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FINAL FEASIBILITY STUDY

DEVELOPMENT OF THE
FARO No. 1 ORE BODY

THE ANVIL MINING CORPORATION, LIMITED

JOB No. 3746-3

FEBRUARY 1967

THE RALPH M. PARSONS CONSTRUCTION CO. OF CANADA LTD.

ENGINEERS-CONSTRUCTORS



The Ralph M. Parsons Construction Co. of Canada, Ltd.

Engineers • Constructors

617 WEST SEVENTH STREET, LOS ANGELES 17, CALIFORNIA

February 17, 1967

Anvil Mining Corporation, Limited
523 West Sixth Street
Los Angeles, California

ATTENTION of Mr. Kenneth Lieber, President

SUBJECT Anvil Mining Corporation, Limited
 Final Feasibility Report
 Our Job No. 3746-3

Gentlemen:

We are pleased to submit herewith fifteen (15) copies, numbered from one to fifteen consecutively, of Volumes No. 1 and No. 2 of our Final Feasibility Report for developing the Faro No. 1 ore body of Anvil Mining Corporation, Limited. The report presents estimates of capital and operating costs for the mining and beneficiation of your lead - zinc ore and storage and shiploading, at tidewater, of lead and zinc concentrates.

The capital and operating cost estimates have been based on the treatment of approximately 2,000,000 tons per year of ore having a grade of about 12.2 percent combined lead and zinc during the first five year period and 3,000,000 tons per year at 9.3 percent combined metal content for the remaining life of the property. Revenues have been developed for the sale of 128,400 and 240,200 tons per year of lead and zinc concentrates, respectively, to custom smelters. A profitability analysis, based on discounted cash flow rates of return, is presented for the Base Case (Case A-4).

The calculated rate of return of 19.1 percent per year and the cash payout period of 4.0 years are based upon an immediate capital expenditure of \$56.1 million. A deferred investment of \$3.1 million will, however, reduce the cash flow in the fifth year of operation. These results would appear to indicate the present refinements in capital and operating costs have substantiated, at equivalent metal prices and capital investments, the economic conclusions drawn from the Preliminary Feasibility Report of April and July, 1966.

The Ralph M. Parsons Construction Co. of Canada, Ltd.

Anvil Mining Corporation

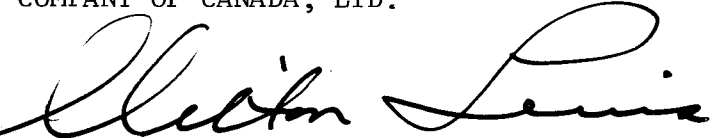
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February 17, 1967

Nothing which has developed in the ten months since the completion of the Preliminary Feasibility Report would impair the optimistic conclusions concerning the complete technical feasibility of producing selective concentrates of lead and zinc of 69 to 70 and 53 to 54 percent grade, respectively, at economical recovery values. On the basis of the cost and revenue input to the present Base Case (Case A-4), it may qualifiedly be concluded that lead and zinc concentrates can be produced by Anvil from their Yukon operation at a cost that is competitive and at discounted cash flow rates of return, on the total investment considered, that appear reasonable. Of course, the final decision for bringing the property into production must rest with Anvil's Management.

Very truly yours,

THE RALPH M. PARSONS CONSTRUCTION
COMPANY OF CANADA, LTD.

By 

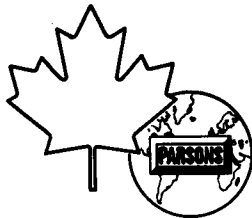
Milton Lewis
Executive Vice President

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Enclosures (15)
As Above



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SECTION 1

SUMMARY

The ensuing report concerns the technical and economic feasibility of developing the Faro No. 1 ore body of the Anvil Mining Corporation, Limited. This lead-zinc property is located in the Whitehorse Mining District of the Yukon Territory, Canada. The principal elements of the study are: (1) capital and operating cost estimates for the mine plant, beneficiation facility and tidewater operation, and (2) profitability analysis of the project economics by discounted cash flow rates of return.

The mining engineering study encompassed the initial and ultimate development of the ore body; proposed mining method; pre-production stripping quantities; waste hauling and dump requirements; calculation of ore reserve tonnages; determination of average assay and mining grades by ore benches; estimate of capital costs; and development of mining costs. This comprehensive study was based upon drill hole data, core logs, assay determinations and geological information provided by Anvil.

The Preliminary Feasibility Report, dated April 26, 1966, indicated the desirability of developing in broader scope and to a greater degree a process facility to produce selective concentrates of lead and zinc for custom smelting. The basic process design criteria used in establishing the quantitative flowsheets were obtained essentially from The Galigher Company report dated **October 3, 1966 and grinding test data from** Mine and Smelter Supply Company, Allis-Chalmers Manufacturing Company and Hazen Research, Inc. **Approximately eighty-five (85) drawings were prepared** in the prosecution of the preliminary engineering-design effort and served as the basis for the capital cost estimate. Likewise, operating cost estimates were developed for the process facility at the minesite and the concentrate storage and handling facility at Haines, Alaska.

The project economics have been evaluated by means of discounted cash flow rates of return. The Base Case (Case A-4) indicates a rate of return of 19.1 percent on the basis of an initial capital expenditure of about \$56.1 million. This rate of return equates to a cash payment period of 4.0 years. The significant economic data for the Base Case are enumerated in Table 1-1.



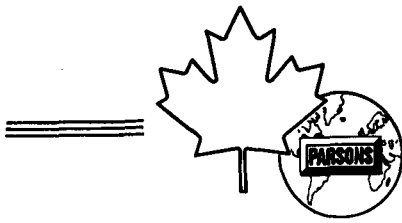
TABLE 1-1

SUMMARY OF ECONOMIC DATA

CASE A-4

	Selective Concentrates	
	69.1% Lead	53.8% Zinc
	<u>Custom Smelting</u>	
<u>Investments in Millions, \$</u>		
Fixed Assets (1)		52.500
Working Capital		3.580
Deferred		<u>3.140</u>
Total		59.220
<u>Metal Prices, ¢/Pound</u>	10.0	10.5
<u>Return on Investment, %</u>		19.1
<u>Cash Payout Period, Years</u>		4.0
<u>Value/Ton of Product, \$</u>		83.02
<u>Costs/Ton of Product, \$</u>		
Cash Costs to Haines (2)		35.67
Cash Costs Haines to Smelter (3)		4.39
Cash Costs (Others)		4.85
Depreciation (4)		<u>()</u>
Total		44.91
<u>Profit/Ton of Product, \$</u>		38.11

- (1) Includes \$2.223 million for preproduction stripping and Anvil's portion of the permanent bridge and access road costs at \$0.463 million, both of which are subject to an early write-off.
- (2) Includes truck transportation to Haines at \$16.31 per dry short ton of concentrates.
- (3) Includes ocean freight costs to and stevedoring charges in Japan at \$3.51 and \$0.70 per dry short ton of concentrates, respectively. Also includes insurance costs for lead and zinc concentrates at \$0.32 and \$0.11 per dry short ton.
- (4) No depreciation is taken until the fourth year of operation.



SECTION 2

INTRODUCTION

On April 26, 1966, The Ralph M. Parsons Construction Company of Canada, Ltd., submitted a report to Anvil Mining Corporation Limited titled, "Preliminary Feasibility Study - Development of the Faro No. 1 Ore Body." Broadly, this report examined the technical and economic feasibility of opening up, developing, mining, concentrating, smelting and refining the ores from the Faro No. 1 Ore Body. In this study, six variations in ore treatment were considered.

These were:

Case A - Production of selective concentrates (55 percent lead and 52 percent zinc) for toll smelting.

Case B - Production of Special High Grade slab zinc by electrolysis and of pig lead in a thermal lead smelter operation using a conventional lead refinery and fuming of the blast furnace slag.

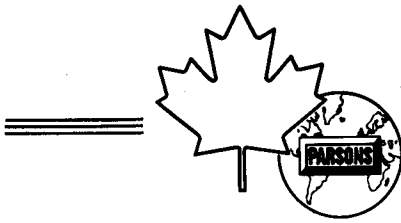
Case C - Production of Special High Grade slab zinc in a facility employing the New Jersey Zinc Company vertical retort process and refining by distillation. Lead is produced in a conventional thermal lead smelter and refinery.

Case D - Treatment of the ore to yield a bulk concentrate for producing crude lead and zinc in a facility employing the Imperial Smelting Process. Zinc is refined to Special High Grade quality by distillation and lead is refined in the conventional manner.

Case A-1-A variation of Case A, predicated upon the production of higher grade selective concentrates (70 percent lead and 55 percent zinc) for toll smelting.

Case B-1-A variation of Case B, considering use of the higher grade selective concentrates to produce Special High Grade zinc by electrolysis and to toll smelt the lead concentrates.

Subsequent to the above-mentioned report, an additional study was conducted and a second report submitted to Anvil Mining Corporation Limited on July 1, 1966. This second report was titled, "Economic Supplement No. 1 to the Preliminary Feasibility Report." The principal objective of this report was to make an incremental analysis on a discounted cash flow basis among concentrating;



smelting and refining; and, smelting and refining with secondary recovery. The capital and operating cost data used in this study was updated and refined from that developed in the initial study. The major refinements considered were:

1. Use of rail transportation of metal products to tidewater.
2. Reduced power costs.
3. Reduced marine transportation and marine insurance rates.
4. Use of the world pricing structure for concentrates and metals marketed outside the limits of the United States.

The three additional cases which were evaluated are described as follows:

Case A-2 - The production of higher grade selective concentrates (70 percent lead and 55 percent zinc) for custom smelting.

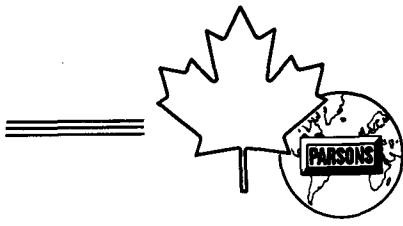
Case B-2 - Production of higher grade selective concentrates for treatment to yield Special High Grade slab zinc from electrolysis and refined pig lead from a conventional lead smelter and refinery with secondary recovery by fuming of the blast furnace slag.

Case B-3 - The same as Case B-2, but excluding the secondary recovery facility.

From an engineering standpoint, the project remained relatively dormant during the period between July and September of 1966. Late in September it was decided to reinitiate the study based on the premise of prosecuting sufficient engineering-design to refine the capital and operating cost estimates and further define the scope of the project. The scope encompassed by the present engineering-design effort includes:

1. Mine development.
2. Crushing plant and coarse ore storage.
3. Concentrator to produce high grade, dried selective lead and zinc concentrates.
4. Concentrate storage and load-out at minesite.
5. All requisite off-sites.
6. Concentrate unloading, storage and ship loading facility at tidewater with ancillary requirements.

The reinitiation of the study was instituted by Anvil Mining Corporation Limited at this time with complementary studies on other phases of the project to permit reexamination of the economics in light of additional data developed,



or being developed. The most significant of these areas were:

1. Sufficient drilling had been completed to more fully delineate the ore body and to permit more definitive calculations of ore reserve tonnages, average assay grades, average mining grades and preproduction stripping requirements.
2. Metallurgical testwork was in progress to evaluate:
 - a. Locked-cycle flotation of a composite sample.
 - b. Locked-cycle flotation of a single core sample.
 - c. Preliminary and confirming grinding test data.
 - d. Preliminary filtration test data.
3. Foster Economic Consultants Ltd. were making a study to appraise the availability of permanent operating personnel and to recommend terms and conditions of employment.
4. E. Miles and Sons were reevaluating the entire problem of transporting the dried concentrates by truck between the plantsite and tidewater (at Haines, Alaska).
5. The Northern Canada Power Commission was making a preliminary power load, **transmission** and cost study covering the supply of power to Anvil.
6. Ametalco was making a comprehensive study of the entire problem of marketing lead and zinc concentrates, which included:
 - a. Short-range marketing outlook for lead and zinc concentrates.
 - b. Tonnage distribution.
 - c. Applicable smelter schedules.
 - d. Metal pricing structure.
 - e. Marine transportation, stevedoring and inland freight rates.
 - f. Marine insurance rates.
 - g. Applicable import duties.



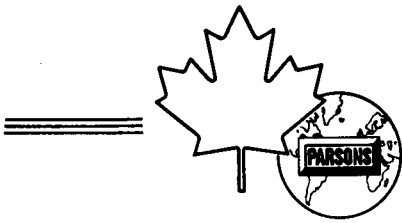
With the new capital and operating cost estimates based on the aforementioned data and the additional economic input from the complementary studies undertaken by Anvil, another Base Case (Case A-4) has been evaluated economically for the production of selective lead and zinc concentrates for custom smelting. This case constitutes the subject matter of this report. The significant criteria, basic to the development of the economics, are enumerated below. These are discussed in detail in subsequent sections of the report.

1. Mill Feed

	<u>Years 1 to 5</u>	<u>Years 6 to 17</u>
Percent Lead	4.9	3.41
Percent Zinc	7.3	5.92
Percent, Combined Lead and Zinc	12.2	9.33
Tonnage, Dry Short Tons x 10 ⁶	2.000	3.000

2. Concentrates

	<u>Years 1 to 5</u>	<u>Years 6 to 17</u>
Lead Concentrate, Percent Lead	69.1	70.0
Lead Concentrate, Percent Recovery	90.5	89.0
Lead Concentrate, Dry Short Tons x 10 ³	128.4	130.1
Zinc Concentrate, Percent Zinc	53.8	54.0
Zinc Concentrate, Percent Recovery	88.5	86.0
Zinc Concentrate, Dry Short Tons x 10 ³	240.2	282.9



3. Metal Pricing Structure (in cents per pound)

	<u>Lead</u>	<u>Zinc</u>
United States	12.5	12.0
World	10.0	10.5



SECTION 3

PROJECT ECONOMICS

In this section on Project Economics, revenues, operating costs, and capital costs are combined to compute rates of return on investment and payout periods. In addition, estimated production costs are consequently available.

The technical feasibility of this project has been discussed elsewhere. This section shows that after due consideration of the several facets of this study it may qualifiedly be concluded that lead and zinc concentrates can be produced in this Yukon operation at a cost that is competitive and at discounted cash flow rates of return, on the total investment considered, that appear reasonable. With the lead and zinc concentrates being custom smelted in Japan, a rate of return of 19.1 percent per year appears realizable on an initial investment considered of approximately \$56 million.¹ This yield corresponds to a cash payout period of 4.0 years. These rates and conclusions are based upon revenues computed from Japanese custom smelter schedules with world prices of lead and zinc at 10.0 and 10.5 cents per pound and are subject to the reliability of the input data and the various assumptions that, of necessity, had to be made.

With cash production costs, after credit for silver and gold revenues, running at 7 cents per pound of combined lead and zinc metal, an operation of this type would appear to be in a good position during a competitive period.

The accompanying Table 3-1 summarizes these data and, in addition, shows the sensitivity of the rate of return to changes in costs, investments, and revenues. It will be noted that with changes in costs and investments to the extent of plus or minus 10 percent, the rate of return would be reduced or increased by an absolute percentage figure of 2 to 3 percent per year. On the other hand, similar percentage changes in revenue would yield increases or decreases of 5 percent per year.

¹Consistent with the entire report, all monetary values are in terms of U.S. dollars.



TABLE 3-1

ECONOMIC SUMMARY
Base Case (A-4)

RATE OF RETURN
ON INVESTMENTS

<u>Base Rate of Return</u>	- %/Yr.	19.1
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Sensitivities
Differential Rates
of Return With

$\pm$ 10% Costs	- %/Yr.	-2.8 +2.5
-----------------	---------	--------------

$\pm$ 10% Investments	- %/Yr.	-2.0 +2.5
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$\pm$ 10% Revenues	- %/Yr.	+4.5 -5.1
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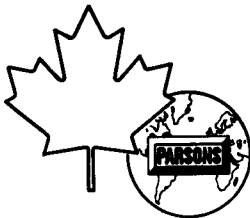
<u>CASH PAYOUT PERIOD</u>	- Yrs.	4.0
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BASIS

<u>Investment</u>		
Immediate	- \$MM	56.1
Fifth Year	- \$MM	3.1

<u>Prices (World)</u>		
Lead	- ¢/lb.	10.0
Zinc	- ¢/lb.	10.5

<u>Concentrate</u> <u>Destination</u>		Japan
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The accompanying Table 3-2 shows the capital outlays estimated for the project. The first item, the exploration and development costs, was estimated to be \$6.2 million up to the initiation of operations. The present review of this project was considered as providing information for the decision to expend funds beyond the sunk costs represented by the exploration and development costs. Consequently, these sunk costs were not part of the investment considered in computing the rate of return and payout period. The next caption shows the estimate of the immediate outlays required which total \$56 million and finally an investment deferred until the fifth year of operation is shown in the amount of \$3 million.

not considered

Revenue and costs are shown in the accompanying Table 3-3 for this case and the method of developing the net realizable value is set forth therein. As additional information, the costs - Haynes to Smelter - have been shown in order to develop the net realization f.o.b. Haynes. Also presenting the cash costs f.o.b. Haynes permits showing the difference between the net realization and cash costs at Haynes. The tabulation also picks up the cost items to show the total estimated cash costs as 7 cents per pound of combined lead and zinc in the concentrates.

It should be pointed out that even if U.S. prices for zinc run 1.5 cents per pound higher, the net realizable value for zinc f.o.b. Haynes is still slightly higher for custom smelting in Japan rather than in the U.S.A. Table 3-12 presented later in this section, shows these computations in more detail. Basically, the 1.5 cents per pound differential is offset by U.S. import duties, higher treatment costs in the U.S., and by the absorption of terminal inland freight costs by Japanese smelters.

*by Japan
seller*

This base case was also evaluated with revenues resulting from lower prices for lead and zinc. A range of 8.5 to 10.0 cents per pound

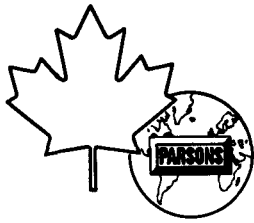


TABLE 3-2
CAPITAL OUTLAYS

	<u>\$MM</u>	<u>\$MM</u>	<u>\$MM</u>
<u>SUNK COSTS</u>			
Exploration and Development Costs			6.2
<u>IMMEDIATE OUTLAYS</u>			
Depreciable Assets	49.8		
Preoperation Stripping	2.2		
Permanent Bridge and Road	0.5		
Net Working Capital	3.6		56.1
Interest During Construction ¹		(3.0)	
<u>DEFERRED INVESTMENTS</u>			3.1

¹This item, not included in the computations, is shown to indicate that a provision may need to be made for some funds of this nature. Interest is implicitly included in the return to total capital.



TABLE 3-3

REVENUES AND COSTS

Per Pound of Lead and Zinc
Content of the Concentrates

	<u>Revenue</u>			<u>Costs</u>
	<u>Pb</u> ¢/lb	<u>Combined</u> ¢/lb	<u>Zn</u> ¢/lb	<u>Combined</u> ¢/lb
World Prices	10.0	10.3	10.5	
Smelter Costs	2.0	- 4.2	5.7	4.2
Realizable Value	8.0	6.1	4.8	
Credits for Other Metals	2.2	+ 0.9	-	(0.9)
Net Realizable Value	10.2	7.0	4.8	
Costs-Haynes to Smelter	0.3	- 0.4	0.4	0.4
Net Realization f.o.b. Haynes	9.9	6.6	4.4	
Cash Costs f.o.b. Haynes		- 3.4		3.4
Difference (Cash)		<u>3.2</u>		
Total Cash Costs				7.1
				<u>7.1</u>

Pb 9 / Zn 10
would give 15%

were extrapolated
from range shown

RATE OF RETURN ON INVESTMENT IN %/YR.

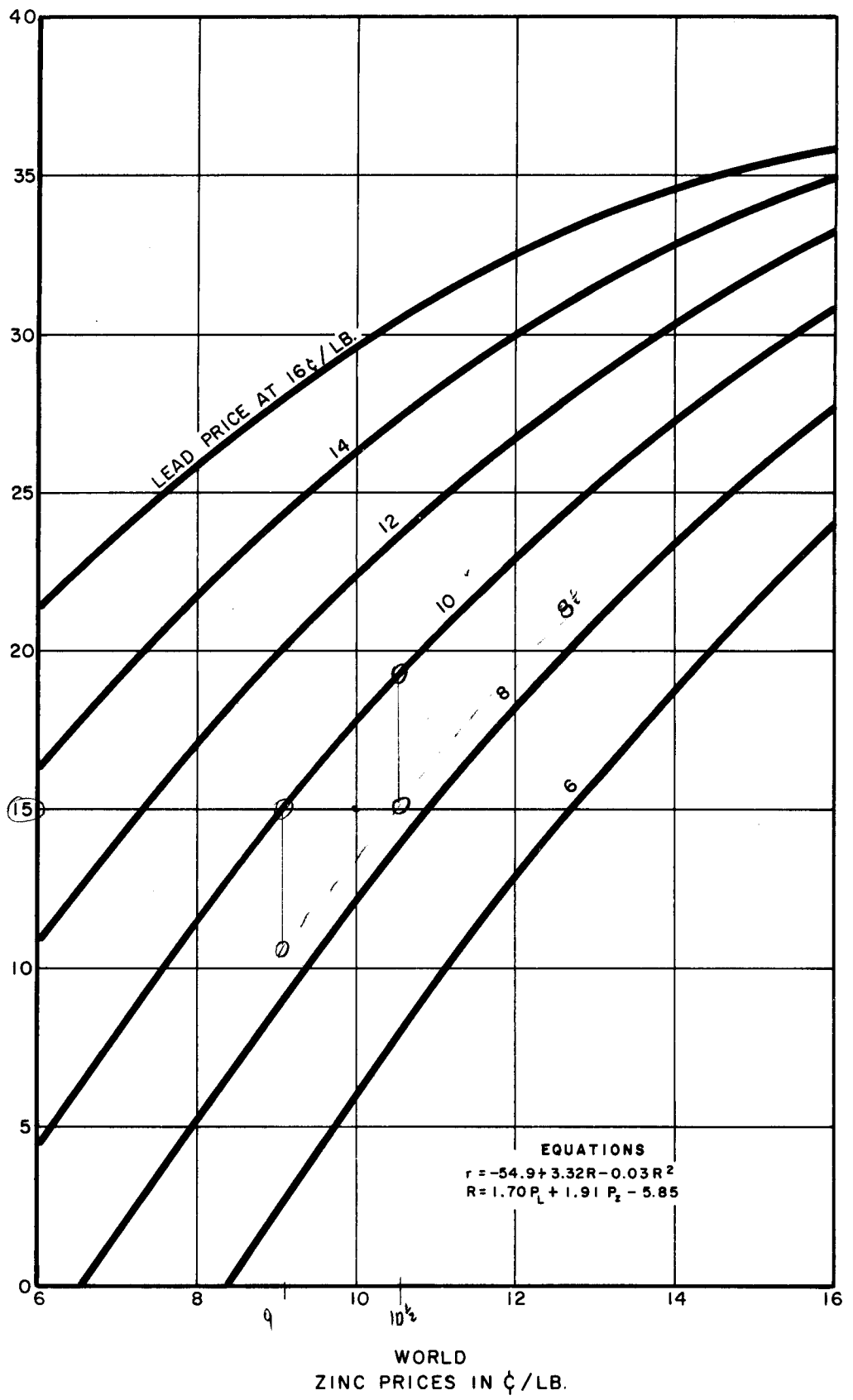
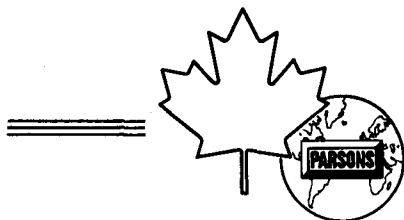


FIGURE 3-1

EFFECT OF PRICE ON RETURN
BASE CASE (A-4)



was covered for lead and 9.0 to 10.5 cents per pound for zinc. These results were used to construct the accompanying parametric curve of Figure 3-1, which shows the rate of return as a function of lead and zinc prices. Rates of return were interpolated within this range and by extrapolation were obtained for a range of 6 to 16 cents per pound for each metal. This is a rather broad extrapolation and the results must be considered as indicative only. ✓

The case presented was based entirely upon custom smelting of the concentrates in Japan. Some consideration was extended to custom smelting of the lead concentrate in Europe, as well as in Japan, and of the zinc concentrate in Japan, Europe, and the U.S. These results are presented in the detailed portion of this section. Sufficient to say, as already indicated, the results are slightly lower when custom smelting is considered in other locations than Japan. *All Japan*

DETAIL

As previously mentioned, the approach to the economics of this project is based upon the principle of a return to total capital, that is, on the return to all investors, both debt and equity. Thus, the cash flow return computed was allowed to include all interest (the return to creditors), as well as the return to equity. The discounted cash flow rate of return is thus an average of the interest rate to the creditors and the rate of return to the equity investors. This approach is particularly convenient when the manner of financing the project is not definitely established, since the amount of debt need not be considered and neither need the interest thereon be deducted from the revenues. At a first look, this avoidance of the method of financing would appear to lose the tax shield benefit of interest. On the other hand, in this project, income tax is forgiven in the first three years and overcome by write-offs in at least the fourth year; thus, obviating the need to include the tax shield benefit in the early years when it would be appreciable. After four years the amount of interest would be smaller and the discount rate higher so that its effect can be neglected without materially affecting the computation of the discounted cash flow rate of return. Also any minor effect of such an omission is to decrease the rate of return.

Other specific views of approaching the economics of this project can better be discussed in the following subsection "Investments". Following the presentation of the investments, the annual costs will be discussed followed by a specific presentation of the method of handling the town site capital and operating costs. Subsequently, the development of the concentrate revenues will be shown along with a reference to the cash flow computations.



Investments

This subsection will present the capital outlays associated with the project and will delineate those specific investments employed as the capital upon which the rate of return was computed. The accompanying Table 3-4 lists the capital outlays associated with this project. It will be noted that it is broken down into Sunk Costs, Immediate Outlays, and Deferred Investments.

The sunk costs in this tabulation are those commonly known as exploration and development costs which, in this case, exceed \$6.2 million. Table 3-13, at the end of this section, shows the derivation of this figure from the 1966 balance sheet of the firm with a projection to October 1, 1969 when full commercial operations are expected to begin. Basically, these costs are costs either already expended or committed and for this reason have been labeled "Sunk Costs". These sunk costs have not been considered as part of the investment under consideration. It was viewed that this study was being prepared to assist in the evaluation of the desirability of committing a rather sizeable investment for the production of lead and zinc concentrates in the Yukon. Since these sunk costs are already expended or committed, they were not included in those investments under consideration as a further commitment. As another view, we are standing at a point in time at which practically all of these sunk costs have been expended and at this point of time a decision is to be made whether further funds should be expended. Thus, the evaluation of this project was based solely upon those funds under review for further commitment.

Those outlays, the commitment for which is under consideration are the ones tabulated as the immediate outlays and the deferred investments. The immediate outlays cover the estimate of the depreciable assets, the preoperation stripping costs, Anvil's portion of the cost for the permanent bridge and road, the estimate of the net working capital required and an estimate of the interest during construction. The deferred investments represent the additions to the mine and mill in the fifth year of operation to accommodate the changes in operation anticipated in the sixth year of operation.

Town site costs have been included in the above item of depreciable assets including a portion of the housing costs. A more



TABLE 3-4
CAPITAL OUTLAYS

	<u>\$MM</u>	<u>\$MM</u>	<u>\$MM</u>
<u>SUNK COSTS</u>			
Exploration and Development Costs			6.212
<u>IMMEDIATE OUTLAYS</u>			
Depreciable Assets	49.814		
Preoperation Stripping	2.223		
Permanent Bridge and Road	0.463		
Net Working Capital	3.580		56.080
Interest During Construction ¹		(3.000)	
<u>DEFERRED INVESTMENTS</u>			3.140

¹This item, not included in the computations, is shown to indicate that a provision may need to be made for some funds of this nature. Interest is implicitly included in the return to total capital.



detailed discussion of the handling of the capital costs for the town site will be presented in a later subsection.

The investment under consideration is the \$56 million tabulated under "Immediate Outlays", and it is upon this investment that the rate of return is calculated. To be sure the deferred investments in the fifth year of operation serve to reduce the cash flow of that year.

We have already mentioned that the exploration and development costs were derived from Table 3-13 at the end of this section. The estimates of the depreciable assets are presented in Section 9 of this report - "Capital Cost Estimate". The town site portion of this capital cost estimate, however, has been modified as will be subsequently pointed out. This same section presents the estimate of preoperation stripping costs¹ which are available for immediate write-off as opportunity permits. It also presents the estimate of the permanent bridge and road cost, Anvil's portion of which is available for early write-off. Finally, the deferred investments too may be found in that section. On the other hand, the computation of the net working capital is presented subsequently in this subsection. Table 3-21, at the end of this section, tabulates the Capital Costs in more detail.

It was estimated that the construction of the facilities would require 24 to 26 months of elapsed time to complete. In addition, it was estimated that there would be a two-month period of non-commercial operation or start-up period. Our cash flow computation in Table 3-15, at the end of this section, shows the operations beginning with the estimated start of commercial operations. Thus, the years of operation in our cash flow table correspond to the anticipated fiscal years of the firm. The two-month start-up or non-commercial operation period is included in the minus first year. Thus, the last year prior to the initiation of commercial operations includes 10 months of construction activity and two months of start-up activity. There then remains some 14 to 16 months of construction activity (or elapsed time) prior to the minus first year just described. For convenience of presentation and computation this initial construction effort was grouped into the minus second year of our tabulation. Since expenditures in the initial months of construction, and

splitting 12/15

¹See also Section 4.



particularly the payment for these expenditures, will not be too large, this simplification was considered permissible. The distribution of construction expenditures between these two years prior to the initiation of commercial operations was based upon the data presented to us but modified by a 45-day accounts payable period.

The two-month start-up period was envisioned as one in which the mill would operate at half capacity with one half of the required personnel, and but one of its mill lines during the first month and likewise during the second month, however, on the second mill line. The production during this period is estimated to be just sufficient to establish the inventory estimated to be required, that is, the ore and concentrate inventories at the plant site and the concentrate inventories at tidewater. The size of these inventories will be further discussed in establishing the net working capital requirement. The costs of this two-month operation were thus, in the main, absorbed by the establishing of inventory and it is for this reason that no pre-production costs are shown. It should be pointed out, however, that a minor cost of \$100,000 has been included in the mill capital costs as run-in expenses and that some minor general and administrative costs for this period have been included in the exploration and development costs.

The amount of the net working capital requirement was developed with the assistance of Anvil and Cyprus personnel. The accompanying Table 3-5 shows the details of this development.

The relationship between the amount of product inventory and the accounts receivable was established by the consideration that for the purpose of this study the title to the product was considered to change hands at the time the vessel is loaded and ready to sail, that is, f.o.b. the Port of Haynes. The accompanying Table 3-6 shows the development of the inventory requirement in tons. Ore inventory of 22,000 tons was taken as likely. For concentrates, two days of production of each concentrate was considered at the plant site and



TABLE 3-5

NET WORKING CAPITAL

Based Upon Case A-4
World Prices Pb 9.0¢/lb.
Zn 10.0¢/lb.

10.0 / 10.5 ?

<u>CURRENT ASSETS</u>		<u>\$ MM</u>
Necessary Cash		1.000
Inventory		
Ore	22,000 Tons @ \$1.929/2.000	0.021
Concentrates		
Plant Site	2,000 Tons @ \$6.737/0.3686	0.037
En Route and at Haynes	31,700 Tons @ \$12.779/0.3686	1.098
Stores and Supplies		
Mining	\$0.340/2	0.170
Oil	0.223/4	0.056
Explosives	0.065/4	0.016
Operating Materials	0.253/4	0.063
Reagents, et cetera	1.798/4	0.450
Plant Mtce. and Operation	0.497/2	0.248
Haynes	0.101/2	0.050
Major Spare Parts		0.250
Contingency		0.040
		1.343
Accounts Receivable	$\frac{(\$27.978 - 1.613)}{12} \times \frac{2}{3}$	1.465
Total Current Assets		4.964



TABLE 3-5 (Cont'd)

NET WORKING CAPITAL

CURRENT LIABILITIES

\$ MM

Accounts Payable

Stores and Supplies	(30 Days)	\$3.277/12	0.273
Truck Transportation	(15 Days)	6.042/24	0.252
Selling Expense	(30 Days)	(26.365 x 0.015)/12	0.033
Coal	(30 Days)	0.600/12	0.050
Power	(30 Days)	1.000/12	0.083

Salaries and Wages (25 Days) (2.487/12) 25/30 0.173

General and Administrative (Excl. of Salaries) 0.020

Taxes and Royalties Due 0.500

Total Current Liabilities 1.384

TOTAL CURRENT ASSETS \$4.964

TOTAL CURRENT LIABILITIES 1.384

Net Working Capital 3.580

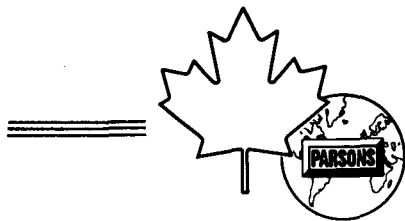


TABLE 3-6

INVENTORY

		<u>Zn</u> Tons	<u>Pb</u> Tons	<u>Total</u> Tons
<u>ORE</u>				
Total				22,000
<u>CONCENTRATES</u>				
Plant Site	2 Days	1,300	700	
En Route	1 Day	650	350	
Haynes		20,000	10,700	
Total		21,950	11,750	33,700
<u>REFERENCES</u>				
Production Rate-Tons/Day		658	352	1,010



one day of production of each concentrate enroute to Haynes. At Haynes zinc concentrate was allowed to build up to an inventory of 20,000 tons of this concentrate. During this period the lead concentrate inventory at Haynes would grow to 10,700 tons. These inventory quantities are evaluated in Table 3-5 under Current Assets.

The related accounts receivable asset was evaluated by considering that for the purposes of this study the concentrates would be sold f.o.b. shipboard at Haynes with allowance for the ocean freight, terminal stevedoring, terminal inland freight, product insurance, and import duties. An accounts receivable period of 20 days was selected as likely considering that 75 percent of the account would be settled on the arrival of the shipping papers in New York and that a one to three month delay might occur for the remainder in order to allow time for assay, et cetera.

The Stores and Supplies Inventory was estimated by considering a 90 day inventory level for operating supplies and a 6 month level for maintenance supplies.

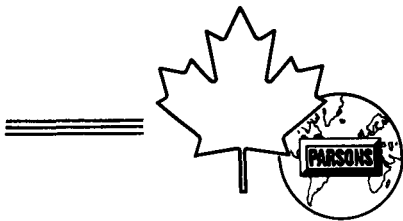
Estimating a necessary cash requirement of \$1 million, the current assets were estimated at just short of \$5 million.

Current liabilities were estimated by considering appropriate accounts payable periods for the items listed. Royalties due is a rather sizeable item since the royalties for the current year are not payable until the following year. The royalties and taxes due item was taken at \$0.5 million.

Combining the estimate for current assets and liabilities, a net working capital of \$3.58 million is obtained.

Annual Costs

Annual costs for the commercial operating years are set forth in detail in Table 3-15 at the end of this section. Mining costs are discussed in more detail in Section 4 - Mining Studies, and the Mill or Plant Costs are discussed in more detail in Section 6 - Plant Operating Costs. The Summary Section (Section 1) covers the items: Truck Transportation to Haynes, Costs at Haynes, Ocean Freight and Stevedoring and Inland Freight, Product Insurance, Import Duties, and Sales Expense. The remainder of the items were computed in this section of the work with the assistance of Anvil and Cyprus personnel or supplied by Cyprus personnel.



Local taxes were computed based upon estimated cost of buildings. Insurance was supplied by Cyprus personnel based upon a capital cost breakdown. General and administrative costs were developed by Anvil and are shown in the accompanying Table 3-7. It will be noted that the Los Angeles expenses include salaries, whereas salaries are excluded in the local general administrative costs. This is because the operating costs previously presented included local general and administrative salaries and wages. Town site costs are discussed in more detail in the next subsection.

The mining royalty or tax was computed according to the schedule provided on the basis that it was a good cost in determining the income tax and that the income tax was a good cost in determining the royalty tax. This required a simultaneous equation approach to the solution.

Income taxes were computed on the basis that tax forgiveness was available for the first three years of commercial operation and that exploration and development costs, preoperation stripping costs, and Anvil's portion of the permanent bridge and road could be written off at the earliest opportunity, that is, in the fourth year and subsequently if required. Since depreciation is postponable, it was not applied until the write-offs no longer eliminated the income tax. The depreciation was computed on the declining balance method according to the allowable maximum percentages shown in Table 3-14 at the end of this section. In the main, the maximum allowance was 30 percent with exceptions for the tidewater facilities and the town site facilities, in which cases the maximum ran from 5 percent for the dock to 20 percent for the major equipment at tidewater. Income tax was taken at 50 percent of the pre-tax profit after depletion; but the $33\frac{1}{3}$ percent depletion, in effect, reduces the income tax to $33\frac{1}{3}$ percent of the profit before taxes and depletion. good

The production costs on a cents per pound basis are tabulated in the accompanying Table 3-8. It is to be noted that these costs are per pound of combined lead and zinc content of the concentrates. They are not expressed on the basis of the lead and zinc recovered by the smelter. Of the cash costs the truck transportation to Haynes and the concentrating costs are the major items. The costs f.o.b. Haynes are added to the costs to deliver the concentrates to the smelter to obtain the costs at that point. The items, stevedoring and inland freight are meant to be the stevedoring at the receiving port and the inland freight from the receiving port to the smelter. Stevedoring at Haynes is included in the item "Cost at Haynes". It is to be noted that the inland freight from the port of receipt to the smelter is expected to be absorbed by the smelter company in all cases



TABLE 3-7

GENERAL AND ADMINISTRATIVE COSTS

	<u>\$MM</u>	<u>\$MM</u>	<u>\$MM</u>
<u>Local Expenses (Excl. of Salaries)</u>			
Communications	0.025		
Insurance-Liability	0.005		
Office Supplies, et cetera	0.012		
Legal Fees	0.005		
Audit Fees	0.005		
Travel	0.005		
Consultants	0.012		
Other	0.005		
		0.074	
<u>Los Angeles Expenses</u>			
Salaries	0.060		
Communications	0.005		
Office Supplies	0.003		
Legal Fees	0.025		
Travel	0.050		
Other	0.005		
		0.148	0.222



TABLE 3-8

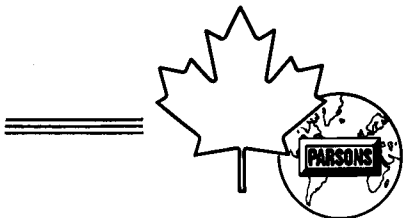
PRODUCTION COSTS

1st Year - Base Case (A-4)

Pb 10.0 ¢/lb, Zn 10.5 ¢/lb

	Costs per Pound of Combined Lead and Zinc Content of Concentrates ¢/lb.
<u>Cash Costs</u>	
Mining	0.443
Concentrating	1.103
Truck Transportation to Haynes	1.386
Costs at Haynes	0.085
Sales Expense	0.099
Local Taxes	0.031
Royalties	0.081
Insurance	0.034
General and Administrative	0.051
Town Site Costs	0.114
Costs f.o.b. Haynes	3.427
Ocean Freight, Stevedoring and Inland Freight	0.356
Product Insurance	0.015
Import Duties	-
Costs to Smelter	3.798
Smelter Costs	3.277
Total Cash Costs	7.075
<u>Depreciation¹</u>	0.672
TOTAL	7.747

¹With Depreciable Assets divided by 17. = number of years of operation



except by the smelters in the U.S.A. Thus, the item inland freight is only pertinent when considering shipment of concentrates to the U.S.

The smelter costs, as might be expected is a sizeable item and includes the treatment costs as well as the effect of not recovering 100 percent of the metal content of the concentrate. The total cash costs estimate adds up to 7.1 cents per pound of combined metal content which represents a competitive cost of the metal laid down to the smelter.

If it is desired to add depreciation to the cash cost, some device must be used since depreciation varies year by year depending upon tax forgiveness, write-offs, and the declining balance. For the purpose of adding depreciation to the cash costs one-seventeenth of the depreciable asset cost was used.

Town Site Costs

The town site costs are separately presented herein for the reason that some of the estimates presented in Section 9, Capital Cost Estimates, have been adjusted for use in this section in line with more recent information. In addition, it was felt desirable at the same time to discuss the manner in which the investment in housing units is handled for the cash flow computations. The accompanying Table 3-9 lists in its initial portion the capital costs by Anvil which are considered as an investment for this project. The amounts tabulated for the dispensary, the recreation hall, and the boiler house are identical with those presented in the referenced Section 9. The estimate in that section for bunk houses was reduced by \$427 thousand on the basis that bunk houses provided for the construction personnel would be subsequently used for the operating personnel. The item for the cost of a school building was not included as an investment for Anvil since it expected the government will provide this facility. This leaves housing and subdivision costs (including utilities other than the boiler house) to be reconciled.

The costs for the houses and subdivision costs (including utility-lines) were estimated by Parsons on the basis that the houses would be constructed at the site. Later information obtained by Anvil showed the possibility of obtaining prefabricated houses constructed in Calgary. It is expected that these prefabricated houses built in two sections could be transported to the housing site and installed and connected for a reasonable cost. These costs are set forth in the accompanying Table 3-11. Further, it is expected that the government will provide subdivision lots including utility lines,

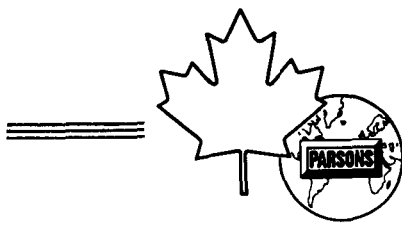


TABLE 3-9

TOWN SITE

Anvil Costs

	<u>\$MM</u>	<u>\$MM</u>	<u>\$MM</u>	<u>\$MM</u>
<u>CAPITAL COSTS BY ANVIL</u>				
Dispensary		0.137		
Recreation Hall		1.100		
Bunk Houses		0.641		
Boiler House		0.650	2.528	
Houses - 300 Units				
Equity	1.770	1.770	1.770	4.298
Mortgage	4.830			
Total		6.600		
<u>ANNUAL COSTS TO ANVIL</u>				
Services (No Personnel Costs)				
Hospital				
School				
Total		0.010	0.010	
Recreation				
Director and Girl		0.015		
Other		0.020	0.035	
Cookhouse and Bunkhouse				
Gross Cost - \$210/man mo.				
Employee Charge - \$75/man mo.				
Net Cost - \$135/man mo.				
75 Men @ - \$135/man mo.		0.122	0.122	
Housing				
300 Units @ \$1100/Yr.		0.330	0.330	0.497
(See Table 3-10 for detail).				



TABLE 3-10
ANNUAL UNIT HOUSING COSTS

PER UNIT

Capital Cost ¹		\$22,000
Loan		
14,000 @ 90%	\$12,600	
5,000 @ 70%	3,500	16,100
Equity		5,900

ANNUAL COSTS PER UNIT

Loan Amortization		
7-1/4% for 25 years - \$116.38/Mo.		
\$116.38/Mo. x 12		1,397
Local Tax		170
Heating		300
Power		180
Insurance		100
Repair		153
Total Annual Cost Per Unit		\$2,300
Annual Personnel Payment Per Unit		1,200
Annual Deficit Per Unit		1,100

¹For detail see Table 3-11

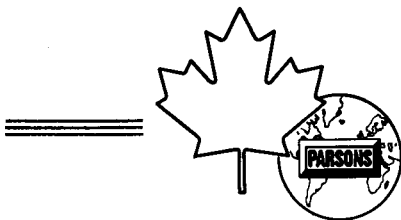


TABLE 3-11
DETAILS OF COST OF HOUSING

1,230 Sq. Ft. f.o.b. Calgary	\$11,500
Extras	670
Freight - Est.	1,500
Excavation	170
Basement	3,450
Finishing	800
Landscaping	500
Services	<u>500</u>
Total	\$19,090
Estimated Lot Cost	<u>2,910</u>
Total, including lot	\$22,000



streets, curbs, et cetera, for approximately \$2,900. This brings the total for each unit to \$22,000 and this is the figure that was used as the housing investment shown in the accompanying Table 3-10. For 300 units accepted as the required amount for this study the total cost for the housing units would be \$6.6 million. As shown in Table 3-9, only the equity portion of this housing cost was listed as the investment by Anvil. We will return to this point in a moment. In the remainder of Table 3-9, the annual costs of operating the town site are shown. The most appreciable factor listed is the cost to Anvil of operating 300 housing units at \$1100 per year.

housing

Table 3-10 shows the unit housing costs, both capital and annual costs. The per unit capital cost of the house was taken at \$22,000. The anticipated loan arrangements set forth leave an equity investment per unit of \$5900. Under annual costs the remaining mortgage of \$16,100 per unit would require a loan amortization payment of approximately \$116 per month when amortized over 25 years at 7-1/4 percent. Other annual costs have been added to the loan amortization costs to attain a total annual cost per unit of \$2300.

It is anticipated that the housing units will be either rented to the employees or sold to them. In either event it is expected that the total monthly expenditure required of each employee so housed would be approximately \$100 leaving an annual deficit per unit of \$1100. For the purposes of this study it was assumed that the employees would purchase these housing units and that the mortgages would be in their name. In effect, then Anvil would be subsidizing the purchase and operation of these houses to the extent of \$1100 per year per unit. Further, since the mortgage would be in the name of the employees, Anvil's investment would be limited to the equity investment shown above.

It is realized that the home mortgage loans are for 25 years, whereas the life of the project is 17 years. No effect of these remaining mortgages (2.5 million dollars) was included in our computations for two reasons. The first reason is that if the remaining mortgages of \$2.5 million would be paid off in the seventeenth year the effect of this payment upon the rate of return is negligible. The second reason is that any responsibility Anvil might have relative to these mortgages would not be a legal one.



Revenue And Cash Flow

The revenues applicable to the base case presented herein are shown in the Cash Flow Table 3-15 at the end of this section. The revenues were obtained by valuing the concentrates f.o.b. Haynes, with ocean freight and terminal stevedoring and terminal inland freight, product insurance, and import duties allowed. The pricing of the concentrate was based upon custom smelter schedules, considering the anticipated lead and zinc contents of the concentrates to be purchased.

It should be noted that the pricing procedure used took into account an inflationary effect which serves to reduce the price of the product over time; that is, the price of the product in effect was reduced in later years of operation. This inflationary effect was employed when considering an escalation of the treatment cost assessed by the smelters. The escalation was applied in equal annual increments for the first six years but held constant thereafter. This is the only instance in this project in which inflationary effects were considered. It is the normal practice to use the conservative approach of working entirely in constant dollars, usually the dollar value prevalent at the beginning of construction. This is a conservative approach in that if revenues and costs were allowed in the computation to inflate to an equal proportional extent the resulting inflated profits would permit a more ready re-payment of the investments involved or, stating it another way, result in a higher rate of return on investment. For this reason this employment of an inflationary effect to reduce the net revenue is pointed out; in that it is inconsistent with the basic procedure employed but at the same time yields a more conservative (lower) rate of return in this instance.

Having the projected revenues and costs, pro forma earnings were developed to which depreciation, depletion and write-offs were added in the normal fashion to obtain a gross cash flow for the years under consideration. In the pertinent years the investments considered were subtracted from the gross cash flow to obtain the net cash flow upon which the discounted cash flow rate of return was computed. The investments considered were the depreciable assets, the net working capital, preoperation stripping, and Anvil's portion of the permanent bridge and road, as is indicated in more detail in Table 3-14 at the end of this section.

Table 3-16, at the end of this section, shows the basic data in terms of tons of concentrate and their destination upon which the



revenues were computed. It will be noted that more cases than the basic one presented herein were considered. As the basic data shows there are two general cases. Case A-4 in which 100 percent of the concentrate is at all times considered to be shipped to Japan and Case A-3 in which 63 percent of the lead concentrate is considered to be shipped to Europe. In addition, in Case A-3 the zinc concentrate was considered to be shipped to Europe, Japan and the United States in the proportions of 33 percent, 25 percent, and 42 percent, respectively.

The effects of varying prices of the two metals over a range are summarized in graphical form in the parametric curve of Figure 3-1 presented in the early part of this section. This figure shows the rate of return as the ordinate, the price of zinc as the abscissa, and the price of lead as a parameter. In effect, the several price combinations provided several revenues which yielded alternate rates of return. These points were used to develop an equation which was then used to prepare the parametric curve of Figure 3-1. There is an institutional difference between Case A-3 in which the concentrates are marketed in three areas as against Case A-4 in which the concentrates are all shipped to Japan. Because of the discontinuity in results thereby attained, the results presented in Figure 3-1 were limited to those obtained for Case A-4 in which all of the concentrates are considered to be shipped to Japan. The results for Case A-3 in which the products were shipped to three areas of the world including the U.S. are illustrated in a parametric curve that has also been prepared and this Figure 3-2 is shown on the accompanying page. In general, the rates of return are slightly lower for the custom smelting in the three areas (A-3) than for custom smelting in Japan alone (A-4). In Case A-3, that portion of the zinc concentrate custom smelted in the United States was considered to carry a 1.5¢/lb higher value for the zinc metal.

In order to illustrate the reason why custom smelting of the lead and zinc concentrates in Europe and of the zinc concentrate in the U.S. even at a higher metal price does not improve the rates of return, the accompanying Table 3-12 has been prepared. In this table the net realization to Anvil f.o.b. Haynes has been computed for the lead concentrate to Japan, and Europe and the zinc concentrate to Japan, Europe and U.S. The net realization f.o.b. Haynes for the lead concentrate to Japan and Europe was based upon the same world price of 10.0 cents per pound and it can be seen that the realization for the shipment to Japan is slightly higher than for the shipment to Europe. The same computation was made for the

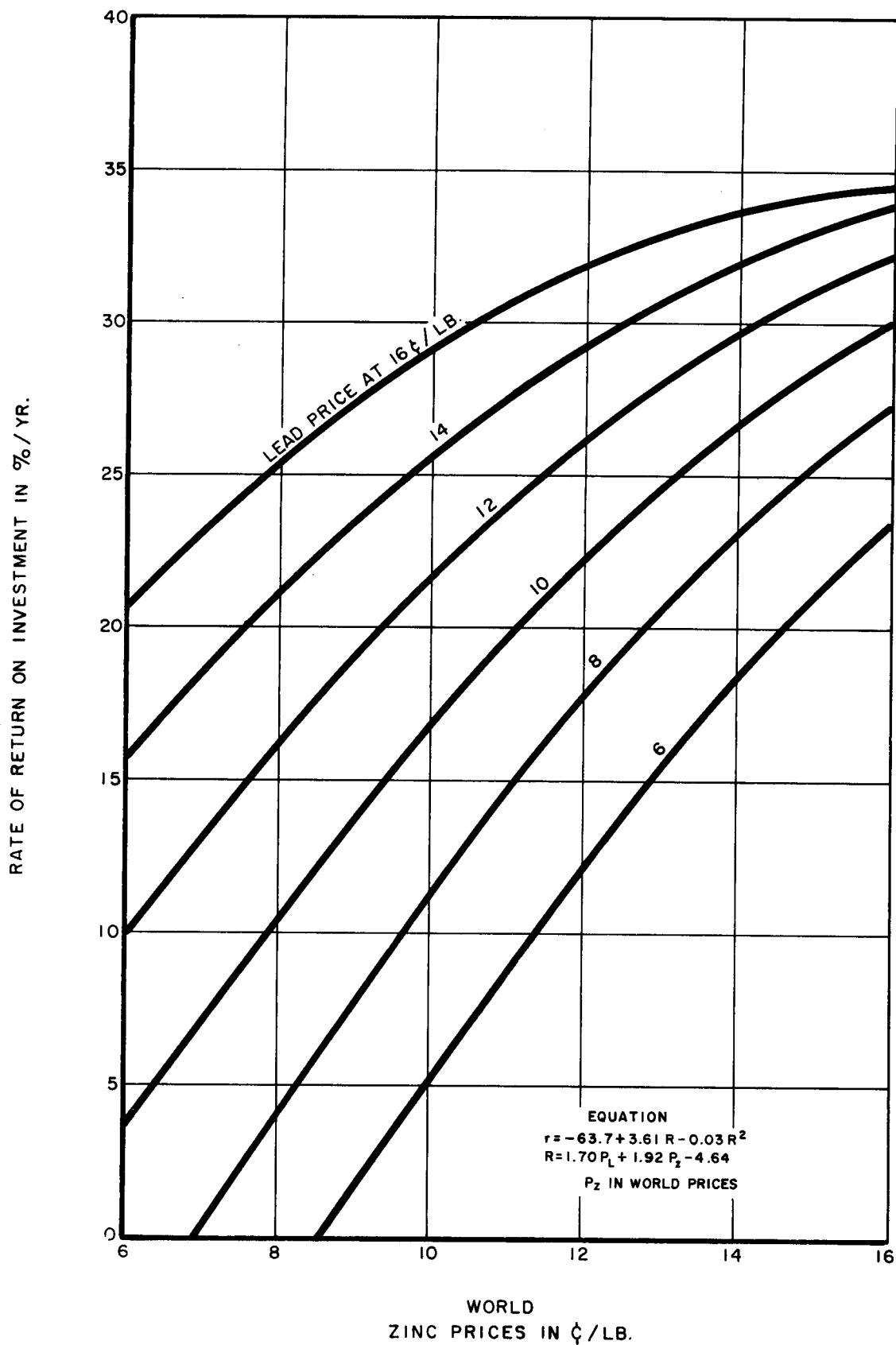


FIGURE 3-2
 EFFECT OF PRICE ON RETURN
 CASE (A-3)

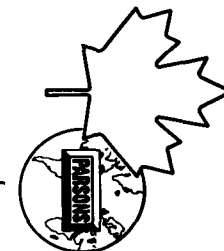


zinc concentrate at 10.5 cents per pound and it can be seen that the realization for the shipment to Japan is appreciably higher than for the shipment to Europe. The final column shows that even if the price of zinc is 1.5 cents per pound higher in the U.S. than in Japan the net realization f.o.b. Haynes does not quite equal that realized from a shipment of zinc concentrate to Japan.

The table also shows that the smelter costs, which include metal not recovered, is much lower for lead than for zinc. That is, the smelter costs for lead are approximately 20 percent of the theoretical value of the lead content of the concentrate, whereas, for zinc the smelter costs are more than 50 percent of the theoretical value of the zinc in the concentrate. In addition, the credit for silver and gold in the lead concentrates brings the net realizable value at the smelter above the theoretical value for the lead in the concentrate. Finally, even though the price of zinc is 0.5 cents per pound higher than the price used for lead the net realization f.o.b. Haynes for lead is more than twice that for zinc. Briefly, the net realization f.o.b. Haynes for lead is almost equal to the theoretical value of the lead in the concentrate.

TABLE 3-12
NET REALIZATION F.O.B. HAYNES

	JAPAN			EUROPE			U.S.A.
	<u>Pb</u> ¢/lb	<u>Avg</u> ¢/lb	<u>Zn</u> ¢/lb	<u>Pb</u> ¢/lb	<u>Avg</u> ¢/lb	<u>Zn</u> ¢/lb	<u>Zn</u> ¢/lb
Price	10.000	10.300	10.500	10.000		10.500	12.000
Smelter Costs	2.009		5.683	1.862		5.723	6.183
Realizable Value	7.991		4.817	8.138		4.777	5.817
Credits for Other Metals	2.237		-	2.215		-	-
Net Realizable Value	10.228	7.020	4.817	10.353	7.047	4.777	5.817
Costs Haynes to Smelter	0.328	0.371	0.401	0.518	0.592	0.642	1.486
Net Realization f.o.b. Haynes	9.900	6.649	4.416	9.835	6.455	4.135	4.331



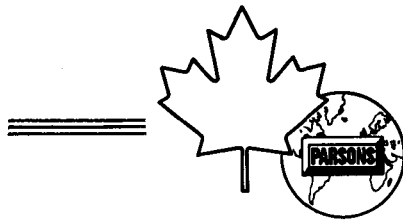


TABLE 3-13

EXPLORATION AND DEVELOPMENT COSTS

Applicable for Write-off for Income Tax Computation

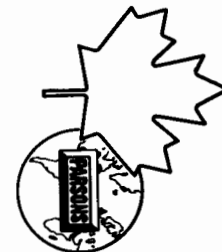
	<u>\$MM (Canada)</u>	<u>\$MM (U.S.)</u>
Per Balance Sheet, December 31, 1966	4.934	
Less estimated non-claimable items	<u>0.028</u> 4.906	
at 92.5%		4.538
Budget to April 30, 1967		0.974
Estimated Expense May 1, 1967 - August 1, 1969		
28 Months at \$25,000 per month		<u>0.700</u>
TOTAL		6.212

TABLE 3-14
CAPITAL COSTS

	<u>Total¹</u> \$MM	<u>Considered Herein</u>			<u>Cash Outflow by Years</u>			
		<u>As Invest-ments</u> \$MM	<u>For Deprec.</u> \$MM	<u>For Write-off</u> \$MM	<u>-2</u> \$MM	<u>-1</u> \$MM	<u>1</u> \$MM	<u>5</u> \$MM
<u>Depreciable Assets</u>								
Mine	4.338	4.338	4.338-30% ²	-	1.900	2.403	-	0.035
Mill	35.073	35.073	35.073-30%	-	17.200	14.768	-	3.105
Transportation Facilities								
Non-mobil	0.427	0.427	0.427-30%	-	0.190	0.237	-	-
Mobil	-	-	-	-	-	-	-	-
Tidewater Facilities								
Major Equipment	1.293	1.293	1.293-20%	-				
Dock	1.332	1.332	1.332-5%	-				
Buildings	4.336	4.336	4.336-10%	-				
Subtotal	6.961	6.961	6.961	-	3.050	3.911	-	-
Town Site Facilities	9.128	4.298	4.298-10%	-	1.900	2.398	-	-
Anvil Superv. and Overhead	1.857	1.857	1.857-30%	-	0.929	0.928	-	-
Total	57.784	52.954	52.954	-	25.169	24.645	-	3.140

¹All costs are exclusive of engineering-constructors contractor's fee.

²Maximum percent of declining balance allowed each year.



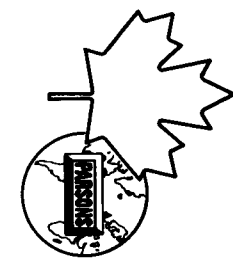


TABLE 3-14 (Cont'd.)

	<u>Total¹</u> \$MM	<u>Considered Herein</u>			<u>Cash Outflow by Years</u>			
		<u>As Invest-ments</u> \$MM	<u>For Deprec.</u> \$MM	<u>For Write-off</u> \$MM	<u>-2</u> \$MM	<u>-1</u> \$MM	<u>1</u> \$MM	<u>5</u> \$MM
<u>Write-offs</u>								
Exploration and Development Costs	6.212	-	-	6.212	-	-	-	-
Preoperation Stripping	2.223	2.223	-	2.223	1.101	1.122	-	-
Preproduction Costs All to inventory								
Permanent Bridge and Road	1.388	0.463	-	0.463	0.230	0.233	-	-
Total	9.823	2.686	-	8.898	1.331	1.355	-	-
Net Working Capital	3.580	3.580	-	-	-	2.110	1.470	-
Interest During Construction	3.000	-	-	-	0.755	2.245	-	-
Grand Total	74.187	59.220	52.954	8.898	27.255	30.355	1.470	3.140

¹All costs are exclusive of engineering-constructors contractor's fee.

TABLE 3-15

Million U.S. Dollars

CASH FLOW
Base Case (A-4)
World Prices Pb 10.0¢/lb, Zn 10.5¢/lb

Years	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
REVENUE																			
Lead Concentrate			18.149	18.149	18.149	18.004	18.004	18.299	18.299	18.299	18.299	18.299	18.299	18.299	18.299	18.299	18.299	18.299	18.299
Zinc Concentrate			12.452	12.452	12.452	12.044	12.044	13.984	13.984	13.984	13.984	13.984	13.984	13.984	13.984	13.984	13.984	13.984	13.984
Total Revenue			30.601	30.601	30.601	30.048	30.048	32.283	32.283	32.283	32.283	32.283	32.283	32.283	32.283	32.283	32.283	32.283	32.283
COSTS																			
Mining			1.929	1.929	1.929	1.929	1.929	2.596	2.596	2.596	2.596	2.626	2.626	2.626	2.626	2.626	2.626	2.626	0.744
Concentrating			4.808	4.808	4.808	4.808	4.808	5.802	5.802	5.802	5.802	5.802	5.802	5.802	5.802	5.802	5.802	5.802	5.802
Truck Transportation to Haynes			6.042	6.042	6.042	6.042	6.042	6.765	6.765	6.765	6.765	6.765	6.765	6.765	6.765	6.765	6.765	6.765	6.765
Costs at Haynes			0.369	0.369	0.369	0.369	0.369	0.380	0.380	0.380	0.380	0.380	0.380	0.380	0.380	0.380	0.380	0.380	0.380
Ocean Freight and Stevedoring			1.552	1.552	1.552	1.552	1.552	1.739	1.739	1.739	1.739	1.739	1.739	1.739	1.739	1.739	1.739	1.739	1.739
Product Insurance			0.067	0.067	0.067	0.067	0.067	0.073	0.073	0.073	0.073	0.073	0.073	0.073	0.073	0.073	0.073	0.073	0.073
Import Duties			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sales Expense			0.433	0.433	0.433	0.428	0.428	0.458	0.458	0.458	0.458	0.458	0.458	0.458	0.458	0.458	0.458	0.458	0.458
Local Taxes			0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133	0.133
Insurance			0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150	0.150
General Administration			0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222	0.222
Town Site Costs			0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497	0.497
✓ Mining Royalty			0.354	0.354	0.354	0.321	0.301	0.191	0.149	0.119	0.097	0.081	0.070	0.062	0.057	0.053	0.050	0.048	0.161
Depreciation			-	-	-	4.632	12.523	8.957	6.441	4.663	3.403	2.504	1.864	1.404	1.074	0.832	0.655	0.525	3.478
Write-Offs			-	-	-	8.898	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Costs			16.556	16.556	16.556	30.048	29.021	27.963	25.405	23.597	22.315	21.430	20.779	20.311	19.976	19.730	19.550	19.418	20.602
Profit before Taxes and Depletion			14.045	14.045	14.045	-	1.027	4.320	6.878	8.686	9.968	10.853	11.504	11.972	12.307	12.553	12.733	12.865	11.681
Depletion			-	-	-	-	0.342	1.440	2.292	2.895	3.322	3.617	3.834	3.990	4.102	4.184	4.244	4.288	3.893
Pre-tax Profit			14.045	14.045	14.045	-	0.685	2.880	4.586	5.791	6.646	7.236	7.670	7.982	8.205	8.369	8.489	8.577	7.788
Income Tax			-	-	-	-	0.342	1.440	2.292	2.895	3.322	3.617	3.834	3.990	4.102	4.184	4.244	4.288	3.893
Earnings			14.045	14.045	14.045	-	0.343	1.440	2.294	2.896	3.324	3.619	3.836	3.992	4.103	4.185	4.245	4.289	3.895
Depreciation			-	-	-	4.632	12.523	8.957	6.441	4.663	3.403	2.504	1.864	1.404	1.074	0.832	0.655	0.525	3.478
Depletion			-	-	-	-	0.342	1.440	2.292	2.895	3.322	3.617	3.834	3.990	4.102	4.184	4.244	4.288	3.893
Write-Offs			-	-	-	8.898	-	-	-	-	-	-	-	-	-	-	-	-	-
Gross Cash Flow			14.045	14.045	14.045	13.530	13.208	11.837	11.027	10.454	10.049	9.740	9.534	9.386	9.279	9.201	9.144	9.102	11.266
Investments Considered																			
Depreciable Assets	25.169	24.645					3.140												
Net Working Capital		2.110	1.470																(1.631)
Preoperation Stripping	1.101	1.122																	
Perm. Bridge and Road	0.230	0.233																	
NET CASH FLOW	(26.500)	(28.110)	12.575	14.045	14.045	13.530	10.068	11.837	11.027	10.454	10.049	9.740	9.534	9.386	9.279	9.201	9.144	9.102	12.897

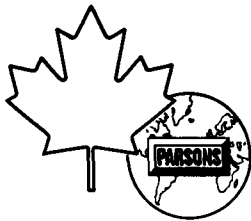


TABLE 3-16

BASIC DATA

2,000 Lb. Tons

<u>Tons Mined</u>		<u>CASE A-3</u>		<u>CASE A-4</u>	
		<u>%</u>	<u>M Tons/Yr.</u>	<u>%</u>	<u>M Tons/Yr.</u>
Yrs. 1-5			2,000		2,000
6-17			3,000		3,000
<u>Pb Conc.</u>					
Yrs. 1-5	Europe	63	80.892	-	-
	Japan	37	47.508	100	128.400
	USA	-	-	-	-
	Total	100	128.400	100	128.400
Yrs. 6-17	Europe	63	81.963	-	-
	Japan	37	48.137	100	130.100
	USA	-	-	-	-
	Total	100	130.100	100	130.100
<u>Zn Conc.</u>					
Yrs. 1-5	Europe	33	100.884	-	-
	Japan	25	79.266	100	240.200
	USA	42	60.050	-	-
	Total	100	240.200	100	240.200
Yrs. 6-17	Europe	33	93.357	-	-
	Japan	25	70.725	100	282.900
	USA	42	118.818	-	-
	Total	100	282.900	100	282.900
<u>Pb Content</u>					
Yrs. 1-5		69.1	88.724	69.1	88.724
Yrs. 6-17		70.0	91.070	70.0	91.070
<u>Zn Content</u>					
Yrs. 1-5		53.8	129.228	53.8	129.228
Yrs. 6-17		54.0	152.766	54.0	152.766



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SECTION 4

STRIPPING AND MINING STUDY

ANVIL PROJECT

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SECTION 4

STRIPPING AND MINING STUDY

SECTION 4A

SUMMARY

On October 11, 1966, the Anvil Mining Company defined, by means of a letter to The Ralph M. Parsons Company, the scope of work for a second feasibility study. This section of the feasibility study is the portion concerning the stripping and mining of the Faro Number One ore deposit. Volumes of materials to be excavated, grade of ore, capital cost of equipment, operating costs, mining plans and operating and maintenance personnel are described in the report. It is proposed to conduct stripping operations on a three shift basis throughout the year while mining will take place on dayshift only throughout the year.

The following pit designs have been prepared:

1. The preproduction pit.
2. A pit for 5 years of operation which would produce about 12,400,000 tons of ore.
3. A recommended final pit from which 41,000,000 tons of ore could be mined.
4. An ultimate pit which could produce 47,000,000 tons of ore.

The quantity of overburden to be removed for each pit stage as well as the average grade and tonnage of ore mined have been calculated. No attempt was made to determine the total quantity of ore in the deposit; Parsons assignment was to design open pits and determine the quantity of ore which could be mined economically from these pits.

Total ore mined in 17 years will be 46,000,000 tons, although not necessarily from one deposit. The mining rate will be 2,000,000 tons per year for the first 5 years and 3,000,000 tons per year for the next 12 years. Preproduction stripping will remove 2,763,000 c.y. of alluvium and 3,777,000 c.y. of cap rock for a total of 6,540,000 c.y. During the first 16 years of mill operation the following average annual quantities will be mined.

	<u>Stripping (c.y.)</u>		<u>Mining (Tons)</u>
	<u>Alluvium</u>	<u>Cap Rock</u>	<u>Ore</u>
1 - 5 Year (per Year)	977,000	2,576,000	2,000,000
6 - 16 Years (per Year)	577,000	3,763,000	3,000,000



Table 4-1 summarized stripping and mining costs.

Table 4-2 shows the total quantities of ore and overburden mined. The grade of ore during the first 5 years will be 12.2% combined lead and zinc. The grade of ore during the next 12 years will be 9.3% combined metals.

During preproduction stripping and during the first 5 years of ore mining, it will be necessary to stockpile a total of 2,790,000 tons of ore in order to achieve an average grade of ore to the mill of 12.2% combined metal. The stockpile will contain 392,000 tons of higher grade ore (11.4% combined lead and zinc) mined during the preproduction stripping period which will be reclaimed during the first 5 years of milling. The remaining 2,398,000 tons will be reclaimed during the 6, 7, 8, and 9 years of milling at the rate of 600,000 tons per year.

The capital cost of the mining equipment is estimated at \$3,830,000 (U.S.). Preproduction stripping is estimated to cost \$2,403,000 (Canadian). All cost figures, with the exception of equipment costs which are in U.S. dollars, are expressed in Canadian currency.

The direct cost of mining ore (including stripping) is as follows:

1 - 5 Years Inclusive	\$0.88 per ton (Canadian)
6 - 9 Years Inclusive	0.93 per ton (Canadian)
10 - 16 Years Inclusive	0.95 per ton (Canadian)

We have developed two means of producing a total of 46,000,000 tons by open pit operations. One method would be by what we have termed the ultimate open pit design which would require the removal of 88,000,000 c.y. of overburden from the Faro Number One deposit. The second and recommended procedure would be to produce about 41,000,000 tons from the Faro Number One deposit and to then obtain about 5,000,000 tons from the Faro Number Two and Three deposits. We estimate that by these means only 72,000,000 c.y. of overburden would have to be removed. The figures in this report are based on the adoption of this plan.

Our operating schedules and equipment list have been developed on the assumption that milling operations would commence in August, 1969. In order to have the preproduction stripping completed by this date, alluvium stripping should start not later than December 1, 1967, and cap rock stripping should commence June 1, 1968.

The necessity of having to operate under cold weather conditions has been incorporated in the design and specifications of mining equipment.



SECTION 4B

INTRODUCTION

A preliminary Feasibility Study was submitted to the Anvil Mining Company on April 26, 1966. This study contained a section in which a stripping and mining program was developed. The report now being submitted is a revision of the first study and the mining section has been revised on the basis of additional drill holes, site inspection, and other information which has been developed subsequent to the first Feasibility Study.

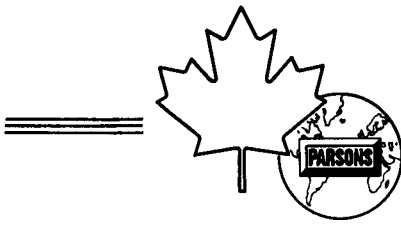
On October 11, 1966, the Anvil Mining Company wrote to The Ralph M. Parsons Company outlining the scope of a second Feasibility Study. The portion of Anvil's instructions regarding the mine were as follows:

"Mining Study

1. Develop open pit mining plans based on geologic sections supplied by Anvil to include:
 - (a) A preproduction stripping plan
 - (b) A waste dump plan
 - (c) A pit design for first 5 years.
 - (d) Final pit design.

The 5 year pit should be designed to exploit the highest grade ore with minimum preproduction stripping requirements.
2. Develop volume estimates for each mining stage specifying quantities of alluvium, waste rock and ore.
3. Develop a recommended equipment list with specifications.
4. Develop mining plans covering volumes of material to be removed year by year and equipment scheduling year by year.
5. Develop capital cost estimates for preproduction stripping and equipment purchase.
6. Develop a schedule of manpower requirements.
7. Develop operating cost estimates including direct cost consisting of operating labor, operating supplies, maintenance labor maintenance supplies, and indirect cost consisting of supervision and general pit expense."

The above written instructions were augmented from time to time by verbal discussions. We were instructed to determine the economic effects of an



increment in pit size. We were also instructed to assume that at least 5,000,000 tons of ore, with a grade equal to the average of the Faro Number One deposit, could be obtained from the Faro Number Two and Three deposits.



SECTION 4C

SCOPE OF THE WORK

In accordance with Anvil's instructions we have performed the following work:

1. Prepared pit plans and sections for the Faro Number One deposit showing:
 - (a) The Preproduction Pit
 - (b) The Five-Year Pit
 - (c) A Pit which would yield approximately 41,000,000 tons of ore
 - (d) An Ultimate Pit which would yield approximately 47,000,000 tons of ore.
2. Prepared a plan showing overburden spoil areas with haulage roads to these areas and to the crusher.
3. Developed quantities of ore and overburden and grade of ore to correspond to the various pit designs.
4. From the drill hole information prepared cross-sections of the ore-body and developed the grade and tonnage of each mining bench.
5. Compiled a list of the major mining equipment required showing the landed and erected cost of the equipment.
6. Prepared operating cost tables and manpower tables for each phase of the mining operation.
7. Developed productivity tables for the major equipment.
8. Prepared basic specifications for all major equipment.



SECTION 4D

DESIGN CRITERIA

The following design criteria were furnished by the client:

1. Staff and hourly paid employees salary and wage rates, hours of work, and the basis for overtime payment. In this regard a 44-hour work week was taken as basic, but the work week is based on 6-8 hour days so that for hourly personnel, payment is for 50 hours per week.
2. The landed cost of diesel oil and gasoline.
3. The specific gravities and densities of alluvium, cap rock and ore.
4. Fifteen percent of alluvium to be considered as cap rock.
5. Minimum temperature -70°F .
6. Pit Criteria - Alluvium slope, to be 1-1/2 to 1 for working and final slopes. Working bench width sufficient for annual stripping and mining requirement, suggested dimensions are: 60' berm, 12' toe and a maximum bench height of 35'. Road widths: 60' with 7% maximum grade. The final pit slope to average 40° .
7. Preproduction Stripping - Develop enough ore for one year's milling.
8. After Production Starts - Each expansion should be sufficient for one year's milling.
9. Alluvium Stripping - Assume twin motor scrapers (e.g. Euclid TS-24) will be used. Give consideration to other methods.
10. For Stripping Cap Rock and Mining Ore - That combination of shovels and trucks which will minimize costs taking into consideration the maximum degree of standardization possible. All mining and stripping operations will be on a year-round basis, 24 hours per day for stripping, and 8 hours per day for mining, working 6 days per week on each.
11. Drilling and Blasting - Assume the use of Bucyrus-Erie Model 40R drills. If this class of drill can be obtained with angle drilling capability, use the angle drill. Holes will be 9" diameter with 18' burden and 22' spacing. Ammonia nitrate/fuel oil mixture will be used for blasting. For secondary drilling, use a rubber tired unit carrying compressor and boom mounted drill with full cab controls e.g., "Travelldrill".

Production Factors

The following figures were used in calculating scraper and truck capacities:



Production Factors (continued)

1. Alluvium

Density of Alluvium	1.68 Tons/bank c.y.
Swell Factor (S.F.)	25%
Dipper Factor (D.F.) if applicable	95%
Dipper Capacity (D.C.) if applicable	According to shovel size
Scraper Capacity (24 c.y. struck)	27 c.y. heaped, loose
Scraper Capacity (bank c.y.)	$= 27 \times \frac{1}{1.25}$
	$= 27 \times .8 = 21.6/\text{bank c.y.}$

If shovel is used then

Dipper Capacity = Dipper size X .8 (S.F.) X .95 (D.F.) bank c.y. e.g., 5 c.y. dipper	$= 5 \times .8 \times .95$
	$= 3.8 \text{ bank c.y.}$
	$= 3.8 \times 1.68$
	$= 6.38 \text{ tons}$
	use 6.40 tons/5 c.y. dipper

2. Cap Rock

Density (S.G. 2.75)	2.3 tons/bank c.y.
Swell Factor (S.F.)	50%
Dipper Factor (D.F.)	85%
One Broken c.y.	$= 2.3 \times .667$
	$= 1.53 \text{ Tons}$
Dipper Capacity - Dipper size X 0.667 (S.F.) X .85 (D.F.) bank c.y. e.g., for a 5 c.y. dipper	$= 5 \times .667 \times .85$
	$= 2.83 \text{ bank c.y.}$
	$= 2.83 \times 2.3 \text{ Tons}$
	$= 6.5 \text{ Tons/5 c.y. dipper}$

3. Ore

Density (S.G. = 3.76)	3.18 tons/bank c.y.
or 8.5 bank c.f.	$= 1 \text{ ton}$
Swell Factor (S.F.)	50%
Dipper Factor (D.F.)	85%
One Broken c.y.	$= 3.18 \times .667$
	$= 2.12 \text{ tons}$
Dipper Capacity - Dipper size X 2.12 X .85 (D.F.) tons e.g., a 5 c.y. dipper	$= 5 \times 2.12 \times .85 \text{ tons}$
	$= 9.01 \text{ tons}$
	use 9.0 tons



Cost Data

Operating costs are in terms of Canadian currency, only the Capital Cost of mining equipment is expressed in U. S. currency. Diesel oil and gasoline delivered to the site have been assumed to cost \$0.32 and \$0.33 per imperial gallon respectively.

The cost of ammonia nitrate, delivered to storage at the mine has been taken to be \$0.06 per pound.

The cost of all labor and supplies are in Canadian currency.



SECTION 4E

OVERBURDEN AND OREBODY CHARACTERISTICS

Description of the Orebody

The Faro Zone No. 1 orebody consists of a tabular massive sulfide body which strikes roughly east and west and plunges south 60° east. About 2400 feet to the southeast along its plunge, it passes under Faro Creek, and is crossed by a northeast trending fault. East of the fault zone the ore bed is as much as two hundred feet deeper.

The original discovery drilling was with 3-1/2 inch diameter rotary drill holes which indicated the massive sulfide zone, but proved unreliable in the exact delineation of the ore. The assay values reported from these holes, while indicative, were generally low. We have used the rotary holes only to help delineate the ore outlined in this report.

The orebody has been blocked out by diamond drilling on four hundred foot centers and partially filled in later on two hundred foot centers. Fifty-seven holes drilled in mineralization and 11 holes in waste have been reported to us.

Our efforts are concerned with those aspects of the orebody which will control the development of a mining program and the projection of capital and operation costs. Based on drill log data, we have estimated the tonnage and grade of ore on each 35 foot bench.

The Nature of the Ore

The drill logs indicate the ore minerals are principally sphalerite and silver bearing galena, along with massive pyrite, pyrrhotite, some chalcopyrite and magnetite. As oxidation of the sulphides near the upper part of the orebody is not mentioned in the drill logs, it is assumed that the mineralization is all sulphide.

~~The ore cut-off used was 5 percent combined lead and zinc; the silver and copper content were not considered in the cut-off criteria. Minor zones of ore within the cap rock, though eventually recoverable, were not included in the reserves. Where the drill hole information on a bench was incomplete, but where good ore was on the benches above and below, the intervening bench was included in calculating reserves and grade. Drill data does not permit closer delineation although careful mining is expected to enhance the ore grade.~~

Some voids and faulting in the east end of the orebody may account for the presence of small unmeasured flows of artesian water observed in this area. The proximity of the Faro Creek drainage may account for the source of this water.



Glacial Debris and Alluvium

The orebody lies on the south slope of the hills bordering Rose Creek about two miles north of the intersection of Faro and Rose Creeks. The surface is covered with glacial till and slopes southeasterly at an average grade of 20 percent. Most of the timber has been cleared from the area.

The glacial debris consists of 75% or more of clayey silt with rock fragments and minor rounded boulders up to 1' X 1-1/2' X 3'. Boulder proportion appears to increase with depth. The bedrock, where exposed by an experimental cut showed bedrock of oxidized quartz-sericite schist with steep bedding or schistosity that was easily cut with the D8 ripper. No perma-frost was reported in the area.

Volume of Materials

The calculation of the volume of materials was performed primarily to provide dimensions for the mining study rather than as a formal presentation of reserves. After trial, we determined that it would be most helpful if the ore tonnage and grade were developed for each 35 foot bench.

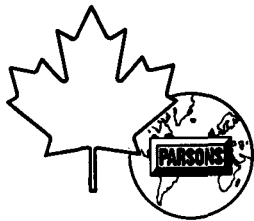
The specific gravity stipulated by the client was 1.68 for alluvium, and 2.3 for cap rock. Massive ore was 3.18 S.G. or 8.5 cubic feet per ton, but a factor of 9.5 cubic feet per ton was used where selective mining was indicated, that is, where ore and waste appeared to commingle.

Grade of Mill Feed

The grade of ore for the first 5-year period or 10,000,000 short tons was 4.9% Pb and 7.3% Zn, or 12.2% combined metal (after dilution). The tonnage and grade for the following 12 years of 36,000,000 short tons assaying 3.4% Pb and 5.9% Zn or 9.3% combined metal. The proposed mining plan is to proceed from west to east because of operating advantages and it is fortunate that a relatively large tonnage of the richer grade ore can be mined during the first five years.

The pit will be developed and equipped so that ore blending can be performed. In practice this means having at least two faces and two shovels available for mining at all times and requires that stripping be maintained one year ahead of mining.

In order to equalize concentrate production during the life of the mine, the mine tonnage of 2,000,000 tons per year will be increased, after the fifth year, to 3,000,000 tons per year as the grade of ore to the mill, subsequent to the fifth year of operation, will average about 9.3 percent combined metals. The increase in tonnage mined will not cause a corresponding increase in stripping because of a better ore to overburden ratio in the east end of the pit. However this situation is only true west of the Faro fault. East of the fault the stripping ratio increases. This subject is discussed in greater detail in Section F.

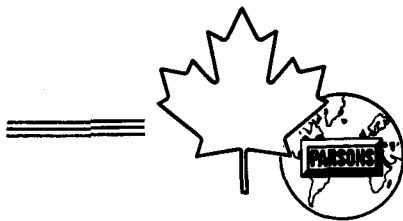


Ore Reserve Calculations

All drill hole logs were examined and weighted averages developed for each 35 foot bench. These calculations, with colored graphic depiction of the drill holes, have been assembled in notebook form. In addition to the client's sections, 25 north-south sections and east-west sections were prepared. These are colored sections and show the grade of ore for each bench.

From the above data, each mining bench was developed. Weighted averages were used to calculate the grade for each bench. Bench areas were determined by means of a planimeter and the tonnage calculated. Ore reserves were obtained, on a weighted basis, by the summation of the individual benches.

Fifty colored sections and bench plans as well as the colored drill hole logs will be delivered to the client. These drawings have not been reproduced and will not appear in this report.



SECTION 4F

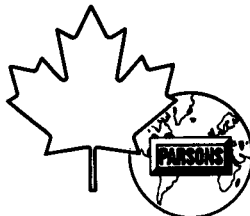
MINING PLAN

General

Some of the highest grade ore occurs in the west one-third of the orebody. Because of the advantages of mining this ore during the tax-free period, this area was selected as the initial mining area. Subsequently, other approaches were examined, but the west end continued to show advantages because of its shallow cover, proximity to the cap rock spoil area and the likelihood of dry conditions because of its higher elevation.

Total reserves which could be mined by open pit methods are estimated to be in the order of 47,000,000 tons, which is sufficient for 17 years of operation at the projected tonnage. However, after about 41,000,000 tons of ore has been uncovered, the stripping ratio becomes much more unfavorable. *optimum life 15 yrs* A pit design which would yield 41,000,000 tons of ore by removing about 62,000,000 c.y. of overburden appears to be the best from a point of economic return. The overall stripping ratio (cubic yards of overburden removed to uncover one ton of ore) for this design is 1.5 to 1. However, in order to obtain an additional 6,000,000 tons it is necessary to remove approximately 26,000,000 c.y. of overburden making the stripping ratio 4.33 to 1 for this (6,000,000 tons) quantity of ore.

Rather than extend the Faro Number One pit to the point where such a high stripping ratio would be encountered (plus the long ore haulage from a deep pit), it would be more economical to obtain 5,000,000 tons from the Faro Number Two and Three deposits. While these deposits have not been completely delineated by drilling, the drilling which has been done shows shallower overburden than the average depth of the overburden on the Faro Number One deposit. Because the information is not as comprehensive for the Faro Numbers Two and Three deposits as it is for the Faro Number One deposit, we have been conservative in our estimates. The stripping ratio at the Faro Number One as stated above, is calculated to be one and one-half cubic yards of overburden for each ton of ore mined from the pit. We have assumed that the stripping ratio at the Faro Numbers Two and Three deposits will be two to one. Therefore, in order to obtain 5,000,000 tons of ore from these deposits it would be necessary to remove approximately 10,000,000 c.y. of overburden. Total stripping in order to make available 46,000,000 tons of ore would consist of 62,000,000 c.y. from the Faro Number One deposit plus 10,000,000 c.y. from the Faro Numbers Two and Three deposits for a total of 72,000,000 c.y. Since the Faro Number One deposit can produce 41,000,000 tons of ore at favorable stripping ratios, and since this amount of ore is sufficient for 15 years of operation at the projected milling requirements, an intensive investigation of where the ore will be obtained after the 15th year of operation does not seem warranted at this time. The evidence on hand shows that an additional 5,000,000 tons of ore can be obtained from the Faro Number One deposit after the extraction of 41,000,000 tons of ore; or 5,000,000 tons of ore can be obtained from the



Faro Number Two and Three deposits. Circumstances 12 to 15 years hence will dictate where the ore can be obtained most economically after 41,000,000 tons of ore have been mined from the Faro Number One deposit.

Based on the criteria given in Section 4E, Volume of Material, Table 4-2 was prepared. It may be summarized as follows:

<u>Period</u>	<u>Alluvium(c.y.)</u>	<u>Cap Rock(c.y.)</u>	<u>Ore (Tons)</u>	<u>Ore Grade Pb + Zn</u>
Preproduction	2,763,000	3,777,000	607,000	11.3%
1 to 5 Years (per Yr.)	977,000	2,576,000	2,000,000	12.2%
6 to 16 Years (Per Yr.)	577,000	3,763,000	3,000,000	9.3%

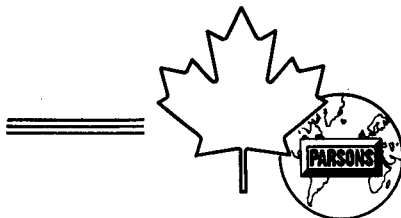
(Note: It is assumed that there would be no stripping operation in the 17th year).

Our studies of equipment led us to the conclusion that 5 c.y. shovels would be the most efficient unit for duty in ore and cap rock and in alluvium if ever required for this service. Dipper sticks and booms of T-1 steel are available for this size shovel. T-1 steel retains a large degree of its strength at low temperatures. In addition, heating strips (e.g. Calrod) and other cold weather protection devices will be incorporated in the shovel design. Electrical power was selected because of efficiency and easier starting after standing in cold weather. However, one diesel shovel has been included to provide flexibility for rapid moves, for operation where there are no power facilities and for use before power facilities (other than temporary or portable) are available. Sixty-five ton diesel ore and waste trucks were selected to match the shovel size. The units selected will have heated bodies, cold weather starting and other winterizing accessories.

Studies were made to determine the productivity of the shovels and trucks in each of the materials hauled. These studies are summarized in the attached tables. Also included are the equipment specifications and recommended extras.

The mining plan proposed is to have two five-yard shovels excavating cap rock, with scrapers removing the alluvium. The spoil roads west of the pit are shown on Drawing 3746-3AA400-4. Cap rock stripping will be a three shift operation with two shovels until the end of the 16th year of pit operation. Ore production will be with one shovel, one shift per day, six days per week, 52 weeks per year during the first five years. Two ore mining shovels are being provided to provide for blending capabilities and to ensure uninterrupted production. After the first five years of mining, two shovels on one shift will be required.

The location of the overburden dumps are shown on Drawing 3746-3AA400-4. The initial alluvium dump has been located adjacent to the cap rock dump. Further drilling may reveal an area south of the mine suitable for alluvium disposal. This will reduce the distance to the dump but until further drilling has been completed we are assuming that the alluvium will have to



be hauled to the west side of the pit. Cap rock for the first five years will be hauled over stripping roads following the 4,350, 4,275 and 4,200 contours west of the pit. After the fifth year, the dump will be built along the 4,150 and 4,000 contours. Dumps are located west of any extensions of the ore body.

Since the pit will be completed as the mining progresses from the west end, it will be possible to build permanent access ramps on the pit walls and floor. These ramps are shown on Drawings 3746-3AA400-3 and 3746-3AA400-5.

The quantity of water to be handled during the mining operations cannot be predetermined. It seems reasonable to assume that the greater part of the subsurface water is associated with Faro Creek and that this flow of water can be reduced by intercepting Faro Creek upstream from the pit and directing it around the eastern edge of the pit. As the alluvium is stripped, seepage can be intercepted at the safety berm and diverted away from the pit. Also, because the upper waste benches are "daylighted" to the east, these benches will tend to intercept and divert surface and subsurface water. Because of the superior height of the west end of the pit over the Faro Creek channel, the drainage plan can be developed after the start of the mining based on better knowledge of field conditions.

Open pit operations would commence by stripping the alluvium on the west end of the pit using scrapers which can be rapidly mobilized. Alluvium stripping will be a year-round operation. The stripping equipment, in all probability, would be used for road building, drainage, and miscellaneous plant site grading operations.

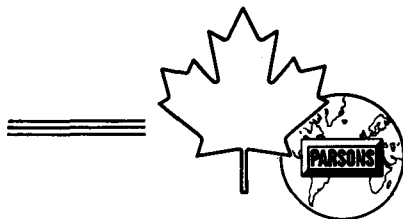
Haulage Roads

Roads will be built using selected rock from the cap rock stripping for the base course and gravel from local pits for the wear surface. Nearly all roads are located on alluvium and on side hills which will allow the construction of suitable drainage structures to keep the subgrade and base course dry.

The main haul roads should be built, to resist to the greatest extent possible, the destructive effects of repeated thawing and freezing. It is recommended that the roads be built in three courses: sub-base, base, and surfacing.

The sub-base course should be laid after the surface of the existing ground has been scraped and cleared of all unsuitable material. The surface will be scarified and compacted to at least 95 percent modified ASSHO specifications. The greatest care must be exercised in preparing for the sub-base.

The sub-base course (selected crushed and screened cap rock) is to be laid in six to ten-inch layers, depending on the type of compaction equipment used and on the size of the rock. The sub-base should be carried a mini-



mum of eighteen inches above the existing grade. There will be no sub-base required when base course can be laid on rock or ore.

A base course five feet thick is to be constructed using selected cap rock material. The first two feet being crushed and screened material from six inches to two and one-half inches. The top three feet to be crushed, screened and graded selected caprock from two and one-half inches to one-half inch in size. This material to be thoroughly compacted and care must be exercised to keep fines out of the base course. Keeping it porous will help to prevent frost "boils".

Road surfacing would consist of six inches of crushed selected cap rock graded from one inch to one-quarter of an inch, and should be treated with a dust suppressing material as required. Detailed road construction specifications applicable to Yukon conditions should be developed prior to commencement of road construction.

Culvert drain pipe and ditches will be required to keep the ground water level at least six inches below the bottom of the base course.

Drilling and Blasting

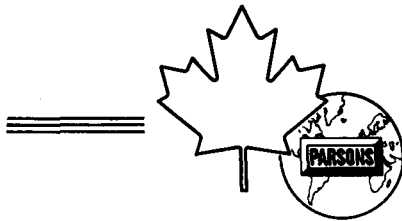
Preliminary stripping tests indicate that the alluvium can be stripped without blasting and with a minimum amount of ripping. Boulders of varying size are reported, some may require blasting but most of them can be removed with the scrapers. All of the alluvium stripping appears to be within the capabilities of scrapers.

The cap rock cores we have examined can be drilled either by rotary or down-the-hole drills. The rock is generally metamorphosed and varies from soft to moderately hard. We have assumed that it can be adequately broken in 35 feet benches with AN/FO explosives with a powder factor of 0.5 lbs./c.y. Some secondary blasting will be required and explosives for this purpose will cost an estimated \$0.01/c.y. in both cap rock and ore. However in our operating cost tables the cost of explosives is not distributed between primary and secondary blasting.

The diamond drilling experience indicates that large sections of the ore consist of dense, massive sulphides. We have assumed that AN/FO consumption will also be 0.5 lbs./c.y. or 0.16 lbs./ton. Three Bucyrus-Erie Model 40R drills have been provided to drill cap rock and ore.

Production

The major mining and stripping equipment consists of one 5 c.y. diesel shovel and three 5 c.y. heavy duty electric shovels loading into 65 ton diesel trucks. Our calculations are based on a 50 minute operating hour, seven productive hours per shift, and 52 weeks of 6 days per week per year. We have used a 75 percent mechanical availability factor for trucks.



The above factors were used to determine the number of equipment hours, the number of units and the cost per equipment operating hours. The hourly operating cost estimates have been summarized in tables which follow. Additional tables show the calculations which establish truck and shovel productivity. These studies were extended through the life of the mine taking into consideration variations in haulage distances for ore and overburden.

Support Equipment

D8 bulldozers were selected for shovel clean-up, spoil bank levelling and general pit service. The bulldozers specifications provide a straight blade, a ripper and cold weather operating accessories.

A Caterpillar No. 16 grader, equipped with a 14 foot blade, scarifier teeth and cold weather equipment was selected for road maintenance.

Pumping Equipment

We have assumed that a pumping rate of 1,000 gpm will handle the volume of seepage water encountered during the stripping and mining operations. Power to the pumps will be supplied by cable from a portable power and control center containing a transformer, breakers and starters with enclosure and heating for cold weather operation.

General Pit Expense

The general pit expense consists of the operation of road maintenance equipment as well as other units such as pick-up trucks, buses, winch trucks, etcetera, which are assigned to pit operations.

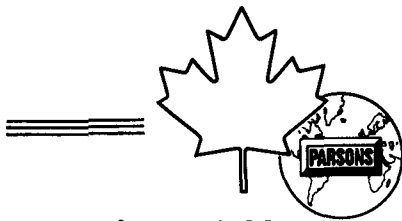
In all equipment items, operating and repair labor is included in the operating cost of the equipment under the headings Operating and Repair Labor. A roster of the men required, their pay scale and payroll overhead is included in the tables. Manpower tables have been prepared to correspond to various stages of pit development.

The hourly payroll roster includes a sufficient number of employees to operate all equipment units on the three shift operation proposed for stripping. The salary and wage rates used have been supplied by the client's consultant.

Pumping costs are a part of general pit expense. We have not allocated a separate charge for pumping, but on the assumption that the pumps we have recommended will keep the mine dry, we believe that there is enough money (in equipment and labor costs) in the general pit expense to provide for the cost of pumping.

Electrical Power Distribution

Power for electric shovels, repair equipment and lighting will be provided



by portable power and control centers containing the following:

- Incoming HV disconnect switch.
- High voltage transformer for repair equipment and lighting.
- High and low voltage breakers.
- Lighting panel.

Power and control centers to be suitably enclosed, grounded and heated for cold weather operation.

Service Building

Service facilities are covered in another section of this report, but in general, a maintenance building has been planned to provide for the servicing and repair operations. This building is to be located in the mill area and will provide service facilities for vehicles used in the mill area as well as for the mining equipment.

The maintenance shop has sufficient area to handle seven dump trucks, or equivalent at one time. Facilities are provided for servicing and major repair, tire changing and repair, lubrications, steam cleaning and painting. In addition to the usual shop tools, the following are recommended:

1. Complete tool sets for servicing and major overhaul of every piece of equipment.
2. Test stands for engine overhaul.
3. Tire vulcanizing and recapping equipment.
4. Overhead traveling bridge crane.

The parking garage will accommodate all dump trucks and service trucks. This area can also be used for parking of tractors and motor graders.

An electrical shop is provided and is equipped for repair and testing of electrical equipment.

A warehouse and storage area for all equipment parts and supplies, tires and steel stock will be provided.

All office, change room and working area are expected to be heated to normal room temperatures.

Indoor parking, storage and warehouse areas should be heated and insulated so as to maintain a temperature above freezing.



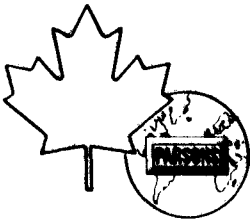
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Table 4-1

SUMMARY
MINING AND STRIPPING COSTS
ANVIL PROJECT

	STRIPPING (c.y.)		MINING (tons)			TOTAL COST
	Alluvium	Cap Rock	Ore	General Pit	Total	Ore to Mill
* Preproduction	\$0.23	\$0.44	\$0.20			
1-5 Years	0.18	0.43	0.17	\$0.16	\$0.33	\$0.88/ton
6-9 Years	0.25	0.45	0.19	0.14	0.33	0.93/ton
9-16 Years	0.25	0.45	0.19	0.14	0.33	0.95/ton
17th Year			0.19	.08	0.27	0.27/ton

* General Pit Expense prorated

Table 4-2

VOLUME OF MATERIALS
SUMMARY
ANVIL MINING COMPANY

<u>PERIOD</u>	<u>STRIPPING (c.y.)</u>			<u>MINING (tons)</u>	<u>TO MILL</u>	
	<u>Alluvium (2)</u>	<u>Cap Rock</u>	<u>Total</u>	<u>To Stockpile</u>	<u>Tons</u>	<u>Grade (4)</u>
Preproduction	2,763,000	3,777,000	6,540,000	392,000 (H.G.) 215,000 (L.G.)		
1-5 Years	<u>4,887,000</u>	<u>12,873,000</u>	<u>17,760,000</u>	<u>2,183,000</u> (L.G.)	<u>10,000,000</u>	12.2%
Sub Total	7,650,000	16,650,000	24,300,000	2,398,000 (L.G.) (3)	10,000,000 (1)	
6-16 Years	<u>6,350,000</u>	<u>41,400,000</u>	<u>47,750,000</u>			
Total Stripping	14,000,000	58,050,000	72,050,000			
6-17 Years Ore Milled					36,000,000	9.3%
From Stockpile					- <u>2,398,000</u>	
Ore Mined					33,602,000	
1-17 Years TOTAL ORE MINED					46,000,000	9.8%

(1) 9,608,000 Tons Mined plus 392,000 From Stockpile = 10,000,000 T
Add Stockpile 2,398,000 T

TOTAL ORE MINED (Preproduction + 1-5 Years) 12,398,000 T

(2) 15% Deducted from Alluvium and Added to Cap Rock.

(3) Stockpile Reclaimed During Year 6, 7, 8, and 9 at Rate of 600,000 T.P.Y.

(4) Per Cent Combined Lead and Zinc.

H.G. = Ore Containing 9% More Combined Metal. (Average Grade 11.4% Pb and Zn)

L.G. = Ore Containing Less Than 9% Combined Metal. 4-T2
(Average Grade 7.9% Pb and Zn)



Table 4-3

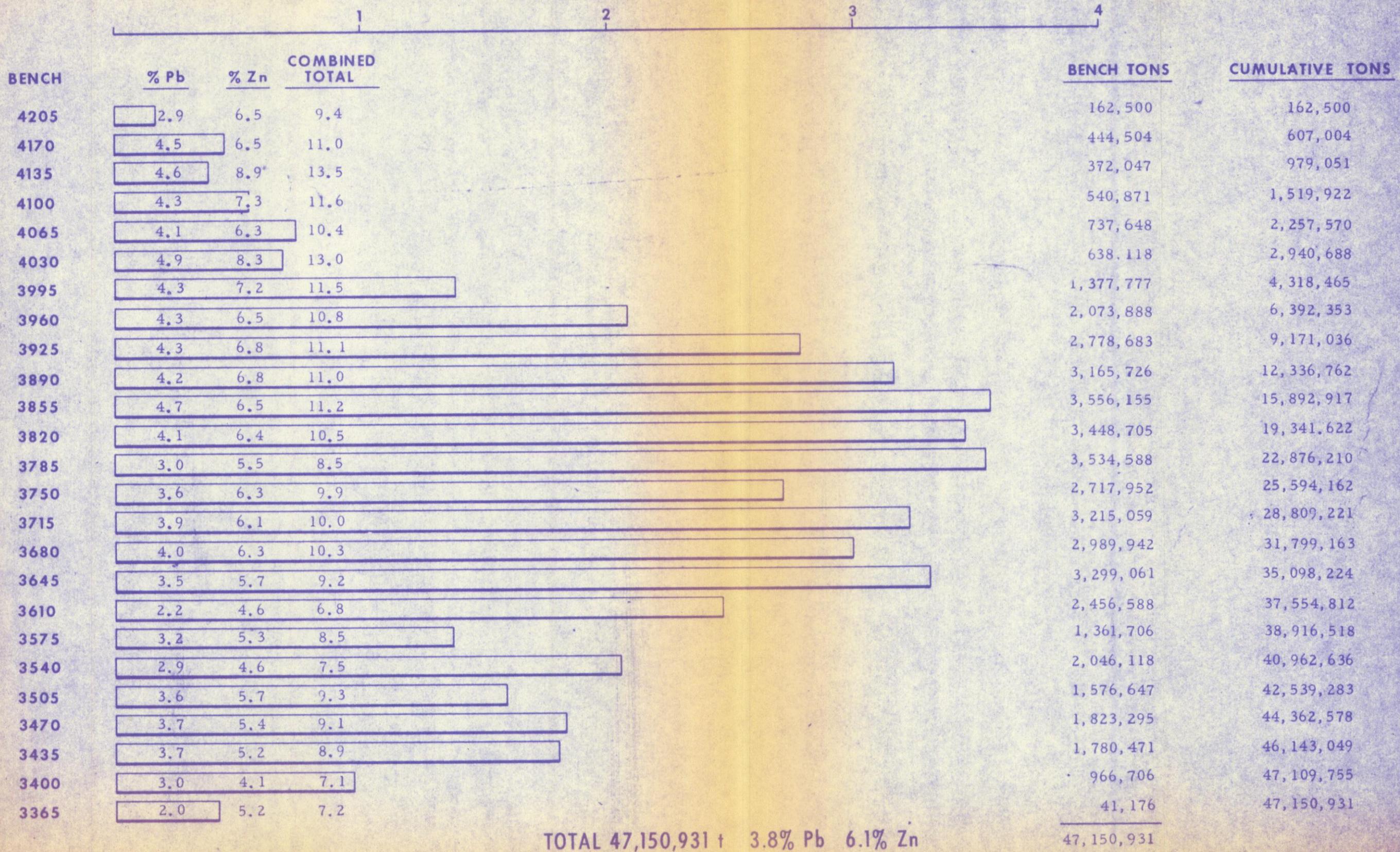
ORE RESERVES
BY BENCH
ANVIL PROJECT

5.0% Combined Pb-Zn = Cut-Off Grade

<u>BENCH</u>	<u>TONS</u>	<u>% Pb</u>	<u>% Zn</u>	<u>COMBINED Pb - Zn</u>
4205	162,500	2.9	6.5	9.4
4170	444,504	4.5	6.5	11.0
4135	372,047	4.6	8.9	13.5
4100	540,871	4.3	7.3	11.6
4065	737,648	4.1	6.3	10.4
4030	683,118	4.9	8.3	13.2
3995	1,377,777	4.3	7.2	11.5
3960	2,073,888	4.3	6.5	10.8
3925	2,778,683	4.3	6.8	11.1
3890	3,165,726	4.2	6.8	11.0
3855	3,556,155	4.7	6.5	11.2
3820	3,448,705	4.1	6.4	10.5
3785	3,534,588	3.0	5.5	8.5
3750	2,717,952	3.6	6.3	9.9
3715	3,215,059	3.9	6.1	10.0
3680	2,989,942	4.0	6.3	10.3
3645	3,299,061	3.5	5.7	9.2
3610	2,456,588	2.2	4.6	6.8
3575	1,361,706	3.2	5.3	8.5
3540	2,046,118	2.9	4.6	7.5
3505	1,576,647	3.6	5.7	9.3
3470	1,823,295	3.7	5.4	9.1
3435	1,780,471	3.7	5.2	8.9
3400	966,706	3.0	4.1	7.1
3365	41,176	2.0	5.2	7.2
GRAND TOTAL	47,150,931	3.8	6.1	9.9

ANVIL PROJECT
TONNAGE AND GRADE DISTRIBUTION BY BENCHES
FARO #1 OREBODY

MILLIONS OF SHORT TONS



STRIPPING AND MINING EQUIPMENT
CAPITAL COST DELIVERED TO SITE
ANVIL PROJECT

	Units	Pound Weight		Equipment Cost U.S.A. Per Unit	Total Equipment Cost U.S.A.	Freight to Dawson Creek Per Unit	Freight to Jobsite	Total Freight Per Unit	Total Freight	Tariff Per Unit	Total Tariff	Erection Cost Each Unit	Total Cost Each Unit	Total Cost All Units
88 Bucyrus 5 Yard Electric Shovels	(3)	273,500	(1)	\$221,600	\$664,800	\$10,900	\$8,200	\$19,100	\$57,300	Free	Free	\$5,700	\$246,400	\$739,200
88 Bucyrus 5 Yard Diesel Shovel	(1)	274,000	(1)	224,200	224,200	11,000	8,300	19,300	19,300	Free	Free	5,700	249,200	249,200
61 Bucyrus 3 Yard Electric Shovel	(1)	116,000	(1)	130,200	130,200	5,000	6,600	11,600	11,600	Free	Free	4,000	145,800	145,800
D8 Caterpillar Dozers	(4)	76,800	(1)	61,100	244,400	3,100	1,500	4,600	18,400	Free	Free		65,700	262,800
Haulpak 65 Ton Trucks	(15)	77,500	(2)	101,400	1,521,000	3,100	200*	3,300	49,500	7,600	114,000		112,300	1,684,500
Bucyrus 40R Rotary Drill	(3)	88,000	(1)	115,900	347,700	3,500	1,800	5,300	15,900	8,700	26,100		129,900	389,700
Athey Travel Drill Sniper	(1)	23,800	(1)	41,500	41,500	1,000	500	1,500	1,500	2,900	2,900		45,900	45,900
TS-24 Rubber Tired Scrapers	(1)	87,200	(1)	94,300	94,300	3,500	500*	4,000	4,000	7,000	7,000		105,300	105,300
Cat 16 Motor Grader	(1)	54,000	(1)	55,900	55,900	2,200	1,100	3,300	3,300	4,200	4,200		63,400	63,400
Powder Truck Bulk Chevrolet	(1)	12,000		10,800	10,800	500	200*	700	700	1,500	1,500		13,000	13,000
Water Truck 2,500 Gal. Chevrolet	(1)	11,500		6,200	6,200	500	200*	700	700	1,100	1,100		8,000	8,000
Grease Truck Lubrivan Chevrolet	(1)	15,000		11,200	11,200	600	200*	800	800	2,000	2,000		14,000	14,000
Winch Truck Tandem H.D. Chevrolet	(1)	15,000		9,500	9,500	600	200*	800	800	1,500	1,500		11,800	11,800
Transport Bus 44 Passenger Chev.	(1)	15,000		14,600	14,600	600	200*	800	800	2,600	2,600		18,000	18,000
4 W.D. Station Wagon Chevrolet	(1)	4,500		3,600	3,600	200	200*	400	400	600	600		4,600	4,600
4 W.D. Pickup 1/2 Ton Chevrolet	(3)	4,200		2,900	8,700	200	200*	400	1,200	500	1,500		3,800	11,400
Power Center Bkrs. Transformers														
1200 HP 15,000/4160V Transformer	(1)	10,000		26,300	26,300	400	100**	500	500	2,000	2,000		28,800	28,800
#6 Type SHD 4160V Power Cable 60% Rubber	(3,000')	3,000		9,500	9,500	200	100**	300	300	700	700		10,500	10,500
PLM Type Cable Connectors 4160V	(3-sets)	500		500	1,500	100	100**	200	600	100	300		800	2,400
5-K.W. Light Plants Diesel	(5)	(500 ea.)		3,700	18,500	100	100**	200	1,000	500	2,500		4,400	22,000

Total Equipment Cost

\$3,444,400

\$188,600

\$170,500

\$26,800

\$3,830,300

(1) List Price less 5%

(2) List Price less 7%

(*) Drive Unit to Jobsite

(**) Haul Unit to Jobsite (handling)

MINING EQUIPMENT
ESTIMATE HOURLY OPERATING COST
ANVIL PROJECT

	5 C.Y. Diesel Shovel	5 C.Y. Electric Shovel	3 C.Y. Electric Shovel	D8 Cat Push Dozer	D8 Cat Cleanup Dozer	Cat 16 Motor Grader	65 Ton Diesel Truck	B/E 40R Rotary Drill	Athey Travel Drill	Euclid TS-24 Scraper	Cat 619 Scraper	Water Truck	Grease Truck	Winch Truck	4 W.D. Bus 44 Pass.	4 W.D. Station Wagon	4 W.D. Pickup 1/2 T.	Bulk Powder Truck	* 988 Cat. Front Loader
LABOR																			
Operating	\$8.23	\$8.23	\$8.23	\$3.85	\$3.66	\$3.66	\$3.85	\$7.23	\$7.23	\$3.85	\$3.85	\$3.69	\$3.74					\$7.58	\$2.96
Repair	5.50	4.50	4.00	4.00	4.00	1.75	3.50	3.00	2.50	2.75	2.55	0.25	1.00	\$0.50	\$0.30	\$0.25	\$0.25	0.35	2.00
Total Labor	\$13.73	\$12.73	\$12.23	\$7.85	\$7.66	\$5.41	\$7.35	\$10.23	\$9.73	\$6.60	\$6.40	\$3.94	\$4.74	\$0.50	\$0.30	\$0.25	\$0.25	\$7.93	\$4.96
OPERATING SUPPLIES																			
Bit Cost								\$10.00	\$1.00										
Rod Cost								.06	1.57										
Fuel	\$3.38			\$3.12	\$3.12	\$0.90	\$3.90	2.60	1.30	\$5.20	\$2.60	\$1.55	\$1.24	\$1.24	\$1.02	\$0.93	\$0.93	\$1.24	\$2.34
Tires						1.00	2.80		0.80	1.95	1.89	0.16	0.18	0.18	0.10	0.12	0.12	0.16	2.63
Electric Power		\$3.20	\$1.49																
Oil	0.30	0.01	0.01	0.16	0.16	0.16	0.30	0.20	0.16	0.25	0.20	0.03	0.03	0.03	0.02	0.03	0.03	0.03	0.21
Grease	0.31	0.31	0.30	0.02	0.02	0.02	0.13	0.25	0.12	0.15	0.15	0.04	0.04	0.04	0.01	0.03	0.03	0.03	0.02
Hydraulic				0.08	0.08	0.04	0.12	0.12	0.10	0.10	0.10		0.01					0.02	0.10
Filters	0.08			0.06	0.06	0.02	0.05	0.02	0.02	0.09	0.06	0.02	0.02	0.02	0.01	0.02	0.02	0.02	0.02
Cable	0.87	0.87	0.67					0.15	0.10					0.05					
Wear Metal	0.65	0.65	0.48	0.75	0.75	0.30	0.40	0.30	0.32	0.50	0.50	0.20	0.25	0.25	0.20	0.10	0.10	0.25	0.30
TOTAL																			
Operating Supplies	\$5.59	\$5.04	\$2.95	\$4.19	\$4.19	\$2.44	\$7.70	\$13.70	\$5.49	\$8.24	\$5.50	\$2.00	\$1.77	\$1.81	\$1.36	\$1.23	\$1.23	\$1.75	\$5.62
Repair Parts	9.08	6.83	6.50	4.85	4.85	2.85	5.15	3.75	3.50	3.75	3.25	0.30	0.50	0.35	0.35	0.25	0.12	0.50	2.25
TOTAL COST	\$28.40	\$24.60	\$21.68	\$16.89	\$16.70	\$10.70	\$20.20	\$27.68	\$18.72	\$18.59	\$15.15	\$6.24	\$7.01	\$2.66	\$2.01	\$1.73	\$1.60	\$10.18	\$12.83

*Estimating Purposes Only

ANNUAL
MINE OPERATING COST
PREPRODUCTION STRIPPING
ANVIL PROJECT

	Number Units Equipment	Hours of Operation	Operate	Labor Repair	Operate	Supplies Repair	Other Costs	Total	Unit Cost
<u>Alluvium (2,762,500 cubic yards)</u>									
619 & TS-24 Scrapers	3	18,134	\$ 69,800	\$ 47,400	\$116,300	\$ 62,000		\$295,500	
D8 Caterpillar Dozers	2	12,090	46,500	48,400	50,700	58,600		204,200	
Total Alluvium			\$116,300	\$ 95,800	\$167,000	\$120,600			
Prorated General Pit Expense								\$499,700	\$0.181
Total Cost of Removing Alluvium								134,000	0.049
									\$0.230/c.y.
<u>Caprock (3,777,500 cubic yards)</u>									
88B Shovel Operations	2	12,467	\$102,600	\$ 56,100	\$ 62,800	\$ 85,100		\$306,600	
Truck Hauling 65 Ton	5	26,955	103,800	94,300	207,600	138,800		544,500	
D8 Caterpillar Dozers	2	12,467	48,000	49,900	52,200	60,500		210,600	
B4OR Rotary Drill	2	7,349	53,100	22,000	100,700	27,600		203,400	
Travil Drill Athey Secondary	5	1,926	13,900	4,800	9,200	6,700		34,600	
Explosives		1,560	11,800	500	151,100	800		164,200	
Total Caprock			\$333,200	\$227,600	\$583,600	\$319,500			
Prorated General Pit Expense								\$1,463,900	\$0.388
Total Cost of Removing Caprock								183,000	0.049
									\$0.437/c.y.
<u>Ore (607,000 tons)</u>									
88B Shovel Operations	1	631	\$ 5,200	\$ 3,500	\$ 3,500	\$ 5,700		\$ 17,900	
Truck Hauling 65 Tons	5	2,592	10,000	9,100	20,000	13,300		52,400	
D8 Caterpillar Dozers	1	631	2,400	2,500	2,600	3,100		10,600	
B4OR Rotary Drill	1	371	2,700	1,100	5,100	1,400		10,300	
Travil Drill Athey Secondary	5	258	1,900	600	1,200	900		4,600	
Explosives		624	4,700	200	12,100	300		17,300	
Total Ore			\$ 26,900	\$ 17,000	\$ 44,500	\$ 24,700			
Prorated General Pit Expense								\$113,100	\$0.186
Total Cost of Removing Ore								9,500	0.016
									\$0.202/ton
<u>Direct Pit Cost Total</u>			\$476,400	\$340,400	\$795,100	\$464,800			
<u>General Pit Expense</u>									
Road Maintenance			\$ 17,600	\$ 7,900	\$ 12,500	\$ 10,900		\$ 48,900	
General Mine Equipment			4,300	4,700	18,300	2,900		30,200	
General Labor			32,800					32,800	
Supervision						4,500		90,300	
Payroll Taxes and Benefits							\$ 85,800	124,300	
Total General Pit								\$326,500	
<u>Total Capital Cost Preproduction Stripping</u>								\$2,403,200	
General Pit Expense (equal to \$0.048 per/c.y. on total quantity stripped) has been prorated to Alluvium and Caprock based on their respective volumes.									

Table 4-7

ANNUAL MINE OPERATING COST (1 to 5 Years Inclusive) ANVIL PROJECT									
	Number Units Equipment	Hours of Operation	Operate	Labor Repair	Operate	Supplies Repair	Other Costs	Total	Unit Cost
<u>Alluvium (977,400 cubic yards)</u>									
619 & TS-24 Scrapers	3	7,971	\$ 30,700	\$ 20,900	\$ 51,100	\$ 27,300		\$130,000	
D8 Caterpillar Dozers	2	2,668	<u>10,300</u>	<u>10,700</u>	<u>11,200</u>	<u>12,900</u>		<u>45,100</u>	
Total Alluvium			\$ 41,000	\$ 31,600	\$ 62,300	\$ 40,200			\$175,100 \$0.179/c.y.
<u>Caprock (2,574,600 cubic yards)</u>									
88B Shovel Operations	2	8,497	\$ 69,900	\$ 38,200	\$ 42,800	\$ 58,000		\$208,900	
Truck Hauling 65 Ton	4	24,336	93,700	85,200	187,400	125,300		491,600	
D8 Caterpillar Dozers	2	8,497	32,700	34,000	35,600	41,200		143,500	
B4OR Rotary Drill	2	5,008	36,200	15,000	68,600	18,800		138,600	
Travil Drill Athey Secondary	5	1,092	7,900	2,700	5,200	3,800		19,600	
Explosives			<u>9,000</u>	<u>400</u>	<u>103,000</u>	<u>600</u>		<u>113,000</u>	
Total Caprock			\$249,400	\$175,500	\$442,600	\$247,700			\$1,115,200 \$0.433/c.y.
<u>Ore (2,358,200 tons)</u>									
88B Shovel Operations	2	2,449	\$ 20,200	\$ 11,300	\$ 12,500	\$ 17,300		\$ 61,300	
Truck Hauling 65 Ton	5	10,077	38,700	30,100	77,400	41,700		187,900	
D8 Caterpillar Dozers	2	2,449	9,400	9,800	10,300	11,900		41,400	
B4OR Rotary Drill	1	1,443	10,400	4,300	19,800	5,400		39,900	
Travil Drill Athey Secondary	5	1,092	7,900	2,700	5,200	3,800		19,600	
Explosives			<u>7,600</u>	<u>400</u>	<u>47,200</u>	<u>500</u>		<u>55,700</u>	
Total Ore			\$ 94,200	\$ 58,600	\$189,000	\$ 80,600			\$405,800 \$0.172/ton
<u>Direct Pit Cost Total</u>									
			\$384,600	\$265,700	\$693,900	\$368,500			
<u>General Pit Expense</u>									
Road Maintenance			\$ 24,800	\$ 11,400	\$ 17,400	\$ 16,600		\$ 70,200	
General Mine Equipment			13,100	7,000	22,000	4,000		46,100	
General Labor			32,800				\$ 32,800	32,800	
Supervision						4,500	119,400	123,900	
Payroll Taxes and Benefits							115,800	<u>115,800</u>	
Total General Pit Expenses									\$388,800 \$0.165/ton
Total Cost Dry Ore									\$2,084,900 \$0.884/ton
									Use \$0.88/ton

ANNUAL
MINE OPERATING COSTS
(6 to 9 Years Inclusive)
ANVIL PROJECT

	Number Units Equipment	Hours of Operation	Labor Operate	Repair	Supplies Operate	Repair	Other Costs	Total	Unit Cost
<u>Alluvium (577,273 cubic yards)</u>									
619 & TS-24 Scrapers	3	5,124	\$ 19,800	\$ 13,400	\$ 32,900	\$ 17,500		\$ 83,600	
D8 Caterpillar Dozers	2	3,416	<u>13,200</u>	<u>13,700</u>	<u>14,300</u>	<u>16,600</u>		<u>57,800</u>	
Total Alluvium			\$ 33,000	\$ 27,100	\$ 47,200	\$ 34,100		\$141,400	\$0.245/c.y.
<u>Caprock (3,763,636 cubic yards)</u>									
88B Shovel Operations	2	12,421	\$102,200	\$ 56,200	\$ 62,800	\$ 85,600		\$306,800	
Truck Hauling 65 Ton	7	39,936	153,800	139,800	307,500	205,700		806,800	
D8 Caterpillar Dozers	2	12,421	47,800	49,700	52,000	60,200		209,700	
B40R Rotary Drill	2	7,322	53,000	22,000	100,300	27,500		202,800	
Travil Drill Athey Secondary	5	1,092	7,900	2,700	5,200	3,800		19,600	
Explosives			<u>9,000</u>	<u>400</u>	<u>15,400</u>	<u>600</u>		<u>160,400</u>	
Total Caprock			\$373,700	\$270,800	\$678,200	\$383,400		\$1,706,100	\$0.453/c.y.
<u>Ore Tons (2,400,000 tons)</u>									
88B Shovel Operations	2	2,493	\$ 20,500	\$ 11,500	\$ 12,700	\$ 17,700		\$ 62,400	
Truck Hauling 65 Ton	6	12,308	47,400	43,100	94,800	63,400		248,700	
D8 Caterpillar Dozers	2	2,493	9,600	10,000	10,400	12,100		42,100	
B40R Rotary Drill	1	1,469	10,600	4,400	20,100	5,500		40,600	
Travil Drill Athey Secondary	5	1,092	7,900	2,700	5,200	3,800		19,600	
Explosives		1,000	<u>7,600</u>	<u>400</u>	<u>48,000</u>	<u>500</u>		<u>56,500</u>	
Total Ore Pit			\$103,600	\$ 72,100	\$191,200	\$103,000		\$469,900	\$0.195/ton
<u>Ore Stock Pile (600,000 tons)</u>									
61B Shovel Operations	1	1,038	\$ 8,500	\$ 4,200	\$ 3,100	\$ 6,700		\$ 22,500	
Truck Hauling 65 Ton	1	2,006	7,700	7,000	15,400	1,000		31,100	
D8 Caterpillar Dozer	1	1,038	<u>4,000</u>	<u>4,200</u>	<u>4,300</u>	<u>5,000</u>		<u>17,500</u>	
Total Cost Stockpile Ore			\$ 20,200	\$ 15,400	\$ 22,800	\$ 12,700		\$71,100	\$0.118/ton
Direct Pit Cost Total			\$530,500	\$385,400	\$939,400	\$533,200			
<u>General Pit Expense</u>									
Road Maintenance			\$ 24,800	\$ 11,400	\$ 17,400	\$ 16,600		\$ 70,200	
General Mine Equipment			13,100	7,000	22,000	4,000		46,100	
General Labor			32,800					32,800	
Supervision						4,500	\$119,400	123,900	
Payroll Taxes and Benefits							145,200	<u>145,200</u>	
Total General Pit Expense									\$418,200 \$0.140/ton
Total Cost Dry Ore								\$2,806,700	\$0.935/ton
									Use \$0.93/ton

Table 4-9

ANNUAL MINE OPERATING COST (10 to 16 Years Inclusive) ANVIL PROJECT									
	Number Units Equipment	Hours of Operation	Labor Operate	Repair	Supplies Operate	Repair	Total Costs	Total	Unit Cost
<u>Alluvium (577,273 cubic yards)</u>									
619 & TS-24 Scrapers	3	5,124	\$ 19,800	\$ 13,400	\$ 32,900	\$ 17,500		\$ 83,600	
D8 Caterpillar Dozers	2	3,416	<u>13,200</u>	<u>13,700</u>	<u>14,300</u>	<u>16,600</u>		<u>57,800</u>	
Total Alluvium			\$ 33,000	\$ 27,100	\$ 47,200	\$ 34,100		\$141,400	\$0.245/c.y.
<u>Caprock (3,763,636 cubic yards)</u>									
88B Shovel Operations	2	12,421	\$102,200	\$ 56,200	\$ 62,800	\$ 85,600		\$306,800	
Truck Hauling 65 Ton	7	39,936	<u>153,800</u>	<u>139,800</u>	<u>307,500</u>	<u>205,700</u>		<u>806,800</u>	
D8 Caterpillar Dozers	2	12,466	<u>47,800</u>	<u>49,700</u>	<u>52,000</u>	<u>60,200</u>		<u>209,700</u>	
B4OR Rotary Drill	2	7,322	<u>53,000</u>	<u>22,000</u>	<u>100,300</u>	<u>27,500</u>		<u>202,800</u>	
Travil Drill Athey Secondary	5	1,092	<u>7,900</u>	<u>2,700</u>	<u>5,200</u>	<u>3,800</u>		<u>19,600</u>	
Explosives		1,000	<u>7,600</u>	<u>400</u>	<u>150,400</u>	<u>500</u>		<u>15,900</u>	
Total Caprock			\$372,300	\$270,800	\$678,200	\$383,300		\$1,704,600	\$0.453/c.y.
<u>Ore (3,000,000 tons)</u>									
88B Shovel Operations	2	3,116	\$ 25,700	\$ 14,900	\$ 16,200	\$ 13,400		\$ 70,200	
Truck Hauling 65 Ton	8	15,385	<u>59,200</u>	<u>53,800</u>	<u>118,500</u>	<u>79,200</u>		<u>310,700</u>	
D8 Caterpillar Dozers	2	3,116	<u>12,000</u>	<u>12,500</u>	<u>13,100</u>	<u>15,100</u>		<u>52,700</u>	
B4OR Rotary Drill	1	1,836	<u>13,300</u>	<u>5,500</u>	<u>25,200</u>	<u>6,900</u>		<u>50,900</u>	
Travil Drill Athey Secondary	5	1,092	<u>7,900</u>	<u>2,700</u>	<u>5,200</u>	<u>3,800</u>		<u>19,600</u>	
Explosives		1,184	<u>8,900</u>	<u>400</u>	<u>60,000</u>	<u>600</u>		<u>69,900</u>	
Total Ore			\$127,000	\$ 89,800	\$238,200	\$119,000		\$574,000	\$0.191/ton
<u>Direct Pit Cost Total</u>			<u>\$532,300</u>	<u>\$387,700</u>	<u>\$963,600</u>	<u>\$536,400</u>			
<u>General Pit Expense</u>									
Road Maintenance			\$ 24,800	\$ 11,400	\$ 17,400	\$ 16,600		\$ 70,200	
General Mine Equipment			<u>13,100</u>	<u>7,000</u>	<u>22,000</u>	<u>4,000</u>		<u>46,100</u>	
General Labor			<u>32,800</u>					<u>32,800</u>	
Supervision						4,500	\$119,400	<u>123,900</u>	
Payroll Taxes and Benefits							<u>145,600</u>	<u>145,600</u>	
Total General Pit Expense								\$418,600	\$0.140/ton
Total Cost Dry Ore								\$2,338,600	\$0.946/ton
								Use	\$0.95/ton

Table 4-10

ANNUAL MINE OPERATING COST (17 Year Inclusive) ANVIL PROJECT									
	Number Units Equipment	Hours of Operation	Operate	Labor Repair	Operate	Supplies Repair	Total Costs	Total	Unit Cost
<u>Ore (3,000,000 tons)</u>									
8B Shovel Operations	2	3,116	\$ 25,700	\$ 14,900	\$ 16,200	\$ 13,400		\$ 70,200	
Truck Hauling 65 Ton	8	15,385	59,200	53,800	118,500	79,200		310,700	
D8 Caterpillar Dozers	2	3,116	12,000	12,500	13,100	15,100		52,700	
B4OR Rotary Drill	1	1,836	13,300	5,500	25,200	6,900		50,900	
Travil Drill Athey Secondary	1	1,092	7,900	2,700	5,200	3,800		19,600	
Explosives		1,184	8,900	400	60,000	600		69,900	
Total Ore			\$127,000	\$ 89,800	\$238,200	\$119,000		\$574,000	\$0.191/ton
<u>Direct Pit Cost Total</u>									
<u>General Pit Expense</u>									
Road Maintenance			\$ 14,200	\$ 6,900	\$ 10,200	\$ 10,000		\$ 41,300	
General Mine Equipment			2,300	3,200	12,100	1,900		19,500	
General Labor			16,500					16,500	
Supervision						4,500	\$ 96,300	100,800	
Payroll Taxes and Benefits							52,200	52,200	
Total General Pit Expense								\$230,300	\$0.077/ton
Total Cost Dry Ore								\$804,300	\$0.268/ton
								Use	\$0.27/ton



Table 4-11

PREPRODUCTION ALLUVIUM STRIPPING SPECIFICATIONS
(TS-24 & 619 SCRAPERS)
ANVIL PROJECT

Quantity cubic yards	2,762,500
Average daily	9,579
Average hourly	457
Annual work days	288
Scrapers required	3
Yards per hour	457
Hours per day	21
Hours annually	18,134
619 Caterpillar Scraper Hours	12,090
TS-24 Euclid Scraper Hours	<u>6,044</u>
Total Scraper Hours	18,134



Table 4-12

ALLUVIUM STRIPPING SPECIFICATIONS
TS-24 EUCLID AND 619 CATERPILLAR SCRAPERS
(1 to 5 YEARS INCLUSIVE)
ANVIL PROJECT

Quantity cubic yards	4,887,000
Average yearly	977,400
Average daily	368
Annual work days	126.4
Scrapers required	3
Yards per hour	368
Hours per day	21
Hours annually	7,971
619 Caterpillar Scraper hours	5,314
TS-24 Euclid Scraper hours	<u>2,657</u>
Total Hours	7,971



Table 4-13

ALLUVIUM STRIPPING SPECIFICATIONS
TS-24 EUCLID AND 619 CATERPILLAR SCRAPERS
(6 to 16 YEARS INCLUSIVE)
ANVIL PROJECT

Quantity cubic yards	6,350,000
Average yearly	577,273
Average daily	7,098
Average hourly	338
Annual work days	81.3
Scraper required	3
Yards per hour	338
Yards per day	7,098
Hours annually	5,124
619 Caterpillar Scraper Hours	3,416
TS-24 Euclid Scraper Hours	<u>1,708</u>
Total Hours	5,124



Table 4-14

ALLUVIUM STRIPPING
RUBBER TIRED SCRAPER PERFORMANCE
PREPRODUCTION PIT
ANVIL PROJECT

TS-24 EUCLID

1.40 Min.	Load Time
1.54 Min.	Pit to Dump
1.49 Min.	Dump to Pit
.70 Min.	Turn and Dump
<u>.15 Min.</u>	Spot Time
5.28 Min.	Total Time
9.50	Trips per Hour
21.60	Yards per Trip
186.20	Yards per Hour
1,303.00	Yards per Shift
3,910.00	Yards per Day

619 CATERPILLAR

.90 Min.
2.00 Min.
1.54 Min.
.70 Min.
<u>.15 Min.</u>
5.29 Min.
9.40
14.40
135.30
947.00
2,841.00



Table 4-15

ALLUVIUM STRIPPING
RUBBER TIRED SCRAPER PERFORMANCE
(1 to 5 YEARS INCLUSIVE)
ANVIL PROJECT

<u>TS-24 EUCLID</u>		<u>619 CATERPILLAR</u>
1.40 Min.	Load Time	.90 Min.
2.34 Min.	Pit to Dump	2.84 Min.
2.25 Min.	Dump to Pit	2.25 Min.
.70 Min.	Turn and Dump	.70 Min.
<u>.15 Min.</u>	Spot Time	<u>.15 Min.</u>
6.84 Min.	Total Time	6.84 Min.
7.30	Trips per Hour	7.30
21.60	Yards per Trip	14.40
158.00	Yards per Hour	105.00
1,106.00	Yards per Shift	735.00
3,318.00	Yards per Day	2,205.00



Table 4-16

ALLUVIUM STRIPPING
RUBBER TIRED SCRAPER PERFORMANCE
(6 to 16 YEARS INCLUSIVE)
ANVIL PROJECT

TS-24 EUCLID

1.40 Min.	Load Time
2.27 Min.	Pit to Dump
2.84 Min.	Dump to Pit
.70 Min.	Turn and Dump
<u>.15 Min.</u>	Spot Time
7.36 Min.	Total Time
6.70	Trips per Hour
21.60	Yards per Trip
145.00	Yards per Hour
1,015.00	Yards per Shift
3,045.00	Yards per Day

619 CATERPILLAR

.90 Min.
2.77 Min.
2.84 Min.
.70 Min.
<u>.15 Min.</u>
7.36 Min.
6.70
14.40
96.40
675.00
2,025.00



Table 4-17

PREPRODUCTION CAP ROCK STRIPPING
SPECIFICATIONS
ANVIL PROJECT

TRUCKS

Quantity Cubic Yards	3,777,500
Average Daily	12,107
Average Hourly	577
Annual Work Days	312
Trucks Required	5
Yards per Hour	701
Hours per Day (5 Trucks)	87
Hours Annually (5 Trucks)	26,955

SHOVELS

Shovels Required (5 Cubic Yards)	2
Yards per Hour (1 Shovel)	303
Yards per Hour (2 Shovels)	606
Hours per Day (2 Shovels)	39.96
Hours Annually (2 Shovels)	12,467



Table 4-18

CAP ROCK STRIPPING
SPECIFICATIONS
(1 to 5 YEARS INCLUSIVE)
ANVIL PROJECT

TRUCKS

Quantity Cubic Yards	12,873,000
Average Yearly	2,574,600
Average Daily	8,252
Average Hourly	393
Annual Work Days	312
Trucks Required	4
Yards per Hour	424
Hours per Day (4 Trucks)	78
Hours Annually (4 Trucks)	24,336

SHOVELS

Shovels Required (5 cubic yards)	2
Yards per Hour (1 Shovel)	303
Yards per Hour (2 Shovels)	606
Hours per Day (2 Shovels)	27.23
Hours Annually	8,497



Table 4-19

CAP ROCK STRIPPING
SPECIFICATIONS
(6 to 16 YEARS INCLUSIVE)
ANVIL PROJECT

TRUCKS

Quantity Cubic Yards	41,400,000
Average Yearly	3,763,636
Average Daily	12,063
Average Hourly	574
Annual Work Days	312
Trucks Required	7
Yards per Hour	660
Hours per Day (7 Trucks)	127.9
Hours Annually (7 Trucks)	39,936

SHOVELS

Shovels Required (5 cubic yards)	2
Yards per Hour (1 Shovel)	303
Yards per Hour (2 Shovels)	606
Hours per Day (2 Shovels)	39.81
Hours Annually (2 Shovels)	12,421



PREPRODUCTION STRIPPING
CAP ROCK
65-TON DIESEL TRUCK PERFORMANCE
ANVIL PROJECT

Load Time	4.67 Min.
Pit Climb Out	1.30 Min.
Pit to Dump	1.22 Min.
Dump to Pit	1.14 Min.
Pit Return	.65 Min.
Turn and Dump	1.00 Min.
Spot Time	<u>.15 Min.</u>
Total Time	10.13 Min.
Trips per Hour	4.90
Yards per Trip	28.60
Yards per Hour	140.14
Yards per Shift	980.98
Yards per Day	2,943.00



Table 4-21

CAP ROCK STRIPPING
65-TON DIESEL TRUCK PERFORMANCE
(1 to 5 YEARS INCLUSIVE)
ANVIL PROJECT

HAUL PAK 65-TON DIESEL TRUCK PERFORMANCE

Load Time	4.67 Min.
Pit Climb Out	3.79 Min.
Pit to Dump	1.22 Min.
Dump to Pit	1.14 Min.
Pit Return	1.42 Min.
Turn and Dump	1.00 Min.
Spot Time	<u>.15 Min.</u>
Total Time	13.39
Trips per Hour	3.70
Yards per Trip	28.60
Yards per Hour	106.00
Yards per Shift	742.00
Yards per Day	2,226.00



CAP ROCK STRIPPING
65-TON DIESEL TRUCK PERFORMANCE
(6 to 16 YEARS INCLUSIVE)
ANVIL PROJECT

HAUL PAK 65-TON DIESEL TRUCK PERFORMANCE

Load Time	4.67 Min.
Pit Climb Out	5.36 Min.
Pit to Dump	.48 Min.
Dump to Pit	.63 Min.
Pit Return	2.43 Min.
Turn and Dump	1.00 Min.
Spot Time	<u>.15 Min.</u>
Total Time	14.72 Min.

Trips per Hour	3.30
Yards per Trip	28.60
Yards per Hour	94.30
Yards per Shift	660.00
Yards per Day	1,980.00



Table 4-23

ORE PRODUCTION
(1 to 5 YEARS INCLUSIVE)
ANVIL PROJECT

TRUCKS

Quantity Tons	11,791,000
Average Yearly	2,358,200
Average Daily	7,558
Average Hourly	1,079
Annual Work Days	312
Trucks Required	5
Tons per Hour	1,170
Hours per Day (5 Trucks)	32.29
Hours Annually	10,077

SHOVELS

Shovels Required (5 cubic yards)	2
Shovel Capacity - Tons per Hour (One Shovel Spare)	963
Hours per Day (1 Shovel)	7.85
Hours Annually	2,449



ORE PRODUCTION
(6 to 9 YEARS INCLUSIVE)
ANVIL PROJECT

TRUCKS

Quantity Tons	9,600,000
Average Yearly	2,400,000
Average Daily	7,692
Average Hourly	1,099
Annual Work Days	312
Trucks Required	6
Tons per Hour	1,170
Hours per Day	39.40
Hours Annually 65-Ton Trucks	12,308

SHOVELS

Shovels required 5 cubic yards	2
Tons per Hour each Shovel	963
Tons per two Shovels	1,926
Hours per Day	7.98
Hours Annually	2,493



ORE PRODUCTION
(10 to 17 Years Inclusive)
ANVIL PROJECT

TRUCKS

Quantity Tons	24,000,000
Average Yearly	3,000,000
Average Daily	9,615
Average Hourly	1,374
Annual Work Days	312
Trucks Required	8
Tons Per Hour	1,560
Hours Per Day (8 Trucks)	49.3
Hours Annually (8 Trucks)	15,385

SHOVELS

Shovels Required (5 cubic yards)	2
Tons Per Hour (1 shovel)	963
Tons Per Hour (2 shovels)	1,926
Hours Per Day	9.98
Hours Annually	3,116



Table 4-26

ORE PRODUCTION
65 TON DIESEL TRUCK DATA
(1 to 5 Years Inclusive)
ANVIL PROJECT

Load Time	3.37 Min.
Pit Climb Out	3.78 Min.
Pit to Mill	1.24 Min.
Pit Return	1.89 Min.
Turn and Dump	1.70 Min.
Spot Time	<u>.15 Min.</u>
TOTAL TIME	13.75

Trips per Hour	3.60
Tons per Trip	65.00
Tons per Hour	234.00
Tons per Shift	1,638.00
Tons per Day	1,638.00



Table 4-27

ORE PRODUCTION
65 TON DIESEL TRUCK DATA
(6 to 9 Years Inclusive)
ANVIL PROJECT

Load Time	3.37 Min.
Pit Climb Out	6.39 Min.
Pit to Mill	.80 Min.
Mill to Pit	.85 Min.
Pit Return	3.19 Min.
Turn and Dump	1.70 Min.
Spot Time	<u>.15 Min.</u>
TOTAL TIME	16.45 Min.

Trips per Hour	3.00
Tons per Trip	65.00
Tons per Hour	195.00
Tons per Shift	1,365.00
Tons per Day	1.365.00



Table 4-28

ORE PRODUCTION
65 TON DIESEL TRUCK DATA
(10 to 17 Years Inclusive)
ANVIL PROJECT

Load Time	3.37 Min.
Pit Climb Out	6.39 Min.
Pit to Mill	.80 Min.
Mill to Pit	.85 Min.
Pit Return	3.19 Min.
Turn and Dump	1.70 Min.
Spot Time	<u>.15 Min.</u>
TOTAL TIME	16.45 Min.

Trips per Hour	3.00
Tons per Trip	65.00
Tons per Hour	195.00
Tons per Shift	1,365.00
Tons per Day	1,365.00



Table 4-29

PREPRODUCTION MINING
ORE TO STOCKPILE
ANVIL PROJECT

TRUCKS

Quantity Tons	607,000
Average Daily	6,741
Average Hourly	963
Annual Work Days	90
Trucks Required	5
Tons per Hour	1,170
Hours per Day (5 Trucks)	28.80
Hours Annually (5 Trucks)	2,592

SHOVELS

Shovels Required (5 cubic yards)	1
Tons per Hour	963
Hours per Day	7
Hours Annually	631



Table 4-30

PREPRODUCTION MINING
ORE TO STOCKPILE
65 TON DIESEL TRUCK DATA
ANVIL PROJECT

Load Time	3.37 Min.
Pit Climb Out	3.78 Min.
Pit to Mill	1.24 Min.
Mill to Pit	1.62 Min.
Pit Return	1.89 Min.
Turn and Dump	1.70 Min.
Spot Time	<u>.15</u> Min.
TOTAL TIME	13.75 Min.

Trips per Hour	3.60
Tons per Trip	65.00
Tons per Hour	234.00
Tons per Shift	1,638.00
Tons per Day	1,638.00



RECLAIMING STOCKPILED ORE
(6 to 9 Years Inclusive)
ANVIL PROJECT

TRUCKS

Quantity Tons	2,400,000
Average Yearly	600,000
Average Daily	1,923
Average Hourly	275
Annual Work Days	312
Trucks Required	1
Tons Per Hour	299
Hours per Day	6.43
Hours Annually	2,006

SHOVELS

Shovels Required (3 cubic yards)	1
Tons per Hour	578
Hours per Day	3.33
Hours Annually	1,038



Table 4-32

RECLAIMING STOCKPILED ORE
65 TON DIESEL TRUCK DATA
(6 to 9 Years Inclusive)
ANVIL PROJECT

Load Time	6.72 Min.
Stockpile to Crusher	1.25 Min.
Crusher to Stockpile	1.00 Min.
Turn and Dump	1.70 Min.
Spot Time	<u>.15</u> Min.
TOTAL TIME	10.82 Min.

Trips per Hour	4.60
Tons per Trip	65.00
Tons per Hour	299.00
Tons per Shift	2,093.00
Tons per Day	2,093.00



Table 4-33

SHOVEL DATA
(FIVE CUBIC YARD SHOVEL)
ANVIL PROJECT

	<u>Cap Rock</u>	<u>Ore</u>
Bank Unit per c.y. (in tons)	2.30	3.18
Loose Unit per c.y. (in tons)	1.53	2.12
Shovel Efficiency, per cent	85.00	85.00
Load per Dipper, c.y.	2.83	2.83
Load per Dipper (in tons)	6.50	9.00
Dippers per 50 Minute Hour	107.00	107.00
Production per 50 Minute Hour, c.y./bank	303.00	303.00
Production per 50 Minute Hour, tons/bank	696.00	963.00
Shovel Hours for 7,558 tons ore/day		7.85
7,692 tons ore/day		7.99
9,615 tons ore/day		9.98



Table 4-34

SHOVEL DATA
(3 CUBIC YARD SHOVEL IN ORE)
ANVIL PROJECT

Bank Unit per c.y. (in tons)	3.18
Loose Unit per c.y. (in tons)	2.10
Shovel Efficiency, per cent	85.00
Load per Dipper, cubic yards	1.70
Load per Dipper (in tons)	5.40
Dippers per 50 Minute Hour	107.00
Production per 50 Minute Hour, c.y./bank	182.00
Production per 50 Minute Hour, tons/bank	578.00
Shovel Hours for 4,046 tons ore/day	7.00
Hours per Shovel Shift	7.00



Table 4-35

DRILLING DATA
ANVIL PROJECT

<u>BUCYRUS ERIE 40R ROTARY DRILL DATA</u>	<u>CAP ROCK</u>	<u>ORE</u>
Hole Diameter (inches)	9.0	9.0
Tons per c.y./bank	2.30	3.18
Bank Height (feet)	35.00	35.00
Burden (feet)	22.00	22.00
Spacing (feet)	18.00	18.00
Yield per foot (cubic yard)	14.70	14.70
Yield per foot (tons)	33.80	46.70
Subdrilling, per cent	12.50	12.50
Net c.y. per foot	12.86	12.86
Net Tons per foot	29.57	40.86
Drilling Rate, feet/hr.	40.00	40.00
Net c.y. per Hour	5.14	5.14
Net Tons per Hour	1,183.00	1,634.00
Bit Life (feet)	2,691.00	3,721.00
Bit Cost per Hour	10.00	10.00
Rod Cost per Hour	.0605	.0605



Table 4-36

PAY RATES
HOURLY PAID EMPLOYEES
ANVIL PROJECT

<u>JOB TITLE</u>	<u>HOURLY</u>	<u>OVERTIME HOURLY</u>
Scraper Operator	\$3.11	\$4.665
Push Cat Operator	3.11	4.665
Motor Patrol Operator	2.96	4.44
Shovel Operator	3.69	5.535
Shovel Oiler	2.96	4.44
Clean up Cat Operator	2.96	4.44
Truck Drivers Heavy	3.11	4.665
Drillers	3.14	4.71
Driller Helpers	2.70	4.05
Powder Man	3.14	4.71
Powder Helper	2.98	4.47
Water Truck Driver	2.70	4.05
Heavy Duty Mechanic	3.69	5.535
Welders	2.84	4.26
Tire Repair Man	3.39	5.085
Lube Service Man	3.02	4.53
Electrician	3.54	5.31
Mechanic Improvers	3.11	4.665
Crane Operator Mobil	2.96	4.44
Crane Oiler Mobil	2.46	3.69
Laborers	2.50	3.75
Payloader Operator	2.96	4.44

Rates Supplied by Anvil Mining Company

4-T36

The Ralph M. Parsons Construction Co. of Canada, Ltd.



Table 4-37

MINE STAFF SALARIES
PREPRODUCTION STRIPPING
ANVIL PROJECT

<u>JOB TITLE</u>	<u>NO.</u>	<u>MONTHLY SALARY</u>	<u>ANNUAL COST</u>
Mine Superintendent	1	\$1,100	\$13,200
Assistant Superintendent	1	900	10,800
Mine Engineer	1	750	9,000
Assistant Engineer	1	500	6,000
Rod Chain	1	375	4,500
Samplers	1	500	6,000
Draftsman	1	525	6,300
Pit Foreman	2	700 (ea.)	16,800
Maintenance Superintendent	1	1,100	13,200
Warehouse Clerk	<u>1</u>	<u>375</u>	<u>4,500</u>
Total	11		\$90,300
Payroll Overhead @ 20%			<u>\$18,000</u>
TOTAL			\$109,300

Salary and wage rates supplied by Anvil Mining Company



Table 4-38

OPERATING AND MAINTENANCE PERSONNEL
PREPRODUCTION STRIPPING
ANVIL PROJECT

<u>JOB TITLE</u>	<u>NO.</u>	<u>HOURLY RATE</u>	<u>RATE/ WEEK</u>	<u>AMT./ WEEK</u>
Scraper Operator	9	3.11	162	\$1,458
Dozer Operator	13	3.11	162	2,106
Shovel Operator	7	3.69	192	1,344
Shovel Oiler	7	2.96	154	1,078
Truck Operator (H.D.)	20	3.11	162	3,240
Rotary Driller	4	3.14	163	652
Secondary Driller	1	3.14	163	163
Powder Man	1	3.14	163	163
Driller Helper	5	2.70	130	650
Powder Man Helper	1	2.98	155	155
Laborers	3	2.50	130	390
Sub Total	71			\$11,399
Motor Grader	1	2.96	154	154
Water & Service Truck	1	2.86	149	149
Sub Total	2			\$ 303
Mechanic (H.D.)	15	3.69	192	2,880
Welders	3	2.84	148	444
Tire Repair	4	3.79	176	704
Electricians	3	3.54	184	552
Improvers	5	3.11	162	810
Lube Service	2	3.02	157	314
Sub Total	32			\$ 5,704
TOTAL	105			\$17,406
Payroll Overhead @ 20%				\$ 3,481
TOTAL/WEEK				<u>\$20,887</u>
TOTAL/YEAR				<u>\$1,086,134</u>



Table 4-39

MINE STAFF SALARIES
(1 to 5 Years Inclusive)
ANVIL PROJECT

<u>JOB TITLE</u>	<u>NO.</u>	<u>MONTHLY SALARY</u>	<u>ANNUAL COST</u>
Mine Superintendent	1	\$1,100	\$13,200
Assistant Superintendent	1	900	10,800
Mine Engineer	1	950	9,000
Assistant Engineer	1	500	6,000
Rod Chain	2	375 (ea.)	9,000
Samplers	2	500 (ea.)	12,000
Draftsmen	2	525 (ea.)	12,600
Pit Foremen	3	700 (ea.)	25,200
Maintenance Superintendent	1	1,100	13,200
Shop Foreman	1	700	8,400
Warehouse Clerk	<u>1</u>	375	<u>4,500</u>
Total	16		\$123,900
Payroll Overhead @ 20%			<u>24,800</u>
TOTAL			<u>\$148,700</u>



Table 4-40

OPERATING AND MAINTENANCE PERSONNEL
(1 to 5 Years Inclusive)
ANVIL PROJECT

<u>JOB TITLE</u>	<u>NO.</u>	<u>HOURLY RATE</u>	<u>RATE/ WEEK</u>	<u>AMT./ WEEK</u>
Scraper Operator	9	3.11	162	\$1,458
Dozer Operator	14	3.11	162	2,268
Shovel Operator	8	3.69	192	1,536
Shovel Oiler	8	2.96	154	1,232
Truck Operator (H.D.)	17	3.11	162	2,754
Rotary Driller	8	3.14	163	1,304
Secondary Driller	1	3.14	163	163
Powder Man	1	3.14	163	163
Driller Helper	9	2.70	130	1,170
Powder Man Helper	1	2.90	155	155
Laborers	6	2.50	130	780
Sub Total	82			\$12,983
Motor Grader	2	2.96	154	308
Water & Service Truck	2	2.86	149	298
Sub Total	4			\$ 606
Mechanic (H.D.)	12	3.69	192	2,304
Welders	3	2.84	148	444
Tire Repair	4	3.39	176	704
Electricians	3	3.54	184	552
Improvers	5	3.11	162	810
Lube Service	2	3.02	157	314
Sub Total	29			\$ 5,128
TOTAL	115			\$18,717
Payroll Overhead @ 20%				\$ 3,743
TOTAL/WEEK				<u>\$22,460</u>
TOTAL/YEAR				<u>\$1,167,920</u>



Table 4-41

MINE STAFF SALARIES
(6 to 9 Years Inclusive)
ANVIL PROJECT

<u>JOB TITLE</u>	<u>NO.</u>	<u>MONTHLY SALARY</u>	<u>ANNUAL COST</u>
Mine Superintendent	1	\$1,100	\$13,200
Assistant Superintendent	1	900	10,800
Mine Engineer	1	750	9,000
Assistant Engineer	1	500	6,000
Rod Chain	2	375 (ea.)	9,000
Samplers	2	500 (ea.)	12,000
Draftsmen	2	525 (ea.)	12,600
Pit Foreman	3	700 (ea.)	25,200
Maintenance Superintendent	1	1,100	13,200
Shop Foremen	1	700	8,400
Warehouse Clerk	<u>1</u>	375	<u>4,500</u>
Total	16		\$123,900
Payroll Overhead @ 20%			<u>24,800</u>
TOTAL			\$148,700



Table 4-42

OPERATING AND MAINTENANCE PERSONNEL
(6 to 9 Years Inclusive)
ANVIL PROJECT

<u>JOB TITLE</u>	<u>NO.</u>	<u>HOURLY RATE</u>	<u>RATE/ WEEK</u>	<u>AMT./ WEEK</u>
Scraper Operator	9	\$3.11	\$162	\$1,458
Dozer Operator	15	3.11	162	2,430
Shovel Operator	9	3.69	192	1,728
Shovel Oiler	9	2.96	154	1,232
Truck Operator (H.D.)	28	3.11	162	4,536
Rotary Driller	7	3.14	163	1,141
Secondary Driller	1	3.14	163	163
Powder Man	1	3.14	163	163
Driller Helper	8	2.70	130	1,040
Powder Man Helper	1	2.98	155	155
Laborers	<u>6</u>	2.50	130	<u>780</u>
Sub Total	93			\$14,826
Motor Grader	2	2.96	154	308
Water and Service Truck	<u>2</u>	2.86	149	<u>298</u>
Sub Total	4			\$ 606
Mechanic (H.D.)	16	3.69	192	3,072
Welders	6	2.84	148	888
Tire Repair	6	3.39	176	1,056
Electricians	3	3.54	184	552
Improvers	7	3.11	162	1,134
Lube Service	<u>5</u>	3.02	157	<u>785</u>
Sub Total	43			\$ 7,487
Total	140			\$22,919
Payroll Overhead @ 20%				<u>\$ 4,584</u>
TOTAL/WEEK				<u>\$27,493</u>
TOTAL/YEAR				<u>\$1,430,156</u>



Table 4-43

MINE STAFF SALARIES
(10 to 16 Years Inclusive)
ANVIL PROJECT

<u>JOB TITLE</u>	<u>NO.</u>	<u>MONTHLY SALARY</u>	<u>ANNUAL COST</u>
Mine Superintendent	1	\$1,100	\$13,200
Assistant Superintendent	1	900	10,000
Mine Engineer	1	750	9,000
Assistant Engineer	1	500	6,000
Rod Chain	2	375 (ea.)	9,000
Samplers	2	500 (ea.)	12,000
Draftsmen	2	525 (ea.)	12,600
Pit Foremen	3	700 (ea.)	25,200
Maintenance Superintendent	1	1,100	13,200
Shop Foreman	1	700	8,400
Warehouse Clerk	<u>1</u>	<u>375</u>	<u>4,500</u>
TOTAL	16		\$123,900
Payroll Overhead @ 20%			<u>24,800</u>
TOTAL			\$148,700



Table 4-44

OPERATING AND MAINTENANCE PERSONNEL
(10 to 16 Years Inclusive)
ANVIL PROJECT

<u>JOB TITLE</u>	<u>NO.</u>	<u>HOURLY RATE</u>	<u>RATE/ WEEK</u>	<u>AMT./ WEEK</u>
Scraper Operator	9	\$3.11	\$162	\$1,458
Dozer Operator	14	3.11	162	2,268
Shovel Operator	8	3.69	192	1,536
Shovel Oiler	8	2.96	154	1,232
Truck Operator (H.D.)	29	3.11	162	4,698
Rotary Driller	8	3.14	163	1,304
Secondary Driller	1	3.14	163	163
Powder Man	1	3.14	163	163
Driller Helper	9	2.70	140	1,260
Powder Man Helper	1	2.98	155	155
Laborers	6	2.50	130	780
Sub Total	94			\$15,017
Motor Grader	2	2.96	154	308
Water and Service Truck	2	2.86	149	298
Sub Total	4			\$ 606
Mechanic (H.D.)	16	3.69	192	3,072
Welders	6	2.84	148	888
Tire Repair	6	3.39	176	1,056
Electricians	3	3.54	184	552
Improvers	7	3.11	162	1,134
Lube Service	5	3.02	157	785
Sub Total	43			\$ 7,487
TOTAL	137			\$23,110
Payroll Overhead @ 20%				\$ 4,622
TOTAL/WEEK				<u>\$27,732</u>
TOTAL/YEAR				<u>\$1,442,064</u>



Table 4-45

GENERAL PIT EXPENSE
PREPRODUCTION STRIPPING
ANVIL PROJECT

<u>ROAD MAINTENANCE</u>	<u>HRS.</u>	<u>LABOR</u>		<u>SUPPLIES</u>	
		<u>OPERATING</u>	<u>MAINTENANCE</u>	<u>OPERATING</u>	<u>MAINTENANCE</u>
Cat 16 Motor Grader	2009	\$ 7,400	\$3,500	\$ 4,900	\$ 5,700
Water Truck	728	6,300	400	3,400	400
D8 Caterpillar Dozer	1000	<u>3,900</u>	<u>4,000</u>	<u>4,200</u>	<u>4,800</u>
		\$17,600	\$7,900	\$12,500	\$10,900
<u>GENERAL EQUIPMENT</u>					
1/2 Ton Pickups	8400		2,000	9,800	1,000
Bus 44 Passengers	900		200	1,100	200
Winch Truck	2009		1,000	3,600	700
Station Wagon	1500		400	1,800	400
Grease Truck	1148	<u>4,300</u>	<u>1,100</u>	<u>2,000</u>	<u>600</u>
		\$ 4,300	\$4,700	\$18,300	\$ 2,900
<u>GENERAL LABOR</u>					
	<u>HRS.</u>	<u>COST</u>			
Six Laborers	13,104	\$32,800			

SUMMARY OF TOTAL ANNUAL GENERAL PIT EXPENSES

Supplies	\$ 44,600
Labor	<u>67,300</u>
TOTAL ANNUAL GENERAL PIT EXPENSE	<u>\$111,900</u>



Table 4-46

**ANNUAL GENERAL PIT EXPENSE
(1 to 16th Year Inclusive)
ANVIL PROJECT**

<u>ROAD MAINTENANCE</u>	<u>HRS.</u>	<u>LABOR</u>		<u>SUPPLIES</u>	
		<u>OPERATING</u>	<u>MAINTENANCE</u>	<u>OPERATING</u>	<u>MAINTENANCE</u>
Cat 16 Motor Grader	3992	\$14,600	\$ 7,000	\$ 9,800	\$11,400
Water Truck	728	6,300	400	3,400	400
D8 Caterpillar Dozer	1000	<u>3,900</u>	<u>4,000</u>	<u>4,200</u>	<u>4,800</u>
		\$24,800	\$11,400	\$17,400	\$16,600
 <u>GENERAL MINE EQUIPEMNT</u>					
1/2 Ton Pickups	8400		2,000	9,800	1,000
Bus 44 Passenger	900		200	1,100	200
Winch Truck	1700		900	3,100	600
Station Wagon	1500		400	1,800	400
Grease Truck	3500	<u>13,100</u>	<u>3,500</u>	<u>6,200</u>	<u>1,800</u>
		\$13,100	\$ 7,000	\$22,000	\$ 4,000
 <u>GENERAL LABOR</u>					
	<u>HRS.</u>	<u>COST</u>			
Six Laborers	13,104	\$32,800			

SUMMARY OF TOTAL ANNUAL GENERAL PIT EXPENSES

Supplies	\$ 60,000
Labor	<u>89,100</u>
TOTAL ANNUAL GENERAL PIT EXPENSES	<u>\$149,100</u>



Table 4-47

GENERAL PIT EXPENSE
(17th Year)
ANVIL PROJECT

<u>ROAD MAINTENANCE</u>	<u>HRS.</u>	<u>LABOR</u>		<u>SUPPLIES</u>	
		<u>OPERATING</u>	<u>MAINTENANCE</u>	<u>OPERATING</u>	<u>MAINTENANCE</u>
Cat 16 Motor Grader	2184	\$ 8,000	\$3,800	\$ 5,300	\$ 6,200
Water Truck	910	3,400	200	1,800	300
D8 Caterpillar Dozer	728	<u>2,800</u>	<u>2,900</u>	<u>3,100</u>	<u>3,500</u>
		\$14,200	\$6,900	\$10,200	\$10,000

GENERAL MINE EQUIPMENT

1/2 Ton Pickups	4368		1,100	5,400	500
Bus 44 Passengers	624		200	900	200
Winch Truck	1560		900	2,800	500
Station Wagon	1560		400	1,900	400
Grease Truck	624	<u>2,300</u>	<u>600</u>	<u>1,100</u>	<u>300</u>
		\$ 2,300	\$3,200	\$12,100	\$ 1,900

<u>GENERAL LABOR</u>	<u>HRS.</u>	<u>COST</u>
Three Laborers	6552	\$16,500

SUMMARY OF TOTAL ANNUAL GENERAL PIT EXPENSES

Supplies	\$ 34,200
Labor	<u>43,100</u>
TOTAL ANNUAL GENERAL PIT EXPENSE	<u>\$ 77,300</u>



EQUIPMENT SPECIFICATION
DIESEL POWER SHOVEL (5 CUBIC YARDS)
BUCYRUS ERIE
ANVIL PROJECT

Model 88B
5 Cubic Yard Shovel

F.O.B. Evansville, Indiana

\$ 210,345.00

OPTIONAL EQUIPMENT INCLUDES

T-1 Steel in Handle	1,800.00
T-1 Steel in Boom	6,210.00
Strip Heaters in Chain Case	2,320.00
Insulated Operators Cab	3,480.00
Set of Kysor Thermostatically Controlled Radiator Shutters	370.00
Aeroquip Hose in Place of Standard Tubing	775.00
Rubber Bumpers for Boom	200.00
GE 3 KW Alternator with Wiring	2,200.00
Air Starting Cummins Diesel	870.00
Akim Hotstart Water and Oil Heater	300.00
Farval Automatic Lube System	<u>7,105.00</u>
	\$ 236,000.00
Less Discount - 5%	<u>11,800.00</u>
TOTAL	\$ 224,200.00

Power Unit Cummins Diesel V12-525
12 Cylinder



EQUIPMENT SPECIFICATION
ELECTRIC POWER SHOVEL (5 CUBIC YARDS)
BUCYRUS ERIE
ANVIL PROJECT

Model 88B
5 Cubic Yard Shovel

F.O.B. Evansville, Indiana

\$ 208,240.00

OPTIONAL EQUIPMENT INCLUDES

T-1 Steel in Handle	1,800.00
T-1 Steel in Boom	6,210.00
Strip Heaters in Chain Case	2,320.00
Insulated Operators Cab	3,480.00
Aeroquip Hose in Place of Standard Tubing	775.00
Rubber Bumpers for Boom	200.00
GE 3 KW Alternator with Wiring	2,200.00
Extra Counter Weight - 5,000 lbs.	940.00
Farval Automatic Lube System	<u>7,105.00</u>
	233,270.00
Less Discount - 5%	<u>11,700.00</u>
TOTAL	\$ 221,600.00



EQUIPMENT SPECIFICATION
ELECTRIC POWER SHOVEL (3 CUBIC YARDS)
BUCYRUS ERIE
ANVIL PROJECT

Model 61B
3 Cubic Yard Shovel

F.O.B. Evansville, Indiana

\$ 122,730.00

OPTIONAL EQUIPMENT INCLUDES

T-1 Steel in Boom 6,210.00

T-1 Steel in Handle 1,800.00

3 KW Heater 270.00

Strip Heater in Gear and Chain Cases 2,320.00

Insulated Operators Cab 3,580.00

Rubber Bumpers on Boom 200.00

\$ 137,110.00

Less Discount - 5% 6,900.00

TOTAL \$ 130,200.00



Table 4-51

EQUIPMENT SPECIFICATION
CATERPILLAR TRACTORS
ANVIL PROJECT

Model D8 Caterpillar
Diesel-Tractor Power Shift Series (H)
F.O.B. Peoria, Illinois \$ 43,600.00

OPTIONAL EQUIPMENT INCLUDES

22" Extreme Service Tracks	850.00
7 Roller Track Frame	788.00
Fuel Tank Cap Lock - Diesel	11.00
Fuel Tank Cap Lock - Gas Starting Engine	10.00
Hydraulic Tank Cap Lock	10.00
Oil Filler Cap Lock	12.00
Radiator Cap Lock	12.00
Fan Blast Deflector	56.00
Instrument Panel Guard	27.00
Steel Final Drive Cases	132.00
Crank Case Guard	299.00
Track Roller Guards	578.00
Front Pull Hook	64.00
Lighting System	151.00
Cylinder Bracket	600.00
Hydraulic Cylinder Right Hand	515.00
Hydraulic Cylinder Left Hand	515.00
Hose Group	180.00
No. 183 Hydraulic Control, 3 Valve	2,345.00
Caterpillar 8A Bulldozer, 5J5717 with Trunnion, 2J584, not installed	4,850.00
No. 8B Caterpillar Ripper, with one Shank Two Additional Teeth	6,490.00
Steel Cab	1,402.00
Cab Insulation	125.00
Cab Heater	234.00
Engine Heater and Crankcase Oil	395.00
	\$ 64,351.00
Less Discount - 5%	3,217.55
TOTAL	\$ 61,100.00



EQUIPMENT SPECIFICATION
MOTOR GRADER
ANVIL PROJECT

Model No. 16 Diesel Motor Grader
Direct Electric Starting System

F.O.B. Peoria, Illinois \$ 52,207.00

OPTIONAL EQUIPMENT INCLUDES

Fuel Tank Lock Cap - Diesel	13.00
Hydraulic Tank Cap Lock	10.00
Oil Filler Cap Lock	12.00
Lighting System - Center Mounted	67.00
Radiator Cap Lock	12.00
No. 16 Caterpillar Ripper - Scarifier Installed	4,380.00
Hydraulic System	121.00
Set of Eleven Scarifier Teeth	410.00
Steel Cab with Tinted Safety Glass	1,071.00
Cab Insulation	125.00
Engine Heater and Crankcase Oil	<u>395.00</u>
	\$ 58,823.00
Less Discount - 5%	<u>2,941.00</u>
TOTAL	\$ 55,900.00

Engine: Caterpillar D-343 Six-cylinder 225 H.P.
Transmission: Caterpillar Power Shift with Torque Divider
Tires Front: 16:00 x 24, 12-ply
Tires Rear: 18:00 x 25, 12-ply



Table 4-53

EQUIPMENT SPECIFICATION
HAULAGE TRUCKS
ANVIL PROJECT

Model 65 HAULPAK

Engines: Cummins VT 12-635 @ 635 H.P.

G.M. 16 VN-71N @ 635 H.P.

Transmission: Allison CLBT - 6061

Tires: 6 each, 21:00-35, 36 Ply Rock Service

F.O.B. Peoria, Illinois

\$ 105,085.00

OPTIONAL EQUIPMENT INCLUDES

Brakes, Dirt and Water Shields	150.00
Cab with Special Extra Heavy-Duty Insulation	180.00
Cab with Extra Heavy-Duty Heater and Defroster (in lieu of stock heater and defroster)	220.00
Kim Hot Start Engine 8,000 watts	480.00
Engine Oil 600 watts	190.00
Transmission Oil 60 watts	200.00
Hydraulic Tank 1,800 watts	120.00
Final Drive 1,200 watts	190.00
Alcohol Injector and Moisture Control	100.00
Hood Sides and Top Covers	320.00
Belly Guards - Engine and Transmission	500.00
Cold Weather Starting Kit (Propane)	35.00
Radiator Shutters	240.00
Skip Resistance Coating on Walkways	40.00
Left and Right Hand Mirrors	70.00
Fire Extinguisher - Mounted	80.00
Under Hood Dash Outside Air Intake System	245.00
Back-up Warning Horn	70.00
Low Temperature (-65°) Hose and Wiring	450.00
	\$ 109,000.00
Less Discount - 7%	7,600.00
TOTAL	
	\$ 101,400.00

The Ralph M. Parsons Construction Co. of Canada, Ltd.



EQUIPMENT SPECIFICATION
ROTARY DRILL
ANVIL PROJECT

Model 40R Bucyrus Erie

Power Unit: Cummins NHRS-6-1P-6 @ 215 H.P.

F.O.B. South Milwaukee, Wisconsin \$ 115,400.00

OPTIONAL EQUIPMENT INCLUDES

Insulated Operators Cab - ARMAFLEX	1,050.00
Cab Heater Blower Type	600.00
Alcohol Vaporizer	145.00
Hydraulic System Filters	455.00
Enclosed Machinery House	1,225.00
Compressor Crankcase Heater	430.00
GE 3 KW Alternator with Wiring	<u>2,200.00</u>
	\$ 122,000.00
Less Discount - 5%	<u>6,100.00</u>
TOTAL	\$ 115,900.00



EQUIPMENT SPECIFICATION
ROTARY DRILL ACCESSORIES
ANVIL PROJECT

Model 40R Bucyrus Erie Accessories

Drill Pipe 7-3/4" x 27'-9" 1,320 lbs. each x 2 = \$4,840.00

Drill Dollars for 9" hole 525 lbs. each x 2 = \$1,410.00

BIT TYPES

META-QUARTZITE

Q 9 J 3-Cutter "Carbide Insert"

Bit Diameter: 9 inch

Bit R.P.M. 60

Weight on Bit: 45,000 lbs.

Bit Life: 1610'

Penetration Rate: 23' per Hour

Bit Cost: \$1,115.00 ea.

QUARTZITIC-CERITE SCHIST

QW 4 H 3-Cutter Tooth Type

Bit Diameter: 9 inch

Bit R.P.M. 50

Weight on Bit: 35,000 lbs.

Bit Life: 2730'

Penetration Rate: 91' per Hour

Bit Cost: \$242.00 ea.



Table 4-56

EQUIPMENT SPECIFICATION
ATHEY TRAVEL DRILL
SNIPER AERIAL ROCK DRILL
ANVIL PROJECT

Model 033
Diesel Power \$ 42,000.00

F.O.B. Raleigh, North Carolina

OPTIONAL EQUIPMENT INCLUDES

Screw Feed P601029 = 120" 910.00

Four Flood Lights 200.00

Twin Cab Heaters 225.00

Anti-Freeze Injection System 350.00

\$ 43,700.00

Less Discount - 5% 2,200.00

TOTAL \$ 41,500.00



Table 4-57

SHOP EQUIPMENT
ANVIL PROJECT

MAJOR SHOP EQUIPMENT

Grinder Floor Mounted: 7.5 H.P. Weight - 1,400 lbs.	\$ 1,000.00
Power Hacksaw Floor Mounted: 1 H.P. Weight - 450 lbs.	500.00
Three Lincoln 300 AMP Welders w/leads 8 Holders - 1,400 lbs.	3,000.00
Allison Torque Tester - Weight 500 lbs.	3,000.00
Diesel Engine Tester - Weight 500 lbs.	2,000.00
Caterpillar: Service and Maintenance Tools for Caterpillar D8 Tractor and No. 16 Motor Grader - 1,500 lbs.	4,000.00
Steam Cleaner Includes Attachments 1,000 lbs.	2,000.00
Truck Tools Include: Tire Stand Repair 1,000 lbs.. . . .	500.00
Complete Engine Overhaul 1,000 lbs.. . . .	1,500.00
Complete Transmission Tools 700 lbs.. . . .	800.00
Rollover Jack Stand 500 lbs.. . . .	500.00
Transmission, Engine Jack Hydr. 800 lbs.. . . .	1,000.00
Cutting Rigs Oxygen & Acetylene 50 lbs. x 5 250 lbs.. . . .	
Price each 50' Double Hose Gauges \$160.00 x 5	800.00
Compressor Shop Type. 1,000 lbs.. . . .	300.00
Hand Tools: Price includes all small tools to equip seventeen Mechanics. 1,700 lbs.. . . .	
\$500.00 each x 16	8,000.00
Service Unit Grease Oils Hydraulics 1,400 lbs.. . . .	2,100.00
	TOTAL 31,000.00
	FREIGHT 700.00
	TARIFF 7,100.00
	GRAND TOTAL \$ 38,800.00

NOTE: This equipment has been included in Section 9 and is repeated here to show the Mining Division Requirements.



CAPITAL COST
MINE PUMPING EQUIPMENT
ANVIL PROJECT

PUMP MODEL 36T-125

6" Discharge Line

Gallons per Minute 500 @ 570' Total Head

Brake Horse Power Required: 103

Pump Price Baseplate and Coupling \$ 1,155

Motor Price 125 H.P. TEFC, 3,600 R.P.M. 2,060

\$ 3,215

ELECTRICAL

Starter size (4) with Breaker (each) \$ 570

Cable size 1/0 4 Conductor Type W 100' 400

Control Switches 100

\$ 1,070

SUMMARY OF PUMPING EXPENDITURES

A. 36T-125 Centrifugal Pump 2 @ \$3,215 \$ 6,430

B. Starters Size 4 2 @ \$570 1,140

C. Power Cable Size 1/0 4 Conductor Type W 400

D. Pipe Plus Couplers (6" 600') with Valves 3,500

E. Power Center Breaker, Transformers
4160V/480V 225 H.P. 3,500F. Flyght Sump Pumps 8 H.P. @ \$1,200 (each) 2,400

TOTAL EQUIPMENT \$17,370



SECTION 5

PROCESS ENGINEERING STUDY

ECONOMIC METALLURGICAL CRITERIA

Two metallurgical balances are used as a basis for the economic evaluation of the Base Case (Case A-4). The first balance estimates the average yearly results to be expected from the concentrator operation during the first five-year period. The second balance covers the operational period from years 6 through 17. These balances are shown in Tables 5-1 and 5-2, respectively.

The total tonnage of ore to be milled is estimated at approximately 46 million tons. The proposed concentrator general arrangement permits the treatment of this ore at a rate of two million tons annually for the first five years. Subsequent to the fifth year of operation, an expansion of concentrator capacity by 50 percent is planned to raise the annual milling rate to three million tons of ore. From the estimated ore reserves, this second period will be about twelve years duration.

These balances indicate the quantities of both lead and zinc concentrate and tailing material that the concentrator is expected to produce on a yearly basis during each of the specified time periods, together with the anticipated grades and percentages of metals recovered in each product from the ore milled. The changes in metallurgical results as affected by the lower grade ores to be treated during the second period are reflected in Table 5-2.

Laboratory test results obtained by The Galigher Company (as recorded in their **October 3, 1966 report**) on **drill cores taken from various parts of** the ore body, and principally their test results on a composited sample of drill cores, have been used as a basis for calculating the material balances. The composite sample was prepared by weighting the drill core components to represent the estimated average composition and types of ore that will be treated during the first five-year period.

In these balances, the grades and tonnages of ore feed to the concentrator conform to the projected mining schedule. The estimated grades of concentrates and metallurgical recoveries have been increased slightly beyond the values attained by Galigher's laboratory test results. This was done to represent the anticipated improvements that would be effected in the operation of the commercial concentrator as opposed to the laboratory results which are probably below optimum.

TABLE 5-1

METALLURGICAL BALANCE FOR FIRST FIVE YEARS

Product	Annual Tonnage (D.S.T.)	Grade of Product					Distribution of Metal				
		% Pb	% Zn	% Cu	OZ/T Ag	OZ/T Au	% Pb	% Zn	% Cu	% Ag	% Au
Mill Feed	2,000,000	4.9	7.3	0.15	1.69	0.010	100.0	100.0	100.0	100.0	100.0
Lead Conc't.	128,400	69.1	5.85	0.52	20.57	0.050	90.5	5.1	22.3	78.1	32.1
Zinc Conc't.	240,200	0.61	53.8	0.49	0.85	0.006	1.5	88.5	39.3	6.0	7.2
Tailings	1,631,400	0.48	0.57	0.07	0.33	0.007	8.0	6.4	38.4	15.9	60.7

TABLE 5-2

METALLURGICAL BALANCE FOR SIXTH YEAR THROUGH LIFE OF PROPERTY

Product	Annual Tonnage (D.S.T.)	Grade of Product					Distribution of Metal				
		% Pb	% Zn	% Cu	OZ/T Ag	OZ/T Au	% Pb	% Zn	% Cu	% Ag	% Au
Mill Feed	3,000,000	3.41	5.92	0.34	1.01	0.010	100.0	100.0	100.0	100.0	100.0
Lead Conc't	130,100	70.0	5.70	0.50	19.9	0.060	89.0	4.2	6.4	85.4	26.0
Zinc Conc't	282,900	0.60	54.0	1.45	0.60	0.006	1.7	86.0	40.2	5.6	5.7
Tailings	2,587,000	0.37	0.67	0.21	0.11	0.008	9.3	9.8	53.4	9.0	68.3



From the relationships shown graphically in Figure 5-1, it can be seen that a considerable improvement in the metallurgical results was obtained by Galigher in going from open-circuit to locked-cycle testing. Because of the limited number of tests which could be made on the small sample quantities available, it is doubtful that even these locked-cycle test results are indicative of optimum metallurgy. Therefore, it is probable that a further improvement will be achieved in the forthcoming pilot plant test program and, also, in actual plant operations. In anticipation of this improvement a lead concentrate of 69.1 percent lead at a recovery of 90.5 percent of the total lead has been used as the basis (for the first five-year period) for the economic evaluation. Likewise, a zinc concentrate of 53.8 percent zinc at a recovery of 88.5 percent has been utilized.

Although this quality of concentrate must be confirmed in the pilot plant operation, its effect on the economics is not great. No economic penalty is involved in marketing lead concentrates of lower grade and only a slight loss of revenue would occur if zinc recoveries dropped to the 85 percent value presently confirmed by laboratory test results.

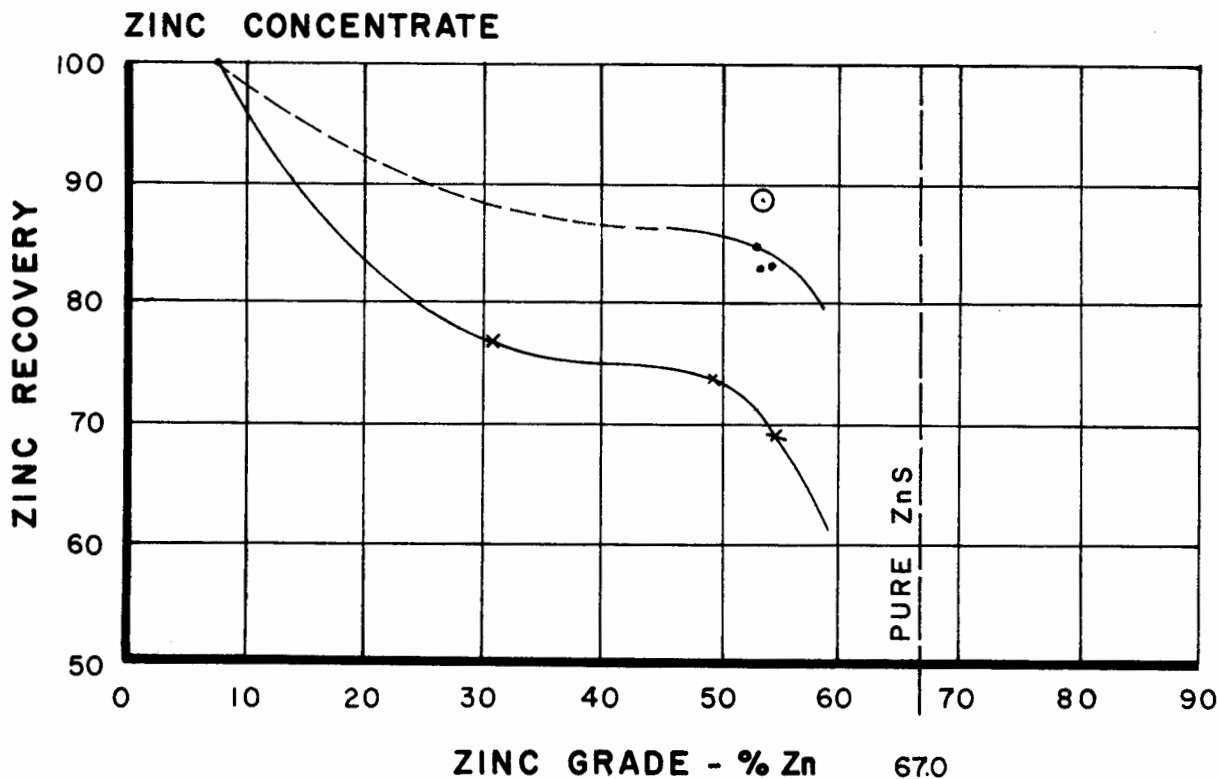
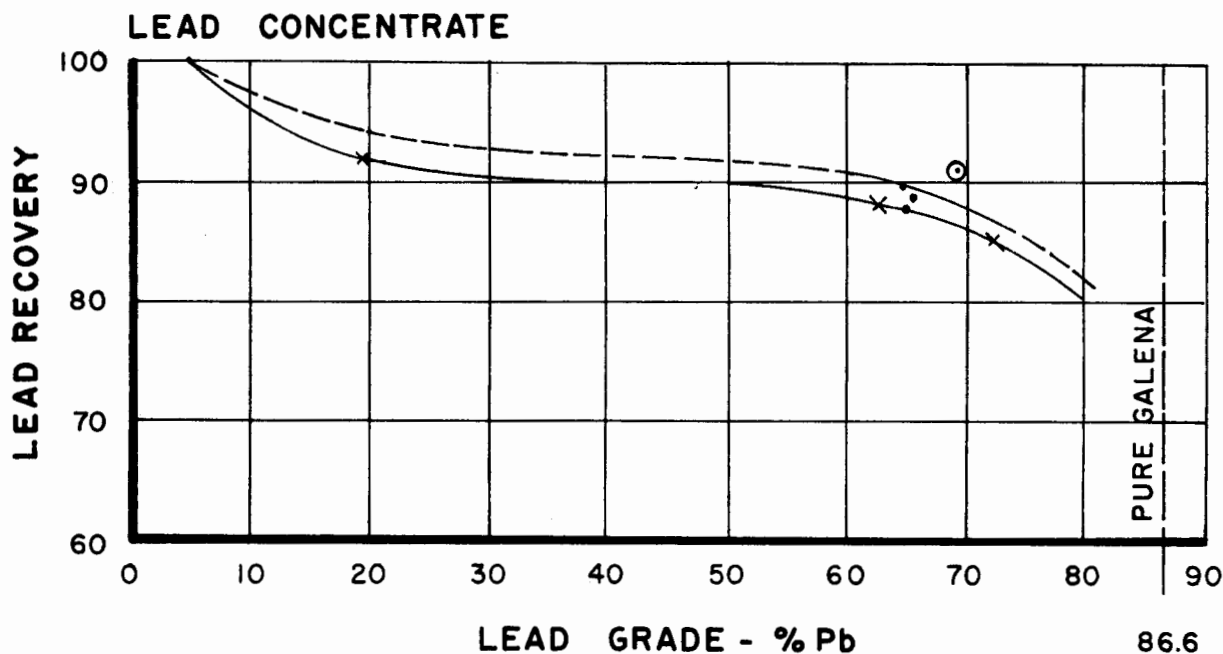
PROCESS DESIGN CRITERIA

The contemplated process comprises four principal steps:

1. Run-of-mine ore is reduced in size to less than 1/2-inch particle size by three stages of crushing and screening.
2. The 1/2-inch size material is further reduced in size by wet grinding in rod and ball mills to the point of maximum economic liberation of the lead and zinc mineral constituents.
3. The liberated ore minerals are selectively separated from the gangue material and selectively concentrated by flotation.
4. The flotation concentrate products are dewatered by thickening, vacuum filtering and thermal drying to yield final products which can be readily handled for shipment.

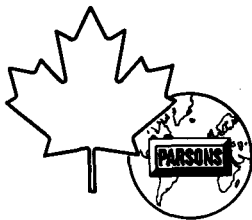
The initial daily average and design milling rates were 4500 and 4800 tons per day, respectively. The conceptual engineering-design and layout work were based on these values with provisions for an ultimate expansion in the sixth year of operation to average and design rates of 6750 and 7200 tons per day, respectively. During January, when confirming grinding test data indicated the original assumed kilowatt hour requirements of the primary grinding circuit were 25 percent higher than the test results, it was decided that the design of the concentrating facilities and economic evaluation of the project should be based on higher average and design

**GRADE - RECOVERY CURVES - GALIGHER LOT No. 1665
COMPOSITE SAMPLE
No. 1.**



- X = OPEN CIRCUIT TEST No. T-2
- = 4 SERIES LOCK-TEST No. T-3
- ⊙ = POINT USED FOR METALLURGICAL BALANCE COVERING FIRST 5 YEARS.

FIGURE 5-1



milling rates. Thus, the average and design milling rates were established at 5500 and 5830 tons per day respectively for years 1 to 5 and 8250 and 8740 tons per day for years 6 through 17, based on the 25 percent over-capacity factor in the primary grinding circuit and the fact that the original materials handling systems were sized on a design capacity of 7200 tons per day.

Primary Crushing and Stockpiling

The run-of-mine ore must be reduced to a nominal minus 1/2-inch in size as feed for the concentrator. This particle sizing requirement is predicated upon optimum operation of the primary grind rod mills. Three stage crushing with screening between stages yields the most economical arrangement in terms of power consumption and equipment sizes. The first stage of crushing will reduce the ore to a nominal 5-inch in size, the second stage to 1-1/4-inch and the third stage crushing results in a final size acceptable as feed for the primary grinding circuit. The screening between stages removes the fine material produced at each crushing stage as a finished, crushed product.

A stockpile of ore is provided as an integral part of the crushing plant. This stockpile will provide storage of from 5 to 6 days ore supply and consists of primary crushed product from which the minus 1/2-inch material has been removed. The removal of fines prior to stockpiling is accomplished to avoid freezing and possible oxidation of the fine material from prolonged exposure.

Grinding Operation

The primary grinding circuit will reduce the particle size of the crushed product (nominally 1/2-inch size) to approximately 80 percent through 110 microns. The Galigher Company's screen analysis accompanying their flotation test work indicates this fineness of grind is sufficient for economic liberation of the lead and zinc minerals.

Two stages of wet grinding are proposed. The first stage reduces the 1/2-inch material to approximately 80 percent minus 14-mesh by utilizing open-circuited rod mills operating at a pulp density of 75 percent solids by weight. The second stage of grinding is accomplished in ball mills operating in closed-circuit with hydrocyclones at a 250 percent circulating load. The hydrocyclone overflow contains the final, sized product from the grinding circuit in a pulp of 38 percent solids by weight.

Grindability tests on drill core samples performed by Mine and Smelter Supply Company, Allis-Chalmers Manufacturing Company and Hazen Research, Inc. are in close agreement as to the work indices of this ore. From these data, the kilowatt hour requirements of the primary grinding circuit have been calculated. The following tabulated results are based upon the work indices evolving from the Allis-Chalmers grinding tests.



1. Rod Mill Grind
 Feed particle size at 80 percent minus 12,700 microns (1/2-inch)
 Product particle size at 80 percent minus 900 microns (14 to 20-mesh)
 Bond Work Index 8.1
 KWH per DST 1.98
2. Ball Mill Grind
 Feed particle size at 80 percent minus 900 microns
 Product particle size at 80 percent minus 110 microns
 Bond Work Index 9.6
 KWH per DST 5.94

The total grinding energy required to reduce rod mill feed to a material size suitable for flotation is 7.92 KWH per DST. Grinding tests were performed on samples with two types of ore structures as determined by geological examination, namely fine grained and medium grained ores. Grinding test results for each type of ore were comparable.

Flotation Operation

The process design criteria used to develop the quantitative flowsheets for the lead and zinc flotation circuits follow closely the Galigher test data. The percent solids of the pulps in the cleaning circuits are particularly low and have been maintained as close to Galigher's test values as practicable.

The flotation operation is divided into the lead and zinc **sections**. Each of these sections contain rougher and scavenger circuits in which lead and zinc minerals are removed from the pulp as low grade concentrate products. These concentrate products are then reduced in size by further grinding in separate ball mill regrind circuits and finally brought up to grade by three stages of cleaning. Flotation reagents are added at various points in the flotation circuits as well as in the primary grinding circuits in preparation for the initial lead rougher-scavenger flotation. Flotation and conditioning times, percent solids and reagent usages are enumerated below.

<u>Circuit</u>	<u>Flotation or Conditioning Time Minutes</u>	<u>Pulp % Solids</u>	<u>Reagents lbs./ton of Feed</u>
Primary Ball Mills		70	3.0 Lime 0.15 NaCN 2.0 Zn SO ₄ 0.05 Z-11
Lead Rougher	3	38	0.03 MIBC
Lead Scavenger	5	38	0.03 MIBC 0.025 Z-11



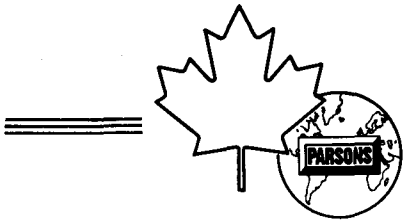
<u>Circuit</u>	<u>Flotation or Conditioning Time Minutes</u>	<u>Pulp % Solids</u>	<u>Reagents lbs./ton of Feed</u>
Lead Regrind		60	0.10 NaCN 0.10 Z-11 0.06 MIBC
Lead Cleaner 1st Stage	6	17	
Lead Cleaner 2nd Stage	5	25	
Lead Cleaner 3rd Stage	5	25	
No. 1 Zinc Conditioner	5	33	2.0 Lime
No. 2 Zinc Conditioner	5	33	1.5 CuSO ₄
No. 3 Zinc Conditioner	5	33	0.08 Z-200
Zinc Rougher and Scavenger	5	33	0.05 No. 65
Zinc Regrind Mill		60	2.0 Lime 1.5 CuSO ₄
Zinc Regrind Conditioner	5	15	1.0 Lime 1.5 CuSO ₄
Zinc Cleaner 1st Stage	7	14	
Zinc Cleaner 2nd Stage	6	34	0.5 Lime
Zinc Cleaner 3rd Stage	6	23	

In most tests the third stage cleaner circuits for both the lead and zinc concentrates are not indicated in the Galigher test work.

Regrind Ball Mill Circuits

The Galigher test work has shown that the rougher/scavenger concentrates of both lead and zinc should be reground for further liberation to obtain optimum upgrading on cleaning. The particle sizing of the reground product has been determined by Dr. Whelan of the University of Utah in conjunction with The Galigher Company as 100 percent through 40 microns with a median particle size of approximately 10 microns. Since Bond's procedure for determining grinding energy requirements is not applicable in this fine size range, the regrind ball mills were arbitrarily sized pending further data to be obtained from the pilot plant milling program. These mills are sized to provide 9.2 KWH of electrical energy per ton of rougher/scavenger lead concentrate and 10 KWH per ton of zinc rougher/scavenger concentrate. There is evidence from some of the Galigher test work that a fine regrind of the concentrates derived from the less finely grained ores may not be required. In this eventuality, a by-pass of the regrind circuits has been provided.

Middling products returned from both the lead and zinc cleaner circuits to their respective scavenger circuits would excessively dilute the scavenger pulps unless a thickening operation is included. High pressure hydrocyclones are proposed for this purpose in the present flowsheet. Cyclone overflows,



containing about 6 percent solids, are used for dilution in the regrind circuits. The cyclone underflows, containing from 21 to 26 percent solids, are returned to the respective scavenger circuits for additional recovery of residual lead and zinc mineralization.

Concentrate Dewatering

The water contained in the lead and zinc concentrate froths will approximate 55 to 57 percent as they leave the flotation section. This retained moisture will be reduced to about 5 percent for shipment as final product.

Dewatering is effected in three separate operations. In the first, the concentrates are thickened by sedimentation to about 50 percent solids prior to filtration. Test data are not yet available on settling rates; therefore for thickener sizing purposes, the following area requirements were assumed: lead concentrates 7.0 square feet per ton per 24 hours, and for zinc concentrates 7.5 square feet per ton per 24 hours. These values are subject to change in final design when more definitive data is developed from the pilot plant testing.

Thickener underflows are pumped to rotary vacuum drum, belt-discharge filters for further removal of water. These filters are sized on the basis of laboratory filtration tests performed in Salt Lake City by Eimco on the lead and zinc concentrates obtained from the Galigher testwork. These test results indicate a solids deposition rate of 45 pounds per hour per square foot at 9 to 10 percent retained moisture for the lead concentrate and 27.5 pounds at 14 to 15 percent water for the zinc concentrate.

The final removal of moisture to reach the desired 5 percent level in the products for shipment is realized in rotary dryers. These rotary dryers are sized, together with exit gas flues and dust recovery systems, to evaporate thermally 1.35 tons per hour from the lead concentrate and 3.59 tons per hour from the zinc concentrate.

Concentrate Storage and Loadout

Storage is provided for 4,000 tons of concentrates. Approximately two thirds of this total is for zinc concentrates, the balance for lead concentrates. This tonnage represents about 3 to 4 days output of concentrates.

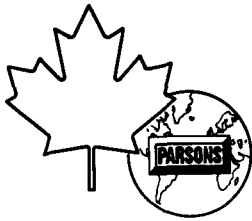


SECTION 6

PLANT OPERATING COSTS

The plant operating costs discussed in this section are categorized into those incurred at the minesite and those incurred at the tidewater facility at Haines, Alaska. In each instance, these costs exclude all costs related to the mining and transportation operations. The basic criteria which have been used in the development of these costs and the sources are enumerated below.

1. The annual tonnage of milled ore for the first five-year period is 2.000×10^6 short tons; for years 6 to 17, 3.000×10^6 short tons.
2. During the first five-year period, the annual lead and zinc concentrate production is 128,400 and 240,200 dry short tons respectively; for years 6 to 17, the comparable values are 130,100 and 282,900.
3. The work classifications, wages, salaries, hours of work, statutory holidays and premium pay as outlined in Foster Economic Consultants, Limited, report of December, 1966, and S. D. Allen's correspondence of December 23, 1966, to R. E. Thurmond, were used in the computation of labor costs at the plantsite.
4. Similar data for the employees located at tidewater in Alaska were used from H. G. Pearce's letter of December 22, 1966, to R. E. Thurmond with Contact Report No. 42 in the computation of labor costs at this operation.
5. Diesel feed for power generation and solar fuel for heating costs were calculated on the basis of pricing quoted in correspondence to R. E. Thurmond from J. Van Zonneveld of British-American Oil Company dated October 26, 1966.
6. Coal costs at the plantsite (no back-haul costs included) were obtained from R. E. Thurmond in a teletype dated November 18, 1966.
7. Rod and ball mill grinding media are priced based on a quotation from The Algoma Steel Corporation, Limited, dated December 9, 1966.
8. Rod and ball mill liners are priced based on a quotation received from Dominion Engineering Limited dated December 23, 1966.



9. Reagent costs were calculated from quotations obtained from Van Waters and Rogers, Limited, December 29, 1966; Cyanamid of Canada, Limited, December 21, 1966; and Dow Chemical of Canada, Limited, December 16, 1966.
10. Power costs were derived from data obtained from E. W. Humphrey of the Northern Canada Power Commission in Ottawa on January 6, 1967.
11. Raw limestone costs (delivered to the plantsite) were obtained verbally from R. E. Thurmond in Los Angeles on December 20, 1966.
12. Types of reagents and estimated usages were obtained from The Galigher Company report dated September 3, 1966.
13. Grinding media and mill liner consumption values were calculated from the grinding test results presented by Allis-Chalmers Manufacturing Company.
14. Freight costs for materials and supplies laid down at the plantsite were based upon a calculated rate of \$1.40 per cwt from North Vancouver to Haines, Alaska. A free backhaul from Haines to the plantsite was assumed. The estimated rate of \$1.40 per cwt was proportioned from the over-all rate of \$2.40 per cwt quoted by the White Pass and Yukon System between North Vancouver and Whitehorse.

PLANTSITE OPERATING COSTS

The plantsite operating costs for the period, years 1 to 5 and years 6 to 17, are shown in Table 6-1.



TABLE 6-1

PLANTSITE OPERATING COSTS*

	Years 1 to 5	Years 6 to 17
Labor	\$0.52	\$0.38
Reagents	0.60	0.58
Grinding Media and Liners	0.29	0.26
Fuel (Coal)	0.32	0.23
Electric Power	0.42	0.28
Maintenance Materials, Operating Supplies and Spare Parts	0.25	0.20
Total	\$2.40	\$1.93

*All costs are in United States dollars per dry short ton of ore.

Based on these unit costs, the annual costs are $\$4.808 \times 10^6$ for the first five-year period and $\$5.802 \times 10^6$ for years 6 to 17.

Labor

The labor costs are predicated on a total of 119 personnel, excluding all mine and transportation personnel. The breakdown of this total is approximately as shown below:

<u>Category</u>	<u>Number of Personnel</u>
Staff, Administrative and Clerical	16
Production and Maintenance Supervision	9
Purchasing and Warehouse	6
Plant Engineering Services	3
Production Crews	53
Technical Services	9
Maintenance, Laborers and Miscellaneous	<u>23</u>
Total	119



The manpower requirements have been developed on a three-shift rotating basis with relief operators and helpers to cover the three extra shifts per week. A normal schedule consists of 48 hours per week and 50 weeks per year. To the basic wage rates and salaries depicted in the aforementioned report of Foster Economic Consultants, Limited, the following factors have been applied as required.

1. Shift differential - average for all three shifts of 4.9 cents per hour for all continuous shift work.
2. Hours of work - 4.1 percent of base rate to cover 44 hours at straight time rate and 4 hours at premium rate of 1 1/2 times base rate.
3. Scheduled Saturday and Sunday work - for continuous shift work average of 3.0 cents per hour; for normally scheduled six-day-per-week operations, 1.5 cents per hour.
4. Holiday pay - 3.8 percent of base rate for eight scheduled holidays at premium rate of 1 1/2 times base rate.
5. To the basic wage rates adjusted in accordance with the above-listed four items and to the salary schedules, a factor of 20 percent has been applied to cover payroll taxes, insurance, vacation, pension and the other welfare and fringe benefits as outlined.

Reagents

The reagent consumption based on the Galigher Company report of September 3, 1966, is as follows.

<u>Reagent</u>	<u>Consumption (pounds per ton of ore)</u>
Sodium Cyanide	0.25
Zinc Sulfate	2.00
Sodium Isopropyl Xanthate	0.175
Methyl Isobutyl Carbonol	0.12
Hydrated Copper Sulfate	3.00
Z - 200	0.16
Frother No. 65	0.05
Calcium Hydroxide	8.50



The calculated cost of reagents is \$0.60 per ton for years 1 to 5 and \$0.58 per ton for years 6 to 17. These costs are lower than those based solely on the reagent consumptions shown and quoted prices obtained. They are, however, realistic and attainable for the following reasons:

1. The Galigher testwork did not investigate the minimum reagent quantities which could be achieved.
2. The Galigher testwork did not investigate other possible reagent consumptions which might be less costly.
3. The Galigher testwork (both batch and locked-cycle) does not reflect the beneficial effects which accrue from the recycle streams encountered in plant operation.
4. In final purchasing negotiations, it is felt that price reductions could be effected for quantity, long-term commitments by Anvil.
5. Zinc and copper sulfates can be produced from available plant materials at a modest capital requirement for about one half the quoted selling prices.

Grinding Media and Liners

The grinding media and liner costs are based on the revised steel consumptions calculated from the grinding test results obtained from Allis-Chalmers Manufacturing Company. The following tabulation shows the consumption values:

STEEL CONSUMPTION (POUNDS PER DRY SHORT TON)

	<u>Grinding Media</u>	<u>Liners</u>
Rod Mill	0.40	0.07
Primary Ball Mills	0.86	0.20
Regrind Ball Mills	0.86	0.20



With grinding media prices of 7.8 cents per pound for the rod mills, 7.9 for the primary ball mills, and 10.4 for the regrind mills, the total grinding media cost for years 1 to 5 is \$0.21 per dry short ton of ore. For liner prices of 22.5 cents per pound for the rod mills and 16.0 cents per pound for the primary and regrind mills, the total liner cost for years 1 to 5 is \$0.08 per dry short ton of ore. Thus, the combined grinding media and liner cost is \$0.29 per dry short ton of ore for this period. For years 6 to 17, the comparable combined cost is estimated at \$0.26 per dry short ton.

Fuel (Coal)

Using a delivered coal price of \$10.00 per ton at the plantsite, the total estimated fuel cost per dry short ton of ore milled is \$0.32. This cost includes process fuel requirements and plant heating requirements. That portion germane to the process requirements is \$0.14 per ton of ore and for plant heating the comparable value is \$0.18 per ton. For years 6 to 17, the equivalent total cost is \$0.23 per dry short ton of ore milled.

Electric Power

The total power cost to the operation is \$832,500 per year. This value has been established as a demand charge irrespective of the energy consumption (and to a more or less degree irrespective of the demand). At the presently contemplated 5,500 ton per day average milling rate the estimated kilowatt hour cost would approximate 11 to 12 mills. On the basis of an annual milled tonnage of 2.000×10^6 dry short tons for years 1 to 5, the unit cost is \$0.42 per ton; for an annual tonnage of 3.000×10^6 dry short tons in years 6 to 17, the unit cost is \$0.28 per ton.

Maintenance Materials, Operating Supplies and Repair Parts

These annual costs have been estimated as a percentage of the total direct field costs. Based on a value of 1 percent of the cost of buildings and structures and 3 percent of equipment, the annual cost is estimated at \$0.497 $\times 10^6$. For the first five-year period, this equates to a unit cost of \$0.25 per dry short ton of ore milled; for years 6 to 17, the comparable unit cost becomes \$0.20 per ton.



TIDEWATER OPERATING COSTS

The tidewater operating costs for the first five-year period and years 6 to 17 are shown below in Table 6-2.

TABLE 6-2

TIDEWATER OPERATING COSTS*

	<u>Years 1 to 5</u>	<u>Years 6 to 17</u>
Labor	\$0.50	\$0.46
Fuel (Oil)	0.23	0.22
Maintenance Materials, Operating Supplies and Spare Parts	<u>0.27</u>	<u>0.24</u>
Total	\$1.00	\$0.92

*All costs are in U. S. dollars per dry short ton of concentrate

Using these unit costs, the annual expenditures for the tidewater operation are estimated at $\$0.369 \times 10^6$ for years 1 to 5, and $\$0.380 \times 10^6$ for years 6 to 17.

Labor

The labor costs are predicated on a total permanent force of 14 personnel at the tidewater site. During outbound movements of concentrates, it is assumed stevedoring gangs will be hired locally on a temporary basis to supplement the permanent forces. **Thirteen outbound movements per year of approximately 30,000 tons each** have been assumed for years 1 to 5 and 15 annually for years 6 to 17. The loading period for each outbound shipment has been estimated at 48 hours. Costs have been developed on the basis of two 12-hour shifts per day with premium time for the four hours worked in excess of eight. Seven-man crews have been established for the first and last 12-hour shift of each movement and five-man crews for the two intervening 12-hour shifts. A com-



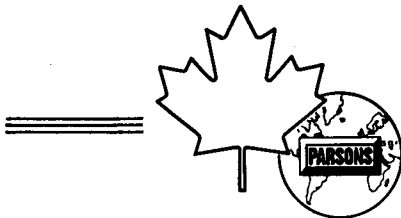
posite rate of \$6.65 per hour including overtime, but exclusive of payroll taxes, insurance, welfare and fringe benefits, has been used in estimating these temporary labor costs. The total labor unit cost of \$0.50 per dry short ton of concentrates is comprised of \$0.40 per ton for the permanent personnel and \$0.10 for the temporary personnel. For years 6 to 17, the respective values are \$0.36 and \$0.10 per dry short ton of concentrates.

Fuel (Oil)

Fuel oil is used at the tidewater site for heating and power generation. The fuel oil heating cost is estimated at \$0.09 per ton of concentrates for years 1 to 5 and \$0.08 per ton for years 6 to 17. Diesel oil costs for power generation are calculated to be \$0.14 per ton of concentrates for each of the time periods considered.

Maintenance Materials, Operating Supplies and Spare Parts

These annual costs have been estimated on the basis of a percentage of the total direct field costs of the complete tidewater facility. Using a value of 1 percent of the cost of buildings and structures and 3 percent of equipment costs, the annual cost amounts to \$0.101 x 10 . For years 1 to 5, this equals a unit cost of \$0.27 per ton of concentrates and for years 6 to 17, the annual unit cost is \$0.24 per ton.

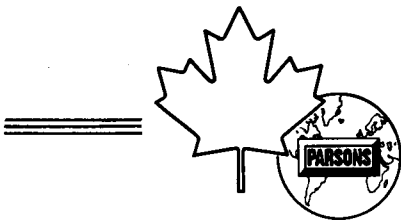


SECTION 7

DESCRIPTION OF PHYSICAL FACILITY

The development of the metallurgical flowsheet and the process design criteria used in sizing the major units of process equipment have been described in Section 5. The following text delineates the physical facilities which constitute the operations at the minesite and tidewater and which served as the basis for the capital cost estimate. For clarity, the following outlines the subject matter of this section.

1. Primary Crushing and Coarse Ore Storage
2. Secondary and Tertiary Crushing
3. Concentrator
 - a. Fine Ore Storage and Primary Grinding
 - b. Lead Flotation Circuit
 - c. Zinc Flotation Circuit
 - d. Concentrate Thickening, Filtration and Drying
4. Concentrate Storage and Load-Out
5. Concentrate Unloading, Storage and Ship Loading (Tidewater)
6. Tailings Disposal
7. Services and Utilities
 - a. Electrical Systems
 - b. Heating, Ventilating, Dust and Fume Control Systems
 - c. Water Supply, Storage and Distribution



- d. Fire Protection System
- e. Nonprocess Structures and Buildings
- f. Miscellaneous Services

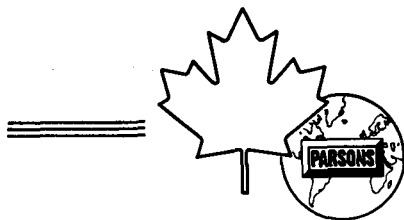
The entire process plant facility is located slightly south and west of the proposed open pit mining operation at an elevation of 3,850 to 3,912 feet. It is bounded on the east by coordinate 10,000E and on the west by 8,500E and essentially lies between coordinates 6,500N and 7,500N. The approximate area encompassed by the facility is 35 acres.

In the development of the general arrangement and plot plan drawings, an attempt has been made to group the process and nonprocess structures in as compact an area as practical to minimize the need for exposed traffic between buildings. Required traffic can be accomplished largely by conveyor tunnels and galleries. The plant road network provides peripheral access to the whole site as well as internal access to the major operation areas and contemplates tie-ins with the mine haul road, permanent access road to the site and existing road from the camp to the pit location.

PRIMARY CRUSHING AND COARSE ORE STORAGE

This circuit has been designed for a maximum rate of 1,400 tons per hour discharge from the primary crusher. Mine-run ore will be hauled from the pit to the dump pocket in rear-dump mine trucks as described in Section 4. The 48-inch gyratory, primary crusher, and auxiliary equipment are located in a concrete structure 42-feet wide by 65-feet long extending approximately 66-feet below grade. A structural steel framed building is located atop this concrete structure and provides dumping access on three sides of the crusher.

Mine-run ore will be crushed to 100 percent passing, 7 1/2 inches, and 80 to 85 percent passing 5 inches in size. The primary crushed ore will drop into a 100-ton capacity surge hopper located under the crusher. The minus 7 1/2-inch material will be withdrawn from the surge hopper at a uniform rate by a 60-inch wide apron feeder and discharged to 494-foot long, 36-inch wide belt conveyor. This belt conveyor elevates and transports the material to a 100-ton (live) capacity surge bin in the west end of the fine crushing plant, located adjacent to the concentrator building. Ore is withdrawn from the bin by two 60-inch wide feeders, each handling 700 tons per hour, and fed to two 72-inch wide, parallel-operating, double-deck screens. Screen undersize (minus 1/2-inch material) is conveyed directly to the rod mill feed bins. Oversize material (minus 7 1/2 inches plus 2 inches and minus 2 inches plus 1/2 inch in size) is conveyed and elevated on a series of three



36-inch wide belt conveyors each rated at 1,215 tons per hour to the coarse ore storage building located intermediate to the primary and fine crushing plants.

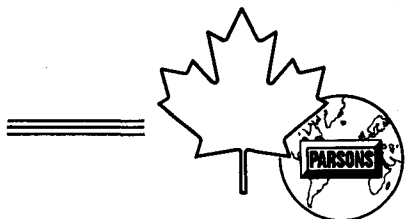
The coarse ore storage building provides a live storage of 30,000 tons. The pile is enclosed by a structure approximately 300 feet long and 90 feet wide with the ridge-line of the building approximately 90 feet above the floor of the reclaim tunnel. Adjacent to the west end of the building is a 38-foot wide by 70-foot long building which serves as the transfer tower for the materials handling systems and houses the dust collection, electrical and auxiliary equipment, and provides the main access to the reclaim tunnel.

Primary crushed ore is stockpiled in the coarse ore storage building by means of a 36-inch wide belt conveyor and tripper having a travel of 217 feet. The pile height is about 50 feet above the reclaim tunnel roof. The building is triangular in section with the sides sloped at 45 degrees. Dead storage of ore is minimized by permitting the natural slope of the excavation to retain the storage pile.

The minus 7 1/2-inch plus 1/2-inch material is withdrawn at the rate of about 840 tons per hour with 48-inch wide by 7-foot long vibrating feeders. Eighteen feeders are provided under the pile, any two of which can sustain the design feed rate. The feeders, along with the 36-inch wide by 284-foot long collecting conveyor are located in the reclaim tunnel under the coarse ore storage building. This tunnel is approximately 40 feet wide, 302 feet long and 11 feet, 6 inches high.

SECONDARY AND TERTIARY CRUSHING

These facilities are located in a steel-framed structure, 60 feet wide by 112 feet long and 90 feet high, located adjacent to the east end of the primary grinding bay of the **concentrator**. Housed in this structure are the coarse ore screens; the secondary crusher and screen; and tertiary crushers and screens. Feed to the secondary crusher is discharged from the collecting conveyor under the coarse ore storage building to a 36-inch wide by 343-foot long belt conveyor which transports and elevates the material to the fine crushing plant. The 7-foot secondary cone crusher is fed by means of a 36-inch wide by 21-foot, 6-inch long retractable belt conveyor. At a closed side setting of 1 1/4 inches, the secondary crusher will produce a product of 60 to 65 percent through 1 1/4 inches and 10 to 15 percent through 1/2 inch in size. The crusher product will discharge to a 72-inch wide screen located under the crusher. Screen undersize (minus 1/2-inch material) is conveyed to the rod mill feed bins and the oversize material (plus 1/2 inch in size) joins the oversize from the two tertiary screens and is returned as tertiary crusher feed.



The feed to the tertiary crushing circuit, including recirculating load, is comprised of plus 1/2-inch material from the secondary crusher screen and the oversize material from the two tertiary screens. This material is conveyed by a series of two 36-inch belt conveyors, at a rate of approximately **1,135 tons per hour, via the transfer tower to a 100-ton (live) capacity surge** bin located above the tertiary crushers. Each tertiary crusher is fed by a 36-inch wide by 24-foot, 6-inch long retractable belt conveyor. The tertiary crushers are 7-foot cone crushers operating at a closed side setting of 3/8 inch and producing a product 85 percent passing 1/2 inches and about 70 percent passing 3/8 inch. Each tertiary crusher will discharge to a 72-inch wide single deck screen located immediately beneath the crusher. The tertiary screen oversize with the plus 1/2-inch material from the secondary screen is recirculated as described previously.

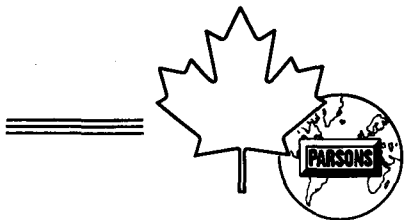
Sized rod mill feed (minus 1/2-inch material) is collected from coarse ore screen, secondary crusher screen, and tertiary crusher screen undersizes onto a 30-inch wide by 296-foot long belt conveyor and conveyed to the transfer house. The material is transferred to a 30-inch wide by 52 foot, 6-inch long conveyor, and then to a 30-inch wide by 545-foot long conveyor with a tripper having a travel of about 156 feet for distribution to the four rod mill feed bins.

The structure which houses the transfer points for the tertiary crusher feed and rod mill feed materials handling systems is a steel-framed structure approximately 26 feet wide, 65 feet long and 38 feet high and is located to the east of the concentrator building.

CONCENTRATOR

The concentrator building is a rectangular, steel-framed structure approximately 242 feet wide by 347 feet long. The proposed arrangement contemplates five operational bays. The northern most bay is the fine ore storage area comprising four 3,000-ton (live) capacity bins for rod mill feed storage. These bins are enclosed in a structure 57 feet wide by 198 feet long and 85 feet high with a floor elevation of 3,882 feet. The next bay is the primary grinding bay which is 60 feet wide by 242 feet long and about 70 feet high. The operating floor is at elevation 3,882 feet and the pump floor at elevation 3,865 feet. This bay encloses all **six primary** grinding mills, pumps, rod and ball storage, and a maintenance and storage area at the extreme eastern end. This bay is serviced by a 20/5-ton capacity travelling bridge crane for its full length. Immediately adjacent to the east end is the fine crushing plant.

To the **south** of the primary grinding bay lies the service and utility bay. This area is 25 feet wide by 242 feet long and consists of three levels.



The lowest level comprises the cable vault. Immediately above this is the switchgear room. The upper level provides the space for the control room, offices, instrument repair and storage. The floor elevation in this bay is 3,853 feet. Adjacent to the service and utility bay is the flotation and regrind area. This area is 110 feet wide by 242 feet long and 65 feet high and has a common floor elevation with the utility bay. In addition to the flotation and regrind circuits, this bay houses the flotation conditioners; reagent storage, preparation and distribution systems; and the lead concentrate thickener. The main operational levels are: regrind mills at 3,853 feet; flotation cells at 3,873 feet; and an auxiliary operating platform at elevation 3,885 feet. This bay is serviced by two travelling bridge cranes. A 20/5-ton capacity crane with a span of about 110 feet is used for the flotation and regrind circuits with a 2-ton unit of about 44-foot span, servicing the copper sulfate and zinc sulfate reagent preparation area.

The final bay of the concentrator is 95 feet wide by 242 feet long and 71 feet high with a floor elevation of 3,847 feet. The principal units housed in this area are the lead and zinc concentrate filters and dryers, boiler plant, coal bunkers for the boilers and dryers, zinc concentrate thickener and sample preparation and storage facilities. The two main operational levels are located at elevations 3,847 and 3,873 feet with auxiliary working levels interspersed. The west end of the bay has the boiler plant with its auxiliaries, the east end has the zinc concentrate thickener, and the center portion contains the zinc and lead concentrate filters and dryers.

The mill operation comprises two parallel primary grinding circuits. Each circuit consists of two fine ore storage bins with belt feeders, one open-circuited rod mill and two ball mills closed-circuited with hydrocyclones. Feed for the rod mills is withdrawn from each fine ore bin by three 36-inch wide belt feeders, two of which are manually adjusted and the third automatically controlled from the belt scale on the rod mill feed conveyor. The discharges of the three belt feeders are collected on a 24-inch by 35-foot long conveyor which in turn discharges to the 24-inch by 65-foot long inclined belt conveyor feeding the rod mill. The rod mill is nominally sized at 9 by 12 feet operating at a pulp density of 2.28 equivalent to 75 percent solids by weight. Rod mill discharge is split and combined with the discharge from one ball mill, diluted to about 56 percent solids and classified at 110 microns in one 20-inch hydrocyclone. Cyclone underflow comprises ball mill feed while the overflows are collected, sampled and become feed to the lead rougher flotation circuit. The ball mills are nominally sized at 9 by 10 feet operating at a pulp density of 2.03 equivalent to 70 percent solids by weight. The discharge from the ball mill returns to the cyclone feed pump sump.

The cyclone overflows after proper reagentizing are subjected to treatment in the lead rougher flotation circuit. Tailings from the rougher circuit



are scavenged in the lead scavenger circuit. Tailings from the lead scavenger circuit constitute feed to the zinc flotation circuit. Lead rougher concentrate combined with scavenger concentrate is further reduced in size in the lead regrind circuit. The lead regrind circuit consists of two 9- by 10-foot ball mills operating in closed circuit with 15-inch hydrocyclones. Overflow from the cyclones (approximately 50 percent through 10 microns) becomes feed to the first stage lead cleaners. The lead cleaner flotation circuit involves three stages of cleaning with the second and third stage tailings being recycled to the first stage feed. The first stage cleaning is carried out in a dilute pulp averaging 14 percent solids by weight. First stage cleaner tailings are thickened in five 10-inch hydrocyclones with the underflow recirculated to the lead scavenger circuit and the overflows used for dilution in the lead regrind circuit. Final lead concentrate from the third stage cleaners at 69 to 70 percent lead by weight is sampled then thickened in a 50-foot diameter thickener.

Tailings from the lead scavenger cells is sampled, thoroughly conditioned and reagentized prior to flotation in the zinc rougher circuit. Tailings from the rougher circuit are scavenged in the zinc scavenger circuit. Tailings from the zinc scavenger circuit constitute final plant tailings and are sampled before ultimate disposal. Zinc rougher concentrate is combined with scavenger concentrate for further size reduction in the zinc regrind circuit. This circuit consists of two 9- by 10-foot ball mills operating in closed-circuit with 15-inch hydrocyclones. Overflow from the cyclones (approximately 50 percent through 10 microns) is further conditioned and reagentized prior to cleaning. The zinc cleaner circuit is a conventional three stage circuit with the second and third stage tailings being recycled to the first stage cleaner feed. Subsequent to thickening in hydrocyclones the first stage cleaner tailings are retreated in the zinc scavenger cells. Cyclone overflow is used for dilution in the zinc regrind and second stage zinc cleaner circuits. The final zinc concentrate from the third stage cleaners averaging 53 to 54 percent zinc by weight is sampled then thickened in a 75-foot diameter thickener.

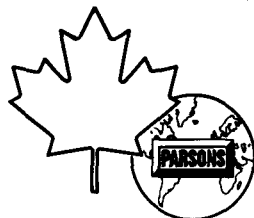
The reagents used in the circuit, the consumptions and points of addition are shown in Table 7-1.



TABLE 7-1

REAGENT CONSUMPTION

<u>Reagent</u>	<u>Total Pounds Per Dry Short Ton Ore</u>	<u>Points of Addition</u>
Zinc Sulfate	2.0	25 percent to each of the four primary grind ball mill circuits
Hydrated Copper Sulfate	3.0	50 percent to the No. 2 zinc rougher-scavenger flotation conditioner 50 percent to zinc regrind circuit conditioner
Calcium Hydroxide	8.5	6 percent to each primary grind rod mill 6 percent to each primary grind ball mill 23 percent to the No. 1 zinc rougher-scavenger flotation conditioner 12 percent to the zinc regrind circuit conditioner 23 percent to the zinc regrind ball mill circuit 6 percent to the second stage zinc cleaner flotation feed
Sodium Cyanide	0.25	15 percent to each of the four primary grind ball mills 40 percent to the lead regrind ball mill circuit
Z-11	0.175	7 percent to each of the four primary grind ball mills 14 percent to the lead scavenger flotation feed 58 percent to the lead regrind ball mill circuit
Z-200	0.16	50 percent to the No. 3 zinc rougher-scavenger flotation conditioner 50 percent to the zinc regrind ball mill circuit

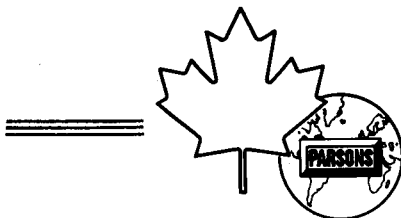


<u>Reagent</u>	<u>Total Pounds Per Dry Short Ton Ore</u>	<u>Points of Addition</u>
MIBC	0.12	25 percent to the lead rougher flotation circuit 25 percent to the lead scavenger flotation circuit 50 percent to the first stage lead cleaner flotation circuit
Frother No. 65	0.05	100 percent to the zinc rougher flotation circuit

Underflow from the lead concentrate thickener at 50 percent solids is further dewatered, on two 11-foot, 6-inch diameter by 14-foot long belt filters, to about 10 to 12 percent retained moisture. The filter cake is conveyed on a 24-inch wide by 51-foot long conveyor to the feed chute of the lead concentrate dryer. The lead concentrate rotary dryer is a direct-fired, pulverized-coal unit 5 feet in diameter by 40 feet long with a combustion chamber of about 20 feet in length. Lead concentrate product for shipment is discharged at plus or minus 5 percent retained moisture. Overflow from the concentrate thickener joins the plant tailings stream for ultimate disposal. The zinc concentrate filtration and dryer operation is identical to that of the lead concentrate except it uses four filters and two dryers.

CONCENTRATE STORAGE AND LOAD-OUT

Storage is provided at the minesite for a total of 4,000 tons of dried concentrates (5 percent retained moisture) of which 2,700 is for zinc concentrates and 1,300 for lead concentrates. Three bins, each 40 feet in diameter by 48 feet high overall with a cylindrical section 20 feet deep, provide the necessary capacity. Concentrates are transported from their respective dryers to the storage bins by means of 24-inch wide by 348-foot long inclined belt conveyors. The zinc concentrate is discharged to a 24-inch by 50-foot long reversible shuttle belt conveyor for discharge to either of the two bins provided. These bins are enclosed in a 70-foot wide by 117-foot long and 95 foot high structural steel framed building. Each bin is provided with a vibrating, whirlpool-type live bottom. Two drive-through aisles are provided, one under the two zinc concentrate bins and one under the lead concentrate bin. After loading, the trucks are weighed on a 50-ton capacity, 60-foot long platform scale.



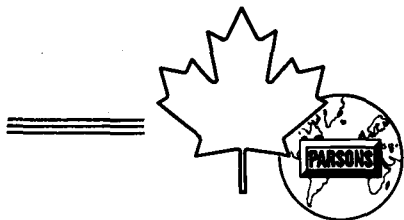
LIME PLANT

The lime plant is a complete entity for the preparation of milk of lime slurry from raw limestone. It is located adjacent to the east end of the fine ore storage bay of the concentrator building in a steel-framed structure 57 feet wide by 156 feet long. The floor slab elevation is 3,865 feet, equivalent to that of the grinding bay in the concentrator. For a distance of 44 feet the roof elevation is maintained the same as that over the fine ore storage bay; for the balance (112 feet) the roof elevation is dropped to 3,952 feet elevation and continues at this height to the east column line of the structure. Enclosed within the building are two 650-ton raw limestone storage bins, a 7-foot diameter by 70-foot long rotary kiln, a 42-inch diameter by 32-foot long rotary cooler, one 150-ton calcined limestone storage bin, two 150-ton coal storage bins, a 2,000 pound per hour coal pulverizer unit, a coal unloading materials handling system, a 5-foot diameter lime ball mill and 24-inch classifier circuit, two 18-foot diameter by 17-foot high milk of lime storage tanks and miscellaneous feeders, pumps and materials handling equipment.

Raw limestone will be received once per week each Sunday from the quarry. Haulage will be effected in truck units similar in size and dumping requirements to the Anvil mine trucks. The limestone is crushed in the primary and secondary crushing plants but will bypass the coarse ore storage building. The stone removed from the crushing circuit for kiln feed will be minus 2-inch plus 1/2-inch in size and will be stored in two 650-ton capacity bins. Under-size (minus 1/2-inch) material is conveyed to a waste storage bin for ultimate disposal by truck. The sized kiln feed will be calcined in a pulverized coal-fired refractory lined, rotary kiln approximately 7 feet in diameter by 70 feet long. Calcined stone after discharge from the kiln will be cooled in an air-swept 42-inch diameter by 32-foot long rotary cooler. The cooled calcine is withdrawn from the bin by an 8-inch wide by 36-inch long vibrating feeder and conveyed on a 14-inch wide by 28-foot long belt conveyor to the ball mill scoop box. The calcined limestone is ground and slaked in a ball mill-classifier circuit using a 5-foot diameter by 6-foot long ball mill and 24-inch diameter spiral classifier. The sand product from the classifier is returned to the mill scoop box and the classifier overflow, after adjustment to the proper percent solids, is stored in the two agitator-equipped 18-foot diameter by 17-foot high storage tanks.

FACILITIES AT TIDEWATER

The tidewater facility which is located in the Haines, Alaska, area provides means for unloading the concentrate trucks; permits a total storage of 20,000 tons each for zinc and lead concentrates; and has a shiploading belt conveyor

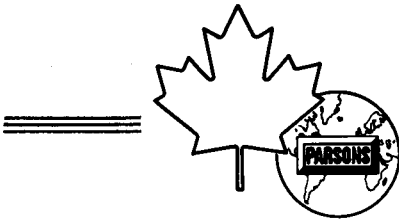


system. Concentrate trucks will unload into a double pocket, dump-hopper onto two vibrating feeders which discharge to an inclined belt conveyor feeding the storage building. The concentrate storage shed is 100 feet wide by 558 feet long and approximately 60 feet high. Zinc and lead concentrates are stored at diametrically opposed ends, each 260 feet in length. Discharge of the concentrates to storage is accomplished by means of a 235-foot long reversible shuttle conveyor. Five draw-off points are located under each storage pile. Concentrates are withdrawn by means of vibrating, whirlpool - type hoppers. To remove the total tonnage from storage will require working the pile with a rubber-tired bulldozer or front-end loader. Concentrates are discharged through the hoppers to a 48-inch wide by 300-foot long collecting conveyor which discharges to a 48-inch wide belt conveyor transporting the concentrates to the ship loader. The ship loader employs a retractable, articulated belt conveyor which discharges material into the ship's hatches. The dock facility at tidewater is a T-shaped, concrete-decked causeway 25 feet in width for a length of 170 feet from shore then enlarging to a 30-foot wide by 50-foot long pier head. Mooring dolphins with powerized capstans are located 350 feet on each side of the causeway centerline. Between the extreme mooring dolphins and pier head, breasting dolphins are spaced on 50-foot centers. The pier head, breasting dolphins and mooring dolphins are interconnected by walkways. The causeway and pier head are supported on steel, H-beam piling.

TAILINGS DISPOSAL AND IMPOUNDMENT

Tailings disposal and impoundment will be in accordance with the arrangements shown on Drawings Nos. 000-1003, 000-3801, and 550-3801. Tailings flow will be generated at three points in the Concentrator Building: zinc scavenger cells, and lead and zinc thickener overflows. Overflows from the lead and zinc thickeners will be transported via separate 12-inch and 10-inch diameter buried lines respectively, to individual settling ponds located southwest of the building. The purpose of the ponds is to allow settling of the concentrates contained in the thickener overflows which may be generated by thickener upset, while excess water is decanted to the tailings impoundment area. Ponds have been sized to provide sufficient capacity for approximately six months storage of settled concentrates, at which period the solids are assumed to be extracted from the ponds by mechanical means. Removal of the solids will renew initial storage capacity of the ponds and increase available concentrate output of the mill.

Tailings flow generated at the zinc scavenger cells, will be collected and transported via a 12-inch diameter underground line to a junction box located immediately south of the thickener overflow settling ponds. Tailings flow from the scavenger cells, based on a design rate of 5,830 tons per day, was



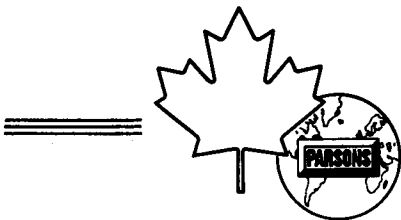
estimated to be 2,000 gallons per minute containing 30 percent solids. At the junction box, the scavenger tails will combine with the decanted water from the settling ponds, which is estimated at 1,570 gallons per minute. From the junction box, the combined flow, estimated at 3,570 gallons per minute containing 20 percent solids, will be directed to the impounding area via a 14-inch diameter pipeline run above-ground on sleepers. Where the pipeline crosses existing washes or deep ravines, light span steel joist trestles will be provided to support the pipe and maintenance vehicles.

The 14-inch diameter pipeline will follow existing contours along the high side of the impoundment area, at a 2 percent gradient to a point terminating at the proposed impoundment dam site. Terminal elevation of the tailings line was set at 3,600 feet to allow raising of the dam and ponding area without requiring future movement of the pipeline. Discharge from the pipeline will be via three spigots and ditches cut into the hillside, to a point located at the toe of the ponding area. The ditches will be trapezoidal shaped and closed on top, with insulated wood covers to allow the tails discharge to flow under any ice that may form on the pond.

The proposed impoundment dam will be located at a point approximately 2 1/2 miles west of the plant site. Initial dam height was set at elevation 3,480 feet to contain approximately five years' tailings storage capacity (approximately 8,000,000 dry short tons), before any further raising of the dam is required. Ultimate storage capacity required for an estimated 20-year plant life is approximately 39,000,000 dry short tons. Maximum height of the initial dam will be approximately 50 feet at its low point in the center of existing Rose Creek. Proposed dam construction consists of an impervious clay core in the center, covered on the upstream and downstream sides with sand and gravel material. Construction of the initial dam should be in accordance with the typical section shown on Drawing No. 550-3801 indicating construction of the maximum foundation required to support the ultimate dam height. This will allow the dam to be raised atop a solid foundation as compared to new fill imposed on slimes which will locate adjacent to the upstream section of the initial dam.

Location of the impounding area is in a valley longitudinally traversed by an existing stream, Rose Creek. To provide sufficient tailings storage capacity in the valley, Rose Creek is to be dammed, and diverted around the impounding area. The diversion dam is to be located at a point upstream of the maximum tailings ponding area. Construction of the diversion dam will be similar to that proposed for the tailings impoundment dam. Elevation of the dam crest has been established at 3,550 feet based on available hydraulic and topographical data.

The diversion ditch will commence at a point upstream of the diversion dam and run parallel to and on the south side of the tailings impoundment area to a point where it will reenter Rose Creek west of the tailings dam.



Capacity of the ditch cross section was set at 1,500 cubic feet per second, based on available hydraulic information. Flood control provisions in the form of spillways are recommended for both the diversion and tailings impoundment dams.

Water discharging from the impoundment area will be removed by means of a decant system run under the tailings dam. Quality of effluent discharged into Rose Creek from the impounding area should be controlled by chemical means, if required, to protect humans, fish and wildlife.

Based on the contemplated expansion commencing in the sixth year of operation from an initial design rate of 5,830 tons per day at a head assay of 12.2 percent combined lead and zinc to an ultimate design rate of 8,740 tons per day of ore averaging 9.3 percent combined lead and zinc, the quantity of tailings to be disposed of will increase by approximately 59 percent. This increase in tailings quantity results in the following:

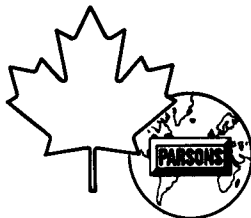
1. The flow rate from the zinc scavenger cells to the junction box will increase from 2,000 to about 3,200 gallons per minute at a density of 30 percent solids by weight.
2. The flow rate (of combined tailings and thickener overflows) from the junction box to the impoundment area will increase from 3,570 gallons per minute to about 4,960 gallons per minute at a density of 20 percent solids.

ELECTRICAL SYSTEMS

The following text briefly describes the electrical distribution system for the 2,000,000-ton per year concentrator as shown on the electrical drawings included in the Addendum to this report.

Utility Connection

The Utility Company will provide the required transmission line from Whitehorse to the plantsite. Between Whitehorse and Carmacks to this plantsite the line



will be a 138-kilovolt guyed V-tower design. It will initially operate at 69 kilovolt and have a conductor size of 636 MCM AC5R to provide for future expansion. The plant main substation will consist of a utility outdoor structure with hardware, power circuit breakers and transformers. Busduct from the secondary side (4,160 volt) of the transformers and associated 4,160-volt substation will be Anvil's responsibility and is included in this estimate.

4,160-Volt Substation

The proposed 4,160-volt substation will consist of indoor metal-clad type switchgear located within an insulated panel-type prefabricated building. For the purpose of this estimate, the 4,160-volt switchgear will contain:

Two 3,000-ampere main incoming breakers, 350-megavolt ampere interrupting capacity, draw-out type.

One 3,000-ampere bus-tie breaker, 350-megavolt ampere, draw-out type.

Two 1,200-ampere breaker, 350-megavolt ampere, draw-out type, to supply power to the 4,160-volt motor starters for the primary grinding and regrind mill motors. Motor starters are located in concentrator electrical switchgear room.

One 1,200-ampere breaker, 350-megavolt ampere, draw-out type, to supply power to the primary crusher switchgear room and on a parallel feeder to the service buildings.

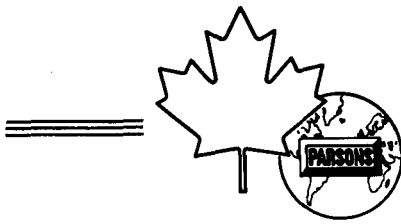
One 1,200-ampere breaker, 350-megavolt ampere draw-out type to supply power to the fine crushing plant switchgear room.

Two 1,200-ampere breakers, 350-megavolt ampere, draw-out type to supply power to the unit substations in the concentrator switchgear room.

One 1,200-ampere breaker, 350-megavolt ampere, draw-out type, to supply power to the raw water pump house switchgear room.

One 1,200-ampere breaker, 350-megavolt ampere, draw-out type to supply power to the mine site via a pole line approximately one mile long.

One lot miscellaneous meters and relays, station battery, battery charger, lighting and provisions for utility metering.



4,160-volt Feeders

The 4,160-volt distribution within the plant will consist of 5-kilovolt Teck cables (multiconductor power cable with aluminum armor and PVC jacket) in Transite conduit below ground and run in heavy duty aluminum cable trays above ground. The 4,160-volt feeders shall terminate at one or more unit substations or high voltage motor starters.

Unit Substations

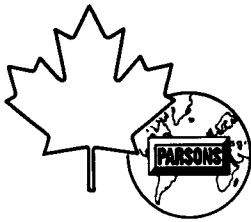
Unit substations will be of the indoor type, metalclad, dead-front. Each unit substation will consist of:

1. High-voltage section, containing a high-voltage fused disconnect switch, capable of interrupting the magnetizing current of the unloaded transformer.
2. 4,160 - 600/347-volt dry-type transformer, Delta-Star connected, grounded through a neutral resistor with a kilovolt ampere rating as shown on drawings. Transformers without fan cooling shall have provision for future fan cooling.
3. Low-voltage switchgear and metering section with an ample amount of breakers to supply the associated motor control centers. Metering shall consist of ammeter and ammeter switch, voltmeter and voltmeter switch and a kilowatt hours meter. Unit substations are located to take into account the center of power load and ease of access.

Motor Control Centers

Motor control centers will be fed from the unit substations and in turn will feed all 600-volt motors throughout the plant, all lighting and instrument transformers, 600-volt heaters, required power panels and starter racks. Motor control centers will be of standard Nema 1A construction, gasketed for dust protection. For this estimate, they have been assumed to be of the 5-or 6-high type (five or six starters, size 1 or 2, in a 90-inch high section).

Motor starters are of the combination type, that is a three-pole moulded case breaker with adjustable magnetic short circuit protection and a three-pole starter with thermal overload protection.



Breakers shall be used for feeder protection above 100 amperes and shall be three-pole, moulded case, with thermal-magnetic (nonadjustable) overload and short-circuit protection.

Three-pole fused disconnect switches shall be used for feeder protection for 100 amperes and below.

The 600-volt system will be designed for selective trip operation.

Power Factor Correction

It is assumed that the heavy motors (400 and 500hp) will be of the synchronous type, 0.8 power factor (leading).

Since the specific terms of the power contract to be negotiated between Anvil and the Utility Company are not known at this time, no provisions for capacitors for power factor correction are included. Should the subject contract contain restrictive power factor clauses, a properly sized bank of capacitors can be added to the system.

Emergency Power

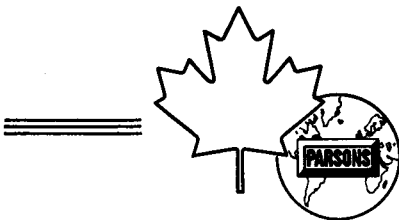
Emergency power will be supplied by two 400-kilowatt, 600-volt, 3-phase, 60-cycle diesel generators centrally located in the concentrator building. These generators will have automatic start-up on loss of the main power supply to plant. Emergency power will be supplied to all necessary plant loads: control room lighting and instrumentation, coarse ore building and fine crushing emergency lighting, one boiler for plant heating, mill office lighting, electrical switchgear room lighting, heating and ventilating, all high bay lighting, dryer and thickener drives, and miscellaneous necessary small loads.

Miscellaneous Systems and Equipment

1. Lighting

Lighting density, fixture types and lighting color shall agree with the latest recommended practices for industrial plants. The average foot candles of lighting intensity to be maintained in specific areas are:

Flotation area, 30 foot candles



Grinding area, 25 foot candles

Control rooms, 75 foot candles

Offices and labs, 75 foot candles

Switch rooms, 20 foot candles

All other generating areas, 20 foot candles

All nonoperating areas within building 10 foot candles

Outdoor lighting, where required, will be provided by floodlights mounted on sides of buildings.

2. Telephones

Conduit for telephone system and telephone installation fee is included in the estimate.

3. Grounding

A grid-radial ground system of embedded bare copper stranded cable and copper-clad steel ground rods will be provided. The layout and dimension of the system will be adequate to prevent excessive potential rise under fault conditions.

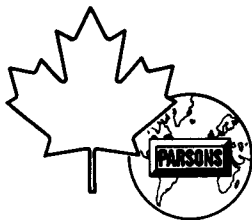
4. Motors

Motors will be generally rated 4,000 volts and 550 volts, 3 phase, 60 cycles, with revolutions per minute as required.

Motors 250 hp and above will be supplied at 4,000 volts; 1/2 hp up to 250 hp will be 550 volts; below 1/2 hp, 110 volts. Motors 550 volts and 110 volts shall be totally enclosed fan-cooled (TEFC) or totally enclosed non-ventilated (TENV). 4,000-volt motors shall be water and dust protected. Generally motors shall have Nema Class B insulation and Nema B design characteristics, unless special requirements dictate otherwise.

Tidewater Facilities

The electrical power supply for the tidewater facilities will consist of two



400-kilowatts, 600-volt, 3-phase, 60-cycle diesel generators. One unit will run continuously and the second will run on peak loads, during load-out operations, approximately 3 to 4 days per month.

HEATING, VENTILATING, DUST AND FUME CONTROL SYSTEMS

Dust Control Systems

Dust control systems will be provided to effectively collect all dust **generated** at discharge chutes, belt transfer points, conveyor **skirtboards**, storage **pile**, bin venting and other dust producing points except at the primary **crusher** dump.

Dry, bag-type dust collectors will be used throughout except in the fine ore storage bin area where wet-type dust collectors will be used.

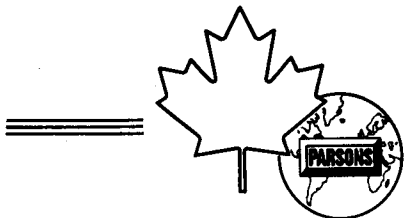
The dust collected in the dry bag collectors will be returned either to conveyor belts or to **storage** bins by means of screw conveyors and rotary valves or by means of rotary **valves** directly. The slurry from the wet-type dust collectors will be piped into the mill circuit.

All dust control systems will discharge all air to atmosphere.

DUST CONTROL SYSTEMS

MINESITE

<u>Building</u>	<u>System</u>	<u>CFM Capacity</u>
Primary Crushing Plant	-	7,400
Coarse Ore Storage	Tripper System	16,000
	Reclaim Feeders	10,000
Fine Crushing Plant	-	64,800
Transfer Tower	-	8,700



<u>Building</u>	<u>System</u>	<u>CFM Capacity</u>
Fine Ore Storage	Tripper Gallery	14,000
	Feeder System	16,000
Concentrate Storage and Load-Out	Lead Concentrate	3,700
	Zinc Concentrate	4,800
Lime Calcining Plant	Limestone System	3,500
	Burned Lime System	2,000
Coal	Boiler Plant	300
	Lime Plant	300

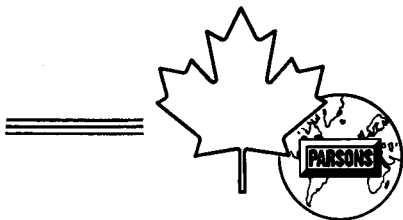
TIDEWATER

Truck Dump	-	2,000
Tripper System	Lead Concentrate	9,600
	Zinc Concentrate	9,600
Reclaim Feeders	Lead Concentrate	12,500
	Zinc Concentrate	12,500
Ship Loading		4,700

Heating and Ventilating

Make-up air, heated in winter, will be provided for all dust and fume control systems as required. All structures including conveyor galleries will be insulated and the following schedule of temperatures will be maintained at an outside temperature of -70F in the winter.

Offices, Laboratories, Toilet and Locker Rooms: 70F
Shops and Warehouse areas: 60F
Concentrator building and all other normally
occupied spaces such as fine crushing plant,
concentrate loadout: 50F



All normally unoccupied spaces such as conveyor galleries, coarse ore storage, transfer tower: 40F

The following ventilation rates of outside air will generally apply:

Winter - two air changes per hour (minimum).

Summer - ten air changes per hour.

All electrical equipment and lube rooms will be pressurized to minimize dust infiltration. Summer ventilation will be provided, in general and as required by powered roof exhausters and operable louvers. Removable insulated panels will be provided to close up all operable louvers in winter to prevent leakage of cold outside air into the buildings. Snow hoods will be provided for all outside air intakes that will be operated in the winter.

Dust and Fume Control Systems

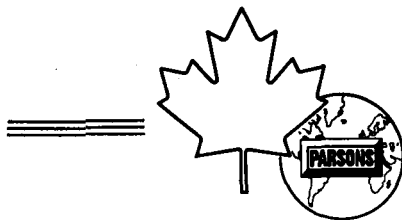
Dust and fume control systems will be provided for the reagents area, laboratory hoods and equipment in the analytical laboratory, metallurgical laboratory, sample preparation and other areas as required. Fume exhaust systems will be provided in the shops and garages for welding, vehicle exhaust and other operations as required.

Boiler Plant

A boiler plant will be provided at the plantsite to supply steam for the plantsite heating system. The boiler plant will consist of three 40,000-pound per hour, stoker-fired, coal-burning boilers; two boilers will carry load during peak heating season, one boiler will serve as standby, plus the necessary boiler feed water system. Coal storage and handling facilities will consist of three coal bins providing 5,000 tons live storage representing two weeks requirement during peak heating season for heating and concentrate drying. A conveyor system will draw off from the bulk storage bins and deliver coal to individual bunkers feeding the boiler stokers and concentrate dryer pulverizing mills. A 50-ton ash storage bin will be provided and ashes will be disposed of by truck.

Plant-site Heating System

The heating system at the plantsite will consist of blower-type heating and



ventilating units with steam coils for heating. Duct systems will be used for distribution of air where required.

Tidewater Facility Heating System

The heating system at the tidewater facility will consist of individual, indirect oil-fired, space-type heating units. Duct systems will be used for distribution of air and heat where required. Fuel oil will be supplied by a central fuel oil distribution system.

NONPROCESS BUILDINGS (MINESITE)

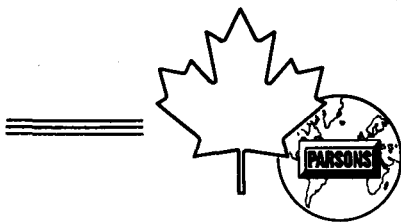
Nonprocess buildings required to support the mill operation are described below.

Combined Administration and Laboratory Building

A structural steel-framed, flat roof building is proposed to house administration, accounting and laboratory personnel and equipment. The building will be a split-level configuration, set into the hillside with the administration and accounting offices on the upper level, and laboratory functions on the lower level.

An area of approximately 7,600 square feet has been provided for the administrative function and an area of 3,500 square feet for the laboratory function. Included in the laboratory area are space accommodations for fire assay, X-ray fluorescent spectography, balance room and wet quantitative and qualitative analytical work. The building will be constructed on concrete foundation and will be enclosed with **galvanized sandwich panels** on metal frames, using 3-inch thick **fiber glass insulation**. Roofing will be built-up type. Partitions within the building will be of dry-wall construction. Windows will be aluminum, double-hung type with thermo-pane panels.

General arrangement of the proposed building is shown on Drawings Nos. 250-3901 A and B.



Combined Garage, Warehouse, Shop and Change Room Building

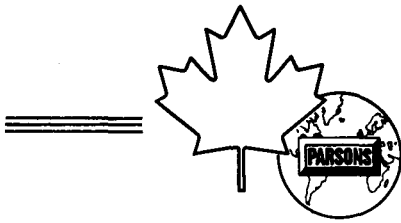
The building proposed will house facilities necessary to support both the mine and mill operations. Arrangement of the building will be in an L-shape, comprising a total area of approximately 58,400 square feet. Based on the requirement to provide sufficient building space for an early construction start, the arrangement proposed can be adapted to provide two separate structures joined by a common set of columns.

The initial building erected would consist of a pre-fabricated, rigid-frame structure 400 feet long by 80 feet wide. For 220 feet of the length, the building would have a 33-foot eave height required to accommodate an under-slung five-ton crane. The remaining length of 180 feet would have a 22-foot eave height. Permanent facilities to be contained within this structure and their respective space requirements are as follows:

<u>Facility</u>	<u>Area (Square Foot)</u>
Machine, Boiler, Welding, Tin, Blacksmith and Pipe Shops	14,400
Truck Storage	11,200
Truck Repair (Large)	3,200
Electrical Shop	800
Carpenter Shop	800
Steam and Paint Room	800
Small Truck Repair	<u>800</u>
Total Area	32,000

Adequate equipment will be provided in each of the facilities outlined to perform the functions required.

The remaining part of the L-shape structure, encompassing an area of 26,400 square feet, can be either structural steel-framed construction or a pre-fabricated rigid frame type. Overall dimensions of the structure proposed would be 220 feet long by 120 feet wide by 22 feet high.



Permanent facilities to be contained within this structure and their respective space allotments are as follows:

<u>Facility</u>	<u>Area (Square Foot)</u>
Warehouse	13,800
Change Room and Lunch Room	4,800
Tire Storage	1,600
Plate Storage	1,600
Sheltered Unloading Area	1,600
Tool Crib	600
Substation	1,200
Sanitary Facilities	600
Receiving Office	<u>600</u>
Total Area	26,400

In addition to the above facilities, office space will be provided in the warehouse area above the receiving office arrangement for supervisory personnel. Total space allowance will amount to approximately 3,000 square feet. Included within the facilities listed will be adequate equipment and hoists required for functional operation of the facilities.

Construction of the buildings described will be similar to that proposed for the Administration Building except that equipment and truck doors will be required. Doors will be sized to handle the equipment involved, with all openings heated by means of combined steam coil type door and space heaters throughout the building.

General arrangement of the buildings proposed are shown on Drawing Nos. 250-3902 A, B and C.



NONPROCESS BUILDINGS (TIDEWATER)

Nonprocess buildings required to support the concentrate loadout operation at Tidewater are described below:

Combined Office, Warehouse, Garage and Powerhouse Building

The building proposed will house all needs of the facility involving administration, warehousing and maintenance requirements. The building will be a rigid-frame steel structure, 180 feet by 60 feet by 22 feet to eave height. Facilities to be contained within the structure and their respective space requirements are as follows:

<u>Facility</u>	<u>Area (Square Foot)</u>
Warehouse	6,000
Garage	3,600
Office	600
Diesel Generator Room	<u>600</u>
Total Area	10,800

Adequate equipment and hoists will be included within the facilities mentioned to properly perform the functions required. Construction of the building will be similiar to that proposed for the minesite, except that heavier steel members will be required to support the excessive snow loads occurring at Haines, Alaska. General arrangement of the building proposed is shown on Drawing No. 600-3801.

PLANT WATER SYSTEM (MINESITE)

Raw water supply for the mill operation will be obtained from a well field located adjacent to Rose Creek, upstream of the proposed tailings diversion dam. Water obtained from the well field will be used for process, fire protection and potable or service water applications.



Based on a 5,830-tons per day ore processing operation, process water requirements are estimated to be about 3,500 gallons per minute. For a mill expansion to 8,740 tons per day, process water needs are estimated to be 4,700 gallons per minute. Fire protection requirements for the mill have been estimated at 1,500 gallons per minute for a period of 120 minutes (two-hour fire), based on two hydrants and a sprinkler system in operation. Potable water requirements have been estimated at 28,000 gallons per day based on a total personnel factor of 280 people.

The well field required to supply water for the above needs has been assumed to comprise six individual wells manifolded together and capable of pumping 600 gallons per minute each. The wells will be located in two rows of three wells, each spaced 500 feet apart, with an assumed maximum depth of 100 feet. Assumptions as to the number of wells, depth, capacity and spacing were made due to the lack of adequate information regarding well development prior to the completion of the feasibility study. Each of the pumps set in the wells will be eight-stage vertical-turbine, centrifugal type with 125 hp motors, capable of pumping 600 gallons per minute against a 600-TDH. In addition, each well will be provided with a 10-foot by 10-foot by 20-foot high, prefabricated building to protect the pump and motor from the extreme temperatures. Each pumphouse will be provided with a one-ton hoist for use in removing the repairing pumps and casings.

Power requirements for the well operation will be supplied from a substation located in the vicinity of the well field. Switchgear for the substation will be housed in a prefabricated steel building, 20 feet by 30 feet by 15 feet high. Standby power, in the form of a 150-kilowatt diesel-generator set, will also be provided to maintain heating and lighting needs of the well field during power outages at times of extreme temperatures. The diesel-generator set and component parts will be housed in the substation building. Oil storage for the diesel-generator set operation will be contained in a 1,000-gallon, buried tank, located adjacent to and outside the substation building.

From the well field, water will be transported via a 14-inch diameter, 1/4-inch wall steel pipe, buried nine feet deep, to a ground-level 500,000 gallon steel storage tank, located immediately south of and adjacent to the concentrator building. The storage tank (50 feet in diameter by 36 feet high) will be set on a ringwall foundation and will be compartmented by means of standpipes into separate storage volumes for fire supply and process, potable and service water supply. The lower level of the tank (approximately 13 feet in depth) will store 180,000 gallons, set aside permanently for fire use; while the upper level of the tank will be used to store a maximum of 320,000 gallons as required for process, potable and service use.

Operation of the well pumps feeding the storage tank will be controlled by level probes located in the tank. Pump operation will be automatically



sequenced to allow the longest running pump to drop out and start last, at commencement of next pumping operation.

Adjacent to the storage tank will be a 40-foot by 60-foot by 22-foot high, prefabricated steel building, housing, five 1,700-gallon per minute, horizontal centrifugal pumps. Two of the pumps, single-stage, double-section type with 200-hp motors, are provided for fire protection distribution, and three pumps, single stage, double sections with 150-hp motors, are provided for mill water distribution. All pumps are rated at a 250 TDH, and include a spare pump for fire protection use and a spare or future pump for mill water use.

From the storage tank, the mill water pumps (two operating) will supply the process, potable and service water distribution systems as required from a common header. In addition, heated water requirements for the flotation circuits (estimated to be 480 gallons per minute) will be accommodated via a separate branch off the common header. An instant steam-operated water heater will be installed in the branch line to raise the passing water temperature, a minimum of 5° F. Heated water requirements will be required only during winter operations. In warm weather operation, a bypass arrangement is provided to pass the flow around the heater.

Details of the proposed plant water supply and distribution systems are shown on Drawings Nos. 200-3201 and 200-3202.

The present water supply system is adequately sized to provide the requirements for the initial plant design rate of 5,830 tons per day. However, changes may be necessary to accommodate the water requirements for the ultimate design rate of 8,740 tons per day. Any, and all, such changes that may be required should be thoroughly studied and reviewed when complete hydrological data is available for potential water sources and when well development data is obtained.

WATER SYSTEM (TIDEWATER)

The plant water system proposed for this facility includes a water supply to be obtained from two wells located east of the concentrate loadout facility. The wells and pumps have been estimated at a capability of 100 gallons per minute against a 100 TDH, with an average depth of 50 feet. Pumps will be 4-stage, vertical-turbine, centrifugal type with 5-hp motors, manifolded together and discharging to an elevated 20,000-gallon steel storage tank located adjacent to and east of the combined office, warehouse and garage building. The elevated tank will be compartmented, by means of standpipes, into individual storage for fire protection supply and potable or service water supply.



Elevation of the tank was set at 75 feet above ground to provide a minimum pressure by gravity on the fire loop of 30 psi for hydrant operation. Pressures above the range afforded as required were assumed to be derived from pumper application from local fire equipment housed in Haines, Alaska. Primary source of water for fire protection, amounting to 18,000 gallons contained in a reserved supply within the tank, would be capable of fighting a fire of approximately 36 minutes duration based on a usage rate of 500 gallons per minute, two hose streams. Makeup to the fire system, during time of use, would be provided from the wells at a rate of 200 gallons per minute amounting to an additional time period of approximately 15 minutes.

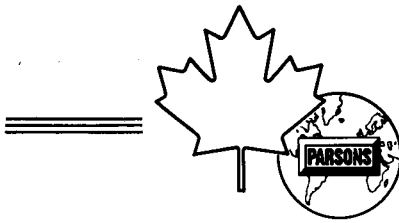
Potable or service water requirements for the facility have been estimated at a rate of 2,000 gallons per day. Water requirements for this need will be contained in the upper level of the elevated storage tank. Operation of the well dump to supply the potable water needs will be controlled from level probes located within the elevated tank. When required, standing water in the tank will be protected against extreme temperatures by means of an oil-fired hot-water furnace and circulator, located at the base of the elevated tank.

Distribution of water to the fire and potable systems will be by gravity via individual systems as shown on Drawing No. 600-3201. The fire protection system proposed includes a closed-loop arrangement around the main nonprocess building with branch lines extended for hydrant application to outlying areas such as the fuel storage facility. In addition, a sprinkler system is proposed for the warehouse portion of the nonprocess building.

PLANT FIRE PROTECTION SYSTEM (MINESITE)

The plant fire protection system has been developed in accordance with recommended practices established by National Board of Fire Underwriters and Canadian Underwriters, where applicable. The primary supply of water to the protection system will be provided by an undisturbed capacity of 180,000 gallons contained in the Raw Water Storage Tank, located adjacent to and south of the Concentrator Building.

Secondary source of water to the fire system can be considered to be the additional supply contained in the storage tank above the reserved fire supply level. The secondary source can be considered a positive supply at all times except during a power outage. During normal operations, the water level in the tank will be assured by level probes controlling the operation of the



well pumps. Should a fire occur during a power outage, positive water supply to the fire system will consist of the reserve 180,000-gallon supply (120 minutes protection) plus an additional minimum 200,000 gallons above the fire level in the tank (approximately 130 minutes additional protection time).

A more positive secondary source of water for the fire system (not included in this report) would be to carry two or more well pumps on the standby power system. Due to cost involved and the adequacy of system proposed, this alternative has not been proposed for this phase of the project.

Fire pumps proposed for the protection system will be rated at 1,700 gallons per minute against a 250 TDH, and will be Underwriter's Approved, horizontal-centrifugal type. Two pumps are proposed, one operating and one standby. Rated capacity of pumps was established on basis of two hydrants and a sprinkler system in operation simultaneously. Hydrants proposed for the project are the "monitor" or "nozzle" type with an approximate capacity of 600 gallons per minute at a pressure of approximately 75 psig. The sprinkler system has been proposed for the warehouse area of the nonprocess building.

The fire pumps will be located in the pumphouse building housing the mill water pumps, adjacent to the storage tank. The distribution system proposed for the plant consists of the closed-loop type run underground at a depth of nine feet, with hydrant branches as required to protect outlying structures such as the Administration Building. Development of the system includes the interlocking of the mill water pumps with the fire loop for the purpose of pressurizing the system during nonfire use. Interlocking consists of flow sensing devices set in the fire loop which will start fire pump operation, at which time mill water pumps will drop out of system.

Details of the proposed fire protection supply and distribution system are shown on Drawing No. 200-3202.

FUEL OIL STORAGE AND DISTRIBUTION (MINESITE)

A fuel oil storage and distribution system has been provided at the minesite to service both the ore haulage and concentrate haulage operations in addition to the diesel-oil fired standby power supply system.

The storage system proposed will include two 30,000-gallon capacity, steel, ground-level tanks, 20 feet in diameter by 14 feet high, to be located immediately north of the proposed garage, warehouse and machine shop building complex.



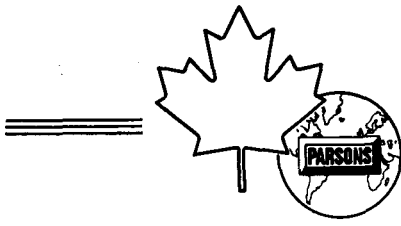
The tanks will be set on ringwall foundations and will be enclosed in earth-dikes as recommended by the National Board of Fire Underwriters for tanks of the capacity proposed. Capacity of the storage tanks was established on the basis of fueling 15 ore trucks and 40 concentrate trucks daily, at a total estimated consumption rate of 3,700 gallons per 24-hour day. The roundtrip distance between the minesite and tidewater is estimated to be 650 miles. Fuel consumption of the concentrate trucks is estimated to be 5 miles per gallon, or a total consumption of 130 gallons per roundtrip. To minimize storage tank requirements, fueling of the trucks will be performed at each end of the haul. Ore truck fuel consumption is estimated to average concentrate truck consumption on the basis of their daily use for either ore haulage or stripping operations.

Storage tanks are to be replenished on the basis of a once per week delivery, via concentrate truck backhaul from Tidewater. Trucks proposed for concentrate haulage will be provided with two 2,000-gallon capacity oil tanks for a total backhaul potential of 4,000 gallons per truck.

Adjacent to the storage tanks, a prefabricated steel building, 20 feet by 30 feet by 15 feet high, will be located to house three 300-gallon per minute oil transfer pumps. The pumps will be interconnected by a valving arrangement to either off-load oil delivery trucks to storage tanks or distribute oil from storage to work tanks located adjacent to the garage end of the building complex. Pumps have been rated at 300 gallons per minutes to allow off-loading concentrate trucks in as short a time as practical.

Work tanks located adjacent to the garage will be buried. Tanks will have a capacity of 5,000 gallons each and will be provided with a 50-gallon per minute oil dispensing unit located at the surface to fuel the haulage trucks. In addition, a 5,000-gallon capacity, buried gasoline storage tank will be located adjacent to the oil work tanks, complete with a 20-gallon per minute dispensing unit to service any gasoline-consuming vehicles.

The standby power supply system will contain two 400-kilowatt diesel-fired generator sets. At full load, each unit will consume approximately 35 gallons per hour, or a total of 560 gallons per eight-hour shift. A 550-gallon capacity day tank has been provided for a power outage of eight hours duration. To sustain standby power generation beyond outages lasting longer than eight hours, oil supply to the engines will be provided from the main storage tanks via a 300-gallon per minute pump carried on the standby power load. All oil and gasoline transfer operations will be properly metered to ensure adequate records of oil consumption, both as to quantity and use. Details of the proposed oil storage and distribution system are shown on Drawing No. 200-3204.



FUEL OIL STORAGE SYSTEM (TIDEWATER)

Fuel oil storage and distribution facilities located at tidewater will be similar to those located at the minesite. In addition to concentrate truck fueling storage requirements, storage will be required for servicing two 400-kilowatt diesel-generator sets providing full power for the facility, plus storage for an oil-fired heating system. Two 40,000-gallon capacity steel ground-level tanks, 22 feet in diameter by 18 feet high, will be provided to store the oil requirements for the facility. Location of the tanks, in addition to an oil transfer pumphouse, will be in an area northwest of the combined office-warehouse building. The tanks will be set on ringwall foundations and will be enclosed in earth-dikes as recommended by the National Board of Fire Underwriters for tanks of the capacity proposed.

Pumping facilities proposed will be similar in arrangement to those proposed for the minesite except that only two 300-gallon **per minute** units are recommended to service the fueling operation. A third pump, of 100-gallon per minute capacity, will be interlocked with the 300-gallon per minute units, but will be primarily used to supply oil to the heating system and diesel-generator unit day tanks. In case of mechanical failure to either of the pumping units, the remaining pumps will act as spares. The building proposed to house the pumping units and metering equipment will be a 20-foot by 30-foot by 15-foot high, prefabricated, rigid-frame steel structure, fully insulated with 3-inch thick fiber glass.

Work tanks and dispensing units will be the same as those proposed for the minesite, both for oil distribution and gasoline storage. All oil and gasoline transfer operations will be properly metered to ensure adequate records of fuel consumption both as to quantity and use.

Details and general arrangement of the proposed system are shown on Drawings Nos. 600-3201 and 600-3801.

PLANT SEWAGE SYSTEM (MINESITE)

The plant sewage system proposed for the mill operation will consist of a collection sewer system and a package-type treatment plant. The sewer collection system will include pipe sizes ranging from 6 inches to 10 inches, run underground with a minimum depth of nine feet. Sewer lines will be extended to collect sewage from all structures housing sanitary facilities, namely primary



crusher, administration building, concentrator, shop-warehouse building and concentrate loadout building.

Collected sewage will be drained to a sewage treatment plant located southwest of the concentrator building. The type of treatment plant proposed uses the activated sludge process providing primary and secondary treatment of the sewage. The package-type plant proposed will include a sewage lift station, comminutor and aeration equipment. The completed treatment plant will be housed in a prefabricated steel structure capable of retaining heat against extreme temperatures, as well as being capable of venting obnoxious gases to atmosphere.

Size of the package plant proposed is rated at 30,000 gallons per day, on the basis of a total personnel requirement of 280 people. Additional sewage treatment needs such as post-chlorination will have to be determined at a later date when information is made available as to the characteristics of the receiving stream. Such information may be required to obtain approval of the system proposed from any governmental agencies having jurisdiction over the territory involved.

SEWAGE TREATMENT (TIDEWATER)

The plant sewage system proposed for the facility will include a collection sewer system and a 1,500-gallon per day septic tank. Sewer lines will vary in size from 6 inches to 8 inches and will be run underground to collect sewage generated at sanitary facilities located in the office building complex, concentrate loadout building and fuel oil pumphouse.

Collected sewerage will be drained to the septic tank located underground in an area southeast of the office building complex. Treated effluent will be discharged to a drainage ditch outletting into Lutak Inlet. Approval of the proposed system may have to be obtained from the authorities having jurisdiction in the area.

Arrangement of the system is shown on Drawing No. 600-3801.



SECTION 8

PRELIMINARY PROJECT SCHEDULE

As a part of the Final Feasibility Report, a preliminary project schedule has been prepared. Upon a decision to proceed with the final engineering-design, a Project Planning and Scheduling Engineer will be assigned to develop the detailed CPM project schedule. This detailed project schedule will permit complete monitoring of the progress of the job. It is essential that a complete monitoring effort be sustained during the progress of the project to ensure:

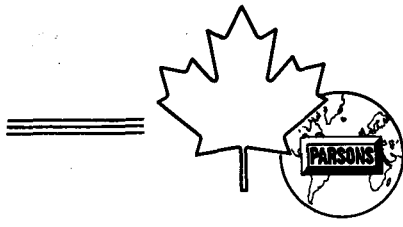
1. The ability to forecast any delaying factors and the extent of delay far enough in advance to permit corrective measures to be undertaken.
2. The proper coordination of the entire engineering-design effort and to produce drawings as required to meet the construction schedule.
3. The procurement and delivery of all materials and equipment to the job site at the time they are required.
4. The proper manpower requirements are maintained over the entire scheduled construction period.
5. The ability to forecast the completion date with reasonable accuracy months before the actual completion.
6. The establishment of proper starting dates for all subcontractors moving onto the job.

The split construction schedule which has been developed for the Anvil Project envisions two construction periods:

May 1967 - November 1967
May 1968 - July 1969

These constitute a construction effort of about 21 to 22 months in an elapsed time of about 26 months. There are several inherent and significant advantages of the proposed split construction schedule as opposed to a continuous field effort. These are:

1. The deliveries of major units of process equipment and plant machinery are not as critical.



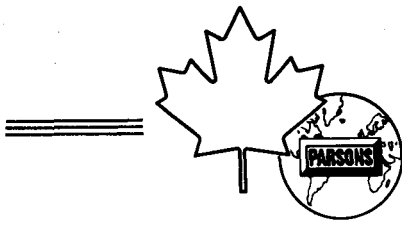
2. The commencement of the major construction effort in May 1968 will permit a scheduled buildup of equipment, machinery and materials in the North Vancouver, Whitehorse/Haines and plant site staging areas. This will result in a **program of priority deliveries to maintain construction target dates.**
3. The erection of the concentrator building steel and installation of major units of process equipment can proceed concurrently from the ground up, thereby assuring more efficient construction services.
4. The proposed schedule permits completion of the final design of the concentrator building steel with sufficient lead time for procurement, detailing and fabrication to obtain firm unit prices and still meet the May 1968 target date for starting erection.
5. The split schedule will provide more overall engineering-design lead time for the project but will impose problems during the initial stages. Sufficient engineering-design effort will have to be prosecuted in the early stages on site preparation (grading and excavation) and concrete to provide a reasonable quantity of field work to make the initial construction period (May 1967 - November 1967) worthwhile.
6. The increase in overall engineering-design lead time combined with the programmed buildup of equipment, machinery and materials in the staging areas will ensure a more efficient use of field labor during the major construction effort commencing May 1968.

In Figure 8-1, the principal activities (engineering, procurement and construction) are shown for the following major elements of the project:

Primary Crusher
Coarse Ore Storage
Fine Crusher and Lime Plant
Concentrator
Concentrate Loadout
Tidewater Facilities
Miscellaneous Facilities

The significant highlights of this preliminary project schedule can be enumerated as follows:

1. Complete necessary temporary facilities, bunk houses, roads, et cetera during May and June, 1967.
2. Start site preparation (grading and excavation) by May 15 - June 1, 1967 and continue until November 1967.



3. Commence pouring concrete for concentrator, garage-shop complex, primary crusher and coarse ore storage July 1, 1967 and continue work through November, 1967.
4. During winter period (November 1967 - May 1968):
 - a. Keep a small maintenance crew only at plant site.
 - b. Maintain a small warehouse crew at Whitehorse to receive and store equipment, machinery and materials for later movement to the site.
5. Recommence construction effort at the site in May 1968 and continue to completion in July 1969.
6. All major steel, mechanical, piping and electrical installation not to begin until May 1968.
7. The field effort at tidewater is not scheduled to begin until June 1968.

To assure completion of the project by July, 1969 and to make certain that the major construction effort commencing May, 1968 will realize the full potential benefit of the inherent advantages of the split construction schedule, it is vital that actions be taken, and decisions made, to utilize the potential engineering-design lead time provided and to undertake during the initial construction period (May 1967 - November 1967) the greatest amount of work practical. Within the framework of these thoughts, the itemized factors below constitute so-called critical elements in this scheduling.

1. Engineering

Final engineering-design should commence at the earliest possible date. To meet the construction requirements of the initial period, engineering-design **for all other aspects of the project other than the concentrator building**, should be started by March 15 - April 1, 1967.

2. Procurement

Long delivery items (10 to 14 month duration) such as gyratory and cone crushers, grinding mills, bridge cranes, major electrical equipment and the like should be placed on order as soon as possible. This early procurement timing will:



- a. Ensure the backlog of equipment at the staging areas necessary to implement the major construction effort beginning May 1968.
- b. Permit the transportation of major units to the plant site during the period when the ice bridge is available.

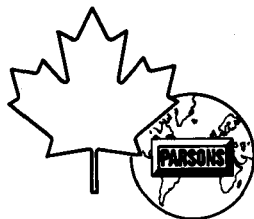
3. Construction

Initiation of the May 1967 construction effort is predicated on the following assumptions and details of work to be undertaken.

- a. Anvil will provide camp facilities (housing and feeding) for up to 60 men beginning April 15, 1967.
- b. Work will be subcontracted through the first construction period up to about November, 1967.
- c. Subcontractors will provide their own camp facilities for the 1967 effort.
- d. Subcontract bid packages (for proposal purposes) should be tendered by about April 1, 1967 for award by May 1, 1967 and move-in by May 15, 1967.

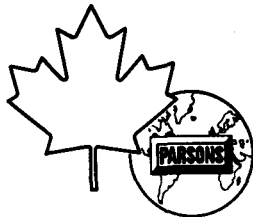
The major work contemplated for this primary period will encompass the items enumerated below.

- a. Locate and open pits for sand and gravel; set up washing, screening and crushing plant; and set up concrete batch plant.
- b. Establish sand and gravel pits and stock piles based on a concrete requirement of 25,000 cubic yards.
- c. Batch and deliver about 10,000 to 15,000 cubic yards of concrete for the concentrator building, garage-shop complex, primary crusher, coarse ore storage, temporary construction requirements and as much other concrete as engineering progress will allow to be poured during this primary period.
- d. Grading at the plant site (excavation and fill) to **accommodate the** listed approximate quantities in cubic yards.



<u>Location</u>	<u>Earth</u>	<u>Excavation (Type)</u>		<u>Fill</u>
		<u>Rippable Rock</u>	<u>Solid Rock</u>	
Coarse Ore Storage	10,000	5,000	30,000	7,600
Site Preparation	25,000	20,500	6,500	250,000
Roads				20,000
Utilities	6,000	6,100	1,300	13,600
Non Process Buildings	7,000	7,640	90	16,800
Fine Crushing	2,000	1,780	710	1,500
Concentrator	12,000	14,380	600	21,800
Lime Plant	1,000	1,200	1,500	1,200
Load-Out Area	600	1,000		3,700
Subtotal	63,600	57,600	40,700	336,200
Temporary Facilities	25,000	25,000		60,000
TOTAL	88,600	82,600	40,700	396,200

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SECTION 9

CAPITAL COST ESTIMATE

ESTIMATE CRITERIA

The capital cost estimate which was made for the final feasibility study is based on the following estimating criteria.

Area Cost Codes

Code 100 - Site Improvement

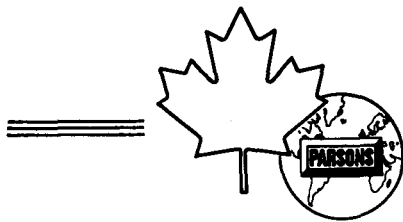
Sub-Code	105	Clearing, Grading and Drainage
	115	Roads, Walks and Parking Areas

Code 200 - Utilities and Services

Sub-Code	205	Water Supply, Storage and Distribution
	210	Plant Drainage and Sewerage System
	215	Switchgear and High Voltage Distribution System to Load Center Disconnects
	220	Yard Lighting
	225	Emergency Power Plant
	230	Plant Communication System
	235	Fire Protection System
	240	Fuel Storage and Plant Heating System
	245	Compressed and Instrument Air System

Code 250 - Nonprocess Buildings

Sub-Code	255	Administration Building
	260	Stores, Warehouse, Shop, Garage and Changeroom
	265	Nonprocess Equipment



Code 300 - Primary Crushing and Coarse Ore Storage

Sub-Code 305 Ore Dump and Primary Crusher
310 Coarse Ore Storage

Code 350 - Secondary and Tertiary Crushing

Sub-Code 355 Secondary Crusher
360 Tertiary Crusher
365 Transfer Tower

Code 400 - Concentrator

Sub-Code 405 Fine Ore Storage
410 Primary Grinding
415 Flotation
420 Regrind
425 Thickening
430 Filtration and Drying
440 Reagent Preparation and Distribution
445 Mill Offices, Control Room, Storage, etc.

Code 435 - Lime Plant

Code 500 - Concentrate Storage and Load-Out at Plant Site

Code 550 - Tailings Disposal

Code 600 - Concentrate Unloading, Storage and Load-Out at
Tidewater

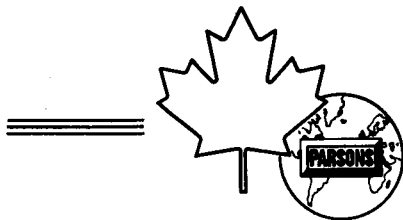
Code 700 - Mine Plant and Auxiliaries

Sub-Code 725 Power Feeder to Mine Site (RMPCo, New York)
730 Diesel and Gasoline Fuel Storage (RMPCo,
New York)

In addition to the above areas the following area codes are included in our capital costs but were provided as indicated:

Code 100 - Site Improvement

Sub-Code 110 Permanent Access Road and Bridge (by Anvil
Mining Corporation, Limited)



Code 700 - Mine Plant and Auxiliaries

Sub-Code	705	Preproduction Stripping (RMPCo, Los Angeles)
	710	Mine Plant Equipment (RMPCo, Los Angeles)
	715	Waste Dump Haul Roads (RMPCo, Los Angeles)
	720	Haul Road to Primary Crusher (RMPCo, Los Angeles)

Code 800 - Transportation System (by Miles & Sons Trucking Company)

Labor Rates and Productivity

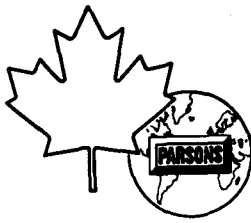
Our Construction Department, after review and discussion with local labor authorities and subcontractors, established an average labor rate of \$4.25 (Canadian) per hour without fringe benefits for the plantsite. For the tidewater area the following labor rates (in United States dollars) without fringe benefits are used:

Boilermakers	\$6.75
Carpenters	6.20
Millwrights	6.55
Electricians	6.85
Ironworkers	6.67
Operating Engineers	6.71
Laborers	5.75
Pipefitters	7.10
Cement Finishers	6.51
Teamsters	5.67

All fringe benefits, lodging and meals are shown under Other Field Costs, Account 8600. A productivity of 65 to 70 percent, based on our statistical data for labor manhours applicable to the Los Angeles area, is used for this project.

Major Equipment

Approximately 90 percent of the cost in this category was quoted by Canadian vendors. The remaining 10 percent was estimated based on previous purchases



and statistical data. Equipment which is indicated as purchased in the United States includes a 22 1/2 percent import duty, where applicable. The cost of the Canadian purchases include freight and export packing, f.a.s. North Vancouver as required. Freight rates from North Vancouver to the jobsite are included in Other Field Costs, Account 9550. All applicable Dominion Sales Taxes are shown in Other Field Costs, Account 9500. The sales tax is figured at an average of 3 percent of material cost including packing, freight and duty (where applicable) based on a 6 percent rate beginning April 1967 and a complete cessation of the sales tax in April 1968.

Concrete

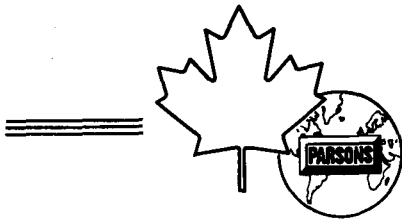
Under this category the cost of excavation, ready-mix concrete, reinforcing, formwork and backfill are included. The cost of ready-mix concrete was calculated based on the following: Cement will be purchased in Canada at a cost of \$1.22 per bag delivered to North Vancouver. The cost for shipping from there to jobsite is included in Other Field Cost, Account 9550. The costs of purchasing, erecting and maintaining a concrete batch plant, gravel crushing plant and a sand washing plant are included in the material cost of the ready-mix concrete. Rebar was quoted by Canadian vendors, f.a.s. North Vancouver, at \$0.07 per pound. Formwork material cost is estimated at \$0.70 per square foot of contact area. The costs for excavation and backfill as shown in our estimate include only direct labor and materials. Cost of equipment to be used for this operation is in Other Field Costs, Account 8300. Quantities for concrete work were obtained from available drawings and supplemented with material take-offs based on similar installations.

Pipe, Valves

Quantities for this account were developed from Piping and Instrumentation Diagrams and general piping arrangement drawings. Canadian quotes on a unit cost basis were obtained from Canadian vendors for pipe and specialities. Other items for which quotes were not obtained were based on statistical data from recently completed jobs in Canada. The cost of these items is f.a.s. North Vancouver.

Structural Steel

Quantities for structural steel were calculated on the basis of similar installations of this type and material takeoffs. This account was broken down



into three main categories and quotes on a unit cost basis obtained from Canadian vendors, f.a.s. North Vancouver. The material cost for heavy steel used in this estimate is \$375 per ton; medium steel depending on type varies from \$440 to \$540 per ton; light steel, stairways, grading, handrails, range from \$750 to \$900 per ton.

Instrumentation

A material list of all instruments divided by areas was furnished by the instrument engineering department and cost estimated from statistical data based on Canadian projects.

Painting

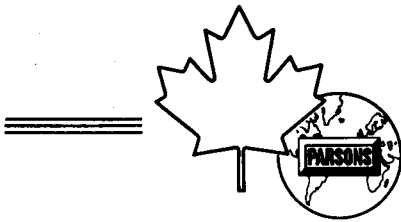
The cost for painting includes one field coat of paint on all structural steel and piping.

Electrical

Single-line electrical diagrams for the project were prepared and material take-offs made from them in connection with the general plot plan and plant layouts. All electrical equipment, cable and wire, was estimated as purchased in Canada and cost includes all these items f.a.s. North Vancouver. Quotations for major items of this account were obtained.

Plant Buildings

In this account the complete cost of buildings including excavation, foundation, structural steel, electrical, utilities and all architectural features is included. Concrete and steel were estimated on the same basis as described previously under these respective headings. Quotes for architectural materials such as siding, roofing materials, doors, windows and partitions were obtained from Canadian vendors and items for which no quotes were obtained were estimated from previous experience on similar jobs in Canada.



Spare Parts

No costs for spare parts are provided for in this estimate nor are there costs for the initial charges for the rod and ball mills.

Site Preparation

Quantities were obtained from topographic plans of the site. Costs were divided into the following categories: earth excavation, decomposed rock excavation, rock excavation (blasting), and compacted backfill. The costs for these items include direct labor only and blasting materials. The equipment necessary to perform this work is in Other Field Costs, Equipment Account 8300.

Other Field Costs

The following accounts are included under this heading:

7100 - Temporary Buildings and Utilities

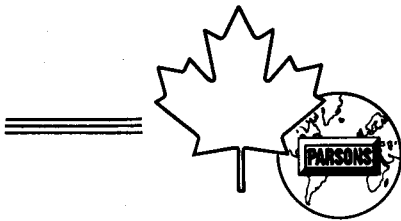
Includes construction warehouse, shops, gate house, change house, field office, and sanitation system during construction. Only a minimum of these buildings will be erected for the starting period of construction and as soon as permanent facilities for the project are erected, these facilities will be used during construction phase of the project. Other items included in this account are winter protection, temporary utilities such as water, gas, air, electrical power, temporary roads and scaffolding.

7300 - Warehousing, Guards, Clean-Up

Included in this account is surveying, warehouse employees, security guards, watchman, truck drivers, janitors, welder qualification costs and safety personnel.

7400 - Start-Up Services

No allowance for this account is included in the estimate.



7500 - Union Welfare and Benefits

In this account the holiday and vacation pay is calculated at \$0.07 per manhour average and \$0.27 per manhour for labor welfare funds. These figures were obtained through our Construction Department. At the tidewater area the fringe benefit for holidays, vacation and labor welfare funds are figures at an average of \$0.45 per manhour.

8100 - Supervision and Field Office Staff

In this account the cost of all field supervision is included. Subsistence and lodging costs as well as transfer and transportation costs for U.S. hired personnel are included. The cost of field office supplies such as furniture, office machines and supplies is included in this account. Salaries for field supervision are based on a 60-hour work week.

8300 - Construction Equipment and Services

The construction equipment in this account is estimated on the basis of renting as much as possible in Canada and supplementing with purchases in Canada. The cost for the maintenance of this equipment as well as an adequate supply of spare parts for the new equipment is provided in this account. Operating supplies such as oil, gas, grease, are included. For the new construction equipment, a salvage value of \$500,000 is deducted from this account.

8400 - Small Tools

Small tool costs are figured as 4 percent of direct labor cost.

8500 - Consumable Supplies

This account is estimated at 6 percent of the total direct labor.



8600 - Subsistence and Lodging

1. Concentrator Site - A camp for 600 men is provided. This camp consists of dormitories, kitchen facilities, and recreational facilities. Cost for these facilities were obtained from Campco. The cost for lodging and meals is estimated at \$7.10 per calendar man day. This cost is to be fully charged to the contractor and no refund will be obtained from craft labor.
2. Tidewater - A separate camp is provided in this area for 80 men. The cost for this camp including kitchen facilities, dorms, and recreational facilities is estimated at \$1,500 per man and subsistence and lodging at \$10 per calendar man day.

8700 - Transportation for Construction Workers

A turnover of 300 percent is estimated for this project. We allowed one round-trip at \$50 in this account.

8800 - Premium Time

All craft labor will be working 60 hours a week for 80 hours pay. Therefore, 33 1/2 percent of total direct labor is estimated as premium time.

9300 - Payroll Taxes and Insurances

This account covers workmen's compensation, bodily injury insurance, liabilities, and all risk insurance.

9500 - Taxes

Dominion sales tax is estimated at 6 percent of all material cost delivered at jobsite for purchases up to April 1, 1968, and no sales tax thereafter.



9550 - Freight

All material cost includes freight to North Vancouver. Freight from North Vancouver to the plantsite is included in this account and estimated at \$48 per ton as a composite rate from North Vancouver to the plant site.

9700 - Duties

Duties for anticipated purchases of equipment and materials outside of Canada are estimated at 22 1/2 percent of material cost.

Home Office Cost

This account includes all costs in contractor's home office for the design and engineering services for this project including contractors overhead and out-of-pocket expenses. The cost for preparation of the various feasibility studies up to this estimate are not included in this cost.

Project Schedule

This Capital Cost Estimate is based on a construction schedule starting May 15, 1967, and ending August 1, 1969.

Escalation

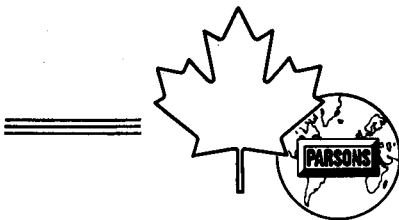
No escalation is applied to this estimate.

Contingencies

No contingencies are applied to this estimate.

Accuracy

Based on the foregoing, it can be assumed that the accuracy of this estimate



is minus 5 percent plus 10 percent, except for the tidewater facilities (Area Code 600) which is plus or minus 20 percent. The Capital Cost Estimate for the tidewater area is in United States dollars; for all other areas, Canadian dollars. All Canadian funds have been converted to United States currency at the exchange rate of one (1) Canadian dollar equaling \$0.925 United States dollars.

SPECIFIC EXCLUSIONS FROM ESTIMATE

1. Power generation and transmission line to plantsite.
2. Primary substation at the plantsite.
3. Cost of property at plantsite and tidewater.
4. Plantsite and tidewater telephone and/or radio communications systems.
5. Plantsite and tidewater fencing and guardhouses.
6. No costs have been provided for the initial rod and ball charges for the primary grind, regrind and line ball mill circuits.
7. No separate allowance has been provided as a contingency.
8. No separate allowance has been provided as an escalation factor beyond calendar year 1967.
9. No value has been included as an engineer-constructor's fee for profit.

ESTIMATE OF CAPITAL COST

Appended to this report in Section 10 (Addendum) are the estimate summary and subsummary sheets for all the direct and indirect costs relevant to the plantsite and tidewater facilities. Table 9-1 summarizes these costs in concise form.



TABLE 9-1

CAPITAL COST ESTIMATE*

(In Millions of United States Dollars)

	<u>Materials and Equipment</u>	<u>Subcontracts</u>	<u>Labor</u>	<u>Total</u>
<u>Plantsite Facility</u>				
Total Direct Costs	11.749	0.051	5.742	17.542
Other Field Costs				<u>11.426</u>
Subtotal				28.968
<u>Tidewater Facility</u>				
Total Direct Costs	2.960	0.020	2.121	5.101
Other Field Costs				<u>1.860</u>
Subtotal				6.961
<u>Home Office Services Cost</u>				
Subtotal				<u>3.000</u>
TOTAL				38.929

* All costs are exclusive of engineering-constructor contractor's fee.

Table 9-2 shows the distribution of all fixed-assets values which have been used for the determination of the discounted cash flow rates of return with the exception of the allotted townsite costs, Anvil's expenditure for overhead and supervision and the net working capital.

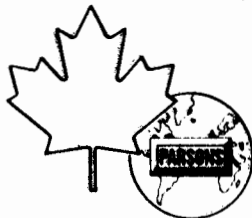


TABLE 9-2

FIXED - ASSETS VALUES*

(In Millions of United States Dollars)

	Time (Years)		<u>Total</u>
	<u>-3, -2, and/or -1</u>	<u>+5</u>	
<u>Plantsite Facility</u>	31.387	3.105	34.492
<u>Tidewater Facility</u>	7.542		7.542
<u>Mine Plant</u>			
Preproduction Stripping	2.223		
Mine Plant and Equipment	4.303		
Faro Creek Diversion	<u>-</u>	<u>0.035</u>	
Subtotal	6.526	0.035	6.561
<u>Transportation System</u>			
Truck Fleet	-		
Maintenance and Repair Shop	0.326		
Mobile Maintenance Equipment	0.022		
Shop Equipment	<u>0.079</u>		
Subtotal	0.427		0.427
<u>Permanent Bridge and Access Road</u>	<u>0.463**</u>		<u>0.463</u>
TOTAL	46.345	3.140	49.485

* All costs are exclusive of engineering-constructor contractor's fee.

** Represents only Anvil's cost (approximately 33 1/3 percent total cost).



Townsite Capital Cost Estimate

During the week of December 19, 1966, the Anvil management requested that a rough, order-of-magnitude capital cost estimate be prepared for certain elements of the proposed townsite for the Anvil Project. Specifically requested to be included within this estimate were the following elements:

1. Family housing (310 three-bedroom units).
2. Single status housing (five 20-man bunkhouses and ancillaries)
3. Elementary school.
4. Dispensary.
5. Recreation facility.
6. Services and utilities.
 - a. Central, stoker-fed, coal-fired steam heating plant with coal storage, ash removal, steam distribution and condensate return systems.
 - b. Roads, sidewalks and curbing.
 - c. Water distribution system (less cost of providing water).
 - d. Sewerage collection system (less treatment plant).
 - e. Power distribution (less primary substation and emergency power plant).

Table 9-3 enumerates these costs in millions of United States dollars exclusive of contractor's fee.

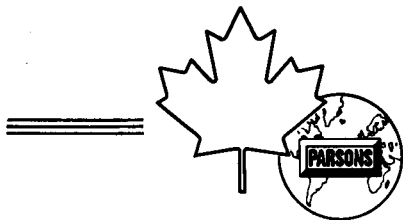


TABLE 9-3

TOWNSITE CAPITAL COSTS*

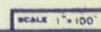
(In Millions of United States Dollars)

<u>Element</u>	
Family Housing	9.136
Bunkhouses	1.068**
Elementary School	0.341
Dispensary	0.137
Recreation Facility	1.100
Services and Utilities	<u>2.836</u>
TOTAL	14.618

* Exclusive of engineering-constructor **contractor's fee**.

** Does not reflect reduction accrued from use of temporary camp facilities at plantsite for permanent bunkhouses.

This topographic map illustrates a hill with a central peak. The map features two types of contour lines: solid lines for surface contours and dashed lines for bedrock contours. A dashed line outlines a pit, and a solid line indicates a bench. Three paths are marked with arrows and labels: 'TO WASTE' (two locations on the left) and 'TO STOCKPILE' (at the bottom right). The map includes a legend in the upper right corner, a scale bar (0 to 200 feet) in the lower right, and a north arrow pointing upwards. Elevation values are marked on the contours, ranging from 4100 to 4525 feet. The map is overlaid with a grid of 10,000-foot and 2,000-foot intervals.



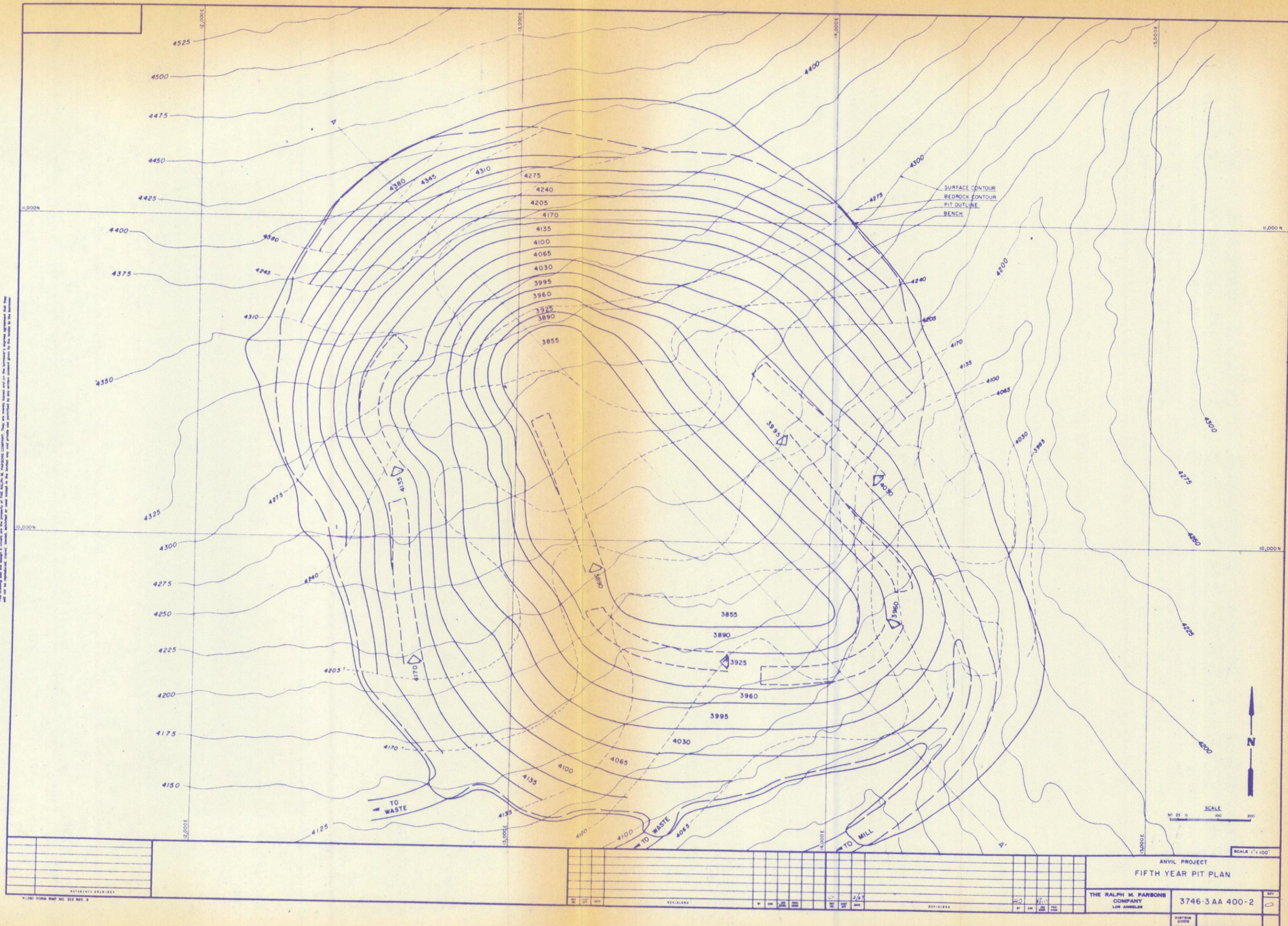
DEFENSE DRAWINGS

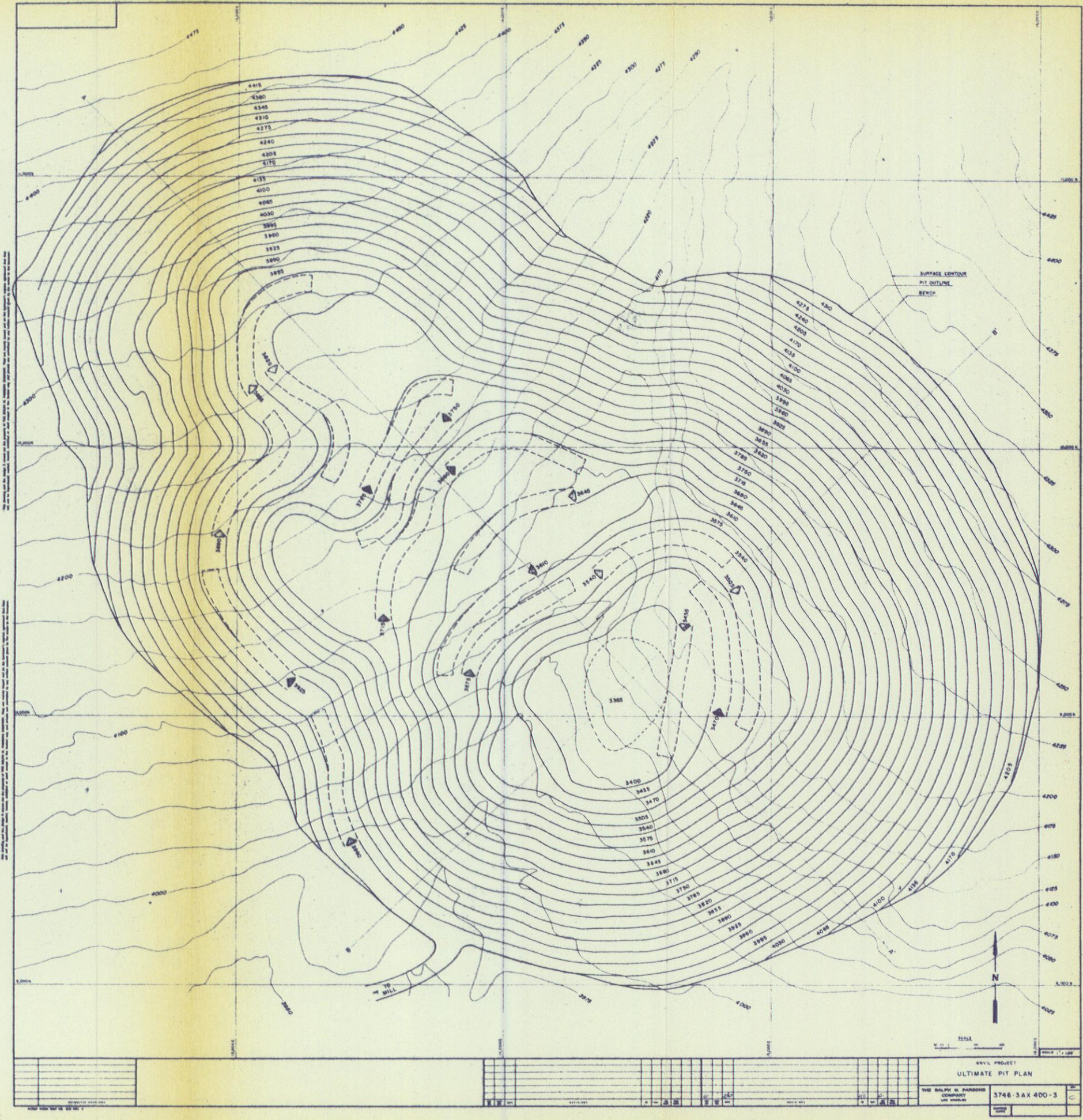
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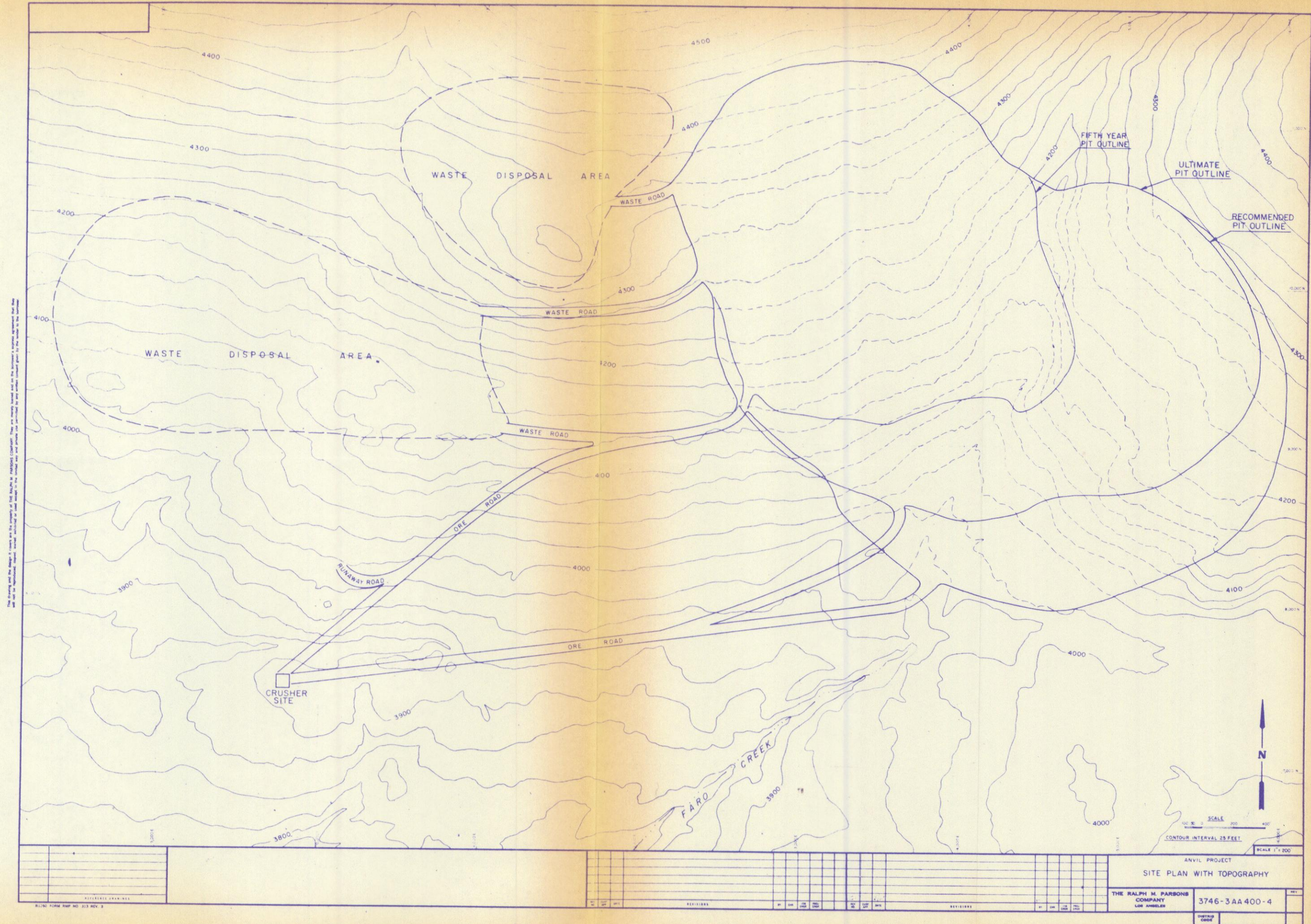
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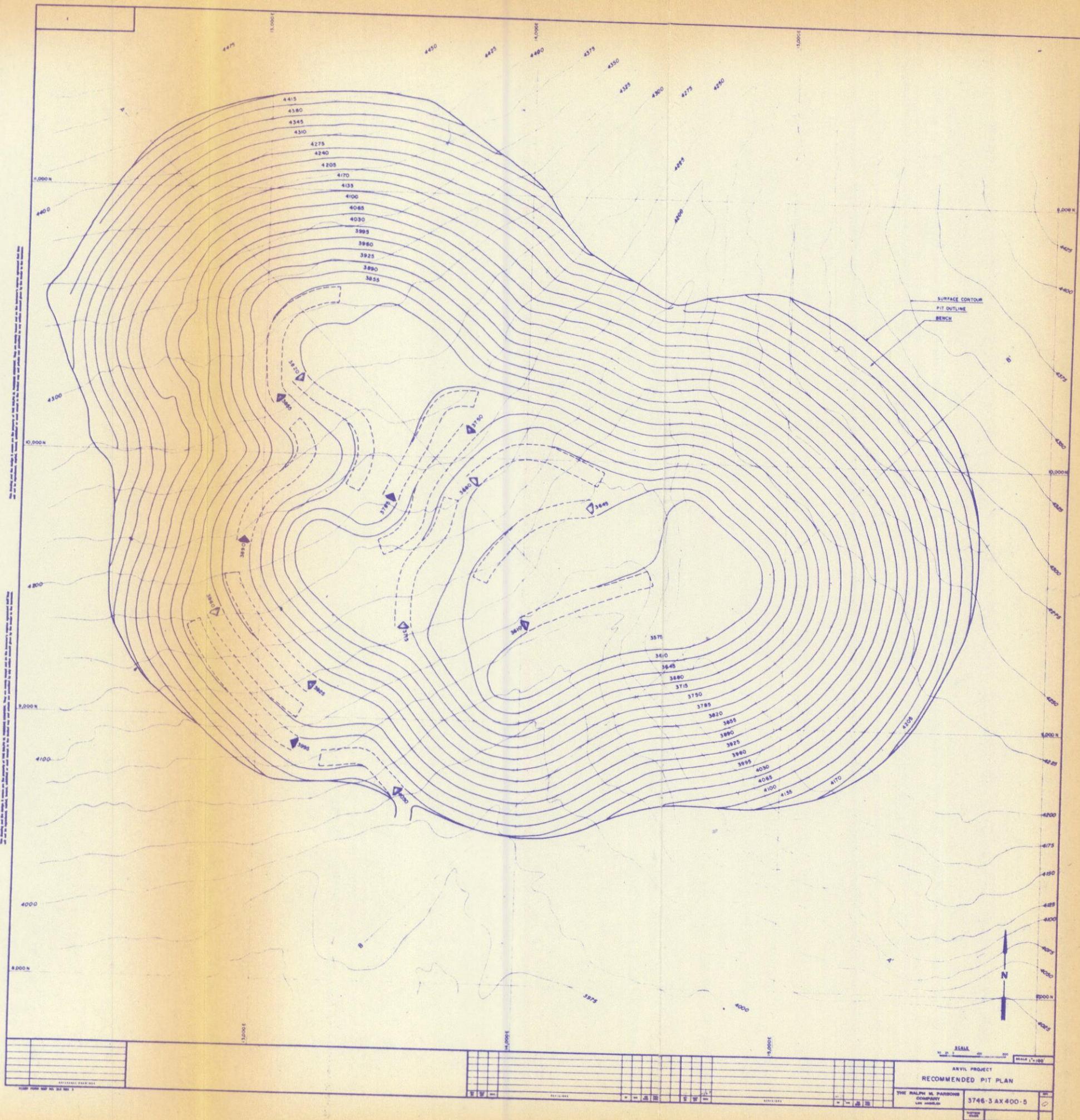
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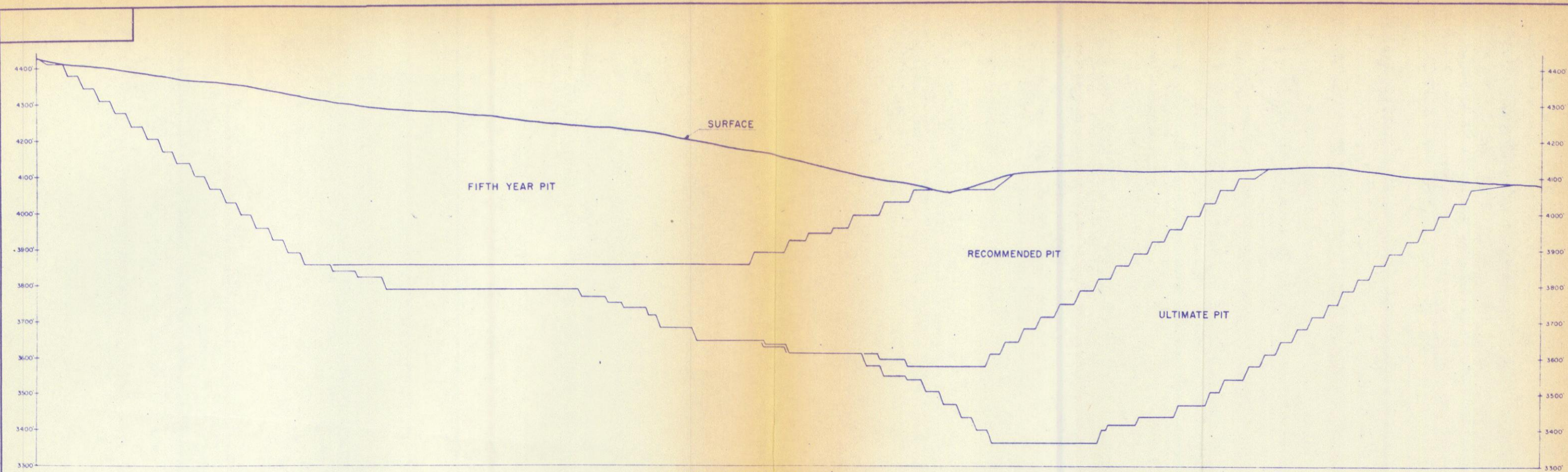
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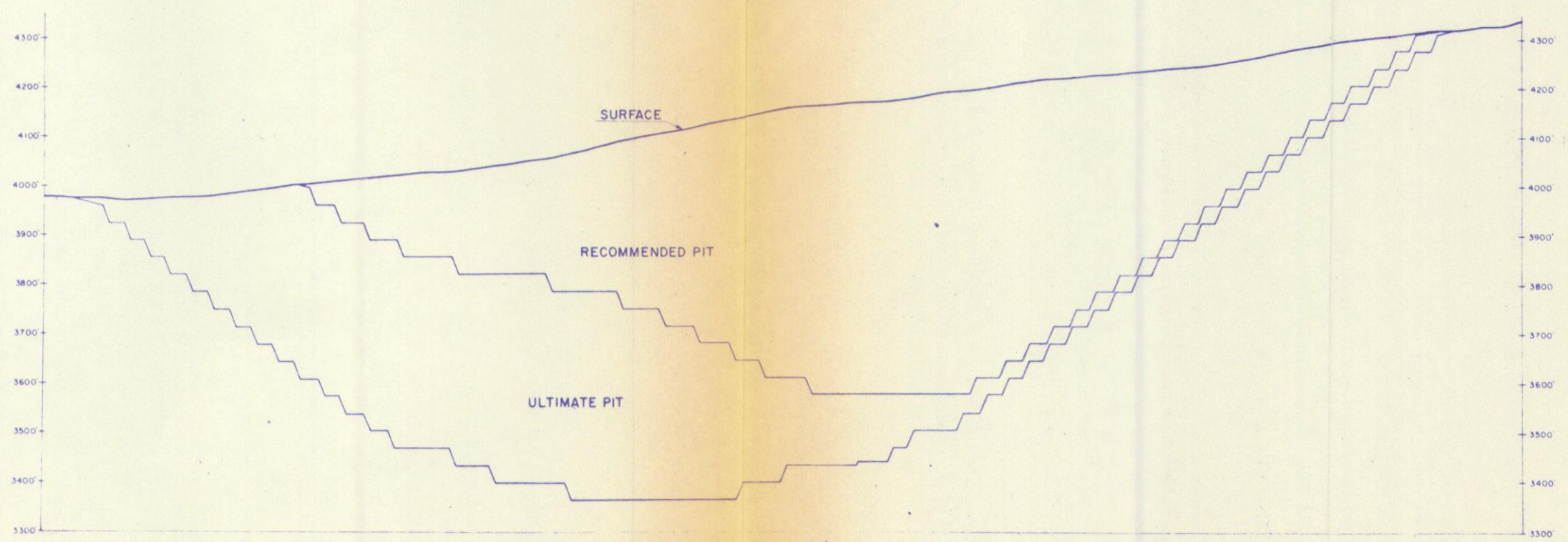




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LONG SECTION A - A'



CROSS SECTION B - B'

SCALE 1" = 100'

REFERENCE DRAWINGS		REVISIONS										ANVIL PROJECT TYPICAL SECTIONS	
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