equipment availability by simply extending the operating times of the remaining vehicles.

The feature common to all alternatives considered, other than the off-highway system was the provision of an additional primary crusher and crushed ore storage facility at the Vangorda Plateau. Thus, while capital and operating costs compared most favourably with other alternatives, the off-highway arrangement did not reduce the risk of production loss that has resulted during shut-downs of the primary crusher. Such disruptions have occurred during scheduled major overhauls of the unit, and when handling wet or frozen ore.

Certain disadvantages were attributed to the system due to the fact that no such equipment is currently in use at the Faro pit. Thus additional operator training programs would be required and warehouse inventory values would increase. Pit ramp slopes would generally be designed at grades lower than those that could be efficiently handled by conventional haulage trucks to reflect the inferior climbing capabilities of the tractor/trailer configuration.

The schist and phyllite waste rock moved during the normal course of the mining, does not exhibit the ideal physical characteristics required of road surfacing material. More suitable material is available locally but at additional expense. Experience has demonstrated that normal trucking operations may be impaired by adverse climatic conditions less likely to affect rail and conveyor haulage systems.

Finally, effective control was considered difficult to maintain over a number of individually operated haulage units travelling over long distances, thus creating the potential for system inefficiencies.

Despite the disadvantages of the off-highway arrangement, the favourable capital and operating costs demand that more detailed studies be conducted to more thoroughly evaluate this generally accepted mode of ore transportation.

6.22 RAIL TRANSPORTATION

The rail system required that conventional haulage trucks be loaded at the face and discharged into a primary crusher located at the Grum minesite. The crushed ore would be delivered to a covered 5,000 tonne stockpile from which

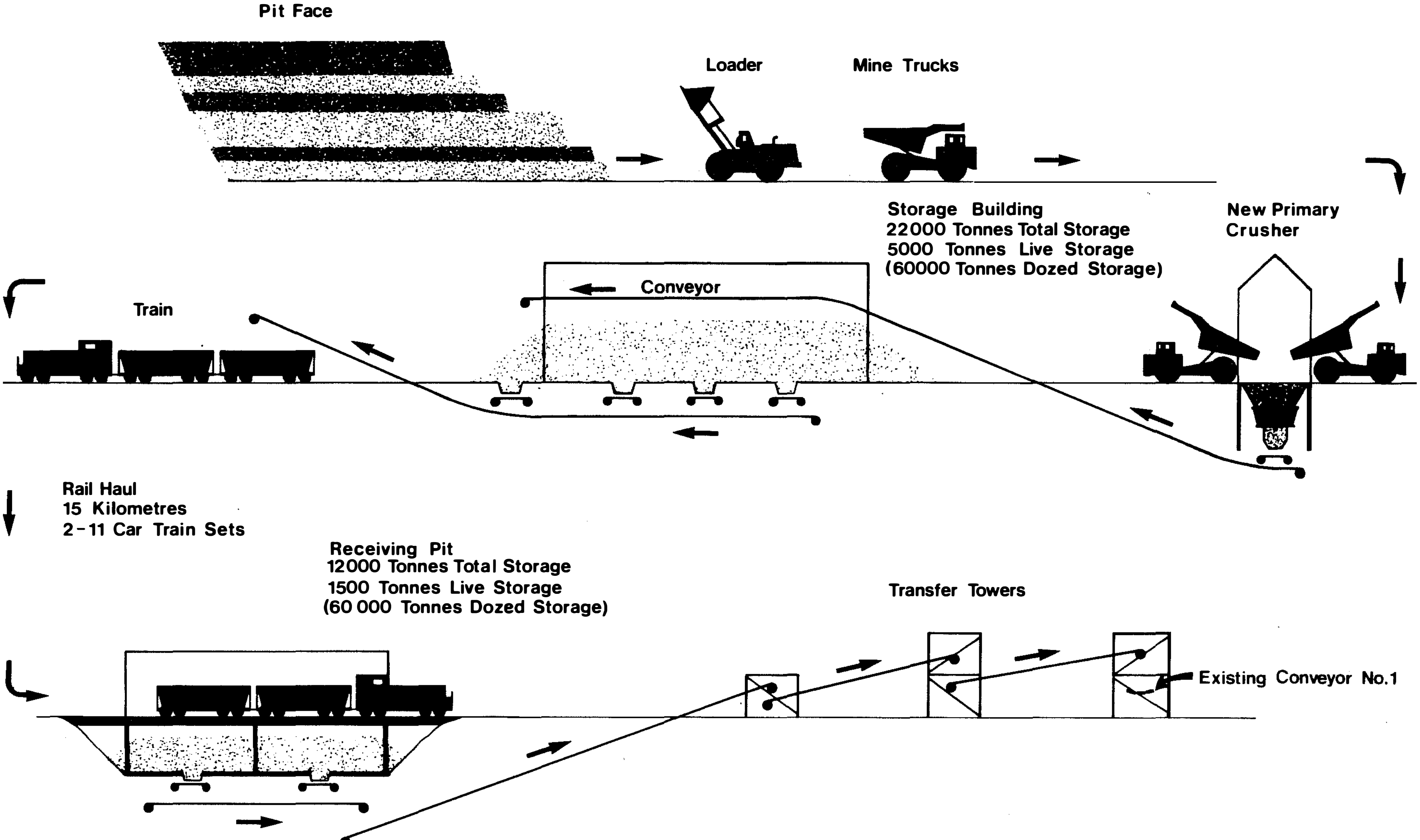


FIGURE 6-3 RAIL HAULAGE SYSTEM

THE DEVELOPMENT OF THE VANGORDA PLATEAU ORE DEPOSITS

material would be drawn by a conveyor belt to load standard gauge, bottom dump rail cars of 80 tonne capacity. (Figure 6-3).

Since the proposed rolling stock included two 800 HP diesel electric locomotives, the system offered flexibility in the selection of train configurations. Thus it would be feasible for each locomotive to pull 11 cars constituting one train or alternatively both locomotives could be operated in tandem to haul one 22 car train. In either case adequate time would be available in one shift to haul the scheduled ore commitments.

The planned rail alignment consisted of a 14.4 kilometre track placed on a 7.3 metre wide rail bed. The line was of favourable grade over most of its length with the exception of a 1.0 km 1 percent adverse grade located at the West end of the line.

The rail cars discharge into a 2,000 tonne capacity stockpile at a trestle dumping station located to the Southwest of the concentrator from whence material was transported by conventional conveyor belt systems and introduced into the existing crusher circuits. The stockpiles were designed to facilitate the removal and replenishment of ore from the system to permit extra-ordinary storage of large volumes of crushed ore prior to major scheduled overhauls of a primary crusher.

The capital cost provided for three spare ore cars and all the track maintenance equipment deemed necessary to support an efficient railroad operation. The rail transportation system offered many tangible advantages in addition to the favourable capital and operating costs. Such systems are simple, well-proven and currently operating successfully in cold climates. Since the proposed locomotives are widely used by major railroad companies across Canada, spare parts are readily available. The rail alternative included provision for the second primary crusher and stockpile facilities and was less prone to disruptions caused by adverse climatic conditions. Supervisory control of the system was considered to be more readily achievable when compared with the off-highway alternative.

Since railroad rolling stock, track structures and ties are normally amortized over a 25-30 year period, there exists an additional cost benefit that is not evident in this report which only considers open pit operations up to 1997. Net present value calculations conducted over extended periods of time to include future underground ore developments could thus tend to favour the rail transportation over the off-highway system.

Disadvantages of the rail alternative include the need for additional training programs and warehouse inventories generated by implementing a new system. Further, the restraints imposed on railroad development by natural topography would almost certainly preclude the extension of the proposed system in an Easterly direction toward the DY and Swim deposits.

6.23 THE CABLE BELT SYSTEM

The cable belt system was developed in England in the early 1950's to meet the need for higher conveyor horsepower than could be obtained with the then standard fabric belt-on-idler conveyors.

Since their introduction, cable belt conveyors have been installed in generally long-haul mining applications in many countries, including the United Kingdom, South Africa, Australia, Canada, Jamaica, Germany and Brazil.

The unique feature of this system is the separation of the driving (tension) medium from the material carrying medium. In conventional systems, the belt must be capable not only of carrying the material load, but also able to transmit the driving tension. In the cable belt system, the material is

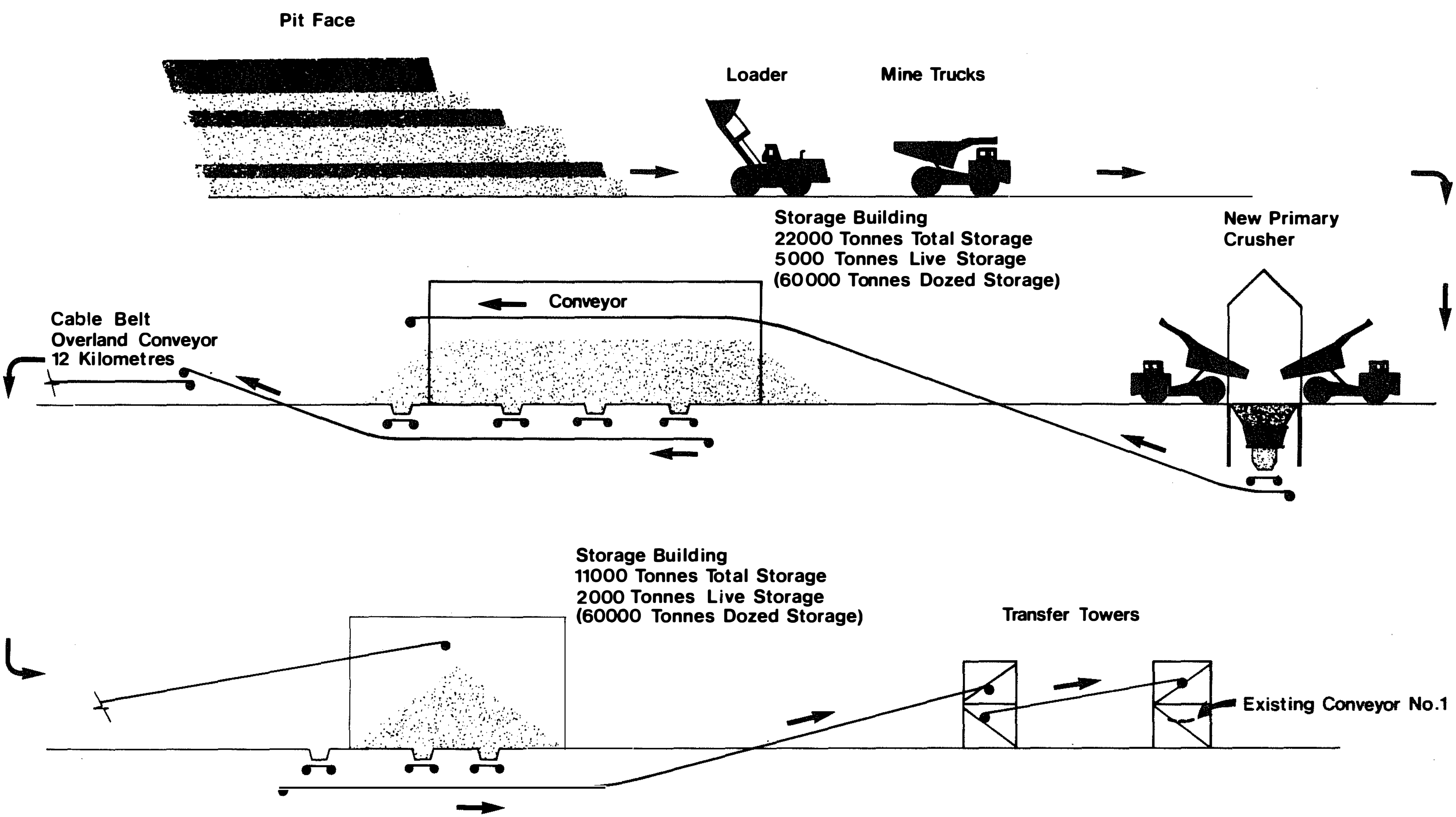


FIGURE 6-4 CABLE BELT HAULAGE SYSTEM
THE DEVELOPMENT OF THE VANGORDA PLATEAU ORE DEPOSITS

carried on a transversely stiffened but longitudinally flexible belt, riding on two endless loops of wire rope that transmit the driving tension. The ropes are supported at intervals by ball bearing pulleys. Molded rubber grooves on the belt grip the wire rope and prevent belt slippage. The wire ropes are driven by Koepe friction wheels at the drive unit, which incorporate a differential to allow for slight inequalities in rope movement. Because the ropes carry the driving tension, only enough tension to prevent folding is applied to the belt.

The mine haulage, crushing and stockpiling facilities included in this system were identical to those described in the rail haulage alternative. Thereafter, the 4,660 tonnes of ore per day would be carried by the 12.8 kilometre long, 76.2 centimetre wide conveyor to a 2,000 tonne covered surge pile. Two 800 HP electric motors would drive the belt at approximately 213 metres per minute.

The conveyor belt systems were found to be less labour intensive than the rail and off-highway alternatives. The relative high operating costs applied to the cable belt were based upon pessimistic estimates of rope life. It is probable that the detailed engineering studies proposed will result in a reduction of these operating costs to levels closer to those reported by cable belt users.

When compared to a conventional conveyor over long distances, the cable belt generally offered reduced maintenance costs, system simplicity and proven reliability. Since the belt itself does not transmit the driving tension, the potential for extensive longitudinal belt rips and the time required to perform low tension belt splices, is minimized. The overall rolling resistance is reduced in the conveyor belt system by eliminating belt flexibility over troughing idlers and reducing the total number of bearings. The net effect of these differences results in a lower power consumption.

There exist two potential disadvantages of the cable belt system when compared with the rail and off-highway trucks. First, the cable belt system is unique and solely dependent upon the performance of one conveyor. Although cable belts have demonstrated high availability at other mines complete spare parts modules must be maintained at the minesite to ensure the rapid change-out of worn or damaged components. Extensive lead times would be required for the purchase of all major mechanical components other than normal operating supplies.

The second potential disadvantage relates to the operation of the system in sub-arctic climates. Despite the universal acceptance of the cable belt for moving bulk materials over long distances there is no known installation that experiences the cold temperatures that prevail in the Yukon winter. Although information to date would suggest that the system is capable of providing satisfactory service under all seasonal conditions, an extensive analysis of the system will be conducted to satisfy these concerns.

6.24 HIGHWAY TRUCKS

This alternative required that off-highway pit trucks be loaded at the face and discharge their loads into a primary crusher located to the South of the Grum deposit. The crushed ore would be delivered to a stockpile from which material would be drawn by a conveyor belt to load trucks, each of which would be capable of hauling 54 tonnes. The trucks would proceed along a 15.2 metre wide road bed to a dumping station located to the East of the concentrator. Ore would be reclaimed by conveyor and thus introduced into the existing crushing circuit. This haulage system would require 12 units operating to deliver the planned tonnage of ore. Figure 6-5.

While it might be considered that the larger number of trucks offered improved system flexibility, it should be noted that

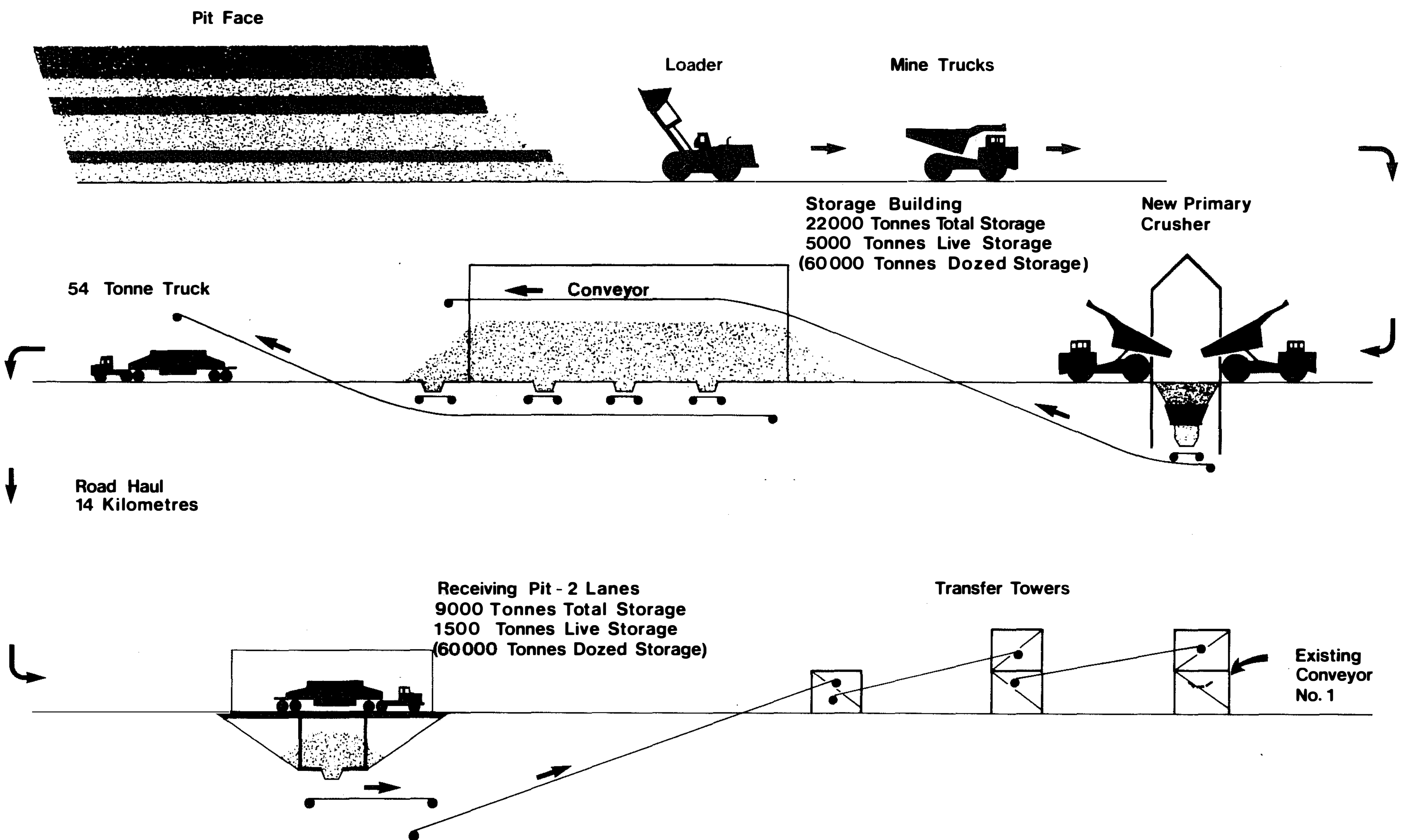


FIGURE 6-5 HIGHWAY TRUCKING SYSTEM
THE DEVELOPMENT OF THE VANGORDA PLATEAU ORE DEPOSITS

under no circumstances could these units be deployed in the Faro pit to be loaded directly from the face. Further the problems discussed previously relating to supervisory control, road maintenance, fuel and tire costs, were also disadvantages of this scheme.

6.25 THE CONVENTIONAL CONVEYOR

As was the case when considering the cable belt and highway trucking, the conventional conveyor system required that pit run ore be hauled to the crusher located at the Grum minesite. Subsequent to crushing and stockpiling, the ore would be delivered to the conventional conveyor. Due to the distance involved, the haulage would be achieved by utilizing two conveyors in series, each of which would be 6.4 kilometres long. The final belt would discharge onto the covered surge pile from which ore would be drawn to be introduced into the existing crushing circuit. Figure 6-6.

The high risk involved in operating conventional conveyor systems over long distances in cold weather negated further consideration of this alternative.

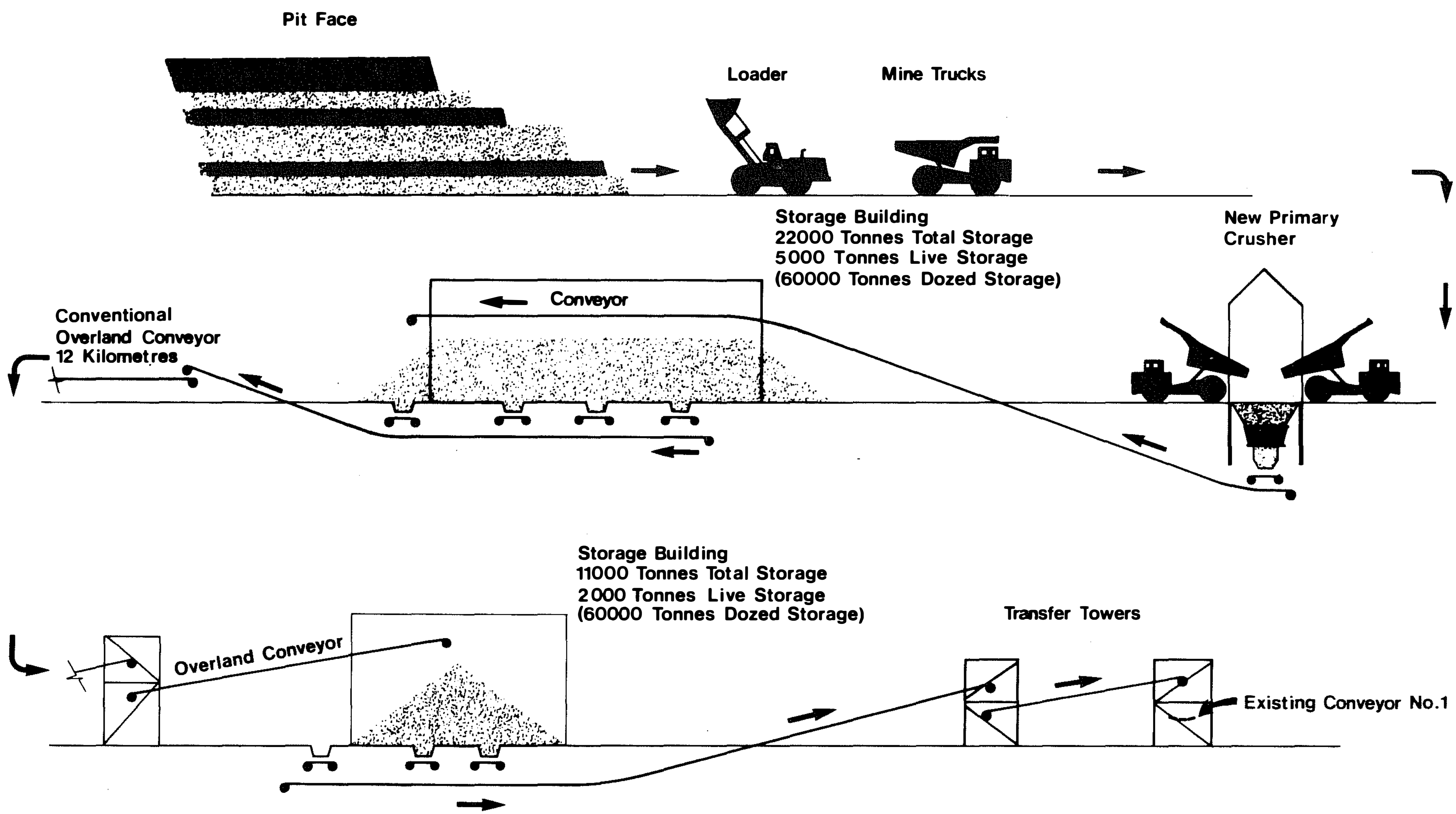


FIGURE 6-6 CONVENTIONAL CONVEYOR HAULAGE SYSTEM
THE DEVELOPMENT OF THE VANGORDA PLATEAU ORE DEPOSITS

6.3 THE SELECTION OF THE ORE HAULAGE SYSTEM

Detailed engineering studies are now in progress to evaluate the off-highway, rail and cable belt conveyor systems. Based upon the results of this work, the input from operating personnel at the minesite and appropriate plant visits, a comprehensive decision analysis will be conducted by mid 1980 to select the transportation system that will haul ore from the Vangorda Plateau to the Mill.

7. MILL MODIFICATIONS

7.1 INTRODUCTION

The original concentrator was modified in 1974 to treat 9,066 tonnes of Faro ore per calendar day and produce a flotation feed grind of about 60%-200 mesh (130 microns) with average ore.

Testwork performed on samples of Grum, Vangorda and Faro ores demonstrated that finer grind, lower flotation pulp densities and increased flotation retention times significantly enhanced the metallurgical performance of each constituent ore type.

The proposed mill modifications were designed to produce optimum conditions for improved metallurgical performance for each ore type. Since each ore type within the various deposits exhibits unique metallurgical characteristics, it would be entirely impractical to design a concentrator specifically to meet the particular requirements of each individual ore type. The proposed modifications will therefore provide a controlled mineral processing environment that will satisfy the principal treatment criteria common to each constituent ore specie.

7.2 PREDICTED PLANT METALLURGY

Predicted plant metallurgical results were based upon a critical analysis of the data generated by laboratory and pilot plant data. It was evident from the results of these studies that certain, readily identifiable ore types within each deposit exhibited differing metallurgical responses. These unique characteristics were considered in developing overall metallurgical performance data for each orebody.

7.21 GRUM

Extensive laboratory scale batch and cycle tests were carried out during 1977 at Lakefield Research on the various Grum deposit ore species, both individually and as composites. The objective of this work was to determine the basic treatment parameters necessary for the various ore types. The laboratory test results indicated that a fine primary grind and very fine regrinding would be required to successfully treat the average Grum ore type.The average results from a typical set of laboratory cycle tests at near optimum conditions are shown below.

FIGURE 7-1

LOCKED CYCLE FLOTATION TEST DATA - GRUM COMPOSITE SAMPLES

DATA SOURCE	CONCENTRATE	ASSAYS %		% DISTRIBUTION	
		Pb	Zn	Pb	Zn
Grum Report	Lead	62.2	*	86.9	*
L.R.1991 Vol.8	Zinc	*	57.4	*	82.6

a) Data from cycle tests 207, 217, 230, 237, 242

b) Tests conducted at 50-60 microns.

The laboratory work was then followed by a series of one tonne/hour pilot plant studies over a three month period at the Lakefield Research facility. These pilot plant tests performed at a 50 microns grind initially yielded poor results but, as the critical nature of the lead regrind mill operation was recognized, the metallurgy gradually improved. Some typical pilot plant results are shown below in Figure 7-2 and the results approximate to the data obtained from the laboratory cycle tests.

FIGURE 7-2

TYPICAL PILOT PLANT RESULTS - GRUM COMPOSITE SAMPLE

TEST NO.	PRODUCT	WEIGHT %	ASSAYS %		% DISTRIBUTION	
			Pb	Zn	Pb	Zn
46	Pb Concentrate	8.12	55.20	12.90	76.80	10.3
	Zn Concentrate	14.77	2.25	50.80	5.70	73.60
	Zn Comb. Tail.	77.11	1.33	2.14	17.50	16.10
	Cyclone O'Flow	100.00	5.74	9.85	100.00	100.00
47	Pb Concentrate	6.90	64.60	9.25	77.90	6.40
	Zn Concentrate	14.69	2.23	55.50	5.70	81.50
	Zn Comb. Tail.	78.41	1.19	1.54	16.40	12.10
	Cyclone O'Flow	100.00	5.72	10.00	100.00	100:00
48	Pb Concentrate	6.75	57.70	12.10	68.50	8.30
	Zn Concentrate	14.63	4.41	53.00	11.30	78.70
	Zn Comb. Tail.	78.62	1.46	1.63	20.20	13.00
	Cyclone O'Flow	100.00	5.69	9.85	100.00	100.00
49	Pb Concentrate	6.99	62.70	9.67	76.00	6.70
	Zn Concentrate	14.00	1.97	56.50	4.70	78.30
	Zn Comb. Tail.	79.01	1.41	1.92	19.30	15.00
	Cyclone O'Flow	100.00	5.77	10.10	100.00	100.00
50	Pb Concentrate	7.46	60.80	10.90	78.10	8.00
	Zn Concentrate	15.34	2.46	53.60	6.50	80.60
	Zn Comb. Tail.	77.20	1.16	1.51	15.40	11.40
	Cyclone O'Flow	100.00	5.81	10.20	100.00	100.00
51	Pb Concentrate	6.86	65.80	8.86	74.40	5.80
	Zn Concentrate	16.48	2.39	52.60	6.50	82.60
	Zn Comb. Tail.	76.66	1.52	1.60	19.10	11.60
	Cyclone O'Flow	100.00	6.07	10.50	100.00	100.00
52	Pb Concentrate	7.47	58.30	11.40	77.50	8.70
	Zn Concentrate	14.08	1.89	54.00	4.70	78.00
	Zn Comb. Tail.	78.45	1.27	1.65	17.80	13.30
	Cyclone O'Flow	100.00	5.62	9.75	100.00	100.00
53	Pb Concentrate	7.17	60.90	10.60	75.50	7.80
	Zn Concentrate	14.13	2.13	54.70	5.20	78.90
	Zn Comb. Tail.	78.70	1.41	1.67	19.30	13.30
	Cyclone O'Flow	100.00	5.78	9.80	100.00	100.00

Notes: a) Data source L.R.2027 Vol. 1.

The sample blended for the pilot plant work was atypical in that it was abnormally rich in sulphide minerals and exhibited very high lead and zinc contents. The pilot plant results were used however, in conjunction with the results of the cycle tests on the various ore composites, to predict plant metallurgical performance. The table below shows the actual results achieved, the inferred results for average Grum ore, and the predicted plant performance allowing for ore dilution.

FIGURE 7-3

PREDICTED PLANT METALLURGY AND PILOT PLANT RESULTS - GRUM ORE

ORE TYPE	CONCENTRATE	ASSAYS Pb%	Zn%	Au g/tonne	Ag g/tonne	DISTRIBUTION % Pb	Zn	Au	Ag
Massive Sulphide Pilot Plant Data	Lead	62	10	4.8	925	77	*	33	72
	Zinc	2.5	56	*	*	*	81	*	*
Average Grum Estimated Metallurgy *	Lead	62	8	5.1	950	80	*	33	72
	Zinc	2.0	56	*	*	*	84	*	*
Predicted Plant Metallurgy **	Lead	60	11	3.5	750	80	*	33	65
	Zinc	2.5	55	*	*	*	83	*	*

* Data source for average Grum metallurgy - Noranda Milling Committee Report December 1977.

** Data source for predicted metallurgy - estimates by Cyprus Anvil Metallurgists.

7.22 VANGORDA

Laboratory flotation testwork has been performed intermittently on samples of Vangorda ore since the discovery of the deposit, twenty seven years ago. The most recent work performed by the Dowa Mining Company, Galigher Company, Brunswick Mining & Smelting Ltd., and Noranda Mines Ltd. produced a broad divergence in results.

Accordingly, an extensive metallurgical test program was designed and carried out in 1979 based upon fresh diamond drill core selected to be representative of the three major ore types identified within the deposit; namely baritic (4G), pyritic (4E), and quartzitic (4A) ores. The program was designed to establish a metallurgical profile for each ore type at various, predetermined ore grades.

The results of this work were used to predict an average metallurgical response for each ore type.

FIGURE 7-4

VANGORDA METALLURGY BY ORE TYPE

ORE TYPE	RELATIVE OCCURRENCE IN DEPOSIT %	PREDICTED METALLURGY ASSAYS Pb	Zn	Au	Ag	Hg	As	DISTRIBUTION Pb	Zn	Au	Ag
4G Baritic	31	53	*	6.0	650	80	0.05	84	*	40	65
		*	55	*	*	440	0.02	*	80	*	*
4E Pyritic	39	49	*	10.0	480	60	0.20	77	*	50	50
		*	51	*	*	250	0.05	*	72	*	*
4A Quartzitic	30	48	*	2.4	600	40	0.50	81	*	15	55
		*	53	*	*	250	0.20	*	79	*	*

Notes: a) Data from best adjusted test data from type testing program.

b) Relative occurrence data calculated from the mine model.

c) Au, Ag, Hg, in g/tonne.

The metallurgical parameters shown above were then averaged on a weighted basis, depending upon the relative occurrence of each ore type within the deposit. The results in Figure show the weighted average metallurgical performance. Since the type 4G ores were generally of a significantly higher grade than either 4E or 4A, it is probable that actual plant results may improve upon the predicted metallurgy shown below.

FIGURE 7-5

PREDICTED PLANT METALLURY - VANGORDA ORE

CONCENTRATE	ASSAYS Pb	Zn	Au	Ag	Hg	As	DISTRIBUTION Pb	Zn	Au	Ag
Lead	50	*	6.5	575	60	0.25	80	*	35	55
Zinc	*	52.8	*	*	300	0.10	*	77	*	*

Notes: a) Metallurgy calculated from weighted average of type testing data.

b) Au, Ag, Hg, in g/tonne.

Vangorda lead concentrates will be characterized by relatively low lead concentrate grades due to extremely fine crystal intergrowths. The concentrates will also exhibit relatively high levels of arsenic, silver and gold.

7.23 FARO ZONE III

Despite the medium to fine grained mineral composition of the Zone III material, laboratory testwork conducted by Lakefield Research on samples of this ore demonstrated that acceptable metallurgical results could be attained under certain conditions. The results of one test series are shown below.

FIGURE 7-6

LABORATORY FLOTATION TEST RESULTS - FARO ZONE III ORE

SOURCE	CONCENTRATE	ASSAYS %		% DISTRIBUTION	
		Pb	Zn	Pb	Zn
Cycle Test No. 18	Lead Zinc	61.5	 52.0	91.2	 87.9
Predicted Metallurgy from Laboratory Work	Lead Zinc	62-68	 52-54	83-91	 87-90

Note: Tests performed at medium-fine grind with a P_{80} 70-75 microns. Data source L.R. 2082.

In conjunction with this work at Lakefield, extensive laboratory testwork was carried out by the metallurgical staff at Faro, (C.A.M.C.) and in Kamloops (K.R.A.L.) on various samples of Faro ores. The objectives of the programs were to confirm that finer grinding improved overall metallurgical performance.

FIGURE 7-7

METALLURGICAL IMPROVEMENTS WITH FINE GRINDING FARO ORES

DATA SOURCE	GRIND RANGE INVESTIGATED		METALLURGICAL IMPROVEMENTS ACROSS RANGE			
	(P_{80} microns)	(%-325#)	LEAD		ZINC	
			GRADE	RECOVERY	GRADE	RECOVERY
Rougher Tests KRAL-78	135 - 40	50 - 85	7.0	-	2.50	-
Rougher Tests CAMC 78-79	125 - 37	45 - 90	11.3	-	5.9	-
Cleaner Tests KRAL-79	125 - 45	45 - 80	2.5	7.0	2.1	9.5
Cleaner Tests CAMC-79	125 - 40	45 - 85	3.2	5.2	4.4	4.0
Plant Test Nov.13-17/78	125 - 60	45 - 70	3.0	5.8	3.0	4.0
Cleaner Tests KRAL-79	145 - 30	30 - 90	-	9.6	-	12.0

Notes: a) - signifies that results were adjusted to a constant grade or recovery for the purposes of analysis of data.

As a result of the encouraging bench scale test data, it was decided to conduct a pilot plant test on ores typical of Zone III material. Two discrete samples consisting primarily of 4E ore types weighing 200 tonnes and 40 tonnes respectively, were selected from the Faro pit and shipped to Lakefield in September 1979.

The primary objectives of the pilot plant were to establish the level of grind and the flowsheet that produced the optimum metallurgical performance, and also to generate data that could be used for predicting the metallurgical results that would be achieved by the modified plant.

Test results generated in this program were adversely affected by oxidation of the samples during the transportation of the two ore samples to the Lakefield facility The larger sample designated Sample No. 1, consumed especially large quantities of soda ash and produced the very poor lead recoveries characteristically exhibited by oxidized ores. Sample No. 2 was found to be slightly less oxidized and under, conditions of fine grind and low pulp densities produced exceptionally good metallurgical results.

The results of selected pilot plant tests performed at pre-determined levels of grind are summarized below.

FIGURE 7-8

METALLURGICAL RESULTS USING DIFFERENT LEVELS OF GRIND - FARO ORE

TEST NO. PILOT PLANT	PRIMARY GRIND P_{80}	PRIMARY GRIND %-200	BULK SAMPLE	PRODUCT	WEIGHT %	ASSAYS % Pb	ASSAYS % Zn	% DISTRIBUTION Pb	% DISTRIBUTION Zn
9,10,24	49	91.3	No. 1	Pb Cleaner Conc. Zn Cleaner Conc. Zn Comb. Tail.	2.58 6.35 91.07	68.4 0.46 0.58	2.91 52.10 0.51	76.0 1.2 22.8	1.9 86.0 12.1
				Head (Calc.)	100.00	2.32	3.85	100.0	100.0
22, 23	77	77.9	No. 1	Pb Cleaner Conc. Zn Cleaner Conc. Zn Comb. Tail.	2.69 6.48 90.83	64.65 0.94 0.57	3.49 49.76 0.41	75.0 2.6 22.4	2.5 87.4 10.1
				Head (Calc.)	100.00	2.32	3.71	100.0	100.0
16, 18, 19, 26, 27.	135	58.0	No. 1	Pb Cleaner Conc. Zn Cleaner Conc. Zn Comb. Tail.	2.84 6.55 90.61	64.6 1.10 0.64	3.65 49.1 0.74	73.0 3.4 23.6	2.3 79.6 18.1
				Head (Calc.)	100.00	2.47	3.94	100.0	100.0
31, 35	59	87.6	No. 2	Pb Cleaner Conc. Zn Cleaner Conc. Zn Comb. Tail.	2.60 6.96 90.44	76.15 0.49 0.43	1.99 55.00 0.52	82.1 1.4 16.5	1.2 88.0 10.8
				Head (Calc.)	100.00	2.41	4.35	100.0	100.0
28, 29	137	57.8	No. 2	Pb Cleaner Conc. Zn Cleaner Conc. Zn Comb. Tail.	2.70 6.65 90.65	71.6 0.69 0.57	3.55 54.15 0.67	77.5 1.8 20.7	2.2 83.7 14.1
				Head (Calc.)	100.00	2.50	4.30	100.0	100.0

Data source L.R.2202 No. 6, Vol. 1.

An analysis of the results indicated that the optimum metallurgical performance was obtained when grinding the flotation feed to about 50 microns.

The predicted lead and zinc plant metallurgical performance was based upon pilot plant results obtained when treating ores at a grind of 50 microns, adjusted to compensate for the measured degree of sample oxidation.

The pilot plant results as recorded, and also adjusted for oxide content are shown below. The predicted plant metallurgy is compared with the 1978, 1979 plant performance.

FIGURE 7-9

PLANT METALLURGY COMPARED WITH PILOT PLANT RESULTS - FARO ZONE III ORE

DATA SOURCE	LEAD		ZINC	
	GRADE % Pb	RECOVERY %	GRADE % Zn	RECOVERY %
Pilot Plant Results	69.1	75.8	52.5	88.8
Adjusted Pilot Plant Results	68.4	86.9	52.1	90.4
Predicted Plant Metallurgy	67.0	87.5	53.5	88.5
1978-79 Plant Performance	61.1	84.0	50.4	80.0

Notes: a) Pilot plant data refers to sample No. 1 only at the optimum grind of 50 microns P_{80}: Test No. 9.

b) Adjusted data involves an empirical correction for the oxide content of the test sample.

c) Plant performance data is an arithmetic average over two years.

The predicted plant metallurgical performance, including metal distribution is shown below.

FIGURE 7-10

PREDICTED PLANT METALLURGY - FARO ZONE III ORE

ORE SOURCE	CONC.	METALLURGY									
		ASSAYS						DISTRIBUTION			
		Pb	Zn	Au	Ag	Hg	As	Pb	Zn	Au	Ag
Faro-Zone III	Lead	67	*	0.6	600	40	0.03	87.5	*	33	65
	Zinc	*	53.5	*	*	300	0.01	*	88.5	*	*

Gold and silver recoveries were deduced from pilot plant and laboratory results. Using silver and gold concentration in the mill feed, predicted by the computer model, the precious metal concentrations in the lead concentrate were calculated. It is probable that, due to the sporadic occurrences of gold in the orebody, that some shipments of lead concentrate will contain payable quantities of gold.

7.3 PLANT DESIGN CRITERIA

The modified plant was designed to be capable of milling all Grum, Vangorda and Faro Zone III ores, individually or concurrently at planned throughput rates to achieve optimum overall metallurgical performance. Due to differing hardness and metallurgical characteristics of each ore type identified within the three deposits, it could be reasonably argued that no one constituent ore type in any blend of mill feed could yield its unique, optimum metallurgical performance. However, since such fundamental operating criteria as grind, retention times, pulp densities and reagent consumptions are similar for these ore types, the total plant performance will represent an acceptable metallurgical compromise.

7.31 OPTIMUM GRIND LEVEL

An instantaneous mill throughput rate of 450 tonnes per hour was used in all grinding calculations to ensure that planned annual mill throughput rates will be attainable under normal circumstances while achieving the target grind of about 50 microns in the flotation feed.

Several hypothetical economic models were constructed and evaluated by varying levels of grind and regrind to reduce capital and operating costs. Metallurgical performance parameters were assigned to each model based upon laboratory and pilot plant test results at the various grind levels. A summary of the appropriate net present value determinations is tabulated below in Figure 7-11.

FIGURE 7-11

PLANT OPTIMUM ECONOMIC GRIND LEVEL

CASE	PRIMARY H.P.	REGRIND H.P.	TOTAL H.P.	GRIND TARGET P_{80}	CAPITAL COST $\$x10^6$	OPERATING COST INCREASE $\$x10^6$	RELATIVE NET PRESENT VALUE $\$x10^6$
1	11,700	1,350	13,050	50	43.0	6.8	153.9
2	9,200	2,350	11,550	70	43.1	5.1	129.8
3	7,700	1,850	10,550	100	38.6	4.3	112.7
4	5,200	1,350	6,550	130	0	0	117.4

Notes: a) Reference - Memo Brown to Taggart Dec. 16, 1979 "Optimum Grind Calculations".

b) Capital cost includes power plant: Clearly case 4 is with no expansion.

c) Operating cost includes manpower, steel and reagents.

d) Data assumes milling a 50:50 blend of Grum and Anvil Zone III ore.

The reported net present values are not absolute, but rather provide a relative comparison between the various flowsheets considered. It was concluded that the flowsheet designed to produce optimum metallurgical performance, also provided the most favourable financial alternative.

7.32 GRINDING CIRCUIT DESIGN

Mill design calculations were based upon standard Bond methods to determine installed horsepower requirements. These calculations indicated that in addition to the existing grinding

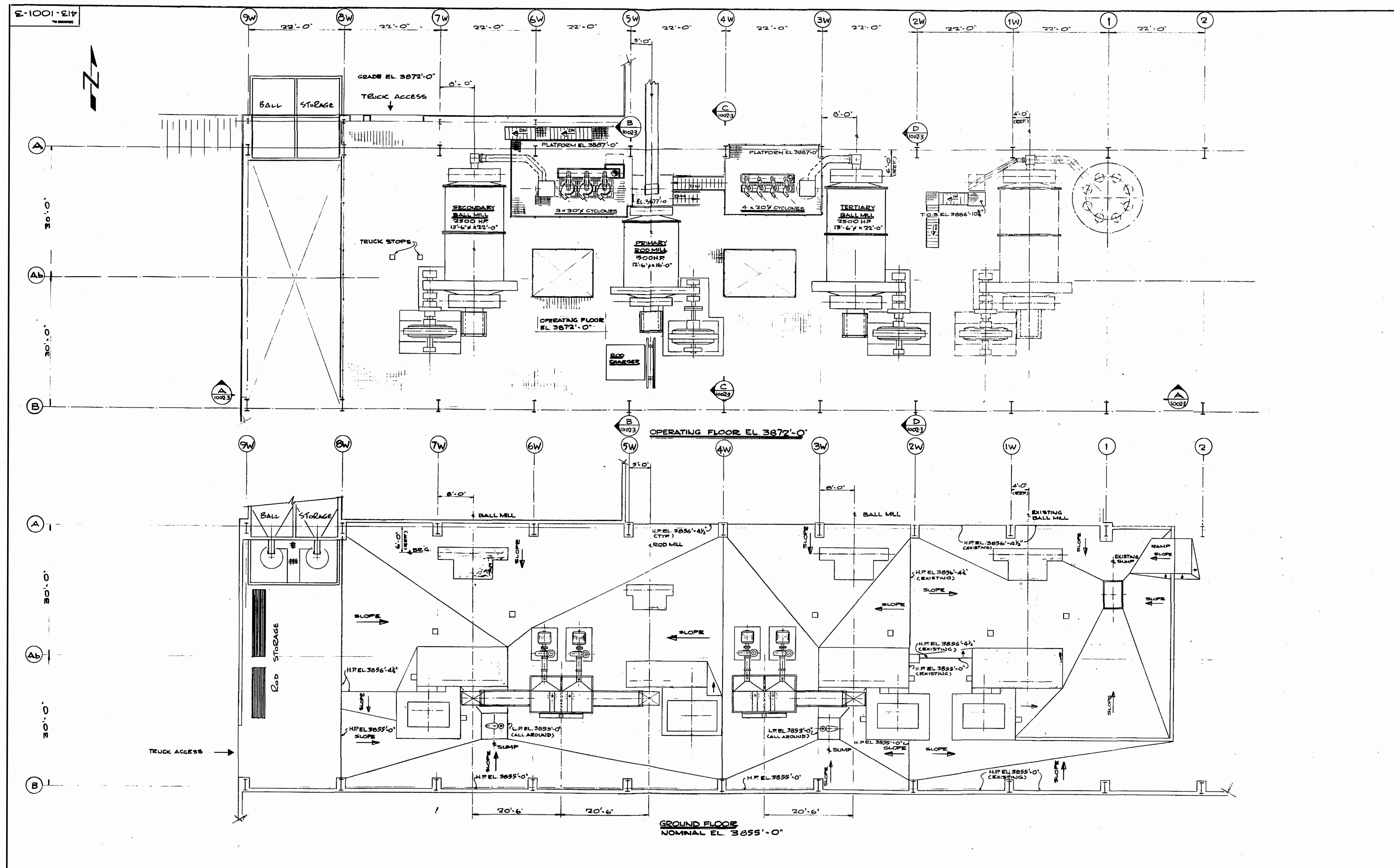


FIG. 7-12

TITLE
MODIFIED MILL GRINDING AREA GENERAL ARRANGEMENT

circuit one new rod mill with a 1500 HP motor, and two in-series ball mills each driven by a 2500 HP motor, would be capable of achieving the target grind of 50 microns at planned throughput rates. All calculations were based on a work index value of, 13.2 kW hr/tonne, the highest value reported during grindability testwork.

Regrind mill capacities will be just sufficient to achieve the required mineral liberation. Ore will be supplied to the new grinding circuit from two additional bins that will effectively supplement the marginal capacity of the existing fine ore storage. Figure 7-12.

7.33 FLOTATION CIRCUIT DESIGN

Results of testwork and experience at other operations has demonstrated that, in addition to fine grinding, certain other factors significantly affect metallurgical performance. Specifically, flotation pulp densities must be reduced and retention times extended when compared with current practice, to create an environment conducive to the efficient selective flotation of the finer minerals.

To achieve both of these objectives, the existing induced air 100 cubic foot flotation cells will be replaced with 1350 cubic foot supercharged machines that will serve as lead and zinc roughers and first cleaners. The existing 200 cubic foot supercharged flotation cells will be converted to second and third stage cleaner circuits for both circuits. The net effect of these changes is summarized below.

FIGURE 7-13

FLOTATION CELL CAPACITY REQUIREMENTS

CIRCUIT	TOTAL CELL CAPACITY (ft^3)	CAPACITY/TONNE (ft^3/tonne)
Anvil Present	29,200	3.22
Anvil Modified	55,200	6.09

The grinding pulps will be aerated prior to lead roughing and conditioners will prepare the mineral surfaces for zinc flotation. The aerators and conditioners form integral sections of their respective rougher flotation cell banks.

Reagents will be received, stored and mixed in a new reagent handling facility to be located immediately to the North of the fine ore storage bins. The plant will offer a much greater degree of environmental security than the existing systems. Basic instrumentation will ensure consistent reagent solution strengths and provide adequate warning of any undesirable condition in the new reagent plant.

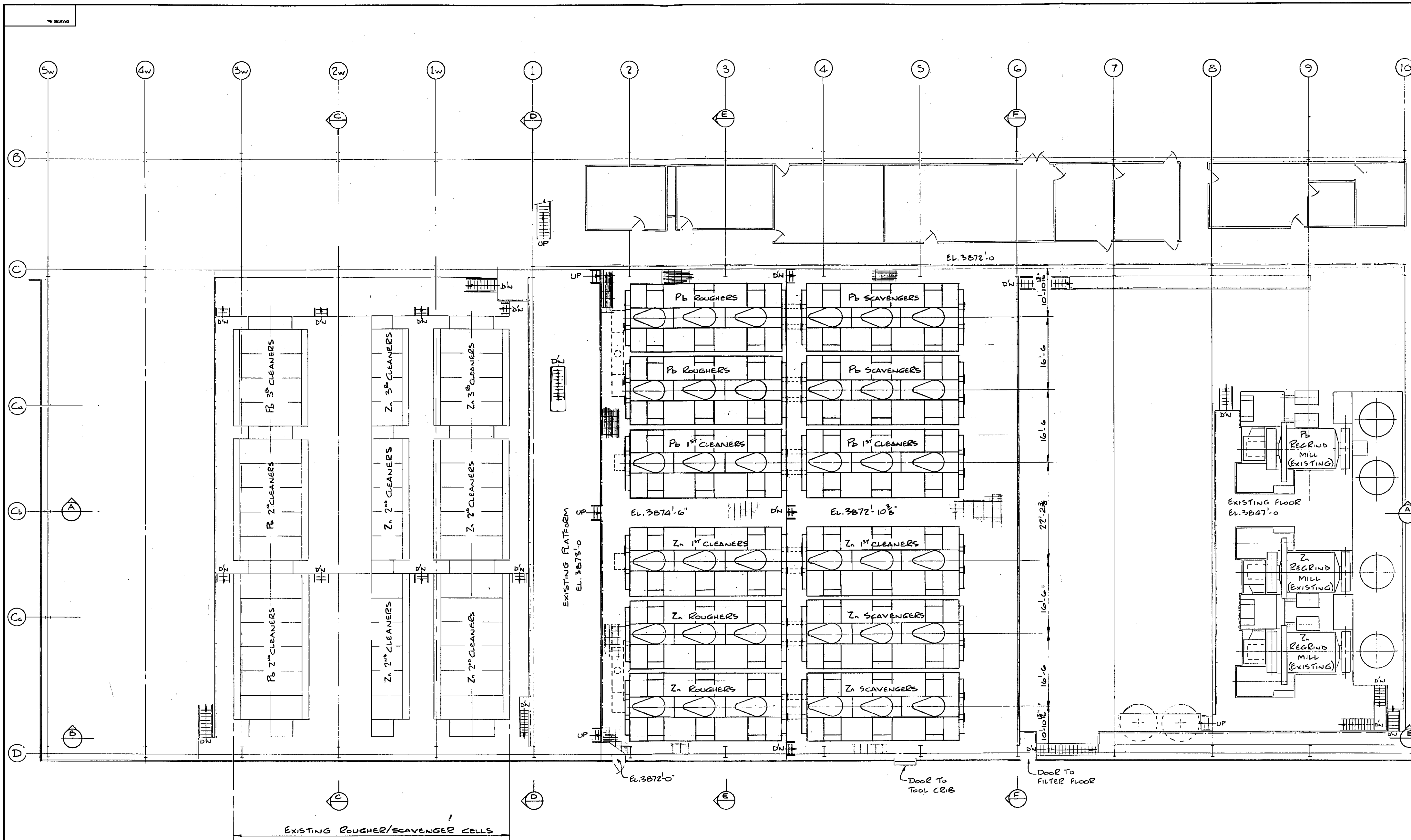


FIG 7-14

TITLE

MODIFIED MILL
FLOTATION & REGRIND
GENERAL ARRANGEMENT

7.34 INSTRUMENTATION

The modified plant will contain some sophisticated but field proven instrumentation to monitor and control some of the key process functions. The equipment selected will provide the minimum information required to control the process and will also form the basis for any new automatic control strategies. Provision of automatic control systems, will be delayed until the necessary strategies are formulated; usually this occurs two or three years after start-up of an on-stream analyzer system.

The principal component of the new instrumentation system will be the Courier 300 on-stream fluorescence analyzer. This unit is capable of measuring the concentration of four elements which will include lead, zinc and iron, in each of fourteen process streams. In addition the unit will measure the pulp density of each of the streams, and print-out all the data on a high speed typewriter.

A particle size monitor of the PSM-200 type will be installed on the lead circuit feed. This unit measures mean particle size and pulp density.

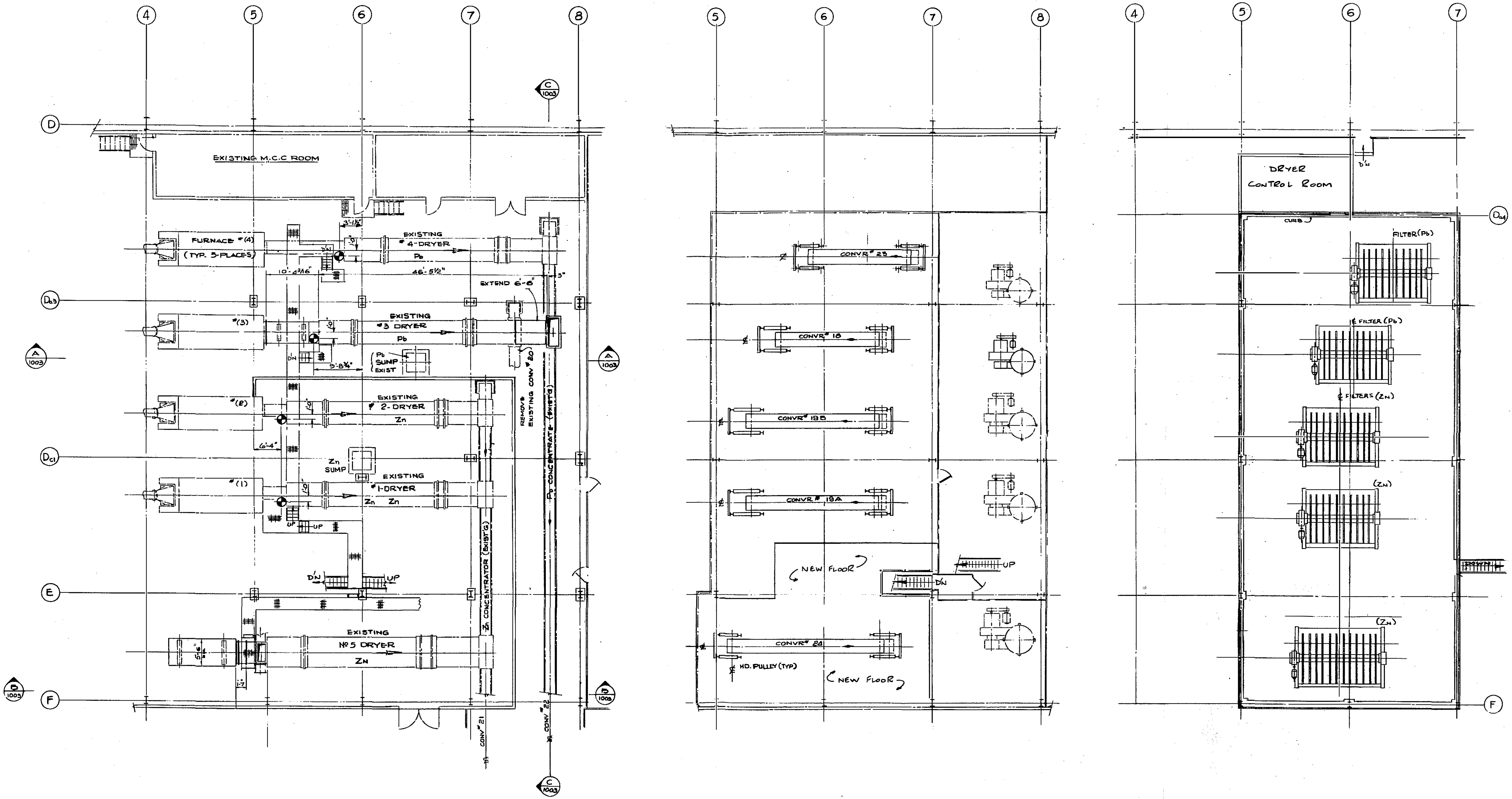
Key reagent flows will be monitored by employing magnetic flow monitoring devices on certain critical reagent lines - e.g. cyanide, soda ash solution, collector and copper sulphate. These flows will be displayed on strip charts and totalized. pH control will be achieved by feed-back systems on the soda ash and lime pressure loops, from pH sensor probes located in the flotation cells.

Mass flow of material into the lead and zinc cleaner circuits will be measured and, using assay data, the circulating cleaner circuit load will be determined.

7.35 DEWATERING CIRCUIT DESIGN

Each concentrate will be thickened and a clarifier used to treat the thickener overflow. The clarifier overflow will be returned to the reclaim water system for re-use. Thickener underflow density control will be effected by means of variable speed diaphragm pumps which eliminate wide fluctuations in filter feed densities.

The filter feed pulps will be heated to improve filtering efficiency. The elevated pulp temperatures effectively reduce surface tension, thereby facilitating improved cake drainage on the filter. The existing filters will be replaced with larger



EXISTING CONCENTRATE DRYERS
BASEMENT FLOOR

NEW FILTER CAKE CONVEYORS
FIRST FLOOR

NEW FILTERS
SECOND FLOOR

FIG. 7-15

TITLE
MODIFIED MILL FILTERING & DRYING GENERAL ARRANGEMENT

units to increase the available filter media area by a factor of three to permit adequate time for cake formation and drainage.

Two additional 5000 CFM vacuum units, identical to the three presently in use will be installed to service the additional filter capacity. An additional 1200 CFM compressor is included in the design to provide air for the filter snap-blow systems and other purposes.

The elimination of bulk concentrates greatly simplifies the layout and thus the operation of the dewatering plant. Contamination between concentrates will be minimized by reducing the number of conveyors, and by dividing the existing basement into two discrete areas.

The allocation of dryers to specific functions is based upon theoretical design calculations that indicate that the existing units can dry the maximum anticipated concentrate production to acceptable moisture contents. The plan figures below are based upon milling planned tonnages of ore for 1984. This year was chosen since the highest annual concentrate output is predicted for 1984.

FIGURE 7-16

THE DEWATERING PLANT CAPACITY

PRODUCT	PLANNED OPERATING RATES (DMT/OPERATING HR)	DESIGN OPERATING RATES (DMT/OPERATING HR)
Lead concentrate Production	21.2	25.2
Zinc concentrate production	36.7	53.2

Grinding circuit availability 92%
Dewatering circuit availability 85%

Day to day changes in plant feed grades require some provision to dewater concentrate above planned levels for short periods of time. To obtain this essential "reserve capacity", dryer slopes have been steepened and the fire box configuration on number 5 dryer has been improved to maximize dryer throughput rates.

New weightometers are to be installed on the loadout belts to accurately monitor the dewatering plant production rates.

8.
POWER SUPPLY

8.1 CYPRUS ANVIL POWER REQUIREMENTS

The estimated power requirements to service the modified mill and mine development programs were calculated by applying the same load factor as presently measured to the designed connected horsepower load for determining peak demand and energy consumption criteria.

The modified mill equipment register and predicted mine equipment loads were used to develop the forecast connected load. For the purposes of this calculation it was assumed that a conveyor would be used to transport ore from the Vangorda Plateau to the mill site, since this alternative represents the worst case from an electrical load stand-point. The peak demand and annual energy consumption were calculated to be 25.0 MW and 155 GW hrs respectively.

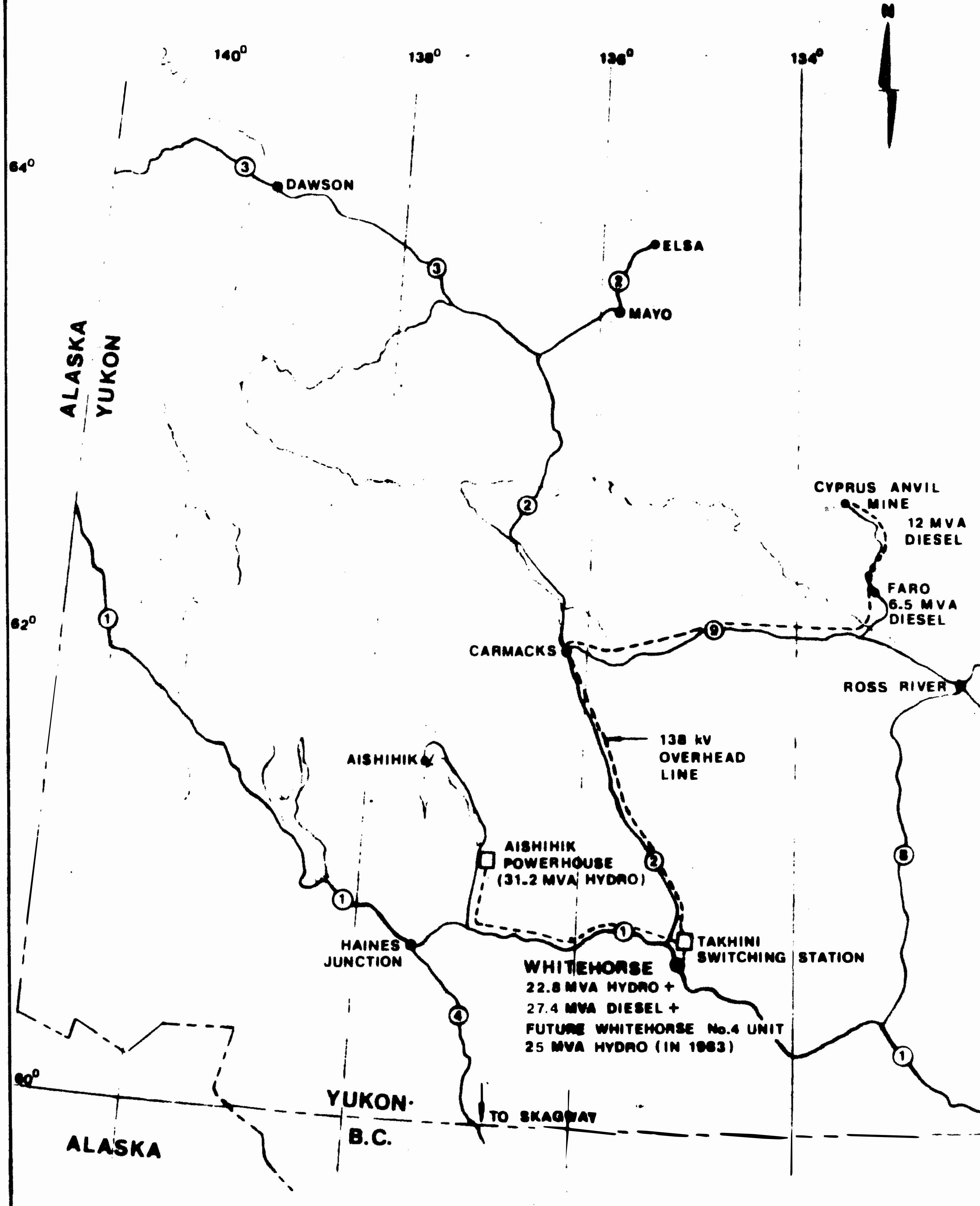


FIG. 8-1

NCPC YUKON POWER SYSTEM

8.2 THE N.C.P.C. POWER GENERATING CAPABILITIES

The N.C.P.C. electrical distribution grid provides power to Faro and the minesite, and also services Whitehorse, Carmacks, Ross River, and a number of smaller communities. (Figure 8-1). The total capacity of the present system is 77.7 MW. Following the installation of a fourth hydroelectric turbine at the Whitehorse Rapids plant in 1983, the system capabilities will be increased to 97.7 MW, exclusive of the proposed local generating capacity. The Figure 8-2 below summarizes the N.C.P.C. generating capabilities and includes the small diesel unit located at Faro, installed primarily to provide emergency domestic power to the communities of Faro and Ross River in the event of a power outage.

FIGURE 8-2

N.C.P.C. GENERATING CAPACITY

SOURCE		PRESENT CAPACITY (MW)	POST MID 1983 CAPACITY (MW)
Hydro:	Aishihik	31.0	31.0
	Whitehorse Rapids	20.0	40.0
	Sub-Total	51.0	71.0
Diesel:	Faro Plant	5.1	5.1
	Whitehorse Rapids	21.6	21.6
	Sub-Total	26.7	26.7
TOTAL		77.7	97.7

Appropriate utility system reserve capacities are normally computed by adding 5 percent of the anticipated peak demand to the capacity of the largest single generator that could be taken "off-line"; in this case one of the two hydroelectric turbines (15.5 MW) located at Aishihik. The forecast system peak demand 1980-81 is 55.5 MW. Thus:

Reserve capacity (MW) = capacity largest unit + (0.05 x peak demand)

= 15.5 + (0.05 x 55.5)

= 18.3 MW

As shown in figure 8-3 below the system has adequate capacity to satisfy the current requirements. The additional 9.0 MW load imposed by the development programs however, will result in a 10.0 MW shortfall in system capacity, which could be largely offset by the installation of a 9 MW thermal generating facility. The temporary reduction in reserve capacity is not sufficient to justify the installation of additional generating equipment, particularly when considering the excess system capacity that will exist once the new hydroelectric turbine is installed and operating by mid 1983.

FIGURE 8-3

N.C.P.C. SYSTEM CAPACITY vs. REQUIREMENTS

CASE	PERIOD	PEAK DEMAND MW	CALCULATED RESERVE MW	REQUIRED GENERATING MW	ACTUAL CAPABILITY MW	SURPLUS (SHORTAGE) MW
1	Forecast 1980/81	55.5	18.3	73.8	77.7	3.9
2 *	Forecast plus Vangorda Development Load	68.8	18.9	87.7	77.7	(10.0)
3	As for 2 plus 9 MW Power Plant	68.8	18.9	87.7	86.7	(1.0)
4	As for 2 plus Additional Hydro Unit at Whitehorse	68.8	18.9	87.7	106.7	19.0

* Includes provision for normal load growth.

From this analysis of the peak demand criteria it was concluded that a 9 MW increase in capacity will be required to satisfy the projected load in 1982 and at least for the first 6 months of 1983. Delays in commissioning the additional 20 MW turbine in Whitehorse would extend the dependency on the generating plant.

Similarly a study of the forthcoming energy production capabilities indicated that sufficient hydroelectric energy would be available to satisfy the system requirements in total by 1984, Figure 8-4 below.

FIGURE 8-4

LOCAL ELECTRICAL ENERGY REQUIREMENTS (GW hrs)

YEAR	1980	1981	1982	1983	1984	1985	1986
C.A.M.C. Requirement	100	100	145	145	145	155	155
N.C.P.C. Contribution	100	100	100	123	145	155	155
Local Generation Requirement	-	-	45	22	-	-	-

The above data assumes that the new hydroelectric turbine will be in service by mid 1983 and will be capable of producing 80 GW hrs/year. This may only be achieved following the resolution of winter and spring problems in the Marwell area of Whitehorse. The data above also assumes that the Whitehorse Copper Mine curtails production in 1983 as scheduled. Delays in a shut-down of this operation for any reason, would extend the dependency of Cyprus Anvil on locally generated energy.

8.3 SUPPLEMENTARY POWER GENERATION

Network analyses of the system were integrated with predicted total grid loadings to establish the need for an additional 9 MW generating facility, that would be required for at least the first two years of operation subsequent to the start-up of the modified mill circuit. A number of power generating alternatives were considered including, coal fired steam generators, hydroelectric generating systems, gas turbines, and diesel engine generators.

A preliminary feasibility study was conducted by Montreal Engineering Ltd. to investigate the viability of coal fired steam generators, for providing short and long term energy supply. The investigation indicated that such generation is worthy of further study in the future as coal quality and reserves are better defined. A coal fired power station could not be commissioned in the time required by the development schedule.

Hydroelectric generating potential in the immediate area was found to be limited. Nevertheless, studies are scheduled to commence in the near future to develop an inventory of possible sites and identify their respective generating potentials.

Gas turbines were found to be cheaper than stationary diesel generating units in terms of capital costs. However, their inferior fuel efficiency characteristics, render these machines more suitable for stand-by and peaking service.

Diesel engine power generating units are normally favoured to supply long term energy needs. These units require a higher initial capital investment, however this disadvantage is offset by the improved reliability and low unit cost of energy produced.

The final plant design is currently in progress to select the most cost efficient and reliable generating system that would provide the incremental power requirements until adequate and cheaper hydro-electric energy is available. Both diesel engine and gas turbine alternatives are under consideration. For this purpose it was assumed that the power plant would be capable of generating up to a maximum of 10 MW.

Despite the installation of suitable compensating and control equipment, line losses in the order of 16 percent were predicted if the additional energy were generated in Whitehorse. Further study proved that it would be cheaper to transport fuel to the Faro area to a plant closer to the load centre, than to incur such substantial transmission losses.

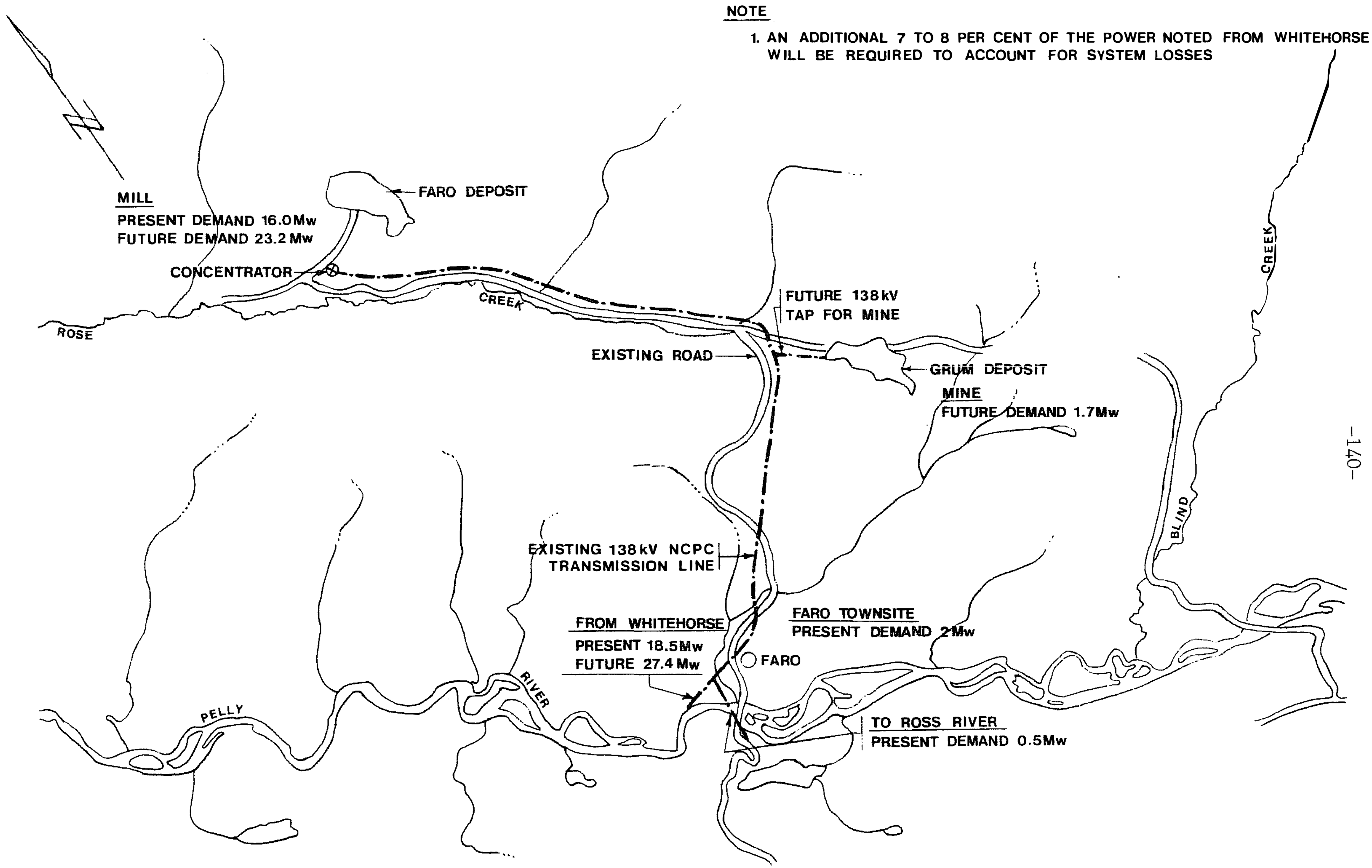


FIG. 8-5

N. T. S.

POWER DISTRIBUTION REQUIREMENTS FARO AREA

Power will be provided to the Vangorda Plateau minesite by means of a branch line taken off the existing 138 kV transmission line between the Town and the Faro minesite. A transformer and sub-station will be located at the Grum minesite.

The power generation and modified distribution system provides for an additional step-down transformer to parallel the 16 MVA unit presently in service. The total cost of the generating station transformers and transmission line to the Grum minesite will be approximately $12.0 million.

8.4 THE COST OF ELECTRICAL ENERGY

The figure below summarizes the annual minesite energy requirement by year and identifies the respective energy sources.

FIGURE 8-6

TOTAL ENERGY PRODUCTION AND COST BY YEAR

YEAR	1980	1981	1982	1983	1984	1985	1986
Total Energy Required (GW hrs)	100	100	145	145	145	155	155
N.C.P.C. Energy Available (GW hrs)	100	100	100	123	145	155	155
Additional Generation Requirements (GW hrs)	-	-	45	22	-	-	-
Total Energy Cost ($ million)	4.61	4.61	8.32	7.71	6.68	7.15	7.15
Unit Energy Cost (¢/KWH)	4.61	4.61	5.74	5.32	4.61	4.61	4.61

Total energy costs were derived by calculating the weighted average value for locally generated energy and that supplied by the existing grid.

8.5 LONG TERM POWER SUPPLY

The N.C.P.C. is presently considering a number of potential hydroelectric sites within the Yukon with a view to commencing staged development programs to provide sufficient energy for long term territorial growth. Areas currently under study are listed below with their respective potential capacities.

FIGURE 8-7

POTENTIAL HYDROELECTRIC DEVELOPMENTS

POTENTIAL SITE	POTENTIAL CAPACITY MW
Eagle Rock	75
Francis River	70
Teslin River	45
Little Salmon River	15

In view of the high transmission losses referred to earlier, and the high cost of thermal generated energy, there will be considerable advantage to Cyprus Anvil if any one of these sites could be developed.

In addition to these studies conducted by the N.C.P.C., Cyprus Anvil will continue to evaluate the feasibility of local hydroelectric and thermal power generating systems.

9. ENVIRONMENTAL CONSIDERATIONS

9.1 INTRODUCTION

The protection of the natural environment and the improvement in working conditions were vital considerations in the design of the long term development program. There are three principal areas of concern in this regard, namely the levels of airborne particulates, of noise in the working environment, and the potential adverse impact future mining developments could impart to the flora, fauna, water quality and aesthetic beauty of the natural surroundings.

9.2 AIRBORNE PARTICULATES

The potential to generate excessive levels of airborne particulates is inherent in the basic processes of crushing, grinding, flotation and concentrate drying. Dust may be generated directly by the process or indirectly by emissions from material spillages that have accumulated in the plant. While all levels of dust are undesirable, the so called "respirable dust", ranging in size from zero to five microns, is the size range deemed most harmful to health.

Significant improvements have been achieved during the last five years in reducing the concentrations of these undesirable, ultra--fine particles. These changes have been realized through the implementation of more effective plant ventilation.

Certain features have been incorporated in the new plant design that will further reduce airborne dust levels. Thus all new flotation cells will be covered and vented to minimize ultra-fine dust generation to atmosphere from flotation froth surfaces.

The most prolific dust source in the filter dryer area has been the transfer conveyor system feeding the dryers. The number of conveyor belt transfer points in the modified dewatering plant has been significantly reduced. In addition a solid floor will be laid directly above the dryers to effectively isolate the dryer section and divide that part of the building into individually ventilated areas. Improved filter and dryer control systems will minimize periods of over-drying and hence dusting conditions, while reducing overall product moisture.Each of the above measures will contribute to much improved air quality and working environment in the dewatering section.

The design of the modified plant will do much to reduce process spillage which is not only unsightly but may also expose employees to elevated levels of dust. The principal sources of process spillage are the grinding and flotation section basements. There is no doubt that the redistribution of mill feed within the expanded grinding circuit, the redesigned flotation launders and improved circuit control facilities will do much to alleviate this problem.

At present the reagent mixing plant is centrally located in the concentrator. Chemical dusts emanating from this vicinity that are not exhausted by the local ventilation system, migrate through the flotation section and the mill mechanical workshop exposing employees to inhalation of this material. The modified plant design provides for a new reagent receiving and mixing facility, physically separated from the operating floors and serviced by a dedicated heating and ventilation system, thus minimizing employee exposure to these potentially harmful chemical dusts.

The existing lunchroom is small, poorly located and difficult to keep clean. Dust carried into the room on work clothing may accumulate on table surfaces and ultimately contaminate food. The modified plant incorporates central air conditioned and pressurized lunchroom facility and a new washroom. Provision is made to hang work coats and coveralls outside the lunchroom to prevent the involuntary transportation of dust and contaminants into the areas where food is consumed.

As a result of the work carried out over recent years, and the measures incorporated in this most recent design to reduce levels of airborne dust, the new plant has the capability of meeting all generally accepted environmental standards.

9.3 NOISE

Certain mineral processing functions typically generate noise levels above that at which hearing protection is required when employees are subjected to prolonged exposures during their normal working shift. Mine safety regulations require that hearing protection be worn by those employees who are exposed to time weighted averages in excess of 85 decibels. Whilst preventive action can be taken to minimize the adverse effects of noise, the wearing of protective devices is frequently uncomfortable and inconvenient. The adverse effect on the overall working environment inevitably results in reduced morale and job interest.

Noise levels will be reduced in the modified mill by the general reduction in the number of small high speed motors, belt-driven sheaves in the flotation area and the increased use of sound proofing materials.

9.4 FRESH WATER CONSUMPTION

The existing water license limits the annual consumption of fresh water at the minesite to 2.0 billion imperial gallons. In order to ensure compliance with this limit, and provide sufficient water to achieve the reduced flotation densities, an internal water recirculation system has been incorporated into the plant design. Thus water overflowing the lead and zinc clarifiers will be collected and used in the grinding and flotation processes. It is estimated that about 2,000 imperial gallons per minute may be recirculated within the modified mill, which would reduce the total fresh water requirement.

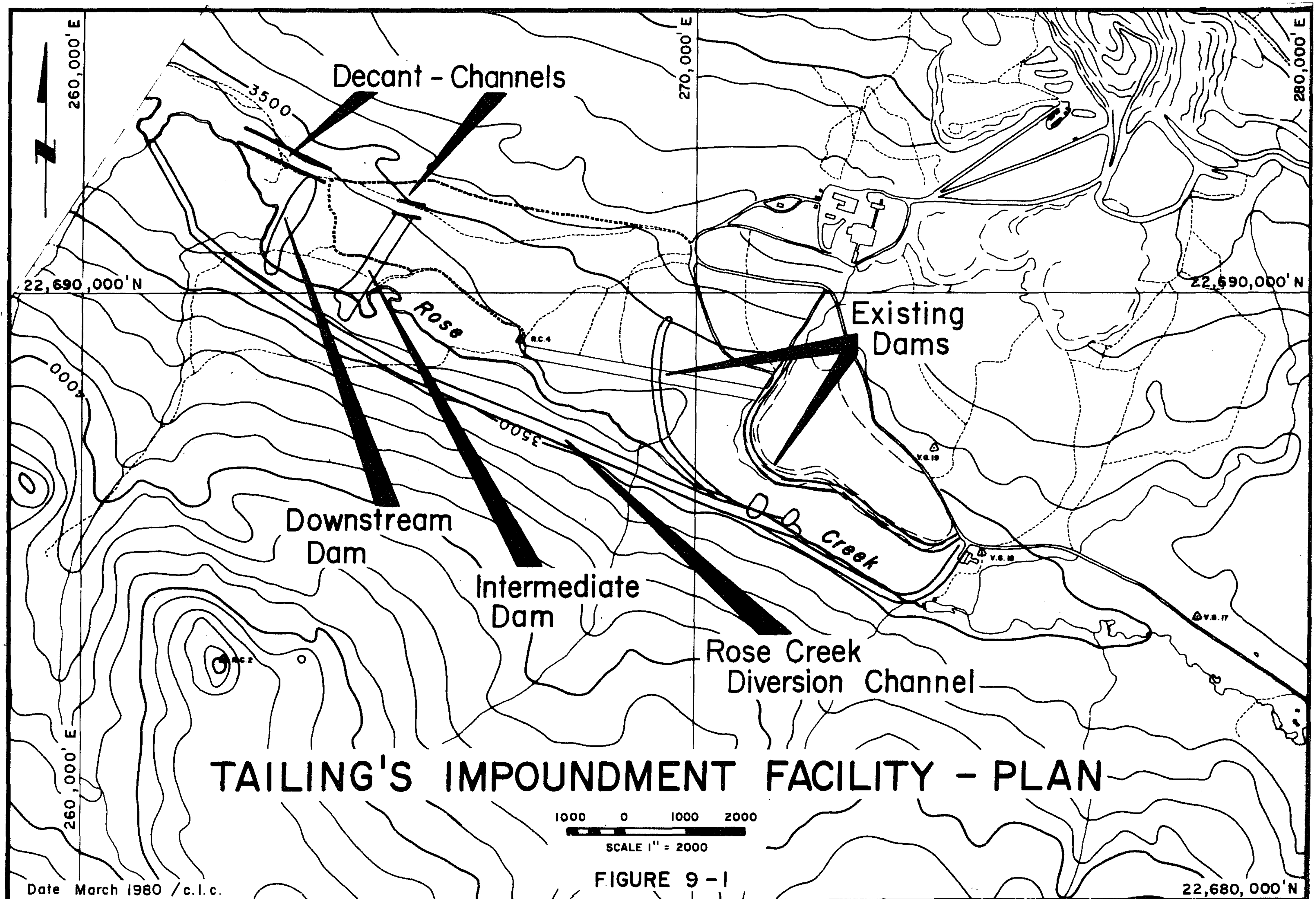
This reduction in fresh water consumption should be considered the first step in the long term management of fresh water usage. Engineering studies, in conjunction with laboratory testwork in progress at the minesite, are proceeding to develop methods by which further reductions in water consumption may be realized to the mutual benefit of the process and the environment.

9.5 TAILINGS DISPOSAL

It will be necessary to construct new tailings impoundment facilities to accommodate the increased volumes of tailings produced when milling ore from the Vangorda Plateau, in addition to feed from the Faro pit.

The finer grind will exert constraints on the overall design of the facility. The physical size of the ponds will be increased not only to accommodate larger volumes of tailings, but also to provide increased retention times for effective settling of the finer suspended particulates. The proposed design reflects the lower ultimate deposition densities of the finer tailings caused by higher levels of fluid entrapment. Finally, since the angle of repose of the tailings will be lower, the expanded pond design provides for increased volumes of material to be stored beneath the water level.

The new tailings disposal facilities shown in Figure 9-1 will provide sufficient storage volume for an additional fifteen years production, during which time the two new dam structures will be raised in 1986 and again in 1990. The existing Rose Creek diversion ditch will be extended along the south side of the valley for approximately 3.2 kilometres. The impoundment area will be contained by two new earth-fill dams. The area up-stream of the intermediate dam will provide storage for the solid tailings. The volume contained between the intermediate dam and the downstream dam will provide sufficient retention time for supernatant clarification to ensure that the final effluent will conform to all pertinent regulatory requirements.



TAILING'S IMPOUNDMENT FACILITY - PLAN

FIGURE 9-1

9.6 THE BIOPHYSICAL ENVIRONMENT

The elevation at the Anvil minesite is 1,070 metres and the climate is characterized by long cold winters and short cool summers. The mine is situated on the south flank of the Anvil Range, an area of rounded mountains cored by granite which reach a maximum elevation of 1,980 metres. Hillsides support an open forest of small white and black spruce above which alpine tundra predominates. Major wildlife species in the area are moose, Dall sheep, woodland caribou, grizzly bear and black bear. Streams support indigenous populations of grayling and provide spawning beds for chinook salmon.

Environmental baseline studies have been designed and are being expanded to assist in the complete understanding of the undisturbed natural environment. The studies are logically divided into specific areas of investigation.

9.61 HYDROLOGY:

Limited stream flow data is available for the Vangorda and Rose Creeks. Continuous recording stations will be constructed to accurately monitor these volumes to provide data for future engineering purposes.

9.62 WATER QUALITY:

Most existing water quality data is concerned with flows in the vicinity of the mine. A regional program of water quality sampling was started in 1979 and includes sites on all watersheds in the Anvil Range, which could be affected by mining developments in the foreseeable future. Sampling has been carried out at approximately 25 sites, twice during 1979 and will be carried out four times per year henceforth. Stream sediments were also sampled during 1979 and will continue to be sampled every year to monitor possible accumulations of heavy metals.

A benthic sampling program will be instituted in co-operation with the Environmental Protection Service in the three major affected drainages in the Anvil Range. Benthic sampling provides a very sensitive method of detecting deteriorations in aquatic environment.

9.63 VEGETATION:

Vegetation mapping has been conducted in a general way (1:50,000 scale) over most of the affected areas of the Anvil

Range to provide an adequate level of detail for most purposes. More intensive studies of vegetation will be carried out to determine species composition and structure of the plant communities and growth rates of tree species for potential use in revegetation. Work commenced in this regard during 1979.

9.64 WILDLIFE:

Low key studies will commence, to monitor fish and ungulate populations in those areas that could be affected by future mining development. Of particular concern is the potential adverse affect, mining activities in Vangorda Plateau may impose upon the resident herd of sheep that have traditionally migrated over this land when travelling between Mount Mye and Sheep Mountain, their respective summer and winter habitats.

All long term production plans will endeavour to minimize detrimental effects upon the indigenous species in this area.

10.
FINANCIAL ANALYSIS

FIGURE 10-1
CYPRUS ANVIL MINING CORPORATION
FINANCIAL EVALUATION OF VANGORDA PLATEAU DEVELOPMENT
Cdn. $ (Millions)

	CASH 1	4B-V	
	Net Cash Flow available for distribution	Net Cash Flow available for distribution	Difference to Case 1
1980	30.538	11.686	(18.852)
81	21.258	8.063	(13.195)
82	18.837	37.794	18.957
83	77.366	76.833	(.533)
84	41.374	38.208	(3.166)
85	39.751	64.206	24.455
86	42.283	46.447	4.164
87	49.023	59.761	10.738
88	40.253	68.625	28.372
89	17.341	72.823	55.482
90		45.527	45.527
91		54.991	54.991
92		60.625	60.625
93		33.927	33.927
94		29.807	29.807
95		24.234	24.234
96		24.551	24.551
97		30.647	30.647
TOTAL	378.024	788.755	410.731
DCF/ROl			36.6%

FIGURE 10-2

CYPRUS ANVIL MINING CORPORATION

VANGORDA PLATEAU DEVELOPMENT

	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
GROSS SALES $(000)																		
Anvil Alone	251,197	239,257	212,404	257,974	278,455	225,628	222,902	227,692	215,027	50,450	-	-	-	-	-	-	-	-
4B-V	251,147	239,759	225,888	293,210	304,708	269,491	271,671	266,968	272,270	293,850	261,816	256,162	263,395	196,908	141,439	128,781	127,924	103,440
NET PROFIT AFTER TAX $(000)																		
Anvil Alone	39,009	29,057	18,034	43,777	46,351	27,503	24,739	27,236	21,533	21,533	1,269	-	-	-	-	-	-	-
4B-V	48,204	38,474	28,300	60,195	68,692	48,102	49,906	47,986	48,655	52,170	32,726	36,694	37,972	24,778	20,207	14,303	13,468	10,827
NET CASH FLOW $(000)																		
Anvil Alone	30,538	21,258	18,837	77,376	41,374	39,751	42,283	49,023	40,253	17,341	-	-	-	-	-	-	-	-
4B-V	11,686	8,063	37,794	76,833	38,208	64,206	46,447	59,761	68,625	72,813	45,527	54,992	60,625	33,927	29,807	24,234	24,551	30,647
UNIT CASH COSTS ¢/LB.																		
F.O.B. TIDEWATER (Adjusted for Ag & Au Credits)																		
Anvil Alone	4.71	12.14	16.26	6.65	5.45	6.38	7.47	6.83	7.92	13.85	-	-	-	-	-	-	-	-
4B-V	4.71	12.10	15.37	6.10	3.89	3.65	3.06	1.36	1.25	2.87	6.24	6.13	5.62	8.15	7.75	8.29	8.74	5.56
PAYABLE METAL PRODUCTION SALES																		
Anvil Alone																		
Zn (000) Lbs.	239,693	259,348	225,667	240,458	240,458	210,154	210,154	225,300	220,665	57,025	-	-	-	-	-	-	-	-
Pb (000) Lbs.	157,155	178,268	149,928	202,222	208,211	166,291	166,291	166,291	160,601	36,209	-	-	-	-	-	-	-	-
Ag (000) Ozs.	2,546	1,904	1,536	2,772	3,156	2,223	2,090	1,945	1,563	266	-	-	-	-	-	-	-	-
Au Ozs.	3,044	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
4B-V																		
Zn (000) Lbs.	239,693	260,060	241,457	270,971	270,971	239,086	234,921	232,471	239,115	280,390	256,050	254,394	260,637	195,501	128,049	119,318	119,054	96,095
Pb (000) Lbs.	157,155	178,566	154,572	212,095	218,333	185,426	190,526	188,259	186,250	187,602	170,787	169,380	173,655	124,057	89,776	80,086	79,231	64,228
Ag (000) Ozs.	2,546	1,907	1,700	3,030	3,459	2,862	2,948	2,835	3,067	3,169	2,591	2,406	2,485	1,916	1,636	1,459	1,444	1,170
Au Ozs.	3,044	-	-	-	-	18,685	19,539	19,160	14,303	8,756	7,699	7,830	8,099	8,276	7,741	6,906	6,832	5,447

FIGURE 10-3

VANCORDA PLATEAU DEVELOPMENT
NET CASH FLOW SENSITIVITIES

ALL PRICES AT SAME % PRICE SENSITIVITY
(CDN. $ MILLIONS)

	- 20		- 15		- 10		- 5		EQUILIBRIUM		+ 5		+10		+ 15		+ 20	
	CASE 1 NCF.	4B-V NCF.	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V
1980	(71.9)	(27274)	7346	(16980)	14111	(6652)	20858	2699	30538	11686	40934	20683	51373	28674	61814	36413	72285	45116
81	1891	(24095)	8064	(15707)	13503	(7398)	18864	860	21258	8063	22864	14742	24376	21434	25800	26956	27109	31405
82	(3718)	5921	4063	14299	9225	22559	14070	30680	18837	37794	23527	44442	28141	50894	32668	56322	37118	61640
83	44238	42032	54546	53500	62346	62461	69885	68814	77366	76833	84782	79851	92169	84975	99509	89350	106795	93690
84	27784	22757	24964	25481	30051	30560	35758	33918	41374	38208	46881	43689	52299	48089	57617	55195	62832	62173
85	18507	38262	25513	46997	30547	52211	35209	58405	39751	64206	44187	68893	48505	73603	52717	78149	56809	82562
86	21944	38072	26926	43820	32116	43122	37235	49330	42283	46447	47259	53368	52163	60008	56999	66546	61767	72979
87	28042	42684	33161	39388	38512	50713	43802	49011	49023	59761	54175	65747	59265	71632	64289	77405	69243	83079
88	21091	40148	25939	51954	30781	51926	35552	62255	40253	68625	44874	74906	49428	81089	53913	87163	58329	93132
89	14530	48401	15756	54404	16353	58870	16880	65888	17341	72823	17738	79641	18073	86369	18336	92976	18536	99484
1990		25995		26143		34808		40237		45527		50694		55741		60655		65448
91		32173		37311		43256		49175		54991		60719		65788		70520		75151
92		35207		42638		48975		55196		60625		65043		70699		76859		82947
93		18356		22698		26959		30419		33927		38451		42164		45449		48643
94		17950		21327		24246		26667		29807		32813		35762		38656		41496
95		12711		15948		18128		21226		24234		27212		30159		33080		35977
96		12932		15891		18463		21520		24551		27566		30557		33519		36463
97		22294		23961		26257		28475		30647		32798		34904		36980		39019
TOT.	173590	404527	226278	503073	277545	599461	328113	694774	378024	788755	427220	881260	475792	972540	523663	1062193	570823	1150406
15%	72472	91973	102016	134370	129795	175124	157205	214133	184654	252021	211855	288453	238779	324299	265373	358994	291639	393295
12%	85011	121878	117635	170182	148527	216714	179007	261508	209428	305140	239541	347289	269332	388802	298747	429117	327784	468925
$/SHARE																		
(15%)	9.44	11.98	13.29	17.50	16.91	22.81	20.48	27.90	24.05	32.83	27.60	37.58	31.11	42.25	34.57	46.77	37.99	51.24
DIFF		27.0%		31.5%		35.0%		36.0%		36.5%		36.0%		36.0%		35.0%		35.0%
(12%)	11.07	15.88	15.33	22.17	19.35	28.23	23.32	34.07	27.28	39.75	31.21	45.24	35.09	50.65	38.92	55.90	42.70	61.09
DIFF		43.5%		44.5%		46.0%		46.0%		45.5%		45.0%		44.5%		43.5%		43.0%
DCFRO%		20.4%		24.1%		28.6%		32.9%		36.6%		39.2%		41.4%		42.4%		43.7%

FIGURE 10-4

VANGORDA PLATEAU DEVELOPMENT

NET CASH FLOW SENSITIVITIES

Ag. Au. AT DOUBLE Pb. & Zn. % Price SENSITIVITY

(CDN. $ MILLIONS)

	CASE 1 -20/40	4B-V -20/40	CASE 1 -15/30	4B-V -15/30	CASE 1 -10/20	4V-B -10/20	CASE 1 -5/10	4B-V -5/10
1980	(13.902)	(41.049)	(.903)	(27.489)	9.525	(13.658)	18.565	(.332)
81	(5.037)	(31.085)	4.923	(20.668)	11.524	(10.526)	17.873	(.691)
82	(9.463)	(.616)	(.313)	9.517	7.340	19.455	13.188	29.172
83	33.105	29.990	46.345	44.573	57.917	58.994	67.681	66.729
84	30.707	10.392	25.839	24.082	26.937	26.746	34.239	32.873
85	13.675	30.958	20.608	37.495	28.464	50.119	34.200	56.569
86	17.279	34.513	23.339	39.238	29.688	39.962	36.040	48.883
87	24.093	30.108	29.984	42.162	36.340	45.845	42.736	46.980
88	17.301	38.867	23.103	39.609	28.919	54.236	34.634	60.305
89	13.875	42.315	15.402	50.825	16.259	51.950	16.844	64.080
1990		19.277		25.663		31.861		38.797
91		28.569		32.939		40.428		47.734
92		32.660		37.638		45.767		53.660
93		14.891		19.294		24.755		29.701
94		13.570		18.124		22.566		25.652
95		8.802		13.013		16.837		20.323
96		9.006		13.240		16.577		20.604
97		19.370		22.512		24.907		27.807
TOTAL	121.633	290.539	188.326	421.767	252.912	546.820	315.999	668.846
15%	40.707	42.325	79.935	99.934	115.915	153.387	150.362	203.705
12%	50.480	65.334	93.408	130.863	133.198	191.811	171.453	249.501
$/SHARE								
(15%)	5.30	5.51	10.40	13.02	15.10	19.98	19.59	26.54
Diff.		4.0%		25.0%		32.0%		35.5%
(12%)	6.58	8.50	12.17	17.05	17.35	24.99	22.34	32.50
Diff.		29%		40.0%		44.0%		45.5%
DCFRO%		15.5%		20.6%		25.9%		31.6%

	EQUILIBRIUM CASE 1	EQUILIBRIUM 4B-V	CASE 1 +5/10	4B-V +5/10	CASE 1 +10/20	4B-V +10/20	CASE 1 +15/30	4B-V +15/30	CASE 1 +20/40	4B-V +20/40
1980	30.538	11.686	44.437	23.531	58.379	33.866	72.322	45.153	86.292	59.127
81	21.258	8.063	22.559	15.910	23.699	23.291	24.664	28.638	25.438	30.393
82	18.837	37.794	24.381	45.646	29.800	52.850	35.116	59.180	40.307	65.364
83	77.366	76.833	86.956	81.164	96.496	87.727	105.968	93.457	115.352	100.854
84	41.374	38.208	48.343	44.985	55.161	51.757	61.819	60.581	68.316	68.274
85	39.751	64.206	45.117	70.413	50.299	76.532	55.296	82.415	60.106	88.068
86	42.283	46.447	48.409	55.323	54.434	63.843	60.345	72.189	66.149	80.358
87	49.023	59.761	55.205	67.466	61.285	74.983	67.260	82.317	73.131	89.468
88	40.253	68.625	45.753	76.797	51.160	84.789	56.453	92.605	61.647	100.244
89	17.341	72.823	17.743	81.384	18.061	89.763	18.281	97.955	18.405	105.960
1990		45.527		52.045		58.359		64.463		69.778
91		54.991		62.096		68.062		73.839		80.228
92		60.625		66.092		73.790		81.427		88.472
93		33.927		39.634		43.897		47.975		52,238
94		29.807		33.753		37.614		41.394		45.083
95		24.234		28.099		31.916		35.684		39.404
96		24.551		28.462		32.333		36.166		39.961
97		30.647		33.435		36.157		38.824		41.439
TOTAL	378.024	788.755	438.903	906.233	498.775	1025530	557.524	1133264	615.147	1244734
15%	184.654	252.021	218.624	298.151	252.126	343.012	285.096	386.669	317.529	429.769
12%	209.428	305.140	246.971	358.534	283.975	410.586	320.373	461.288	356.158	511.238
$/SHARE										
(15%)	24.05	32.83	28.48	38.84	32.85	44.69	37.14	50.37	41.37	55.99
Diff.		36.5%		36.5%		36.0%		35.5%		35.5%
(12%)	27.28	39.75	32.17	46.71	37.00	53.49	41.74	60.09	46.40	66.60
Diff.		45.5%		45.2%		44.5%		44.0%		43.5%
DCFRO%		36.6%		39.9%		42.1%		43.6%		45.6%

FIGURE 10-5

VANGORDA PLATEAU DEVELOPMENT SENSITIVITIES

	- 10% FEED GRADE		EQUILIBRIUM		+ 10% FEED GRADE		+ 10 IN OPERATING COSTS		EQUILIBRIUM		+ 10 CAPITAL	
	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V	CASE 1	4B-V
1980	18.127	(.830)	30.538	11.686	44.853	23.888	19.008	.422	30.538	11.686	28.006	7.004
81	16.799	(2.329)	21.258	8.063	23.557	17.319	16.382	(2.921)	21.258	8.063	20.980	4.706
82	12.205	27.531	18.837	37.794	25.304	47.020	11.272	27.001	18.837	37.794	18.870	38.216
83	66.020	64.957	77.366	76.833	88.591	82.717	69.826	70.134	77.366	76.833	77.415	76.073
84	33.132	31.800	41.374	38.208	49.397	45.921	33.884	34.035	41.374	38.208	41.353	36.032
85	33.075	55.852	39.751	64.206	46.089	70.834	34.233	57.582	39.751	64.206	39.693	64.755
86	34.865	47.435	42.283	46.447	49.470	56.680	36.137	50.788	42.283	46.447	42.235	51.156
87	41.608	47.681	49.023	59.761	56.281	68.459	43.162	48.234	49.023	59.761	49.074	56.066
88	33.551	58.323	40.253	68.625	46.725	77.999	34.537	62.334	40.253	68.625	40.301	68.666
89	16.962	62.591	17.341	72.823	17.604	82.807	16.500	65.694	17.341	72.823	17.501	73.038
1990		37.743		45.527		53.033		39.427		45.527		45.055
91		46.502		54.991		63.135		48.603		54.991		54.934
92		52.358		60.625		67.326		54.866		60.625		60.653
93		29.061		33.927		40.429		29.531		33.927		33.987
94		25.005		29.807		34.337		25.951		29.807		29.838
95		19.709		24.234		28.698		20.551		24.234		24.238
96		19.985		24.551		29.067		20.791		24.551		24.624
97		27.687		30.647		33.545		28.298		30.647		30.799
TOTAL	306.334	651.060	378.024	788.755	447.871	923.215	315.021	681.319	378.024	788.755	375.428	779.842
15%	145.448	196.879	184.654	252.021	223.220	304.415	149.557	206.898	184.654	252.021	182.309	244.961
12%	165.914	241.545	209.428	305.140	252.145	365.896	170.617	253.635	209.428	305.140	207.028	297.711
$/SHARE												
(15%)	18.95	25.65	24.05	32.83	29.08	39.66	19.48	26.95	24.05	32.83	23.75	31.91
DIFF		35.5%		36.5%		36.5%		38.5%		36.5%		34.5%
(12%)	21.61	31.47	27.28	39.75	32.85	47.67	22.23	33.04	27.28	39.75	26.97	38.78
DIFF		45.5%		45.5%		45.0%		48.5%		45.5%		44.0%
DCFROR%		31.0%		36.6%		40.4%		32.7%		36.6%		33.2%

FIGURE 10-6

CYPRUS ANVIL MINING CORPORATION

FINANCIAL EVALUATION OF VANGORDA PLATEAU DEVELOPMENT

CANADIAN $000

CASE 1

Year	Total Prod'n Costs	Sales Less Op. Cash Costs	Capital	Pre-Pdn Stripping	N.C.F. Avail. P.A.	Cumulative
1980	54,935	76,953	19,265	-	30,538	
1981	56,360	55,683	13,475	-	21,258	51,796
1982	58,267	38,190	2,100	-	18,837	70,633
1983	49,597	90,114	2,000	-	77,366	147,999
1984	52,291	96,805	2,000	-	41,374	189,373
1985	36,576	77,125	2,000	-	39,751	229,124
1986	37,885	73,043	2,000	-	42,283	271,407
1987	32,862	78,392	1,000	-	49,023	320,430
1988	29,175	71,998	-	-	40,253	360,683
1989	7,338	12,768	-	-	17,341	378,024
1990	-	-	-	-	-	-
1991	-	-	-	-	-	-
1992	-	-	-	-	-	-
1993	-	-	-	-	-	-
1994	-	-	-	-	-	-
1995	-	-	-	-	-	-
TOTAL	415,286	671,071	43,840	-	378,024	
N.P.V. @ 15%					184,654	

DETAILS OF ADDITIONAL CAPITAL

	1980	1981	1982	1983-86	1987	1988-89	Total
Prior Year	1488	-	-	-	-	-	1488
Mine	4546	3875*	500	500	300	-	11221
Mechanical	1243	400	400	400	300	-	3943
Environment	715	250	250	200	200	-	2215
Mill	508	500	500	500	200	-	3708
Townsite	6885	1250	250	200	-	-	9185
Other	1880	200	200	200	-	-	3080
Sub-total	17265	6475	2100	2000	1000	-	34,840
Additional Spending For Case 1 Comparisions Assuming VanGorda Plateau is not Developed.							
Townsite	-	1000	-	-	-	-	1000
Tailings Dam	2000	6000	-	-	-	-	8000
Sub-total	2000	7000	-	-	-	-	9000
TOTAL	19,265	13,475	2100	2000	1000	-	43,840

* Note Case 1 - Faro only 5 Trucks 1981
" 1A- Faro only 5 Trucks 1981
" 1B- Faro only 5 Trucks 1981
" 2A- Faro only 5 Trucks 1981
" 2B- Faro only 5 Trucks 1981
3B-G/V - Faro only 1 Truck 1981
4B-G/V - Faro only 5 Trucks 1981

Reclamation cost under enviorment operating costs.

Tailings lifts under mill cost after 1980.

FIGURE 10-7

CYPRUS ANVIL MINING CORPORATION

FINANCIAL EVALUATION OF VANGORDA PLATEAU DEVELOPMENT

CANADIAN $000

CASE 4B-V

Year	Total Prod'n Costs	Sales Less Op. Cash Costs	Capital Addit. to 1.	Pre-Pdn Stripping	N.C.F. Avail. P.A.	Cumulative
1980	54,935	76,953	27,860	-	11,686	-
1981	56,360	55,891	26,381	-	8,063	19,749
1982	61,560	50,122	-	-	37,794	57,543
1983	55,032	111,370	18,150	-	76,833	134,376
1984	53,182	123,671	38,250	4,000	38,208	172,584
1985	46,464	103,536	6,120	2,520	64,206	236,790
1986	46,779	106,160	1,000	7,111	46,447	283,237
1987	37,225	112,211	2,000	12,050	59,761	342,998
1988	40,005	115,698	3,000	7,500	68,625	411,673
1989	45,815	122,041	3,000	-	72,823	484,446
1990	48,475	97,045	3,000	-	45,527	529,973
1991	44,645	96,618	3,000	-	54,991	584,964
1992	43,405	102,088	2,000	-	60,625	645,589
1993	48,514	99,343	1,000	-	33,927	679,516
1994	54,042	88,201	1,000	-	29,807	709,323
1995	28,512	65,644	1,000	-	24,234	733,557
1996	-	-	-	-	24,551	758,108
1997	-	-	-	-	30,647	788,755
TOTAL	764,950	1,526,592	136,761	33,181	788,755	
N.P.V. @ 15%					252,021	

DETAILS OF ADDITIONAL CAPITAL:

Mill	33,381
Power Plant	12,000
Tailings Dam	12,000
Less Alternate Tailings Dam in Case 1	(8,000)
Sub-total	49,381
Townsite	7,000
Less Townsite Capital Case 1	(1,000)
Equipment - Mine (Excl. Anvil Overhauls)	15,705
Mine Shop Etc.	6,515
Transportation System	34,300
Additional Continuing Capital	23,000
Other	1,861
Sub-total	87,381
TOTAL	136,762

CASE PRINT OUTS

FIGURE 10-8 CASE I CASH FLOW REPORT

		1980	1981	1982	1983	1984	1985	1986	1987	1988	1989
	39										
1. SALES LESS CONVERSION	130	165.709	148.774	130.749	176.362	185.965	146.718	143.995	145.021	134.255	31.152
2. + LOAN DRAWDOWN	15	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
3. - LOAN REPAYMENTS	135	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
4. + BUYERS RECD. ADJUSTMEN	102	7.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	11.830
5. + OPENING CASH ON HAND	103	10.457	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000
6. CASH AVAILABLE	117	183.166	150.774	132.749	178.362	187.965	148.718	145.995	147.021	136.255	44.982
USE OF CASH											
7. PRODUCTION COSTS	104	74.376	78.712	78.179	71.868	74.780	56.012	57.321	53.061	48.901	12.164
8. ORE BODY ROYALTY	119	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
9. GENERAL AND ADMN. COSTS	105	14.380	14.380	14.380	14.380	14.380	13.580	13.480	13.380	13.080	5.195
9.5 RECLAMATION	400	0.000	0.000	0.000	0.000	0.000	0.000	.150	.188	.275	1.025
10. INTEREST - TERM DEBT	31	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
11. - HOUSING	222	-1.059	.475	.475	0.000	0.000	0.000	0.000	0.000	0.000	0.000
12. PRINCIPAL HOUSING MORTGA	108	.164	.164	4.606	0.000	0.000	0.000	0.000	0.000	0.000	0.000
13. CAPITAL EXPEND. - ANVIL	391	19.265	13.475	2.100	2.000	2.000	2.000	2.000	1.000	0.000	0.000
14. V+G-RELATED - OTHER	389	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
15. EXPLORATION +FEASIBILITY	207	6.050	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
16. PRE-PRODUCTION STRIPPING	225	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
17. TORRENS SHARES	110	.454	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
18. YUKON ROYALTY	221	5.840	5.145	3.669	2.245	7.020	7.686	5.350	4.958	5.483	5.174
19. FED. INCOME TAX	99	25.032	15.165	8.503	8.503	46.411	27.688	23.411	23.411	26.262	2.178
20. DIVIDENDS	111	6.126	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
21. PREPAID EXPENSES	113	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.905
22. FLOAT ADJUSTMENT	257	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
23. TOTAL	114	150.628	127.516	111.913	98.996	144.591	106.967	101.712	95.998	94.002	27.641
24. CLOSING CASH ON HAND	115	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	0.000
25. CASH AVAIL. FOR DISTRIB.	116	30.538	21.258	18.837	77.366	41.374	39.751	42.283	49.023	40.253	17.341
DISCOUNTED NCF											
26. 10 PERCENT	49	27.761	17.568	14.152	52.842	25.690	22.438	21.698	22.869	17.071	6.686
27. 12 PERCENT	42	27.266	16.946	13.408	49.167	23.477	20.139	19.127	19.799	14.516	5.583
28. 15 PERCENT	38	26.554	16.074	12.386	44.234	20.570	17.186	15.896	16.026	11.442	4.286
CUMULATIVE NPV											
29. 10 PERCENT	321	27.761	45.330	59.482	112.324	138.014	160.453	182.150	205.020	222.091	228.777
30. 12 PERCENT	324	27.266	44.212	57.620	106.787	130.264	150.403	169.530	189.329	203.645	209.428
31. 15 PERCENT	320	26.554	42.628	55.014	99.248	119.818	137.004	152.899	168.925	180.367	184.654
32. PRICES - LEAD (US C/LB)	418	43.000	42.500	42.500	42.500	42.500	42.500	42.500	42.500	42.500	42.500
33. ZINC (US C/LB)	419	39.226	44.000	50.000	50.000	50.000	50.000	50.000	50.000	50.000	50.000
34. SILVER (US $/OZ	420	24.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000
35. GOLD (US $/OZ	421	456.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000
36. CND. DOLLAR PER U.S. DOL	422	1.121	1.050	1.025	1.025	1.025	1.025	1.025	1.025	1.025	1.025

ANVIL ALONE CASE 1 POST CMC MTG 11/04/80

FIGURE 10-8 CASE I CASH FLOW REPORT (CON'T)

		39	1990	1991	1992	ACCUM
1.	SALES LESS CONVERSION	130	0.000	0.000	0.000	1408.698
2.	+ LOAN DRAWDOWN	15	0.000	0.000	0.000	0.000
3.	- LOAN REPAYMENTS	135	0.000	0.000	0.000	0.000
4.	+ BUYERS RECD. ADJUSTMEN	102	0.000	0.000	0.000	18.830
5.	+ OPENING CASH ON HAND	103	0.000	0.000	0.000	28.457
6.	CASH AVAILABLE	117	0.000	0.000	0.000	1455.985
USE OF CASH						
7.	PRODUCTION COSTS	104	0.000	0.000	0.000	605.375
8.	ORE BODY ROYALTY	119	0.000	0.000	0.000	0.000
9.	GENERAL AND ADMN. COSTS	105	0.000	0.000	0.000	130.615
9.5	RECLAMATION	400	0.000	0.000	0.000	
10.	INTEREST - TERM DEBT	31	0.000	0.000	0.000	0.000
11.	- HOUSING	222	0.000	0.000	0.000	-.109
12.	PRINCIPAL HOUSING MORTGA	108	0.000	0.000	0.000	4.934
13.	CAPITAL EXPEND. - ANVIL	391	0.000	0.000	0.000	43.840
14.	V+G-RELATED - OTHER	389	0.000	0.000	0.000	0.000
15.	EXPLORATION +FEASIBILITY	207	0.000	0.000	0.000	6.050
16.	PRE-PRODUCTION STRIPPING	225	0.000	0.000	0.000	0.000
17.	TORRENS SHARES	110	0.000	0.000	0.000	.454
18.	YUKON ROYALTY	221	0.000	0.000	0.000	52.570
19.	FED. INCOME TAX	99	0.000	0.000	0.000	206.565
20.	DIVIDENDS	111	0.000	0.000	0.000	6.126
21.	PREPAID EXPENSES	113	0.000	0.000	0.000	1.905
22.	FLOAT ADJUSTMENT	257	0.000	0.000	0.000	0.000
23.	TOTAL	114	0.000	0.000	0.000	1059.963
24.	CLOSING CASH ON HAND	115	0.000	0.000	0.000	18.000
25.	CASH AVAIL. FOR DISTRIB.	116	0.000	0.000	0.000	378.023
DISCOUNTED NCF						
26.	10 PERCENT	49	0.000	0.000	0.000	228.777
27.	12 PERCENT	42	0.000	0.000	0.000	209.428
28.	15 PERCENT	38	0.000	0.000	0.000	184.654
CUMULATIVE NPV						
29.	10 PERCENT	321	228.777	228.777	228.777	
30.	12 PERCENT	324	209.428	209.428	209.428	
31.	15 PERCENT	320	184.654	184.654	184.654	
32.	PRICES - LEAD (US C/LB)	418	0.000	0.000	0.000	32.731
33.	ZINC (US C/LB)	419	0.000	0.000	0.000	37.171
34.	SILVER (US $/OZ	420	0.000	0.000	0.000	15.692
35.	GOLD (US $/OZ	421	0.000	0.000	0.000	312.000
36.	CND. DOLLAR PER U.S. DOL	422	0.000	0.000	0.000	.798

FIGURE 10-9 CASE I INCOME STATEMENT (CND$M.)

	39	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989
1. SALES - LEAD	401	75.687	79.436	65.259	87.919	90.512	72.358	72.358	72.358	69.894	15.763
2. - ZINC	402	105.436	119.842	115.659	123.235	123.235	107.704	107.704	115.469	113.091	29.225
3. - SILVER	403	68.517	39.979	31.486	56.821	64.708	45.566	42.840	39.865	32.042	5.462
4. - GOLD	404	1.557	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
5. PRIOR YEAR ADJUSTMENT	405	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
6. TOTAL SALES	406	251.197	239.257	212.404	267.974	278.455	225.628	222.902	227.692	215.027	50.450
7. COSTS - PRODUCTION	104	74.376	78.712	78.179	71.868	74.780	56.012	57.321	53.061	48.901	12.164
8. - ORE BODY ROYALTY	119	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
9. - CONVERSION	407	85.489	90.483	81.654	91.612	92.490	78.911	78.907	82.671	80.772	19.298
10. - GENERAL AND ADMI	105	14.380	14.380	14.380	14.380	14.380	13.580	13.480	13.380	13.080	5.195
10.5 - RECLAMATION	400	0.000	0.000	0.000	0.000	0.000	0.000	.150	.188	.275	1.025
11. - TOTAL	408	174.245	183.575	174.213	177.860	181.650	148.503	149.858	149.300	143.028	37.682
12. SALES LESS OP. CASH COST	30	76.953	55.683	38.190	90.114	96.805	77.125	73.043	78.392	71.998	12.768
13. DEFERRED DEVELOP. COSTS	303	-8.293	-13.668	-11.907	-7.301	-5.533	7.774	10.064	10.867	14.072	4.031
14. + DEPRECIATION CHARGES	301	10.531	13.014	12.439	12.758	13.276	12.526	11.031	10.566	10.566	2.831
15. + AMORT. CHARGES	302	6.174	7.052	7.052	7.052	7.052	7.052	7.052	7.052	7.052	1.889
16. + INVENTORY CHANGE	88	3.940	.225	-.274	.512	-.036	.070	.009	.183	.145	1.519
17. NONCASH PROD. CHARGES	304	12.353	6.624	7.310	13.022	14.758	27.422	28.156	28.668	31.835	10.271
18. OPERATING PROFIT	306	64.600	49.059	30.880	77.092	82.047	49.703	44.887	49.724	40.163	2.497
19. - INTEREST	272	-1.059	.475	.475	0.000	0.000	0.000	0.000	0.000	0.000	0.000
20. - EXPLORATION W/O	208	1.750	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
21. + EXCHANGE GAIN/LOSS(-)	200	-.981	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
22. NET PROFIT BEFORE TAX	307	62.928	48.584	30.405	77.092	82.047	49.703	44.887	49.724	40.163	2.497
23. YUKON MINING ROYALTY	220	5.145	3.669	2.245	7.020	7.686	5.350	4.958	5.483	4.791	.383
24. FEDERAL INCOME TAX	96	13.777	12.288	8.503	27.457	30.720	24.425	23.411	25.696	23.977	2.178
25. DEFERRED TAXES	289	4.996	3.570	1.623	-1.162	-2.710	-7.574	-8.221	-8.691	-10.138	-1.332
26. TOTAL TAXES	308	23.919	19.527	12.371	33.315	35.696	22.201	20.148	22.488	18.630	1.228
27. NET PROFIT AFTER TAX	311	39.009	29.057	18.034	43.777	46.351	27.503	24.739	27.236	21.533	1.269
28. FEED GRADE (PERCENT)	409	7.24	8.18	7.25	8.20	8.30	7.00	7.00	7.30	7.10	5.90
29. CONCENTRATE PROD. - LEAD	410	114.713	130.296	112.474	149.046	153.734	120.921	120.921	120.921	116.234	25.117
30. (000 DMT) - ZINC	411	213.968	260.170	225.961	239.908	239.908	207.489	207.489	223.698	218.295	56.457
31. - BULK	412	34.724	35.000	35.000	35.000	35.000	35.000	35.000	35.000	35.000	0.000
32. - TOTA	413	363.406	425.466	373.434	423.954	428.642	363.410	363.410	379.620	369.529	81.575
33. SALES (000 DMT) - LEAD	414	118.028	130.296	112.474	149.046	153.734	120.921	120.921	120.921	116.234	28.342
34. - ZINC	415	236.434	260.170	225.961	239.908	239.908	207.489	207.489	223.698	218.295	61.005
35. - BULK	416	39.271	35.000	35.000	35.000	35.000	35.000	35.000	35.000	35.885	0.000
36. - TOTA	417	393.734	425.466	373.434	423.954	428.642	363.410	363.410	379.620	370.414	89.348

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ANVIL ALONE CASE 1 POST CMC MTG 11/04/80
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FIGURE 10-9 CASE I INCOME STATEMENT (CON'T)

	39	1990	1991	1992	ACCUM
1. SALES - LEAD	401	0.000	0.000	0.000	701.543
2. - ZINC	402	0.000	0.000	0.000	1060.600
3. - SILVER	403	0.000	0.000	0.000	427.286
4. - GOLD	404	0.000	0.000	0.000	1.557
5. PRIOR YEAR ADJUSTMENT	405	0.000	0.000	0.000	0.000
6. TOTAL SALES	406	0.000	0.000	0.000	2190.986
7. COSTS - PRODUCTION	104	0.000	0.000	0.000	605.375
8. - ORE BODY ROYALTY	119	0.000	0.000	0.000	0.000
9. - CONVERSION	407	0.000	0.000	0.000	782.288
10. - GENERAL AND ADMI	105	0.000	0.000	0.000	130.615
10.5 - RECLAMATION	400	0.000	0.000	0.000	
11. - TOTAL	408	0.000	0.000	0.000	1519.915
12. SALES LESS OP. CASH COST	30	0.000	0.000	0.000	671.071
13. DEFERRED DEVELOP. COSTS	303	0.000	0.000	0.000	.107
14. + DEPRECIATION CHARGES	301	0.000	0.000	0.000	109.537
15. + AMORT. CHARGES	302	0.000	0.000	0.000	64.480
16. + INVENTORY CHANGE	88	0.000	0.000	0.000	6.294
17. NONCASH PROD. CHARGES	304	0.000	0.000	0.000	180.418
18. OPERATING PROFIT	306	0.000	0.000	0.000	490.653
19. - INTEREST	272	0.000	0.000	0.000	-.109
20. - EXPLORATION W/O	208	0.000	0.000	0.000	1.750
21. + EXCHANGE GAIN/LOSS(-)	200	0.000	0.000	0.000	-.981
22. NET PROFIT BEFORE TAX	307	0.000	0.000	0.000	488.031
23. YUKON MINING ROYALTY	220	0.000	0.000	0.000	46.730
24. FEDERAL INCOME TAX	96	0.000	0.000	0.000	192.433
25. DEFERRED TAXES	289	0.000	0.000	0.000	-29.639
26. TOTAL TAXES	308	0.000	0.000	0.000	209.524
27. NET PROFIT AFTER TAX	311	0.000	0.000	0.000	278.507
28. FEED GRADE (PERCENT)	409	0.00	0.00	0.00	5.65
29. CONCENTRATE PROD. - LEAD	410	0.000	0.000	0.000	1164.379
30. (000 DMT) - ZINC	411	0.000	0.000	0.000	2093.343
31. - BULK	412	0.000	0.000	0.000	314.724
32. - TOTA	413	0.000	0.000	0.000	3572.446
33. SALES (000 DMT) - LEAD	414	0.000	0.000	0.000	1170.919
34. - ZINC	415	0.000	0.000	0.000	2120.357
35. - BULK	416	0.000	0.000	0.000	320.156
36. - TOTA	417	0.000	0.000	0.000	3611.432

ANVIL ALONE 78/79 METAL CASE 1 POST CMC MTG **FIGURE 10-10 CASE I PRODUCTION STATISTICS** 11/04/80

		1980	1981	1982	1983	1984	1985	1986	1987	1988	1989
FEED GRADE (PERCENT)											
LEAD	1	2.83	3.00	2.60	3.40	3.50	2.80	2.80	2.80	2.70	2.00
ZINC	2	4.41	5.18	4.65	4.80	4.80	4.20	4.20	4.50	4.40	3.90
TOTAL	7	7.24	8.18	7.25	8.20	8.30	7.00	7.00	7.30	7.10	5.90
CUBIC MTRS MOVED (000S)											
WASTE AND LOW GRADE	8	9102	10872	10022	8268	7560	2678	1828	1530	340	0
ORE	9	882	846	818	818	818	818	818	818	818	220
TOTAL	10	9984	11718	10840	9086	8378	3496	2646	2348	1158	220
TONNES MILLED (000S)	11	3322	3404	3404	3404	3404	3404	3404	3404	3404	912
MILL FEED (000 LBS)											
LEAD	12	207260	225134	195116	255152	262656	210125	210125	210125	202620	40212
ZINC	13	322974	388356	348957	360214	360214	315187	315187	337701	330196	78413
METAL RECOVERED (000 LBS)											
LEAD - SELECT	18	147262	175223	145404	200438	206742	162616	162616	162616	156312	33778
- BULK	20	13779	13889	13889	13889	13889	13889	13889	13889	13889	0
- TOTAL	23	161041	189112	159293	214327	220631	176505	176505	176505	170201	33778
ZINC - SELECT	25	238357	289079	249325	266566	266566	230545	230545	248555	242552	62731
- BULK	26	21282	21605	21605	21605	21605	21605	21605	21605	21605	0
- TOTAL	29	259639	310685	270930	288171	288171	252150	252150	270161	264157	62731
GOLD IN PB SEL (000 OZ)	42	4	3	2	3	3	2	2	2	2	1
SILVER IN PB SEL (000 OZ)	30	2404	1915	1530	2802	3198	2225	2092	1947	1558	261
SILVER IN BULK (000 OZ)	32	229	175	175	175	175	175	175	175	175	0
RECOVERIES (PERCENT)											
LEAD - SELECT	66	71.05	77.83	74.52	78.56	78.71	77.39	77.39	77.39	77.15	84.00
- BULK	68	6.65	6.17	7.12	5.44	5.29	6.61	6.61	6.61	6.85	.00
- TOTAL	71	77.70	84.00	81.64	84.00	84.00	84.00	84.00	84.00	84.00	84.00
ZINC - SELECT	73	73.80	74.44	71.45	74.00	74.00	73.15	73.15	73.60	73.46	80.00
- BULK	74	6.59	5.56	6.19	6.00	6.00	6.85	6.85	6.40	6.54	.00
- TOTAL	77	80.39	80.00	77.64	80.00	80.00	80.00	80.00	80.00	80.00	80.00
CONCENTRATE GRADES (PERCENT)											
LEAD SELECT - LEAD	114	58.23	61.00	58.64	61.00	61.00	61.00	61.00	61.00	61.00	61.00
-SILVER (OZ/DMT)	116	20.96	14.70	13.60	18.80	20.80	18.40	17.30	16.10	13.40	10.40
-GOLD (OZ/DMT)	118	.03	.02	.02	.02	.02	.02	.02	.02	.02	.02
ZINC SELECT - ZINC	123	50.53	50.40	50.05	50.40	50.40	50.40	50.40	50.40	50.40	50.40
-IRON	125	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00
BULK -LEAD	130	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	0.00
-ZINC	131	27.80	28.00	28.00	28.00	28.00	28.00	28.00	28.00	28.00	0.00
-SILVER (OZ/DMT)	132	6.59	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	0.00
-IRON	133	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	0.00
CONCENTRATE PRODUCED (DMT)											
LEAD	160	114713	130296	112474	149046	153734	120921	120921	120921	116234	25117
ZINC	161	213968	260170	225961	239908	239908	207489	207489	223698	218295	56457
BULK	162	34724	35000	35000	35000	35000	35000	35000	35000	35000	0
TOTAL DMT	165	363406	425466	373434	423954	428642	363410	363410	379620	369529	81575

C.4B-V REV G-V DEV 85 (FINE GRIND 82) 11/04/80

FIGURE 10-11 CASE 4B-V CASH FLOW REPORT

	39	1980	1981	1982	1983	1984	1985	1986	1987	1988	1969
1. SALES LESS CONVERSION	130	165.709	148.995	145.856	202.937	213.595	185.351	187.619	183.899	189.927	204.012
2. + LOAN DRAWDOWN	15	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
3. - LOAN REPAYMENTS	135	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
4. + BUYERS RECD. ADJUSTMEN	102	7.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
5. + OPENING CASH ON HAND	103	10.457	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000
6. CASH AVAILABLE	117	183.166	150.995	147.856	204.937	215.595	187.351	189.619	185.899	191.927	206.012
USE OF CASH											
7. PRODUCTION COSTS	104	74.376	78.724	81.354	77.187	75.544	67.435	67.780	58.007	60.355	67.791
8. ORE BODY ROYALTY	119	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
9. GENERAL AND ADMN. COSTS	105	14.380	14.380	14.380	14.380	14.380	14.380	13.680	13.680	13.680	13.680
9.5 RECLAMATION	400	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	.500
10. INTEREST - TERM DEBT	31	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
11. - HOUSING	222	-1.059	.475	.475	0.000	0.000	0.000	0.000	0.000	0.000	0.000
12. PRINCIPAL HOUSING MORTGA	108	.164	.164	4.606	0.000	0.000	0.000	0.000	0.000	0.000	0.000
13. CAPITAL EXPEND. - ANVIL	391	17.265	6.475	2.100	3.000	3.000	3.000	3.000	3.000	3.000	3.000
14. V+G-RELATED - OTHER	389	29.860	33.381	0.000	17.150	37.250	5.120	0.000	0.000	0.000	0.000
15. EXPLORATION +FEASIBILITY	207	5.359	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
16. PRE-PRODUCTION STRIPPING	225	0.000	0.000	0.000	0.000	4.000	2.520	7.111	12.050	7.500	0.000
17. TORRENS SHARES	110	.454	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
18. YUKON ROYALTY	221	5.840	6.368	4.182	3.185	10.945	12.867	6.605	6.356	6.992	8.025
19. FED. INCOME TAX	99	16.715	.965	.965	11.202	30.269	15.822	42.997	31.045	29.775	38.193
20. DIVIDENDS	111	6.126	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
21. PREPAID EXPENSES	113	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
22. FLOAT ADJUSTMENT	257	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
23. TOTAL	114	169.480	140.932	108.062	126.104	175.387	121.144	141.172	124.138	121.302	131.189
24. CLOSING CASH ON HAND	115	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000
25. CASH AVAIL. FOR DISTRIB.	116	11.686	8.063	37.794	76.833	38.208	64.206	46.447	59.761	68.625	72.823
DISCOUNTED NCF											
26. 10 PERCENT	49	10.623	6.663	28.395	52.478	23.724	36.243	23.835	27.879	29.104	28.077
27. 12 PERCENT	42	10.434	6.428	26.901	48.829	21.680	32.529	21.010	24.136	24.747	23.447
28. 15 PERCENT	38	10.162	6.097	24.850	43.929	18.996	27.758	17.461	19.536	19.507	18.001
CUMULATIVE NPV											
29. 10 PERCENT	321	10.623	17.287	45.682	98.160	121.885	158.127	181.962	209.841	238.945	267.021
30. 12 PERCENT	324	10.434	16.861	43.763	92.591	114.272	146.801	167.811	191.947	216.694	240.141
31. 15 PERCENT	320	10.162	16.258	41.109	85.038	104.034	131.792	149.254	168.790	188.297	206.298
32. PRICES - LEAD (US C/LB)	418	43.000	42.500	42.500	42.500	42.500	42.500	42.500	42.500	42.500	42.500
33. ZINC (US C/LB)	419	39.226	44.000	50.000	50.000	50.000	50.000	50.000	50.000	50.000	50.000
34. SILVER (US $/OZ	420	24.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000
35. GOLD (US $/OZ	421	456.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000
36. CND. DOLLAR PER U.S. DOL	422	1.121	1.050	1.025	1.025	1.025	1.025	1.025	1.025	1.025	1.025

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FIGURE 10-11 CASE 4B-V CASH FLOW REPORT (CON'T)

	39	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
1. SALES LESS CONVERSION	130	179.967	175.074	180.064	135.170	99.626	90.373	89.705	72.567	0.000	0.000
2. + LOAN DRAWDOWN	15	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
3. - LOAN REPAYMENTS	135	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
4. + BUYERS RECD. ADJUSTMEN	102	0.000	0.000	0.000	0.000	4.000	0.000	0.000	7.830	0.000	0.000
5. + OPENING CASH ON HAND	103	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	0.000	0.000
6. CASH AVAILABLE	117	181.967	177.074	182.064	137.170	105.626	92.373	91.705	82.397	0.000	0.000
USE OF CASH											
7. PRODUCTION COSTS	104	68.717	64.727	63.971	52.217	40.354	35.852	35.808	24.298	0.000	0.000
8. ORE BODY ROYALTY	119	0.000	0.000	.705	2.377	2.351	2.174	2.131	1.898	0.000	0.000
9. GENERAL AND ADMN. COSTS	105	13.680	13.680	13.680	13.680	10.301	10.301	10.301	7.905	0.000	0.000
9.5 RECLAMATION	400	.525	.050	.325	.438	.588	.950	1.488	1.038	0.000	0.000
10. INTEREST - TERM DEBT	31	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
11. - HOUSING	222	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
12. PRINCIPAL HOUSING MORTGA	108	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
13. CAPITAL EXPEND. - ANVIL	391	3.000	3.000	2.000	1.000	1.000	1.000	0.000	0.000	0.000	0.000
14. V+G-RELATED - OTHER	389	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
15. EXPLORATION +FEASIBILITY	207	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
16. PRE-PRODUCTION STRIPPING	225	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
17. TORRENS SHARES	110	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
18. YUKON ROYALTY	221	9.601	6.684	6.816	7.504	4.000	2.289	2.004	3.459	0.000	0.000
19. FED. INCOME TAX	99	38.917	31.942	31.942	24.027	15.225	13.573	13.422	11.248	0.000	0.000
20. DIVIDENDS	111	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
21. PREPAID EXPENSES	113	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.905	0.000	0.000
22. FLOAT ADJUSTMENT	257	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
23. TOTAL	114	134.440	120.083	119.439	101.243	73.819	66.139	65.153	51.750	0.000	0.000
24. CLOSING CASH ON HAND	115	2.000	2.000	2.000	2.000	2.000	2.000	2.000	0.000	0.000	0.000
25. CASH AVAIL. FOR DISTRIB.	116	45.527	54.991	60.625	33.927	29.807	24.234	24.551	30.647	0.000	0.000
DISCOUNTED NCF											
26. 10 PERCENT	49	15.957	17.522	17.561	8.934	7.136	5.274	4.857	5.512	0.000	0.000
27. 12 PERCENT	42	13.088	14.115	13.894	6.942	5.446	3.953	3.576	3.985	0.000	0.000
28. 15 PERCENT	38	9.786	10.278	9.853	4.795	3.663	2.590	2.281	2.476	0.000	0.000
CUMULATIVE NPV											
29. 10 PERCENT	321	282.978	300.500	318.061	326.995	334.130	339.404	344.262	349.774	349.774	349.774
30. 12 PERCENT	324	253.229	267.344	281.238	288.180	293.625	297.579	301.154	305.140	305.140	305.140
31. 15 PERCENT	320	216.084	226.362	236.215	241.010	244.673	247.263	249.544	252.021	252.021	252.021
32. PRICES - LEAD (US C/LB)	418	42.500	42.500	42.500	42.500	42.500	42.500	42.500	42.500	0.000	0.000
33. ZINC (US C/LB)	419	50.000	50.000	50.000	50.000	50.000	50.000	50.000	50.000	0.000	0.000
34. SILVER (US $/OZ	420	20.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000	0.000	0.000
35. GOLD (US $/OZ	421	400.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000	0.000	0.000
36. CND. DOLLAR PER U.S. DOL	422	1.025	1.025	1.025	1.025	1.025	1.025	1.025	1.025	0.000	0.000

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C.4B-V REV G-V DEV 85 (FINE GRIND 82) 11/04/80
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FIGURE 10-11 CASE 4B-V CASH FLOW REPORT (CON'T)

	39	2000	ACCUM
1. SALES LESS CONVERSION	130	0.000	2850.446
2. + LOAN DRAWDOWN	15	0.000	0.000
3. - LOAN REPAYMENTS	135	0.000	0.000
4. + BUYERS RECD. ADJUSTMEN	102	0.000	18.830
5. + OPENING CASH ON HAND	103	0.000	44.457
6. CASH AVAILABLE	117	0.000	2913.733
USE OF CASH			
7. PRODUCTION COSTS	104	0.000	1094.496
8. ORE BODY ROYALTY	119	0.000	11.636
9. GENERAL AND ADMN. COSTS	105	0.000	234.528
9.5 RECLAMATION	400	0.000	
10. INTEREST - TERM DEBT	31	0.000	0.000
11. - HOUSING	222	0.000	-.109
12. PRINCIPAL HOUSING MORTGA	108	0.000	4.934
13. CAPITAL EXPEND. - ANVIL	391	0.000	57.840
14. V+G-RELATED - OTHER	389	0.000	122.761
15. EXPLORATION +FEASIBILITY	207	0.000	5.359
16. PRE-PRODUCTION STRIPPING	225	0.000	33.181
17. TORRENS SHARES	110	0.000	.454
18. YUKON ROYALTY	221	0.000	113.721
19. FED. INCOME TAX	99	0.000	398.243
20. DIVIDENDS	111	0.000	6.126
21. PREPAID EXPENSES	113	0.000	1.905
22. FLOAT ADJUSTMENT	257	0.000	0.000
23. TOTAL	114	0.000	2090.977
24. CLOSING CASH ON HAND	115	0.000	34.000
25. CASH AVAIL. FOR DISTRIB.	116	0.000	788.756
DISCOUNTED NCF			
26. 10 PERCENT	49	0.000	349.774
27. 12 PERCENT	42	0.000	305.140
28. 15 PERCENT	38	0.000	252.021
CUMULATIVE NPV			
29. 10 PERCENT	321	349.774	
30. 12 PERCENT	324	305.140	
31. 15 PERCENT	320	252.021	
32. PRICES - LEAD (US C/LB)	418	0.000	36.452
33. ZINC (US C/LB)	419	0.000	42.058
34. SILVER (US $/OZ	420	0.000	17.333
35. GOLD (US $/OZ	421	0.000	345.524
36. CND. DOLLAR PER U.S. DOL	422	0.000	.884

C.4B-V REV G-V DEV 85 (FINE GRIND 82) 11/04/80

FIGURE 10-12 CASE 4B-V INCOME STATEMENT (CND$M.)

	39	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989
1. SALES - LEAD	401	75.687	79.569	67.287	92.228	94.929	80.621	82.828	81.847	80.993	81.599
2. - ZINC	402	105.436	120.148	123.747	138.873	138.873	122.532	120.397	119.141	122.546	143.700
3. - SILVER	403	68.517	40.042	34.854	62.110	70.906	58.677	60.435	58.123	62.867	64.961
4. - GOLD	404	1.557	0.000	0.000	0.000	0.000	7.661	8.011	7.855	5.864	3.590
5. PRIOR YEAR ADJUSTMENT	405	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
6. TOTAL SALES	406	251.197	239.759	225.888	293.210	304.708	269.491	271.671	266.968	272.270	293.850
7. COSTS - PRODUCTION	104	74.376	78.724	81.354	77.187	75.544	67.435	67.780	58.007	60.355	67.791
8. - ORE BODY ROYALTY	119	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
9. - CONVERSION	407	85.489	90.764	80.031	90.274	91.113	84.140	84.052	83.069	82.343	89.838
10. - GENERAL AND ADMI	105	14.380	14.380	14.380	14.380	14.380	14.380	13.680	13.680	13.680	13.680
10.5 - RECLAMATION	400	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	.500
11. - TOTAL	408	174.245	183.868	175.766	181.841	181.037	165.955	165.511	154.756	156.379	171.809
12. SALES LESS OP. CASH COST	30	76.953	55.891	50.122	111.370	123.671	103.536	106.160	112.211	115.891	122.041
13. DEFERRED DEVELOP. COSTS	303	-8.290	-13.666	-11.905	-7.299	-5.747	.453	.724	8.770	7.590	3.727
14. + DEPRECIATION CHARGES	301	9.084	13.035	12.325	13.918	17.512	17.049	15.253	14.694	15.086	15.537
15. + AMORT. CHARGES	302	2.430	3.008	3.008	3.008	3.351	5.651	6.509	8.085	9.167	9.167
16. + INVENTORY CHANGE	88	4.029	.330	-.277	.376	-.016	.035	.046	.152	-.129	-.095
17. NONCASH PROD. CHARGES	304	7.253	2.707	3.151	10.003	15.101	23.188	22.532	31.700	31.714	28.336
18. OPERATING PROFIT	306	69.700	53.184	46.971	101.367	108.571	80.347	83.628	80.512	84.177	93.705
19. - INTEREST	272	-1.059	.475	.475	0.000	0.000	0.000	0.000	0.000	0.000	0.000
20. - EXPLORATION W/O	208	1.750	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
21. + EXCHANGE GAIN/LOSS(-)	200	-.981	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
22. NET PROFIT BEFORE TAX	307	68.028	52.709	46.496	101.367	108.571	80.347	83.628	80.512	84.177	93.705
23. YUKON MINING ROYALTY	220	6.368	4.182	3.185	10.945	12.867	6.605	6.356	6.992	8.025	9.601
24. FEDERAL INCOME TAX	96	2.583	.965	6.084	20.530	15.822	29.409	30.680	29.775	33.984	40.728
25. DEFERRED TAXES	289	10.873	9.088	8.928	9.697	11.189	-3.769	-3.314	-4.241	-6.486	-8.794
26. TOTAL TAXES	308	19.823	14.235	18.196	41.172	39.879	32.245	33.722	32.526	35.523	41.535
27. NET PROFIT AFTER TAX	311	48.204	38.474	28.300	60.195	68.692	48.102	49.906	47.986	48.655	52.170
28. FEED GRADE (PERCENT)	409	7.24	8.18	7.25	8.20	8.30	7.68	7.69	7.60	7.60	8.31
29. CONCENTRATE PROD. - LEAD	410	114.713	130.296	119.733	151.148	155.593	156.096	160.995	158.818	149.752	142.034
30. (000 DMT) - ZINC	411	213.968	260.440	251.202	270.284	270.284	240.226	236.009	233.525	237.416	275.505
31. - BULK	412	34.724	35.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
32. - TOTA	413	363.406	425.736	370.935	421.432	425.677	396.322	397.004	392.346	387.169	417.539
33. SALES (000 DMT) - LEAD	414	118.028	130.296	119.733	151.148	155.593	156.096	160.995	158.818	149.752	142.034
34. - ZINC	415	236.434	260.440	251.202	270.284	270.284	240.226	236.009	233.529	237.416	275.505
35. - BULK	416	39.271	35.885	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
36. - TOTA	417	393.734	426.621	370.935	421.432	425.877	396.322	397.004	392.346	387.169	417.539

C.4B-V REV G-V DEV 85 (FINE GRIND 82) 11/04/80

FIGURE 10-12 CASE 4B-V INCOME STATEMENT (CON'T) (CND$M.)

	39	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
1. SALES - LEAD	401	74.319	73.709	75.560	54.042	39.108	34.887	34.515	27.969	0.000	0.000
2. - ZINC	402	131.226	129.916	133.577	100.195	65.625	61.151	61.015	49.249	0.000	0.000
3. - SILVER	403	53.115	49.327	50.938	39.278	33.531	29.912	29.593	23.989	0.000	0.000
4. - GOLD	404	3.157	3.210	3.321	3.393	3.174	2.831	2.801	2.233	0.000	0.000
5. PRIOR YEAR ADJUSTMENT	405	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
6. TOTAL SALES	406	261.816	256.162	263.395	196.908	141.439	128.781	127.924	103.440	0.000	0.000
7. COSTS - PRODUCTION	104	68.717	64.727	63.971	52.217	40.354	35.852	35.808	24.298	0.000	0.000
8. - ORE BODY ROYALTY	119	0.000	0.000	.705	2.377	2.351	2.174	2.131	1.898	0.000	0.000
9. - CONVERSION	407	81.849	81.088	83.331	61.738	41.813	38.408	38.219	30.873	0.000	0.000
10. - GENERAL AND ADMI	105	13.680	13.680	13.680	13.680	10.301	10.301	10.301	7.905	0.000	0.000
10.5 - RECLAMATION	400	.525	.050	.325	.438	.588	.950	1.488	1.038	0.000	0.000
11. - TOTAL	408	164.771	159.544	162.012	130.450	95.407	87.685	87.946	66.012	0.000	0.000
12. SALES LESS OP. CASH COST	30	97.045	96.618	101.383	66.458	46.032	41.096	39.977	37.429	0.000	0.000
13. DEFERRED DEVELOP. COSTS	303	3.998	4.858	5.897	2.410	-1.649	2.419	2.419	5.497	0.000	0.000
14. + DEPRECIATION CHARGES	301	16.067	16.712	17.260	13.547	9.084	9.445	9.445	7.252	0.000	0.000
15. + AMORT. CHARGES	302	9.167	9.167	9.167	7.041	4.583	4.583	4.583	3.519	0.000	0.000
16. + INVENTORY CHANGE	88	-.224	.060	.061	-.246	-.145	-.058	-.016	2.409	0.000	0.000
17. NONCASH PROD. CHARGES	304	29.009	30.798	32.385	22.752	11.874	16.390	16.432	18.677	0.000	0.000
18. OPERATING PROFIT	306	68.036	65.820	68.998	43.706	34.158	24.706	23.545	18.751	0.000	0.000
19. - INTEREST	272	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
20. - EXPLORATION W/O	208	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
21. + EXCHANGE GAIN/LOSS(-)	200	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
22. NET PROFIT BEFORE TAX	307	68.036	65.820	68.998	43.706	34.158	24.706	23.545	18.751	0.000	0.000
23. YUKON MINING ROYALTY	220	6.684	6.816	7.504	4.000	2.289	2.004	1.979	1.480	0.000	0.000
24. FEDERAL INCOME TAX	96	32.173	31.942	33.807	22.162	15.225	13.573	13.422	11.248	0.000	0.000
25. DEFERRED TAXES	289	-9.096	-9.632	-10.285	-7.234	-3.563	-5.173	-5.324	-4.804	0.000	0.000
26. TOTAL TAXES	308	29.761	29.126	31.026	18.928	13.951	10.403	10.078	7.924	0.000	0.000
27. NET PROFIT AFTER TAX	311	38.276	36.694	37.972	24.778	20.207	14.303	13.468	10.827	0.000	0.000
28. FEED GRADE (PERCENT)	409	7.55	7.49	7.70	7.49	7.99	7.32	7.28	7.25	0.00	0.00
29. CONCENTRATE PROD. - LEAD	410	128.653	127.792	131.194	95.943	71.442	63.731	63.050	47.887	0.000	0.000
30. (000 DMT) - ZINC	411	251.889	249.332	256.263	191.292	124.241	115.770	115.513	88.689	0.000	0.000
31. - BULK	412	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
32. - TOTA	413	380.542	377.124	387.457	287.235	195.683	179.501	178.564	136.576	0.000	0.000
33. SALES (000 DMT) - LEAD	414	128.653	127.792	131.194	95.943	71.442	63.731	63.050	51.112	0.000	0.000
34. - ZINC	415	251.889	249.332	256.263	191.292	124.241	115.770	115.513	93.237	0.000	0.000
35. - BULK	416	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
36. - TOTA	417	380.542	377.124	387.457	287.235	195.683	179.501	178.564	144.349	0.000	0.000

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C.4B-V REV G-V DEV 85 (FINE GRIND 82) 11/04/80
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FIGURE 10-12 CASE 4B-V INCOME STATEMENT (CON'T)

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		39	2000	ACCUM
1.	SALES - LEAD	401	0.000	1231.697
2.	- ZINC	402	0.000	1987.345
3.	- SILVER	403	0.000	891.177
4.	- GOLD	404	0.000	58.659
5.	PRIOR YEAR ADJUSTMENT	405	0.000	0.000
6.	TOTAL SALES	406	0.000	4168.877
7.	COSTS - PRODUCTION	104	0.000	1094.496
8.	- ORE BODY ROYALTY	119	0.000	11.636
9.	- CONVERSION	407	0.000	1318.431
10.	- GENERAL AND ADMI	105	0.000	234.528
10.5	- RECLAMATION	400	0.000	
11.	- TOTAL	408	0.000	2664.993
12.	SALES LESS OP. CASH COST	30	0.000	1503.884
13.	DEFERRED DEVELOP. COSTS	303	0.000	.206
14.	+ DEPRECIATION CHARGES	301	0.000	242.305
15.	+ AMORT. CHARGES	302	0.000	105.196
16.	+ INVENTORY CHANGE	88	0.000	6.294
17.	NONCASH PROD. CHARGES	304	0.000	354.000
18.	OPERATING PROFIT	306	0.000	1149.883
19.	- INTEREST	272	0.000	-.109
20.	- EXPLORATION W/O	208	0.000	1.750
21.	+ EXCHANGE GAIN/LOSS(-)	200	0.000	-.981
22.	NET PROFIT BEFORE TAX	307	0.000	1147.261
23.	YUKON MINING ROYALTY	220	0.000	107.881
24.	FEDERAL INCOME TAX	96	0.000	384.111
25.	DEFERRED TAXES	289	0.000	-31.940
26.	TOTAL TAXES	308	0.000	460.051
27.	NET PROFIT AFTER TAX	311	0.000	687.210
28.	FEED GRADE (PERCENT)	409	0.00	6.58
29.	CONCENTRATE PROD. - LEAD	410	0.000	2168.872
30.	(000 DMT) - ZINC	411	0.000	3881.854
31.	- BULK	412	0.000	69.724
32.	- TOTA	413	0.000	6120.450
33.	SALES (000 DMT) - LEAD	414	0.000	2175.412
34.	- ZINC	415	0.000	3908.868
35.	- BULK	416	0.000	75.156
36.	- TOTA	417	0.000	6159.436

CONSOL C.4B-V REV V-G DEV 85 (FINE GRIND 82) **FIGURE 10-13 CASE 4B-V PRODUCTION STATISTICS** 11/04/80

		1980	1981	1982	1983	1984	1985	1986	1987	1988	1989
FEED GRADE (PERCENT)											
LEAD	1	2.83	3.00	2.60	3.40	3.50	3.13	3.22	3.18	3.14	3.16
ZINC	2	4.41	5.18	4.65	4.80	4.80	4.56	4.48	4.43	4.46	5.15
TOTAL	7	7.24	8.18	7.25	8.20	8.30	7.68	7.69	7.60	7.60	8.31
CUBIC MTRS MOVED (000S)											
WASTE AND LOW GRADE	8	9102	10872	10022	8268	7560	4400	4300	1333	2400	4800
ORE	9	882	846	818	818	818	814	814	814	857	924
TOTAL	10	9984	11718	10840	9086	8378	5214	5114	2147	3257	5724
TONNES MILLED (000S)	11	3322	3404	3404	3404	3404	3402	3402	3402	3402	3402
MILL FEED (000 LBS)											
LEAD	12	207260	225134	195116	255152	262656	234377	241127	238127	235261	237002
ZINC	13	322974	388731	348957	360214	360214	342002	335627	331877	334474	386253
METAL RECOVERED (000 LBS)											
LEAD - SELECT	18	147262	175223	162707	223258	229824	195376	200776	198376	196084	197476
- BULK	20	13779	13889	0	0	0	0	0	0	0	0
- TOTAL	23	161041	189112	162707	223258	229824	195376	200776	198376	196084	197476
ZINC - SELECT	25	238357	289380	285761	318789	318789	281454	276546	273658	281312	329871
- BULK	26	21282	21605	0	0	0	0	0	0	0	0
- TOTAL	29	259639	310985	285761	318789	318789	281454	276546	273658	281312	329871
GOLD IN PB SEL (000 OZ)	42	4	3	2	3	3	21	22	21	16	10
SILVER IN PB SEL (000 OZ)	30	2404	1915	1820	3189	3641	3018	3109	2994	3228	3336
SILVER IN BULK (000 OZ)	32	229	175	0	0	0	0	0	0	0	0
RECOVERIES (PERCENT)											
LEAD - SELECT	66	71.05	77.83	83.39	87.50	87.50	83.36	83.27	83.31	83.35	83.32
- BULK	68	6.65	6.17	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
- TOTAL	71	77.70	84.00	83.39	87.50	87.50	83.36	83.27	83.31	83.35	83.32
ZINC - SELECT	73	73.80	74.44	81.89	88.50	88.50	82.30	82.40	82.46	84.11	85.40
- BULK	74	6.59	5.56	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
- TOTAL	77	80.39	80.00	81.89	88.50	88.50	82.30	82.40	82.46	84.11	85.40
CONCENTRATE GRADES (PERCENT)											
LEAD SELECT - LEAD	114	58.23	61.00	61.64	67.00	67.00	56.77	56.57	56.66	59.39	63.07
-SILVER (OZ/DMT)	116	20.96	14.70	15.20	21.10	23.40	19.34	19.31	18.85	21.56	23.48
-GOLD (OZ/DMT)	118	.03	.02	.02	.02	.02	.13	.14	.13	.11	.07
ZINC SELECT - ZINC	123	50.53	50.40	51.60	53.50	53.50	53.14	53.15	53.15	53.75	54.31
-IRON	125	3.67	3.67	3.67	3.67	3.67	7.33	7.33	7.33	11.00	11.00
BULK -LEAD	130	18.00	18.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
-ZINC	131	27.80	28.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
-SILVER (OZ/DMT)	132	6.59	5.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
-IRON	133	4.67	4.67	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00
CONCENTRATE PRODUCED (DMT)											
LEAD	160	114713	130296	119733	151148	155593	156096	160995	158818	149752	142034
ZINC	161	213968	260440	251202	270284	270284	240226	236009	233529	237416	275505
BULK	162	34724	35000	0	0	0	0	0	0	0	0
TOTAL DMT	165	363406	425736	370935	421432	425877	396322	397004	392346	387169	417539

CONSOL C.4B-V REV V-G DEV 85 (FINE GRIND 82) **FIGURE 10-13 CASE 4B-V PRODUCTION STATISTICS (CON'T)** 11/04/80

		1990	1991	1992	1993	1994	1995	1996	1997
FEED GRADE (PERCENT)									
LEAD	1	2.87	2.85	2.92	2.77	3.15	2.81	2.78	2.75
ZINC	2	4.69	4.65	4.78	4.72	4.84	4.51	4.50	4.50
TOTAL	7	7.55	7.49	7.70	7.49	7.99	7.32	7.28	7.25
CUBIC MTRS MOVED (000S)									
WASTE AND LOW GRADE	8	4700	4383	4000	4000	4000	2500	2500	581
ORE	9	924	924	924	735	515	515	515	396
TOTAL	10	5624	5307	4924	4735	4515	3015	3015	977
TONNES MILLED (000S)	11	3402	3402	3402	2613	1701	1701	1701	1306
MILL FEED (000 LBS)									
LEAD	12	214876	213376	219001	159463	118126	105376	104251	79172
ZINC	13	351752	348377	358502	271914	181501	169126	168751	129564
METAL RECOVERED (000 LBS)									
LEAD - SELECT	18	179776	178295	182795	130586	94501	84301	83401	63343
- TOTAL	23	179776	178295	182795	130586	94501	84301	83401	63343
ZINC - SELECT	25	301236	298228	306632	230002	150646	140375	140063	107538
- TOTAL	29	301236	298228	306632	230002	150646	140375	140063	107538
GOLD IN PB SEL (000 OZ)	42	9	9	9	9	8	7	7	5
SILVER IN PB SEL (000 OZ)	30	2727	2534	2616	2017	1722	1536	1520	1154
RECOVERIES (PERCENT)									
LEAD - SELECT	66	83.66	83.56	83.47	81.89	80.00	80.00	80.00	80.00
- TOTAL	71	83.66	83.56	83.47	81.89	80.00	80.00	80.00	80.00
ZINC - SELECT	73	85.64	85.60	85.53	84.59	83.00	83.00	83.00	83.00
- TOTAL	77	85.64	85.60	85.53	84.59	83.00	83.00	83.00	83.00
CONCENTRATE GRADES (PERCENT)									
LEAD SELECT - LEAD	114	63.38	63.29	63.20	61.74	60.00	60.00	60.00	60.00
-SILVER (OZ/DMT)	116	21.20	19.83	19.94	21.02	24.10	24.10	24.10	24.10
-GOLD (OZ/DMT)	118	.07	.07	.07	.09	.11	.11	.11	.11
ZINC SELECT - ZINC	123	54.25	54.26	54.28	54.54	55.00	55.00	55.00	55.00
-IRON	125	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00
-IRON	133	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00
CONCENTRATE PRODUCED (DMT)									
LEAD	160	128653	127792	131194	95943	71442	63731	63050	47887
ZINC	161	251889	249332	256263	191292	124241	115770	115513	88689
TOTAL DMT	165	380542	377124	387457	287235	195683	179501	178564	136576

C.4B-V REV G-V DEV 85 (FINE GRIND 82) WITH INTEREST 14/04/80

FIGURE 10-14 CORPORATE CASH FLOW 1980-89
(CASE 4B-V INCLUDING INTEREST, EXPLORATION & DIVIDENDS)

	39	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989
1. SALES LESS CONVERSION	130	165.709	148.995	145.856	202.937	213.595	185.351	187.619	183.899	189.927	204.012
2. + LOAN DRAWDOWN	15	0.000	1.705	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
3. - LOAN REPAYMENTS	135	0.000	0.000	1.705	0.000	0.000	0.000	0.000	0.000	0.000	0.000
4. + BUYERS RECD. ADJUSTMEN	102	7.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
5. + OPENING CASH ON HAND	103	10.457	2.000	0.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000
6. CASH AVAILABLE	117	183.166	152.700	144.152	204.937	215.595	187.351	189.619	185.899	191.927	206.012
USE OF CASH											
7. PRODUCTION COSTS	104	74.376	78.724	81.354	77.187	75.544	67.435	67.780	58.007	60.355	67.791
8. ORE BODY ROYALTY	119	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
9. GENERAL AND ADMN. COSTS	105	14.380	14.380	14.380	14.380	14.380	14.380	13.680	13.680	13.680	13.680
9.5 RECLAMATION	400	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	.500
10. INTEREST - TERM DEBT	31	0.000	.107	.085	0.000	0.000	0.000	0.000	0.000	0.000	0.000
11. - HOUSING	222	-1.059	.475	.475	0.000	0.000	0.000	0.000	0.000	0.000	0.000
12. PRINCIPAL HOUSING MORTGA	108	.164	.164	4.606	0.000	0.000	0.000	0.000	0.000	0.000	0.000
13. CAPITAL EXPEND. - ANVIL	391	17.265	6.475	2.100	3.000	3.000	3.000	3.000	3.000	3.000	3.000
14. V+G-RELATED - OTHER	389	29.860	33.381	0.000	17.150	37.250	5.120	0.000	0.000	0.000	0.000
15. EXPLORATION +FEASIBILITY	207	5.359	6.500	6.500	6.500	6.500	6.500	6.500	6.500	6.500	6.500
16. PRE-PRODUCTION STRIPPING	225	0.000	0.000	0.000	0.000	4.000	2.520	7.111	12.050	7.500	0.000
17. TORRENS SHARES	110	.454	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
18. YUKON ROYALTY	221	5.840	6.368	3.646	2.834	10.664	12.239	7.270	6.547	6.822	7.844
19. FED. INCOME TAX	99	16.715	0.000	0.000	8.019	26.407	13.553	27.531	28.565	26.958	34.202
20. DIVIDENDS	111	6.126	6.126	6.126	6.126	6.126	6.126	6.126	6.126	6.126	6.126
21. PREPAID EXPENSES	113	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
22. FLOAT ADJUSTMENT	257	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
23. TOTAL	114	169.480	152.700	119.273	135.196	183.872	130.873	138.997	134.476	130.981	139.643
24. CLOSING CASH ON HAND	115	2.000	0.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000
25. CASH AVAIL. FOR DISTRIB.	116	11.686	0.000	22.878	67.741	29.724	54.478	48.622	49.423	58.946	64.370
DISCOUNTED NCF											
26. 10 PERCENT	49	10.623	0.000	17.189	46.268	18.456	30.751	24.951	23.056	24.999	24.817
27. 12 PERCENT	42	10.434	0.000	16.284	43.051	16.866	27.600	21.994	19.961	21.256	20.725
28. 15 PERCENT	38	10.161	0.000	15.043	38.731	14.778	23.552	18.279	16.156	16.756	15.911
CUMULATIVE NPV											
29. 10 PERCENT	321	10.623	10.623	27.812	74.080	92.537	123.288	148.238	171.295	196.293	221.111
30. 12 PERCENT	324	10.434	10.434	26.718	69.769	86.635	114.235	136.229	156.190	177.446	198.172
31. 15 PERCENT	320	10.161	10.161	25.204	63.936	78.714	102.266	120.545	136.701	153.457	169.368
32. PRICES - LEAD (US C/LB)	418	43.000	42.500	42.500	42.500	42.500	42.500	42.500	42.500	42.500	42.500
33. ZINC (US C/LB)	419	39.226	44.000	50.000	50.000	50.000	50.000	50.000	50.000	50.000	50.000
34. SILVER (US $/OZ)	420	24.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000	20.000
35. GOLD (US $/OZ)	421	456.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000	400.000
36. CND. DOLLAR PER U.S. DOL	422	1.121	1.050	1.025	1.025	1.025	1.025	1.025	1.025	1.025	1.025

C.4B-V REV G-V DEV 85 (FINE GRIND 82) WITH INTEREST 14/04/80

FIGURE 10-15 CORPORATE BALANCE SHEET

	250	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989
ASSETS												
1. CASH AND SHORT TERM DEP.	316	10.457	2.000	0.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000	2.000
2. ACCOUNTS RECEIVABLE	333	19.811	11.830	11.830	11.830	11.830	11.830	11.830	11.830	11.830	11.830	11.830
3. PREPAID EXPENSES	335	.330	.330	.330	.330	.330	.330	.330	.330	.330	.330	.330
4. INVENTORIES-SUPP./SP. PA	339	6.112	6.112	6.112	6.112	6.112	6.112	6.112	6.112	6.112	6.112	6.112
5. -PRODUCT	344	6.294	2.265	1.935	2.212	1.836	1.852	1.817	1.770	1.619	1.748	1.842
6. TOTAL CURRENT ASSETS	346	43.004	22.537	20.207	22.484	22.108	22.124	22.089	22.042	21.891	22.020	22.114
7. INVESTMENTS	348	35.539	35.539	35.539	35.539	35.539	13.539	13.539	13.539	13.539	13.539	13.539
8. DEPRECIABLE ASSETS	349	101.592	148.717	188.573	184.673	204.823	245.073	253.193	256.193	259.193	262.193	265.193
9. PROVISION FOR DEPRECIATI	351	33.780	42.864	55.899	68.223	82.141	99.654	116.703	131.955	146.649	161.735	177.271
10. SUB-TOTAL	352	67.812	105.853	132.674	116.450	122.682	145.419	136.490	124.238	112.544	100.458	87.922
11. DEFERRED CHARGES	353	41.228	53.127	73.293	97.698	111.497	149.744	158.311	171.198	180.978	187.388	190.161
12. PROVISION FOR AMORTIZATI	354	14.202	16.632	19.640	22.648	25.655	29.007	34.658	41.167	49.252	58.419	67.586
13. SUB-TOTAL	355	27.026	36.495	53.653	75.050	85.841	120.737	123.653	130.031	131.726	128.969	122.575
14. SURPLUS OPERATING CASH	376	0.000	11.686	11.686	34.564	102.305	132.029	186.507	235.129	284.551	343.497	407.867
15. TOTAL ASSETS	356	173.381	212.110	253.759	284.087	368.475	433.849	482.278	524.978	564.251	608.483	654.016
LIABILITIES AND SHAREHOLDERS EQUITY												
16. SHORT TERM BORROWING	357	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
17. CUR. PORTION LONG TERM D	358	.638	.184	.184	-.000	-.000	-.000	-.000	-.000	-.000	-.000	-.000
18. ACCOUNTS PAYABLE + ACCRU	359	8.347	8.347	8.347	8.347	8.347	8.347	8.347	8.347	8.347	8.347	8.347
19. INCOME TAX	360	14.132	0.000	0.000	4.010	12.854	0.000	6.989	4.012	1.222	4.213	6.747
20. ROYALTY	361	5.840	6.368	3.646	2.834	10.664	12.239	7.270	6.547	6.822	7.844	9.400
21. TOTAL CURRENT LIABILITIE	362	28.957	14.899	12.177	15.190	31.866	20.586	22.606	18.906	16.391	20.404	24.493
22. TORRENS DEBENTURES	372	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
23. HOUSING MORTGAGE	373	4.750	4.586	4.422	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
24. BANK DEBT	374	0.000	0.000	1.705	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
25. L.T. DEBT (EXCL. CUR. PO	375	4.750	4.586	6.127	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
26. DEFERRED TAX	364	27.549	38.422	48.438	57.395	69.762	82.224	86.325	88.141	86.902	83.414	77.616
27. TOTAL LIABILITIES	365	61.256	57.907	66.742	72.585	101.628	102.810	108.931	107.047	103.293	103.818	102.109
28. CAPITAL STOCK	366	3.913	3.913	3.913	3.913	3.913	3.913	3.913	3.913	3.913	3.913	3.913
29. CONTRIBUTED SURPLUS	367	4.067	4.067	4.067	4.067	4.067	4.067	4.067	4.067	4.067	4.067	4.067
30. RETAINED EARNINGS	369	104.145	146.223	179.037	203.522	258.868	323.059	365.367	409.952	452.978	496.685	543.927
31. TOTAL SHAREHOLDERS EQUIT	370	112.125	154.203	187.017	211.502	266.848	331.039	373.347	417.932	460.958	504.665	551.907
32. TOTAL LIABILITIES AND SHAREHOLDERS EQUITY	371	173.381	212.110	253.759	284.087	368.475	433.849	482.278	524.978	564.252	608.483	654.016