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September 24, 1984 SEP 24 1984

Mr. I.D. Bayer
President and Chief Executive Officer
Kerr Addison Mines Limited
P.O. Box 91
Commerce Court West
Toronto, Ontario
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Dear Ian:

Re: CYPRUS ANVIL MINING CORPORATION (CAMC)

Enclosed is a copy of a mine evaluation study carried out by Pincock, Allen and Holt (PAH).

We have reviewed the PAH report and have discussed the content with Cyprus Anvil staff. Based on the review and discussion we believe the following observations are relevant in considering the PAH cash flow calculations.

1. PAH have carried out a thorough, but somewhat conservative review of minesite costs. This conservatism is understandable given the relatively high cost of pre-June, 1982 operations. However, given the dramatic improvement in operating costs which has been demonstrated in the stripping operations over the past 18 months, it is the view of CAMC that PAH's estimates can be bettered in a number of areas. The principal areas where there may be some "fat" on the PAH numbers are labor costs, townsite costs, tire costs, fuel usage, maintenance costs and blasting.

   Preliminary 1985 budget estimates completed by CAMC's minesite staff indicate mining costs will be less than those forecast by PAH. CAMC is in the process of reviewing both the PAH forecast and CAMC budget and rationalizing the differences.

2. PAH's forecast of concentrate recovery and grade is largely based on results during the short period between the mill expansion and shut-down. In the view of CAMC staff it does not fully reflect the impact of planned mill improvement expenditures and the fact that the expanded mill was never fully tuned up prior to shut-down.

3. PAH assumes that Cyprus Anvil concentrates will be trucked to Haines Alaska. It is the view of CAMC management that there is a reasonable chance that trucking to Skagway may be achieved on start-up, and very a good chance that this route will be available in the longer term. Based on PAH's estimate of $46 per tonne to Skagway this would result in an annual saving of $2-1/2 - $3 million. CAMC has received trucking bids as low as $39 per tonne; a further saving of $3 million per year.

4. CAMC's market analysis indicates that smelter tolls may be 5-10% lower than those used by PAH. In addition potentially significant credits for gold in lead concentrates are not considered.

5. The PAH report is based on the existing Faro pit plan which was developed in the late 70's. Because new reserves have been found on the perimeter of the pit, economic conditions have changed, and the pre-stripping program has been undertaken, it is likely that the existing plan can be improved upon. During the next few months CAMC plans to review and update the mining plan.

6. Neither the Vangorda Plateau reserves (open pit and underground) nor the exploration potential of the extensive block of claims in the Anvil District have been included in the evaluation since these were beyond the scope of PAH's work.

In the event you have questions on the study, please contact the undersigned and I will attempt to obtain answers from PAH. Similarly, if you wish to review CAMC's detailed budget and discuss the items which CAMC believes should be adjusted, I will arrange to supply the additional information and/or arrange a meeting with CAMC staff.

Yours very truly,

Gary Sugar

Gary A. Sugar

GAS:lgr

Encl.

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CYPRUS ANVIL MINE

REVIEW AND EVALUATION

Prepared for

DOME PETROLEUM LIMITED

Pincock, Allen & Holt, Inc.

August 1984

TABLE OF CONTENTS

TABLE OF CONTENTS
(Con't)

LIST OF TABLES

1.0 INTRODUCTION

1.0 INTRODUCTION

Mr. E. L. Forgues, Vice-President of Dome Petroleum Limited and President of Cyprus Anvil Mining Corporation, (CAMC), a subsidiary of Dome, requested in June 1984 that Pincock, Allen & Holt, Inc. (PAH) review and evaluate the Cyprus Anvil mine at Faro, Yukon. This report presents the results of our work.

Dome made available to PAH ore reserve estimates, production schedules, cost records and budgets, metallurgical test results, and other information for use in our review. We had free access to records and to Dome and CAMC personnel. We made such independent estimates of equipment, manpower, supplies, and costs as were possible and necessary.

We would like to thank Mr. Forgues and his staff for their cooperation and assistance. We were also assisted by Mr. Robert E. Thurmond, independent mining consultant, who was formerly President of Cyprus Anvil Mining Corporation. His assistance was most valuable.

The monetary unit used throughout this report is the Canadian dollar, unless the text specifically states otherwise. When it has been necessary to convert from US dollars to Canadian dollars, the rate of US$0.75/Cdn dollar has been used. In converting LME lead prices to US dollars, an exchange rate of US$1.31/British pound was used.

All revenues and costs in this report are in 1984 dollars.

2.0 SUMMARY & RECOMMENDATIONS

2.0 SUMMARY AND RECOMMENDATIONS

PAH has evaluated the Anvil project using the following assumptions:

a) mining of the Faro open pit reserves indicated to be minable by present planning, but excluding certain potential Faro pit reserves, and excluding Vangorda Plateau reserves;

b) an electric power cost of $0.07 per kwh, (present cost is $0.06 per kwh);
YTD June '84 0.0593

c) truck haulage of concentrates from Faro to Haines, Alaska, with a backhaul by truck of operating supplies;

d) an investment in 1984 and 1985 of about $30 million, including $15.3 million for mine, mill, and townsite capital, recruiting and environmental expense, $10 million in working capital, and about $4.4 million for stripping 4.4 million DMT of waste in November and December 1984.

Our evaluation is based on achievement of an average milling rate of 11,160 DMT of ore per day, and on metallurgical performance improved over historical performance with respect to mill recoveries and concentrate grades. We consider the milling rate, recoveries and concentrate grades to be reasonable in light of testwork and planned plant modifications. The evaluation also is based on operating costs which reflect improvement over past operations, but which we believe are attainable with good management.

Our conclusions are as follows:

1. With the above assumptions, at present metal prices of US $0.45 per pound for zinc, US $0.22 per pound for lead, and US $7.50 per ounce for silver, the project has the potential to achieve payout of the incremental investment of about $30 million needed to commence operations in about two years from start-up. It would generate $231 million in pre-interest and pre-tax net cash flow over the life of the reserves included in the presently designed Faro open pit mining phases. The discounted cash flow return on investment (ROI) represented by the pre-tax cash flows is 78.7%. Dome management informs us that because of CAMC's tax position, little or no taxes would be payable for several years. In this case the after-tax

ROI would also be very high. At a 15 percent discount rate, the project at present metal prices has a net present value of $114 million.

2. Additional ore reserve evaluation and mine planning should be undertaken to optimize the operation and improve the project's cash flows.

3. The proven Faro open pit reserves are limited. They will last about seven years at the 11,160 DMT per day milling rate. The Vangorda Plateau reserves could more than double the life of the project. These reserves and the Faro deposit would than represent a long term secure source of concentrates. Canada is a favorable country from a political standpoint. There is a concentrator in place which can be brought up to the 11,160 DMT/day milling rate for a relatively modest outlay. While Faro is remote, townsite and infrastructure are in place, and roads to tidewater are there. These factors are certainly a plus when compared to the costs of grassroots mine developments in other areas.

4. CAMC management has expressed its intention not to proceed with start-up unless truck transportation of concentrates to Haines or Skagway, with load and length limits set at economic levels, is permitted by the

Canadian and Alaskan authorities, or rail transportation at competitive rates is assured on a long term basis, a favorable labor contract is signed, a new electric power agreement is signed, and smelting contracts on reasonable terms are assured. We agree that it is necessary to resolve these matters favorably. Because of tight cash flows in the early years, CAMC management should continue to pursue trucking to Skagway.

5. It is technically feasible to start the project up at a milling rate of 9,300 tpd in December, 1984, and to perform the modifications needed to reach the average 11,160 tpd rate, with temporary shutdowns as required, by the third quarter of 1985.

6. Smelting contracts must be negotiated before startup that will ensure an outlet for the concentrates. An outside consultant with hands-on recent experience in marketing lead and zinc concentrates should be engaged to assist in developing and executing a concentrate marketing plan.

Absolute must.

7. Truck haulage is technically feasible over either the Skagway or Haines roads. If it is decided to use the Haines road, a port facility must be constructed. No cost estimate is available, but the cost could be in the

neighborhood of $5 million, assuming no dredging is required. The operating cost could be about $3,120,000 yearly, including depreciation.

3.0 MINING

3.0 MINING

3.1 Summary

CAMC estimates that the minable reserves of the Anvil pit are as follows:

Open pit ore	26,432,000 DMT	4.3% Zn, 2.9% Pb, 36.1 g/t Ag
Stockpiles	1,400,000 DMT	4.7% Zn, 2.9% Pb, 37.6 g/t Ag
Total	27,832,000 DMT	4.3% Zn, 2.9% Pb, 36.2 g/t Ag

Waste stripping is currently being done. CAMC estimates that approximately 80 million DMT of waste will remain to be mined as of December 31, 1984. This would amount to a stripping ratio for pit ore of approximately 3.03 to 1.00.

Within the allotted time and money, the establishment of confidence in ore reserve calculations has been limited to a check of methods used by CAMC personnel. Within these limitations, PAH believes the reserve estimates are reasonable for this evaluation. A complete check of the reserves would require substantially more time than is available for this report. CAMC is preparing to run an updated, detailed ore body model in 1985 as described in Section 3.2 of this report.

CAMC has designed a number of mining phases for development of the pit. The phases appear to be well engineered, but should be incorporated into a detailed life-of-mine mining plan. The mining schedule currently used by CAMC appears to have sufficient lead time to allow for normal deviations, such as unexpected equipment downtime. The schedule indicates four to five months ore developed ahead of mining. Normally PAH recommends a six-month cushion.

In general mine operations compare favorably with those of other open pit mines. In Section 3.4 of this report, a few areas for improvement or review are highlighted with the aim of reduction of operating costs.

Our review of the major mining equipment available at the mine indicates that the present fleet is adequate for the remaining life of the Anvil pit with one exception. One additional rotary blasthole drill must be acquired for use in 1985 through 1987. The drill is expected to cost about $1,800,000.

PAH estimates that mine operating costs will be about $1.05/DMT of rock mined in 1985, rising to $1.24 by 1987, and will peak at slightly over $2.00/DMT in 1990. Total annual mining costs will decrease after 1987, due to the large reduction in the stripping ratio in 1988 and following years. Our mining cost estimates assume that truck haulage of concentrates will be possible, and that some savings over present delivered supply costs will be achieved by backhauling fuel, tires, and explosives. The estimates also assume a $0.07/kwh power cost.

3.2 Ore Reserves

Within the allotted time and money, the establishment of confidence in ore reserve calculations has been limited to a check of methods used by CAMC personnel. Within these limitations, PAH believes the reserve estimates are reasonable for this evaluation and would not expect a detailed grass roots calculation to result in a major change in reserves. A complete independent review of the geologic and minable reserves would require considerably more time for examination of the raw assay data and a rerun of the computer mine modeling.

CAMC management is presently planning a rerun of the geologic model early in 1985. This "F-4" model will primarily be based upon a 140 x 140 foot grid spacing while the previous "F-3" model was developed from drill hole information based upon a 140 x 280 foot grid. The F-3 model used 50 x 50 foot blocks. The F-4 model will use 35 x 35 foot blocks. In both cases the bench height is 20 feet.

Minable reserves, exclusive of stockpile tonnage, have been calculated by CAMC at 26,432,000 dry metric tonnes. Table 3.1 shows the tonnage and grade of the reserves along with the projected mining schedule.

TABLE 3.1

ZONE 3 - MINE PLAN TONNAGE & GRADE

CYPRUS ANVIL MINE

	Mining Rate/Day Total Material BCY	Total Ore/Day Metric Tonnes	Total Waste BCY/Day	Total Waste Metric Tonnes/Day	S/R	% Zn	% Pb	Ag Grams Per Tonne	Tonnes Ore From Pit Per/Year (000)	Total Tonnes Waste/Year (000)	In-Pit Tonnes Ore Remaining Year End (000)	In-Pit Tonnes Waste Remaining Year End (000)	Remaining S/R
1984 - Nov 1 to End of Year (61 Days)	34,000	SP 7,000*	34,000	71,400	---				(SP 427)	4,355	26,432	80,295	3.03:1
1985 - Jan 1 to Apr 17 (105 Days)	34,000	SP 9,300*	34,000	71,400	---				(SP 973) (SP 1,400)*	7,497	26,432	72,798	2.75:1
Apr 18 to Aug 8 (109 Days)	34,000	9,300	30,793	64,665	6.95:1	4.4	2.7	36.7	1,014	7,049	25,418	65,749	2.59:1
Aug 9 to Dec 31 (141 Days)	34,000	11,473	30,044	63,092	5.50:1	4.4	2.7	36.7	1,617 2,631	8,096 23,442	23,801	56,853	2.39:1
1986 (355 Days)	34,000	11,473	30,044	62,516	5.62:1	4.4	2.9	33.4	4,073	22,398	19,728	34,455	1.75:1
1987 (355 Days)	32,750	11,473	28,794	60,407	5.27:1	4.0	2.8	35.7	4,073	21,466	15,655	12,989	0.83:1
1988 (355 Days)	15,000	11,473	11,044	23,192	2.02:1	4.5	3.4	45.3	4,073	8,233	11,582	4,756	0.41:1
1989 (355 Days)	7,500	11,473	3,544	7,442	0.65:1	4.7	3.2	40.2	4,073	2,642	7,509	2,114	0.28:1
1990 (355 Days)	5,500	11,473	1,544	3,242	0.28:1	4.6	3.0	35.8	4,073	1,151	3,436	963	0.28:1
1991 (299 Days)	4,490	11,473	1,554	3,221	0.28:1	4.1	2.4	26.6	3,436	963	-	-	-
								TOTAL PIT ORE	26,432	84,650			
								TOTAL PIT AND STOCKPILE	27,832				

* Material supplied to mill from high grade and low grade stockpiles

The inverse-distance-squared interpolation method in massive sulphide environments typically tends to define overall deposit reserves and grades to within acceptable tolerance limits but can deviate substantially at the single mining unit (block) scale. Although this method has been used at CAMC for the mine model, it was determined that it cannot be depended upon for short-range mine planning. Individual ore-rock type "kriging" might enhance the accuracy of the reserve interpolation.

CAMC appears to have done a detailed job on the geology of the deposit. Additional data, not available in 1981 for assay interpolation in the F-3 model, will be used for the F-4 model re-run planned for early 1985.

Geologic sections have been prepared 141.5 feet apart with the longitudinal sections at 135 degrees off of true north and cross sections at 45 degrees from true north.

The present geologic model of the ore deposit incorporates the information derived from diamond drill core recovery of 95 percent in the ore zone and from the analysis of core lengths of nearly all 5 feet or more. A Sperry-Sun shot camera was used to determine inclination of structure. Logs of the drill holes list six different ore types and grades, lithologic features, various structural features and foliation of material.

Ore columns have been composited at 20 foot intervals and drill hole data is stored on magnetic tapes for easy access.

Table 3.2 represents the ore reserve estimate prepared by the Geology Department at Faro as of July, 1984.

It will be noted that the grade of the minable reserves is slightly less than the grade of the geologic reserves. The geologic reserves include a deep portion of the ore body which has not been included in the minable reserves because of its high stripping ratio. This excluded material amounts to approximately 2,000,000 metric tonnes and dips westerly from the bottom of the west pit wall. This lens is quite high grade, with roughly 13 percent combined lead and zinc and 71 grams/tonne of silver. The thickness of an average of 24 feet might make possible the recovery of the ore by underground methods utilizing a decline driven from the bottom of the pit. The technical and economic feasibility of this should be investigated by CAMC.

An area of slope instability on the eastern pit wall presents a problem along with a possible loss of minable reserves. At present time the failure and sloughing of the pit wall has blocked the mining of approximately 1,500,000 metric tonnes which have been included in the minable reserves in Tables 3.1 and 3.2 and in the mine plan.

TABLE 3.2

ORE RESERVES

CYPRUS ANVIL MINE

	Tonnage DMT	Assays Zn	Pb	Ag
Geological Reserves, cutoff 4% combined lead and zinc combined	33,000,000	4.6%	3.0%	36.0 g/t
Minable Reserves, cutoff 4% combined lead and zinc	26,432,000	4.3%	2.9%	36.1 g/t
Stockpiles[1]				
Oxide	1,363,000	4.7%	2.9%	37.6 g/t
CFSP II	22,000	3.8%	2.2%	33.0 g/t
Total of minable reserves and stockpiles	27,817,000	4.3%	2.9%	36.2 g/t

[1] There is a difference of 15,000 tons between this stockpile tonnage and the stockpile tonnage of 1,400,000 tons used in the CAMC mining plan. This is not significant.

Optional approaches to this problem are as follows:

1) Do nothing and sacrifice 1,500,000 tonnes of ore now carried as minable reserves.
2) Clean off much of the spill material and step in with final pit limit. This would lose some ore, the quantity of which would be dependent upon the extent of the final pit limit.
3) Push final limit back 200 feet and mine the push back at a 36.5 to 38 degree slope. Approximately 3.5 to 4 million tonnes of ore would be uncovered, but the stripping ratio of 7.8 to 8.9:1 makes this option marginal.
4) Bring in contractor to strip approximately 1,000,000 cubic yards of loose overburden to allow better examination of fault zone. This last option is the preferable plan of action. Options 2 and 3 require the removal of the loose overburden in any case. At present the footwall of the fault along which the failure occurs has shown very little evidence of failure and it is possible that the pit wall could be maintained at a slope considerably steeper than 38 degrees which in turn could provide at a much reduced cost, much of the 3-1/2 to 4 million metric tonnes of ore calculated to be recovered with a 200 foot push back. This would

improve the stripping ratio which at 38 degrees is a prohibitive 7.8:1 to 8.9:1 when the 200 foot push back is considered.

In any case, further efforts must be made to divert water flowing in a diversion channel which has been cut above the east slope. This water flows over the Big Indian Fault exposure and is obviously a large part of the problem although normal ground water also accounts for some of the instability.

It may well be that grouting of the channel particularly in the faulted area, is the only good solution to the channel water penetration. Bentonite or drilling mud is probably not a permanent fix. Plastic sheeting has already been tried without success. It is estimated that the stripping and channel grout work could be accomplished for $1,500,000. We have allowed for this in our capital expenditure budget in our cash flow projections.

More frequent ore grade update and short range mine planning could be facilitated greatly by better use of the computer (IBM-PC) capability at the mine. Compatability between the existing Tektronix terminals and the IBM-PC microcomputer requires investigation. Assuming the Tektronix terminals at the mine could be used with the microcomputer, a graphics software interface package will be required for graphics capabilities. The on-site microcomputer can enhance the CAMC operation by cutting their computer overhead cost, and recording and maintaining production data as it is produced.

3.3 Mine Planning

Mine planning was reviewed by checking the existing pit maps, projected mine tonnage removal in chronological sequence, and pit bench tonnage.

Information available for review consisted of:

1) Individual phase designs at 1" = 100' for phases NA, OA, PA, 7D, UA, WA, and YA.

2) Tabulated phased reserves remaining as of November 30, 1984.

3) A graphical paper mining schedule of remaining reserves for November 1984 through September 1991.

Using this information, PAH has reviewed the phase designs, the mining schedule, and pit operating geometry resulting from the current mining schedule.

The current sequence of phases is based on minimizing stripping required to reach ore as shown by the following sequence of phased reserves:

Phase	Waste Stripping To Reach Ore KTonnes[1]	Ore and Internal Waste: Ore KTonnes	% Pb	% Zn	gpt Ag	Waste KTonnes[1]	W:0 Ratio	Total W:0 Ratio
NOP[2]	-	3,909	2.7	4.3	35.6	17,279	4.42	4.42
7D	4,767	1,856	2.9	4.6	31.7	6,163	3.32	5.89
UB	8,106	5,624	2.7	4.0	33.9	8,984	1.60	3.04
WA	18,749	9,872	3.2	4.5	41.0	14,274	1.45	3.35
YA	2,915	4,971	2.7	4.4	31.4	3,213	0.65	1.23
SUBTOTAL	34,537	26,232	2.9	4.4	36.2	49,913	1.90	3.22
Ramp		200				200	1.00	
TOTAL	34,537	26,432				51,113		

1 At 2.10 tonnes waste per bcy.

2 Phases NA, OA, and PA will be mined concurrently and may be considered as a single phase (NOP).

Based on the above sequence, each phase has been designed, complete with access, on a stand alone basis. This means that any phase under consideration could be mined as the last pit increment and does not depend on succeeding phases for access. This design philosophy is good engineering practice as it allows many scheduling options to be evaluated without major re-designs. When phases are combined, as with NA, OA, and PA, or advance stripping is scheduled, some of the individual phase access will not be necessary, but it is better to have access designed and not implemented rather than to suddenly require access that has not been previously evaluated. In summary, PAH considers the phases themselves which have been designed by CAMC to be well engineered but a need exists to incorporate the various phase calculations into a complete mining plan. The total tonnage of waste to be removed in all phases is 84,650,000 DMT and the ore to be mined is 26,432,000. These phase tonnages correspond to the waste and ore tonnages shown in Table 3.1. The figures in Table 3.1 have been used in our cash flow projections.

The mining schedule currently being used appears to have sufficient lead time to cover normal deviations, such as unexpected equipment downtime. A preliminary review of available figures indicates cushion of between four and five months between ore stripped and ore required. PAH normally recommends maintaining a six month cushion but in general, roughly concurs with the scheduled stripping rates.

Mine operating geometry was reviewed by integrating the CAMC graphical paper schedule with the individual phase designs to develop pit composites showing pit configurations at the end of each calendar year. In the process of developing these composites, PAH has essentially mined the deposit on paper, with each bench of each phase being reviewed to insure accessibility. Tables 3.3 through 3.9 show which benches of the various phases are mined during each year from 1985 through 1991. These tables have been derived from the CAMC graphical schedule. Pit composites assembled by PAH showing annual operating geometry at the ends of 1984 through 1991 are included at the end of this section. Note that partial benches (those benches being mined at the end of a year) are not shown on these drawings as PAH does not have sufficient information to draw meaningful partials. The only significant conflict between the paper schedule and operating geometry occurs in 1987. The paper schedule shows Phase YA starting in May to mine out the access ramp developed in Phase WA. This ramp must be maintained until Phase WA joins the final pit access ramp on the 3670 bench. Consequently, Phase YA cannot start until September, 1987. Since there is adequate ore stripped in Phase WA, this deferral of stripping will not be a problem. Development of the annual composites indicated several areas where temporary access must be left for a time and later mined out. In each case, access is developed and removed within the same year, and consequently does not show on the year-end configurations. In order to point out these areas, PAH has developed 'intermediate' composites for 1986, 1987, and 1988. Areas requiring advance consideration prior to mining have been noted on the composites and on the tables.

TABLE 3.3

SOURCE OF MATERIAL MINED 1985
CYPRUS ANVIL MINE

Month	NA Bench	OA Bench	PA Bench	7D Bench	UB Bench	WA Bench	YA Bench
Jan	3830	3830	3830 3810	3830	-	-	-
Feb	3830 3810	3810	3810 3790	3830 3810	-	-	-
Mar	3810	3810 3790	3790	3810	-	-	-
Apr	3810 3790	3790 3770	3790 3770	3810 3790	-	-	-
May	3790 3770	3770	3750	3790 3770	-	-	-
June	3770 3750	3770 3750	3730	3770	-	-	-
July	3750 3730	3750 3730	3730 3710	3770 3750	-	-	-
Aug	3710 3690 3670	3730 3710	3710 3690	3750 3730	-	-	-
Sept	3670	3710 3690	3690 3670	3730 3710	3830	-	-
Oct	-	3690 3670 3650	3670 3650	3710 3690	3830	-	-
Nov	-	3650 3630	3650 3630	3690	3830 3790	-	-
Dec	-	-	3630 3610 3590	3690 3670	3790	-	-

NA, OA, and PA mined concurrently.

TABLE 3.4

SOURCE OF MATERIAL MINED 1986
CYPRUS ANVIL MINE

Month	NA Bench	OA Bench	PA Bench	7D Bench	UB Bench	WA Bench	YA Bench
Jan	-	-	3590 3570	3670 3650	3790 3750	-	-
Feb	-	-	3570 3550	3650 3630	3750	3990 3950	-
Mar	-	-	-	3630 3610	3750 3710	3950	-
Apr	-	-	-	3610 3590	3710	3950	-
May	-	-	-	3590 3570	3710* 3670	3950	-
June	-	-	-	3570 3550	3670*	3950	-
July	-	-	-	3550 3530	3670* 3650	3950 3910	-
Aug	-	-	-	3530 3510	3630* 3610	3910	-
Sept	-	-	-	-	3610 3590	3910	-
Oct	-	-	-	-	3590 3570	3910 3870	-
Nov	-	-	-	-	3570 3550	3870	-
Dec	-	-	-	-	3550 3530	3870 3830	-

*Some of this material is tied up in temporary access ramp (3710 to 3630) and will not be available until August/September, when ramp is removed.

TABLE 3.5

SOURCE OF MATERIAL MINED 1987
CYPRUS ANVIL MINE

Month	NA Bench	OA Bench	PA Bench	7D Bench	UB Bench	WA Bench	YA Bench
Jan	-	-	-	-	3530	3830	-
Feb	-	-	-	-	3530	3830 3790	-
Mar	-	-	-	-	3530 3510	3790	-
Apr	-	-	-	-	3510	3790 3750	-
May	-	-	-	-	3510 3490	3750	3790* 3750
June	-	-	-	-	3490	3750 3710	3710
July	-	-	-	-	3490 3470	3710	3710 3670
Aug	-	-	-	-	3470	3710 3690	3670
Sept	-	-	-	-	3470 3450	3690 3670	3670
Oct	-	-	-	-	3450 3430	3670 3650	3670 3630
Nov	-	-	-	-	3430	3650 3630	3630
Dec	-	-	-	-	-	3630	3630

*Phase YA mines out the access ramp developed in Phase WA. This ramp must be maintained until Phase WA joins the final pit access on the 3670 bench. Consequently, Phase YA cannot start until September, 1987.

TABLE 3.6

SOURCE OF MATERIAL MINED 1988
CYPRUS ANVIL MINE

Month	NA Bench	OA Bench	PA Bench	7D Bench	UB Bench	WA Bench	YA Bench
Jan*	-	-	-	-	-	3610 3590	3630 3590
Feb*	-	-	-	-	-	3590	3590
Mar*	-	-	-	-	-	3570 3550	3590
Apr*	-	-	-	-	-	3550 3530	3590
May*	-	-	-	-	-	3530	3590 3550
June*	-	-	-	-	-	3530	3550
July*	-	-	-	-	-	3530	3550
Aug	-	-	-	-	-	3530 3510	3550 3510
Sept	-	-	-	-	-	3510	3510
Oct	-	-	-	-	-	3510	3510
Nov	-	-	-	-	-	3510 3490	3510
Dec	-	-	-	-	-	3490	3510

*Must maintain temporary access in north end to 3670 until Phase WA/YA connects with bottom fo UB (5810 bench).

WA and YA mined concurrently for entire year and balance of mine life.

TABLE 3.7

SOURCE OF MATERIAL MINED 1989
CYPRUS ANVIL MINE

Month	NA Bench	OA Bench	PA Bench	7D Bench	UB Bench	WA Bench	YA Bench
Jan	-	-	-	-	-	3490	3510 3490
Feb	-	-	-	-	-	3490	3490
Mar	-	-	-	-	-	3490	3490 3470
Apr	-	-	-	-	-	3490 3470	3470
May	-	-	-	-	-	3470	3470
June	-	-	-	-	-	3470	3470 3450
July	-	-	-	-	-	3470	3450
Aug	-	-	-	-	-	3470	3450 3430
Sept	-	-	-	-	-	3470 3450	3430
Oct	-	-	-	-	-	3450	3430
Nov	-	-	-	-	-	3450	3430 3410
Dec	-	-	-	-	-	3450	3410

TABLE 3.8

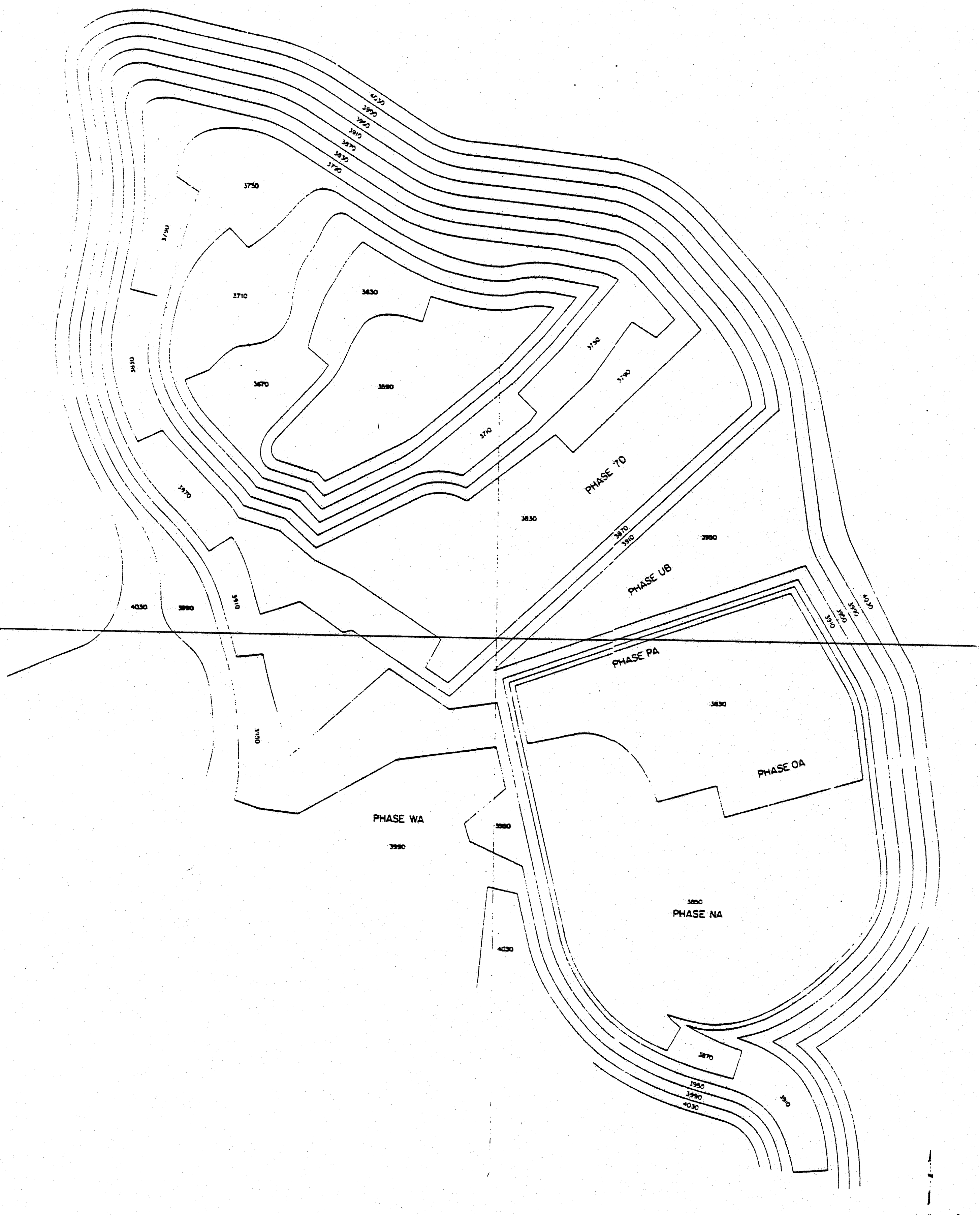
SOURCE OF MATERIAL MINED 1990
CYPRUS ANVIL MINE

Month	NA Bench	OA Bench	PA Bench	7D Bench	UB Bench	WA Bench	YA Bench
Jan	-	-	-	-	-	3450	3410
Feb	-	-	-	-	-	3450 3430	3410
Mar	-	-	-	-	-	3430	3410
Apr	-	-	-	-	-	3430	3410
May	-	-	-	-	-	3430	3410 3390
June	-	-	-	-	-	3430	3390
July	-	-	-	-	-	-	3390
Aug	-	-	-	-	-	-	3390
Sept	-	-	-	-	-	-	3390
Oct	-	-	-	-	-	-	3390 3370
Nov	-	-	-	-	-	-	3370
Dec	-	-	-	-	-	-	3370

TABLE 3.9

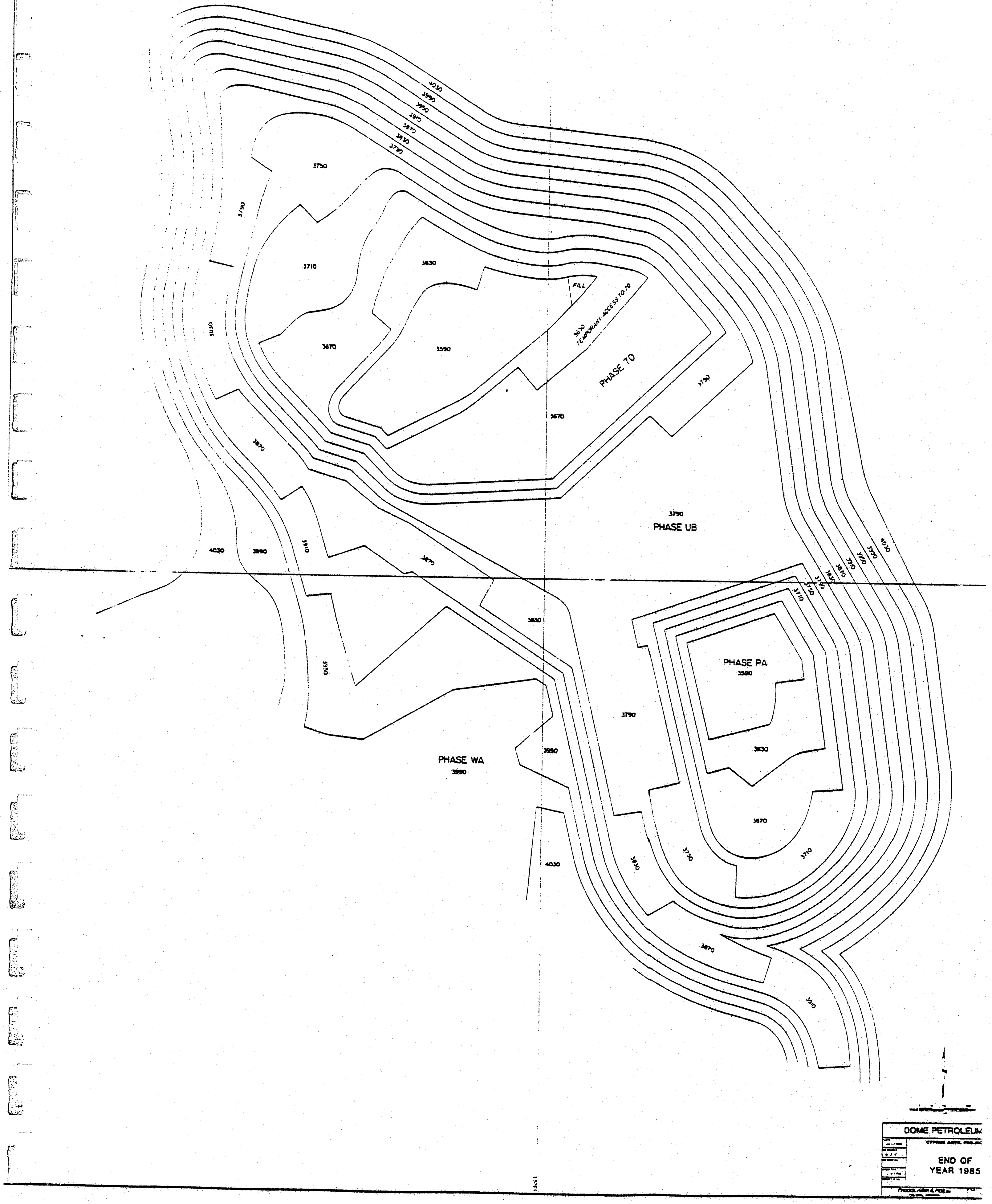
SOURCE OF MATERIAL MINED 1991
CYPRUS ANVIL MINE

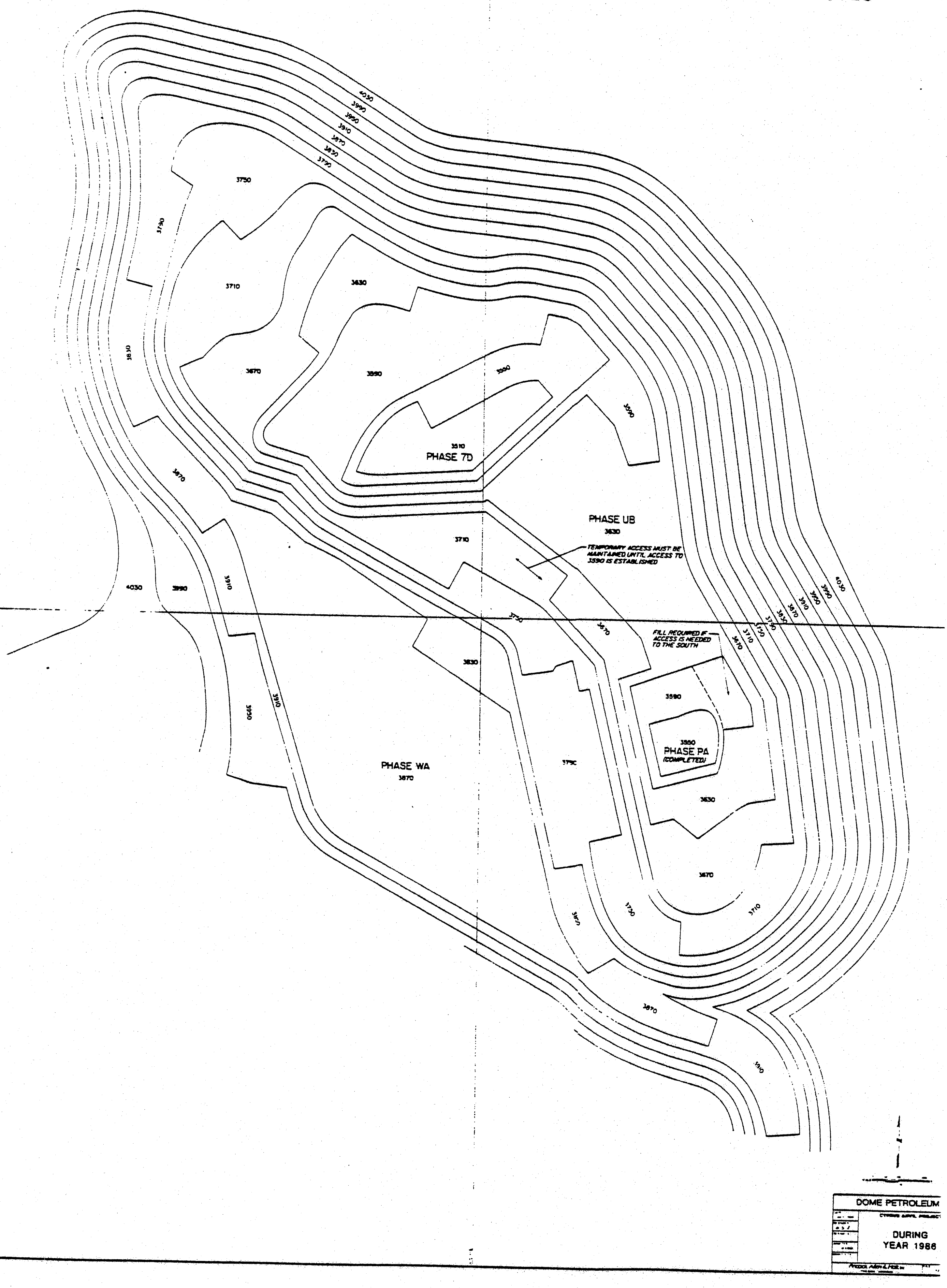
Month	NA Bench	OA Bench	PA Bench	7D Bench	UB Bench	WA Bench	YA Bench
Jan	-	-	-	-	-	-	3370
Feb	-	-	-	-	-	-	3370 3350
Mar	-	-	-	-	-	-	3350
Apr	-	-	-	-	-	-	3350 3330
May	-	-	-	-	-	-	3330

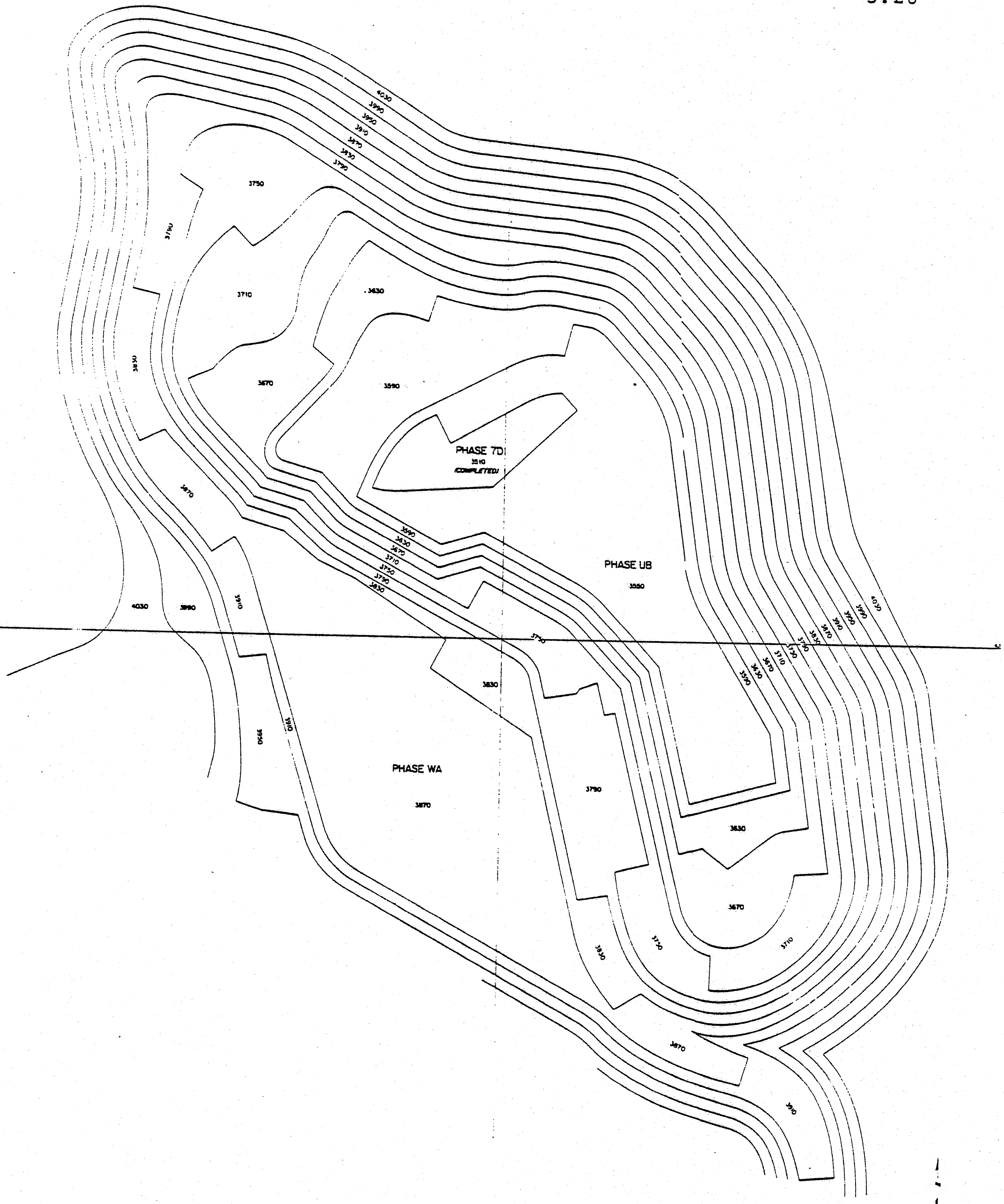


DOME PETROLEUM

END OF
YEAR 1984


3750
3710
3630
3670
3590
FILL
TEMPORARY ACCESS TO 70
PHASE 70
3670
3870
3790
PHASE UB
4030
3990
3910
3830
PHASE PA
3590
3790
3630
PHASE WA
3990
4030
3670
3710
3870
DOME PETROLEUM
END OF
YEAR 1985

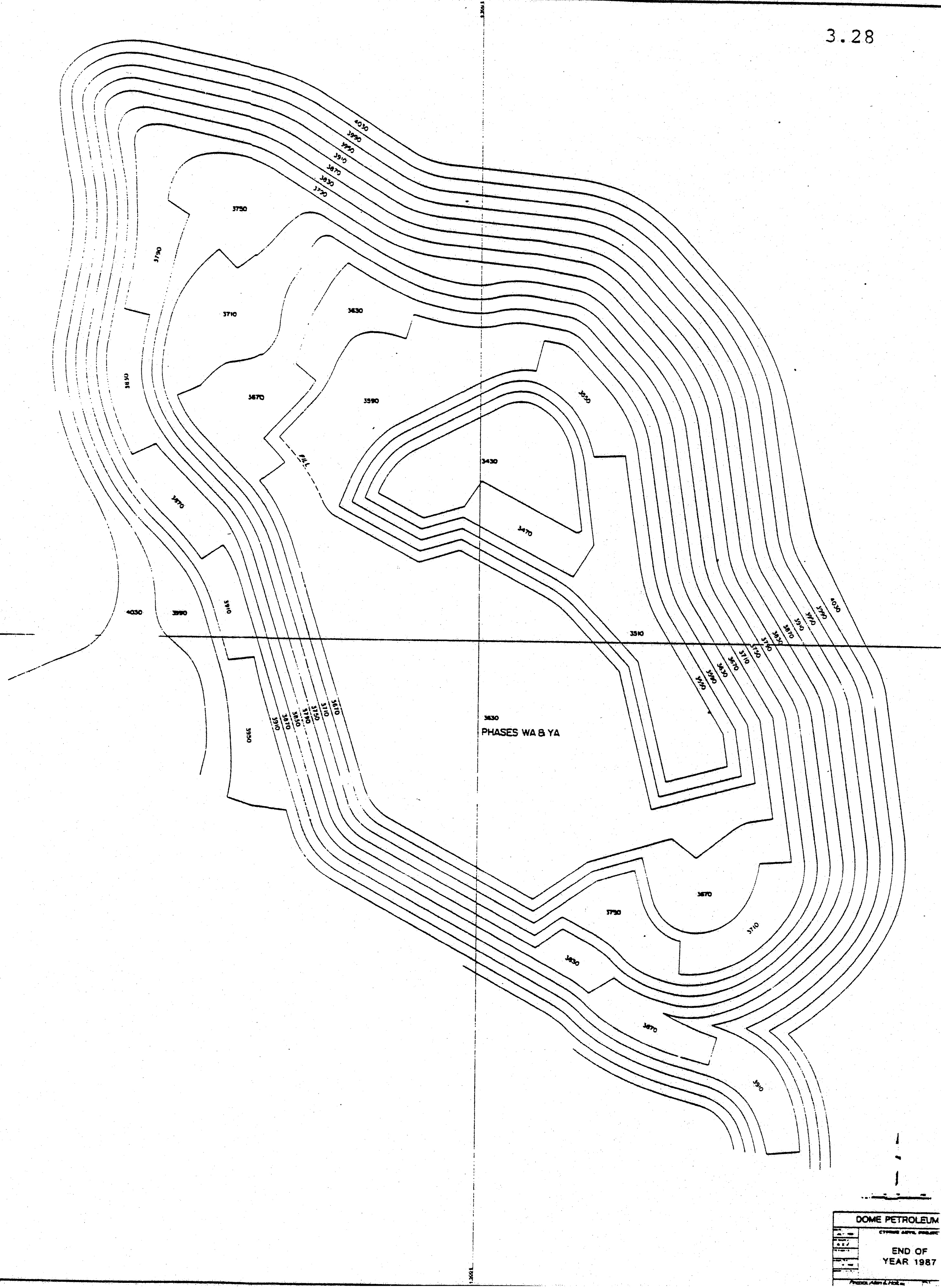

PHASE 7D
3510
PHASE UB
3630
TEMPORARY ACCESS MUST BE MAINTAINED UNTIL ACCESS TO 3590 IS ESTABLISHED
FILL REQUIRED IF ACCESS IS NEEDED TO THE SOUTH
3550
PHASE PA
(COMPLETED)
PHASE WA
3870
DOME PETROLEUM
DURING
YEAR 1986

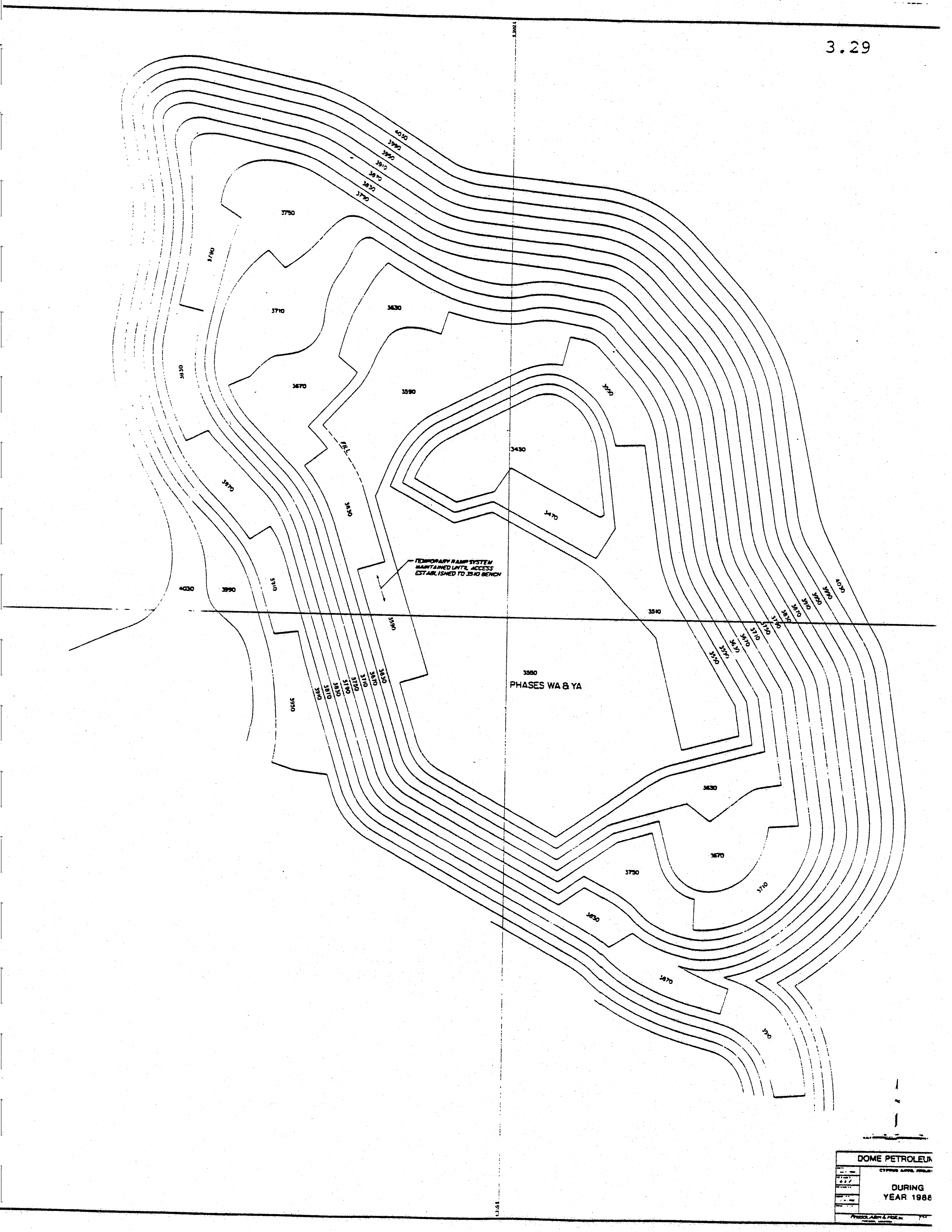


DOME PETROLEUM

END OF
YEAR 1986

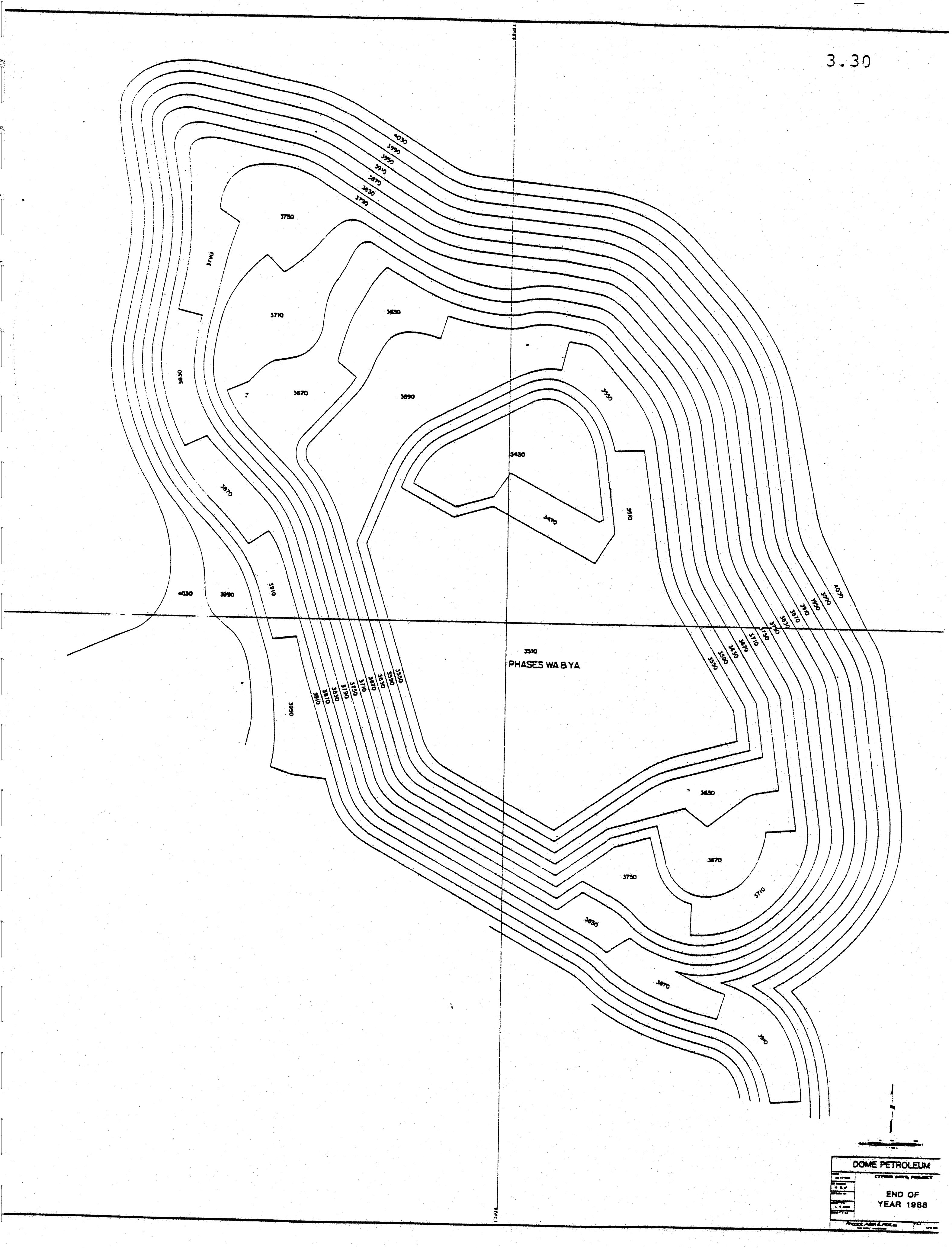

4030
3990
3950
3910
3870
3830
3790
3750
3710
3630
3590
3430
3470
3670
3510
PHASE WA
PHASE YA
THIS ACCESS IS REQUIRED FOR WA MATERIAL —
UNTIL ACCESS TO 3670 IS ESTABLISHED.
CONSEQUENTLY, YA CANNOT START UNTIL SEPT/87
DOME PETROLEU
DURING
YEAR 198

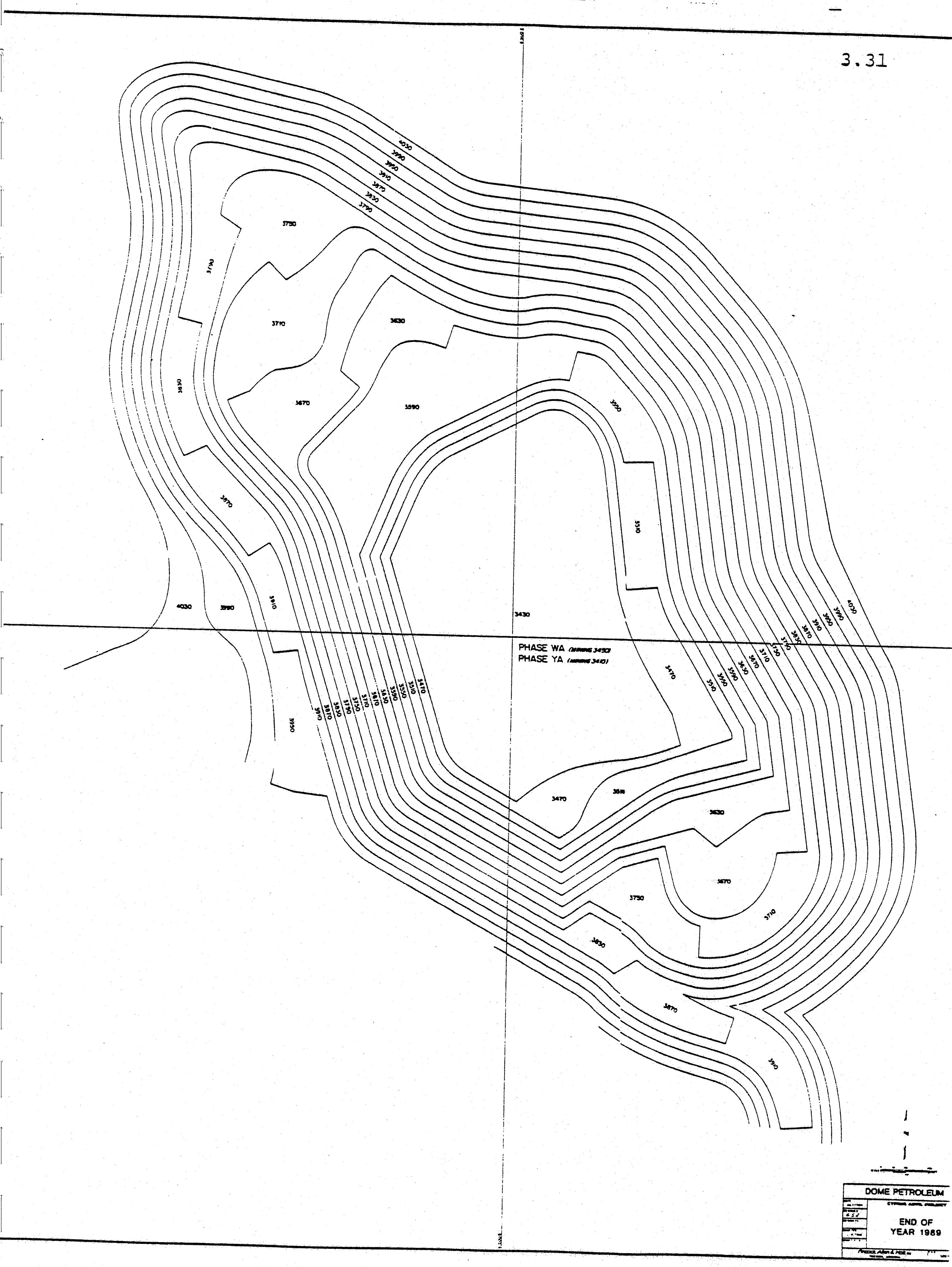

4030
3990
3950
3910
3870
3830
3790
3750
3710
3670
3630
3590
3550
3510
3470
3430
FILL
PHASES WA & YA
DOME PETROLEUM
END OF
YEAR 1987

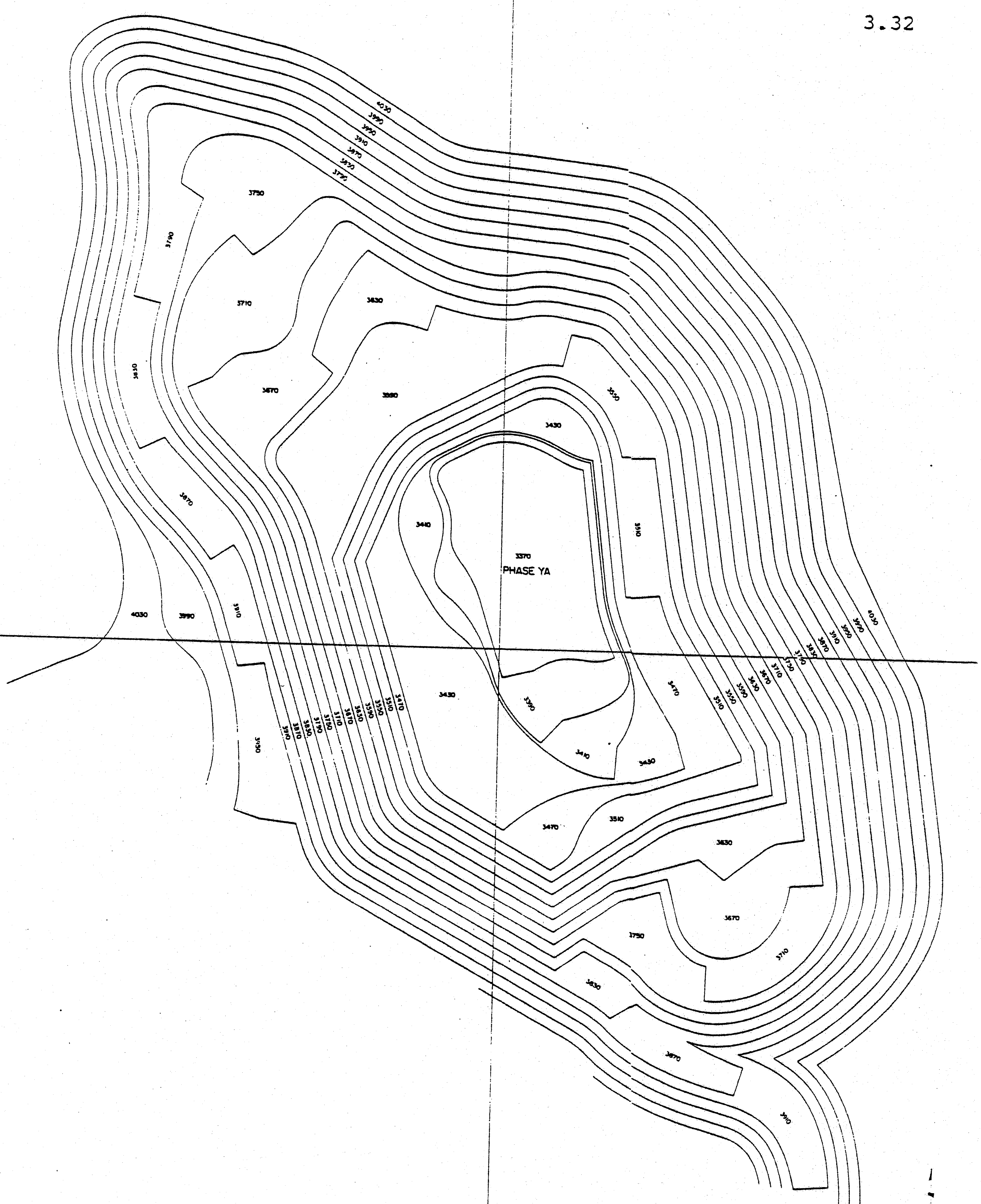


TEMPORARY RAMP SYSTEM MAINTAINED UNTIL ACCESS ESTABLISHED TO 3510 BENCH
PHASES WA & YA
4030
3990
3950
3910
3870
3830
3790
3750
3710
3670
3630
3590
3550
3510
3470
3430
FILL
DOME PETROLEUM
DURING
YEAR 1988

3.30

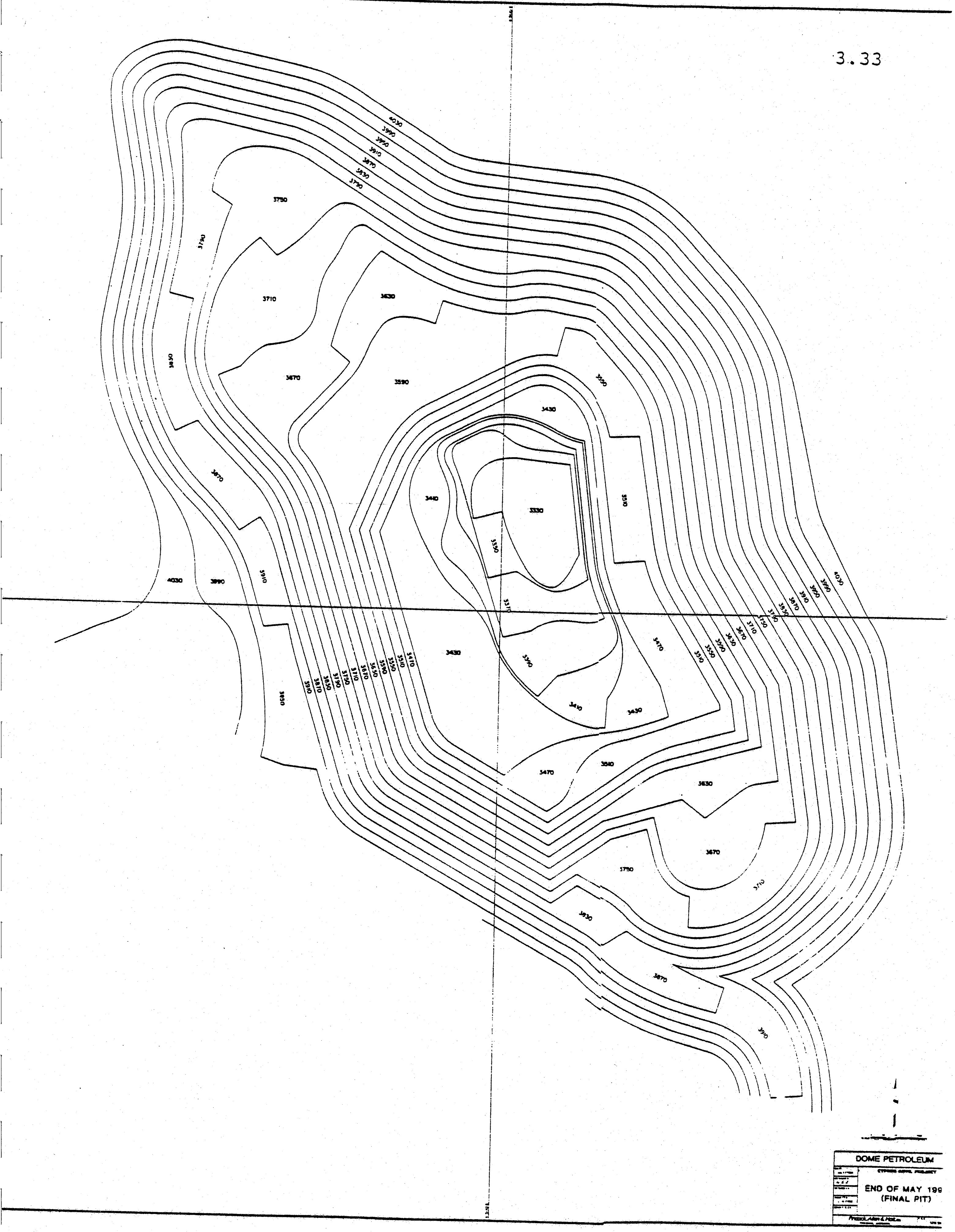

PHASES WA & YA
4030
3990
3950
3910
3870
3830
3790
3750
3710
3670
3630
3590
3550
3510
3470
3430
DOME PETROLEUM
END OF
YEAR 1988


4030
3990
3950
3910
3870
3830
3790
3750
3710
3630
3670
3590
3550
3510
3430
3470
PHASE WA
PHASE YA
DOME PETROLEUM
END OF
YEAR 1989



DOME PETROLEUM

END OF
YEAR 1990


DOME PETROLEUM
END OF MAY 199
(FINAL PIT)

3.4 Mine Operations

The following Table 3.10, portrays the mine production schedule through mine life from November 1, 1984 until the end of milling in 1991.

TABLE 3.10

PRODUCTION SCHEDULE
(000 TONNES)
CYPRUS ANVIL MINE

	Pit Ore	STPL Ore	Waste	Total
1984 (Nov-Dec)	0	427	4,355	4,782
1985	2,631	973	23,442	27,046
1986	4,073	0	22,398	26,471
1987	4,073	0	21,466	25,539
1988	4,073	0	8,233	12,306
1989	4,073	0	2,642	6,715
1990	4,073	0	1,151	5,224
1991 (Jan-Nov)	3,436	0	963	4,399
TOTAL	26,432	1,400	84,650	112,482

In general mine operations are favorably comparable to other open pit mines. Since the last labor contract negotiations, improved work practices have greatly increased productivity. Stripping volume is somewhat above budget and the pit will be well within the projected ore to waste ratio by year end.

The establishment of two twelve (12) hour shifts/day and a shift change at the equipment has improved the effective operating time to 17.5 hours in a 24-hour day. Non-effective time per shift consists of the following:

Contractual:	Lunch - 2 x 30 min. ea	=	60 min
	Work Breaks - 2 x 10 min. ea	=	20 min
Work Related:	Walk around inspection	=	10 min
			90 min (1.5 hrs)
	Time remaining 10.5 hrs		

Normal operating delays for blasting, shovel clean-up, equipment moves, foreman instruction, personal breaks and waiting time is taken as 10 minutes per hour remaining or 105 minutes per shift. Therefore, total effective minutes per shift is estimated to be 525 minutes, or 8.75 hours.

For comparison an average open pit mine operating three (3) shifts with only one lunch/shift has the same effective working time per day based on 350 effective minutes per shift.

Obviously, if in the labor contract negotiations to be held in September, 1984, the second lunch period can be reduced to a 10 minute work break or eliminated entirely, then productivity can be expected to increase.

Some problems occur in Faro mining operations, one of which is lack of communications. For several reasons, mine supervision personnel have been promoted from the rank and file equipment operators while young engineers employed out of college have been placed immediately into some engineering discipline and their progress is up through the engineering department. As a result, the engineers fail to understand equipment capabilities, mining expediencies and mine operator psychology. Conversely, the operators have little understanding of engineering processes, designs or tolerances.

As a first step to improve this situation, present management plans to use new engineers in the radio tower on the edge of the pit to minimize haulage transport delays and in turn they will get some pit operating experience. Employees to handle this job have been included in the manning schedule.

A period of time spent as a shift supervisor would add further to the operating knowledge of the young engineer.

In addition to lack of mine operating experience, the Faro engineers have little or no exposure to maintenance practices and procedure.

Another serious communications problem exists between various departments. For example, when the separate capabilities of the 120-ton WABCO and the 170-ton Euclid trucks was discussed, the operations manager was apparently unaware that good time trials had been run on the haulage fleets and any additional tests were redundant. It would have been sufficient to talk to General Electric and General Motors engineers to determine what could be done to increase the performance of the WABCO 120-ton trucks.

Some quite good engineering studies of various things, including possible dewatering of ore and the proper approach to the east side slide area, have been made and came to light only after interrogation.

It was noted that although mine haul road surfaces are in rather good condition, the state of bench and dump surfaces is substandard. Poor upkeep of loading, hauling and dumping surfaces results in a slower, more expensive operation. Rough, uneven bench surfaces leading away from a loading area cause tire damage due to the rough roads themselves and due to spill rocks from preceeding trucks which were caused to fall by the rough roads. Spill can be prevented in two ways:

1) Underload the trucks.

2) Maintain good haulage surfaces.

Obviously, the underloading of trucks is not the answer to the problem, hence it is imperative that improved road surfaces be established and maintained.

Some thought has been given to reducing the sub-grade blasting by one (1) foot to five (5) rather than six (6) feet so that shovel operators can't dig below grade. This solution is roughly equivalent to burning the barn down to get the horses out of it.

The proper answer to the problem is management's insistence that the front line supervision and the shovel operators cooperate to keep the shovels on grade. Level plates have been placed in the pit but are obviously not used correctly. The present period of reduced operation allows an opportunity to establish desirable pit surfaces before pit activity is increased.

The 2,821 hour average tire life for the Euclid trucks in recent months, as compared to an industry average of 4,000-4,500 hours, is an indication of the seriousness of the problem.

The considerable amount of water found in the ore zone continues to present problems. Attention is drawn to the April 8, 1982 report by Mr. R. Lopaschuk, "Justification of Pit Dewatering Using In-Pit Wells", in which the problem and possible solutions are clearly drawn by the author.

At the present time, carbide insert drill bits are used exclusively for blasthole drilling at Faro. Nine and 7/8-inch carbide bits cost approximately $3,000 (U.S.) as against $1,000 (U.S.) for a similar size milled-tooth bit and a milled tooth bit can be re-tipped for $60-100.00.

In hard rock, the carbide insert bit will show an economic advantage by outdrilling the softer bit by more than 3 to 1. In softer rock the carbide bit will suffer bearing failure before it loses its gauge and the mill-tooth bit will prove the better choice. It is suggested that the cheaper bit be tried in the soft rock areas to determine comparative costs.

Drill production performance was developed in the following manner:

Assume:

525 effective minutes per shift
140 feet per hour average down hole drilling
  (average of actual drill information)
5 minutes-moving and setting up between holes
  depth of hole = 46 ft

Calculation of effective drilling time:

140 ÷ 46 = 3.04 holes/hr or 19.74 min/hole
19.74 + 5.00 = 24.74 actual min/hole

$$\frac{525 \text{ min/shift}}{24.74 \text{ min/hole}} = 21.22 \text{ holes/shift}$$

21.22 x 46 ft = 976 ft/shift
976 ft/shift ÷ 140 ft/hr = 6.97 hrs (7) effective drilling time per shift

A powder factor of 0.44 kg/BCY was given for ore and 0.40 kg/BCY for waste. These numbers translate to 0.968 lbs/BCY and 0.88 lbs/BCY. Calculating a given 23 ft x 23 ft pattern on a 20 ft ore bench: 23 ft x 23 ft x 20 ft = 10,580 ft^3 or 391.9 yd^3; 391.9 x 0.968 = 379.4 lbs of explosive required with a 9-7/8 in diameter blasthole. The weight of explosives per foot of 9-7/8 in hole is about 29.2 lbs of 0.88 g/cc ANFO.

Hence: 379.4 lbs ÷ 29.2 lbs/ft = 13 ft of powder rise with a 24 ft total hole depth. The amount of stemming is 24 ft - 13 ft or 11 ft.

With 23 ft of burden on the blasthole and only 11 ft of stemming it would be expected that the hole would be overloaded and that a blast would produce considerable "flyrock". Because in actual practice this is not so, some of the basic information provided appears to be misunderstood. Local management should investigate this inconsistency.

A possibility exists that the cycle times supplied to PAH personnel were overestimated for hauls in the last years. Should the haul distances and profiles prove less arduous, haulage costs could be expected to be reduced from the PAH estimates. A properly designed mining plan for the mine life will provide more accurate data for the determination of the haul profiles.

The preventive maintenance program at Faro is favorably comparable to those of other open pit mining organizations where the following schedules exist:

Equipment	Frequency
Shovels	Once a week
Drills	Once a week
Trucks	500 hrs (essentially once a month)
Dozers & Graders	500 hrs (essentially once a month)
Tire Rotation	On inspection
Engine Oil Analysis	500 hrs @ oil change

The Maintenance Department at Faro has adopted the maintenance program recommended by the Euclid Truck Company to use for its various truck inspections and lubrications. A copy of the reporting form is attached.

CYPRUS ANVIL MINING CORPORATION
EUCLID R-170 HAUL TRUCKS
MECHANICAL P.M. & LUBESERVICE

UNIT NO. ________ DATE SCHEDULED ________ DATE STARTED ________ DATE COMPLETED ________
LUBESERVICEMEN ________ MECHANICS ________ WORK ORDER # ________ HOUR METER ________
________ ________ SUPPLY CODE ________

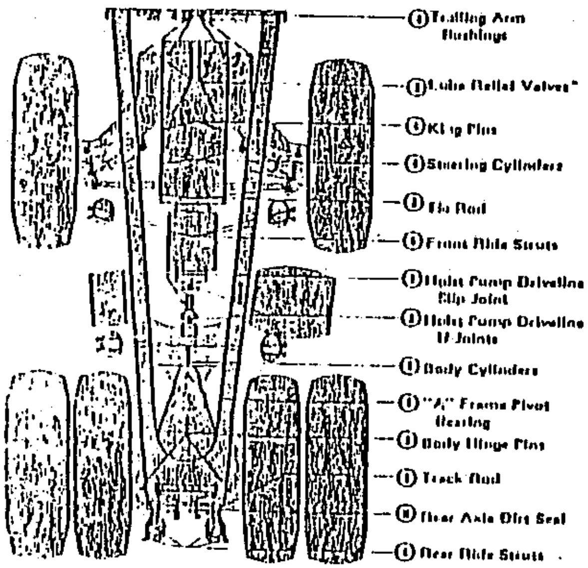
( ) INDICATES OK/SERVICED

SERVICE REQUIRED

250 HOUR LUBESERVICE	( )	250 HOUR P.M.	( )
500 HOUR LUBESERVICE	( )	500 HOUR P.M.	( )
1000 HOUR LUBESERVICE	( )	1000 HOUR P.M.	( )
3000 HOUR LUBESERVICE	( )	3000 HOUR P.M.	( )

LUBRICATION CHART:

CIRCLED NUMBERS INDICATE THE NUMBER OF FITTINGS



EL-11407B

Fig. 1 Lubrication Points

*Lubrication Relief Valves are shown for reference only. They are installed to prevent over greasing from damaging the trailing arm bushing seals. Lubricate the trailing arm bushing until grease comes out of the respective relief valve.

(R) INDICATES REPAIR COMPLETE

250 HOUR LUBESERVICE

CHANGE:
- ENGINE OIL ( )
- ENGINE OIL FILTERS (FULL FLOW & BYPASS) ( )
- FUEL FILTERS ( )
- WHEEL MOTOR OILS ( )
- WATER FILTERS ( )
- HYDRAULIC TANK FILTERS ( )
- STEERING OIL TANK FILTER ( )
- FRONT WHEEL BEARING OILS ( )

WASH/CLEAN:
- HYDRAULIC & STEERING TANK BREATHERS ( )
- FUEL TANK FILTER NECK SCREEN ( )
- WHEEL MOTOR HOUSING ( )
- WHEEL MOTOR HOUSING BREATHER & MAGNETIC PLUGS ( )

DRAIN:
- WATER/SEDIMENT AT FUEL TANK ( )
- AIR RESERVOIR ( )

CHECK/FILL:
- HYDRAULIC OIL ( )
- COOLANT LEVEL (-50 DEGREES F) ( )
- BATTERIES ( )

AIR STARTER:
- REMOVE INLET LINE & POUR ONE CUP OF DIESEL FUEL INTO AIR MOTOR. RECONNECT AIR INLET LINE & OPERATE STARTER WITH CAUTION. ( )

GREASE/OIL CAN:
- ALL POINTS (SEE DIAGRAM) ( )

500 HOUR LUBESERVICE

250 HOUR LUBESERVICE ( )

STARTER:
- REMOVE PLUGS AT HOUSING COVER AND GEARCASE. INSTALL GREASE FITTINGS. WITH HAND GREASE GUN APPLY ONE SHOT OF GREASE TO HOUSING COVER AND THREE TO GEARCASE. REINSTALL GREASE PLUGS. ( )

(X) INDICATES REPAIRS REQUIRED

1000 HOUR LUBESERVICE

250 AND 500 HOUR LUBESERVICE ( )

CHANGE:
- HYDRAULIC OIL) HAS TO SUIT BRAKE ( )
- STEERING TANK OIL) SYSTEM TEMP. WISE ( )
- HIGH PRESSURE FILTER ( )
- INNER AIR FILTERS ( )
- OUTER AIR FILTERS - IF REQUIRED ( )

STARTER:
- REMOVE BUSHING OIL PLUG FROM BENDIX, SATURATE BUSHING WITH LOW OIL AND REPLACE PLUG. ( )

MOISTURE EJECTOR VALVE:
- REMOVE, DISASSEMBLE, CLEAN AND LUBRICATE. CHANGE FILTER. ( )

3000 HOUR LUBESERVICE

250, 500 AND 1000 HOUR LUBESERVICE ( )
DRAIN/FLUSH COOLING SYSTEM ( )

250 HOUR P.M. INSPECTION

ENGINE:
CHECK OIL LEAKS - LOCATION ________ ( )
CHECK COOLANT LEAKS - LOCATION ________ ( )
CHECK FILTER HOUSING LEAKS ( )
CHECK AIR INDUCTION PIPING SECURITY ( )
CHECK AIR FILTER INDICATORS ( )
V BELTS - REPLACE/ADJUST AS REQUIRED ( )
- FAN (NO ADJUSTMENT REQUIRED) ( )
- ALTERNATOR (100 FT. LB.) ( )
- WATER PUMP (100 FT. LB.) ( )

CHECK SHUTTER OPERATION ( )
CHECK STARTER LUBRICATOR ( )
CHECK THROTTLE CONTROL ( )
TURBO CHARGERS - LEAKS ( )
- VIBRATION ( )

COMPRESSOR - MOUNTING ( )
- LEAKAGE AT LINES, ETC. ( )

3.5 Mine Equipment

Major mine operating equipment consists of:

(2) Blast Hole Drills	Both model M4 Marion Drill one electric (1979) and one diesel electric (1976).
(4) Electric Shovels (15 Cy Yd)	Three of the shovels are P&H 2100 and one machine is a Marion 191M.
(22) Haulage Trucks (120 Ton)	All these units are Wabco 120 models.
(8) Haulage Trucks (170 Ton)	All these units are Euclid R170 models.
(3) Caterpillar (D9H)	1975, 1978 & 1980 models
(1) Komatsu (D-355A)	1981 model
(1) Caterpillar (854)	1976 R.T. Dozer
(2) Front-End Loaders (10 Cy Yd)	Two units are Le Tourneau 800
(3) Caterpillar Motor Graders	Two Model 16G - Both 1980 and on Model 14G 1982.

Table 3.11 indicates that the present fleet is adequate in all areas except blasthole drills.

Thought has been given to the purchase of a used drill, to make up for the lack of capacity in 1985, 1986 and 1987 and a Marion M5, 1975, has been located. This drill, which has

TABLE 3.11

CYPRUS ANVIL

EQUIPMENT FLEET

	PRIMARY UNITS					AUXILIARY UNITS					
	Drill	Shovel	F.E.L	Truck	Truck	F.E.L.	R.T.D.	Dozer	Dozer	Road	Water
PERIOD	9 7/8	15 yd.	Stpl.	120 ton	170 ton	10 yd.		D-9	D-8	Grader	Truck
1984 (NOV-DEC)	2	3	1	4	8	1	1	2	1	3	1
1985	3	3	1	5	8	1	1	2	1	3	1
1986	3	3	1	8	8	1	1	2	1	3	1
1987	3	3	1	8	8	1	1	2	1	3	1
1988	3	3	1	8	8	1	1	2	1	3	1
1989	3	3	1	8	8	1	1	2	1	3	1
1990	3	3	1	8	8	1	1	2	1	3	1
1991 (JAN-OCT)	3	3	1	8	8	1	1	2	1	3	1

No units are taken out of the fleets unless scrap age is reached.

been offered for $500,000, <u>does not</u> have a cold weather main frame. Dismantling, freight and re-erection costs are estimated at $250,000 which brings the machine within about $500,000 of a new unit with a cold weather frame which could be used later on the Vangorda Plateau without frame problems. Only in the event of a planned cessation of mining after 1991 is a recommendation made for purchase of the used machine.

As an aid to the establishment of good haulage surfaces, a portable rock crusher is recommended should a decision be made to resume milling operations. The cost of such a unit would be in the neighborhood of $175,000-200,000.

Some difficulty exists in the joint operation of the Wabco and Euclid haulage trucks. Tests conducted on the two types of units indicate normally loaded speeds up an eight percent grade of seven plus mph for the Euclid and approximately 5.4 mph for the Wabco.

At present and for the early years of milling, little or no eight percent road grade will be encountered. By 1988 the daily required waste tonnage will have dropped to a point where the necessary use of the Wabco trucks at Faro will be sporadic at most.

Present plans call for continuation of the use of the larger haulage units for the more demanding waste haul with the 120-ton trucks being used first to haul ore from the stockpile to the crusher and then ore from the pit. These Wabco units necessary for use during 1985, 1986 and 1987 can have the 150 mm fuel injectors replaced by 180 mm injectors and increase the horsepower from 1,000 to 1,200. To do this it would also be necessary to put inner coolers in the manifolds to prevent shortened engine life. This change can be performed for approximately $10,000 per unit and should only be done for a maximum of seven trucks.

A further problem to having the Wabco and the Euclid trucks running together is the slower speed of the Wabco on a flat haul. This can easily be changed by adjustment to the speed servicing card in the 772 wheel motors. Adjustment up to 38 mph can be made if so desired.

The geometry of the Faro ore body's Zone 3 is such that little can be done to level out equipment requirements. Stripping ratios drop from 7 to 1 for part of 1985 to 5.6 to 1 in 1986 to 0.41 to 1 in 1989. Obviously, this results in progressively reduced demand for the mining machinery. If a decision is made to mine the Grum or Vangorda deposits the excess equipment can be utilized. The alternative is to sell the unnecessary units.

With the exception of the rotary drill fleet, no additional major equipment is required because of the projected short mine life of the Faro deposit and somewhat of a surplus of equipment. In the event that the Vangorda stripping is planned during the mine life of the Faro deposit, the replacement needs of used equipment at Faro should be reevaluted.

Tables 3.12A and 3.12B show substantiating calculations of required truck operating shifts.

Tables 3.12C, 3.12D, 3.12E, 3.12F, 3.12G, 3.12H and 3.12I show substantiating calculations of required drill and shovel operating productivity.

Tables 3.13 through 3.23 portray required fleet sizes of major production equipment through 1991.

TABLE 3.12 A

CYPRUS ANVIL

TRUCK HAULAGE SUMMARY
170-TON TRUCKS

	Material	Tonnes (x1000)	Cycle Time	Loads/ Shift	Tonnes/ Trshft	Reqd Op Shs
1984 (NOV-DEC)	ORE	0	0.00	0.00	0	0
	WASTE	3580	14.00	37.46	5319	673
	RE REH.	0	0.00	0.00	0	0
	TOTAL	3580			5319	673
1985	ORE	0	0.00	0.00	0	0
	WASTE	21100	14.00	37.49	5324	3963
	RE REH.	0	0.00	0.00	0	0
	TOTAL	21100			5324	3963
1986	ORE	0	0.00	0.00	0	0
	WASTE	18300	16.00	32.49	4614	3966
	RE REH.	0	0.00	0.00	0	0
	TOTAL	18300			4614	3966
1987	ORE	0	0.00	0.00	0	0
	WASTE	20000	14.00	37.50	5325	3756
	RE REH.	0	0.00	0.00	0	0
	TOTAL	20000			5325	3756
1988	ORE	4073	20.00	25.98	3689	1104
	WASTE	8233	17.00	30.50	4331	1901
	RE REH.	0	0.00	0.00	0	0
	TOTAL	12306			4095	3005
1989	ORE	4073	20.85	24.99	3548	1148
	WASTE	1800	30.00	17.49	2483	725
	RE REH.	0	0.00	0.00	0	0
	TOTAL	5873			3136	1873
1990	ORE	4073	20.85	24.99	3548	1148
	WASTE	1151	30.00	17.47	2481	464
	RE REH.	0	0.00	0.00	0	0
	TOTAL	5224			3241	1612
1991 (JAN-NOV)	ORE	3436	22.50	22.98	3263	1053
	WASTE	963	30.00	17.48	2482	388
	RE REH.	0	0.00	0.00	0	0
	TOTAL	4399			3053	1441

TABLE 3.12 B

CYPRUS ANVIL

TRUCK HAULAGE SUMMARY
120-TON TRUCKS

	Material	Tonnes (x1000)	Cycle Time	Loads/ Shift	Tonnes/ Trshft	Reqd Op Shs
1984 (NOV-DEC)	ORE	427	10.00	52.23	5405	79
	WASTE	775	14.00	37.44	3875	200
	RE REH.	0	0.00	0.00	0	0
	TOTAL	1202			4308	279
1985	ORE	3604	18.04	29.09	3011	1197
	WASTE	2342	14.00	37.46	3877	604
	RE REH.	1578	10.00	52.40	5423	291
	TOTAL	7524			3597	2092
1986	ORE	4073	25.50	20.49	2121	1920
	WASTE	4098	21.00	25.00	2507	1584
	RE REH.	2444	10.00	52.48	5431	450
	TOTAL	10615			2685	3954
1987	ORE	4073	28.00	18.49	1914	2128
	WASTE	1466	18.50	27.99	2897	506
	RE REH.	2444	10.00	52.48	5431	450
	TOTAL	7983			2589	3084
1988	ORE	0	0.00	0.00	0	0
	WASTE	0	0.00	0.00	0	0
	RE REH.	2444	10.00	52.48	5431	450
	TOTAL	2444			5431	450
1989	ORE	0	0.00	0.00	0	0
	WASTE	842	17.00	30.48	3154	267
	RE REH.	2444	10.00	52.48	5431	450
	TOTAL	3286			4583	717
1990	ORE	0	0.00	0.00	0	0
	WASTE	0	0.00	0.00	0	0
	RE REH.	2444	10.00	52.48	5431	450
	TOTAL	2444			5431	450
1991 (JAN-NOV)	ORE	0	0.00	0.00	0	0
	WASTE	0	0.00	0.00	0	0
	RE REH.	2068	10.00	52.45	5428	381
	TOTAL	2068			5428	381

TABLE 3.12 C

CYPRUS ANVIL

DRILLING EQUIPMENT PRODUCIVITY

DRILLTYPE	9 7/8 inch
MATERIAL TO DRILL	ORE
IN-SITU DENSITY OF MATERIAL	3.80 Tonne/M3

BLASTING

Bench Height	20.00 ft
Subgrade Drilling	4.00 ft
Hole Depth	24.00 ft
Hole Diameter	9.90 in.
Explosives Sp. Gr.	0.88 g/cc
Powder Factor	0.16 kg./tonne
Powder Rise	9.25 ft
Stemming Height	14.75 ft
Column Load	13.27 kg./ft
Burden	17.29 ft
Spacing	17.29 ft
Tonnes Drilled per Hole	643 tonnes

DRILLING PRODUCTIVITY

Tonnes per Foot Drilled	26.90 tonnes/ft.
Penetration Rate	74.00 ft/hr.
Down-the-Hole Drilling Time per Shift	7.00 hrs.
Feet Drilled per Operating Shift	518.00 ft
Tonnes Drilled per Operating Shift	13934 tonnes

TABLE 3.12 D

CYPRUS ANVIL

DRILLING EQUIPMENT PRODUCIVITY

DRILLTYPE	9 7/8 inch
MATERIAL TO DRILL	WASTE
IN-SITU DENSITY OF MATERIAL	2.75 Tonne/M3
BLASTING	
Bench Height	40.00 ft
Subgrade Drilling	6.00 ft
Hole Depth	46.00 ft
Hole Diameter	9.90 in.
Explosives Sp. Gr.	0.88 g/cc
Powder Factor	0.19 kg./tonne
Powder Rise	25.60 ft
Stemming Height	20.40 ft
Column Load	13.27 kg./ft
Burden	23.95 ft
Spacing	23.95 ft
Tonnes Drilled per Hole	1786 tonnes
DRILLING PRODUCTIVITY	
Tonnes per Foot Drilled	38.91 tonnes/ft.
Penetration Rate	140.00 ft/hr.
Down-the-Hole Drilling Time per Shift	7.00 hrs.
Feet Drilled per Operating Shift	980.00 ft
Tonnes Drilled per Operating Shift	38131 tonnes

TABLE 3.12 E

CYPRUS ANVIL

LOADING EQUIPMENT PRODUCTIVITY

Material: ORE

TYPE OF EQUIPMENT:

Shovel	11.47 M3
Truck	108.94 tonne

MATERIAL TO LOAD:

In-Situ Density	3.80 tonne/M3
Swell Factor	1.30
Loose Density	2.92 tonne/M3
Bench Height	6.10 m.

PRODUCTIVITY PER LOAD:

Bucket Size	11.47 M3
Bucket Fill Factor	90.00 %
Tonnes per Bucket	30.14 tonnes
Truck Size	108.94 tonnes
Theoretical Buckets per Load	3.33 buckets
Actual Buckets per Load	4 buckets
Swing Cycle Time	30.00 sec.
Spot Time between Loads	10.00 sec.
Total Time per Load	2.17 min/load

PRODUCTIVITY PER SHIFT:

Effective Minutes per Shift	525.00 min./shift
Truck Loads per Shift	241.94 loads/shift
Truck Load Factor	92.00 %
Average Truck Load	100.22 tonnes
Shovel Production per Oprtg. Shift	24247 tonnes

TABLE 3.12 F

CYPRUS ANVIL

LOADING EQUIPMENT PRODUCTIVITY

Material: WASTE

TYPE OF EQUIPMENT:

Shovel	11.47 M3
Truck	108.94 tonne

MATERIAL TO LOAD:

In-Situ Density	2.75 tonne/M3
Swell Factor	1.25
Loose Density	2.20 tonne/M3
Bench Height	12.20 m.

PRODUCTIVITY PER LOAD:

Bucket Size	11.47 M3
Bucket Fill Factor	90.00 %
Tonnes per Bucket	22.71 tonnes
Truck Size	108.94 tonnes
Theoretical Buckets per Load	4.41 buckets
Actual Buckets per Load	5 buckets
Swing Cycle Time	30.00 sec.
Spot Time between Loads	10.00 sec.
Total Time per Load	2.67 min/load

PRODUCTIVITY PER SHIFT:

Effective Minutes per Shift	525.00 min./shift
Truck Loads per Shift	196.63 loads/shift
Truck Load Factor	92.00 %
Average Truck Load	100.22 tonnes
Shovel Production per Oprtg. Shift	19706 tonnes

TABLE 3.12 G

CYPRUS ANVIL

LOADING EQUIPMENT PRODUCTIVITY

Material: ORE

TYPE OF EQUIPMENT:

Shovel	11.47 M3
Truck	154.33 tonne

MATERIAL TO LOAD:

In-Situ Density	3.80 tonne/M3
Swell Factor	1.30
Loose Density	2.92 tonne/M3
Bench Height	6.10 m.

PRODUCTIVITY PER LOAD:

Bucket Size	11.47 M3
Bucket Fill Factor	90.00 %
Tonnes per Bucket	30.14 tonnes
Truck Size	154.33 tonnes
Theoretical Buckets per Load	4.71 buckets
Actual Buckets per Load	5 buckets
Swing Cycle Time	30.00 sec.
Spot Time between Loads	10.00 sec.
Total Time per Load	2.67 min/load

PRODUCTIVITY PER SHIFT:

Effective Minutes per Shift	525.00 min./shift
Truck Loads per Shift	196.63 loads/shift
Truck Load Factor	92.00 %
Average Truck Load	141.98 tonnes
Shovel Production per Oprtg. Shift	27918 tonnes

TABLE 3.12 H

CYPRUS ANVIL

LOADING EQUIPMENT PRODUCTIVITY

Material: WASTE

TYPE OF EQUIPMENT:

Shovel	11.47 M3
Truck	154.33 tonne

MATERIAL TO LOAD:

In-Situ Density	2.75 tonne/M3
Swell Factor	1.25
Loose Density	2.20 tonne/M3
Bench Height	12.20 m.

PRODUCTIVITY PER LOAD:

Bucket Size	11.47 M3
Bucket Fill Factor	90.00 %
Tonnes per Bucket	22.71 tonnes
Truck Size	154.33 tonnes
Theoretical Buckets per Load	6.25 buckets
Actual Buckets per Load	7 buckets
Swing Cycle Time	30.00 sec.
Spot Time between Loads	10.00 sec.
Total Time per Load	3.67 min/load

PRODUCTIVITY PER SHIFT:

Effective Minutes per Shift	525.00 min./shift
Truck Loads per Shift	143.05 loads/shift
Truck Load Factor	92.00 %
Average Truck Load	141.98 tonnes
Shovel Production per Oprtg. Shift	20310 tonnes

TABLE 3.12 I

CYPRUS ANVIL

LOADING EQUIPMENT PRODUCTIVITY

Material:ORE STPL

TYPE OF EQUIPMENT:

Loader	7.65 M3
Truck	108.94 tonne

MATERIAL TO LOAD:

In-Situ Density	3.80 tonne/M3
Swell Factor	1.30
Loose Density	2.92 tonne/M3
Bench Height	0.00 m.

PRODUCTIVITY PER LOAD:

Bucket Size	7.65 M3
Bucket Fill Factor	90.00 %
Tonnes per Bucket	20.10 tonnes
Truck Size	108.94 tonnes
Theoretical Buckets per Load	4.99 buckets
Actual Buckets per Load	5 buckets
Swing Cycle Time	45.00 sec.
Spot Time between Loads	10.00 sec.
Total Time per Load	3.92 min/load

PRODUCTIVITY PER SHIFT:

Effective Minutes per Shift	525.00 min./shift
Truck Loads per Shift	133.93 loads/shift
Truck Load Factor	92.00 %
Average Truck Load	100.22 tonnes
Loader Production per Oprtg. Shift	13422 tonnes

TABLE 3.13

CYPRUS ANVIL

DRILLING EQUIPMENT REQUIREMENTS

9 7/8 inch Drill

Period	Kt.	T./ Shift	Req.d Shifts	Avail. Shifts	Mech. Avail	Units Required Oprtg.	Util.	Fleet
1984 (NOV-DEC)	4355	38202	114	120	80%	0.94	47%	2
1985	26073	32429	804	710	80%	1.13	38%	3
1986	26471	30001	880	710	79%	1.24	41%	3
1987	25539	29870	855	710	78%	1.20	40%	3
1988	12306	24224	508	710	76%	0.72	24%	3
1989	6715	18550	362	355	76%	1.02	34%	3
1990	5224	16224	322	355	75%	0.91	30%	3
1991 (JAN-OCT)	4399	16173	272	291	74%	0.93	31%	3

No drills are taken out of the fleet unless scrap age is reached.

TABLE 3.14

CYPRUS ANVIL

LOADING EQUIPMENT REQUIREMENTS

Shovel 11.5 M3

Period	Kt.	T/ Shift	Req.d Shifts	Avail. Shifts	Mech. Avail	Units Required Oprtg.	Util.	Fleet
1984 (NOV-DEC)	4355	20199	216	120	81%	1.80	60%	3
1985	26073	20590	1266	710	80%	1.78	59%	3
1986	26471	20730	1277	710	77%	1.80	60%	3
1987	25539	20812	1227	710	75%	1.73	58%	3
1988	12306	22324	551	710	75%	0.78	26%	3
1989	6715	24220	277	355	75%	0.78	26%	3
1990	5224	25789	203	355	75%	0.57	19%	3
1991 (JAN-OCT)	4399	25802	170	291	75%	0.58	19%	3

No shovels are taken out of the fleet unless scrap age is reached.

TABLE 3.15

CYPRUS ANVIL

LOADING EQUIPMENT REQUIREMENTS

Loader 7.7 M3

Period	Kt.	T/ Shift	Req.d Shifts	Avail. Shifts	Mech. Avail	Units Required		
						Oprtg.	Util.	Fleet
1984 (NOV-DEC)	427	13422	32	120	80%	0.27	27%	1
1985	2551	13422	190	710	80%	0.27	27%	1
1986	2444	13422	182	710	80%	0.26	26%	1
1987	2444	13422	182	710	80%	0.26	26%	1
1988	2444	13422	182	710	80%	0.26	26%	1
1989	2444	13422	182	355	73%	0.51	51%	1
1990	2444	13422	182	355	70%	0.51	51%	1
1991 (JAN-OCT)	2068	13422	154	291	67%	0.53	53%	1

TABLE 3.16

CYPRUS ANVIL

TRUCK REQUIREMENTS

(120 - ton Trucks)

Period	Kt.	T./ Shift	Req.d Shifts	Avail. Shifts	Mech. Avail	Units Required Oprtg.	Util.	Fleet
1984 (NOV-DEC)	1202	4308	279	121	75%	2.31	58%	4
1985	7524	3597	2092	710	75%	2.95	59%	5
1986	10615	2685	3954	710	75%	5.57	70%	8
1987	7983	2589	3084	710	73%	4.34	54%	8
1988	2444	5431	450	710	71%	0.63	8%	8
1989	3286	4583	717	355	71%	2.02	25%	8
1990	2444	5431	450	355	71%	1.27	16%	8
1991 (JAN-OCT)	2068	5428	381	291	71%	1.31	16%	8

No trucks are taken out of the fleet unless scrap age is reached.

TABLE 3.17

CYPRUS ANVIL

TRUCK REQUIREMENTS

(170 - ton Trucks)

Period	Kt.	T./ Shift	Req.d Shifts	Avail. Shifts	Mech. Avail	Units Required Oprtg.	Util.	Fleet
1984 (NOV-DEC)	3580	5319	673	121	75%	5.56	70%	8
1985	21100	5324	3963	710	75%	5.58	70%	8
1986	18300	4614	3966	710	74%	5.59	70%	8
1987	20000	5325	3756	710	71%	5.29	66%	8
1988	12306	4095	3005	710	70%	4.23	53%	8
1989	5873	3136	1873	355	70%	5.28	66%	8
1990	5224	3241	1612	355	70%	4.54	57%	8
1991 (JAN-OCT)	4399	3053	1441	291	69%	4.95	62%	8

No trucks are taken out of the fleet unless scrap age is reached

TABLE 3.18

CYPRUS ANVIL

AUXILIARY EQUIPMENT REQUIREMENTS

Front-End Loader

Period	Reqd. shifts	Avail. shifts	Mech. avail.	Units required Oprg.	Util.	Fleet
1984 (NOV-DEC)	34	120	60%	0.28	28%	1
1985	260	710	60%	0.37	37%	1
1986	210	710	60%	0.30	30%	1
1987	210	710	60%	0.30	30%	1
1988	110	710	60%	0.15	15%	1
1989	100	710	60%	0.14	14%	1
1990	100	710	60%	0.14	14%	1
1991 (JAN-OCT)	60	291	60%	0.21	21%	1

No units are taken out of the fleet unless scrap age is reached

TABLE 3.19

CYPRUS ANVIL

AUXILIARY EQUIPMENT REQUIREMENTS

Rubber-Tired Dozer

Period	Reqd. shifts	Avail. shifts	Mech. avail.	Units required Oprg.	Util.	Fleet
1984 (NOV-DEC)	62	120	65%	0.52	52%	1
1985	365	710	65%	0.51	51%	1
1986	365	710	63%	0.51	51%	1
1987	365	710	63%	0.51	51%	1
1988	300	710	62%	0.42	42%	1
1989	150	355	62%	0.42	42%	1
1990	150	355	60%	0.42	42%	1
1991 (JAN-OCT)	60	291	60%	0.21	21%	1

No units are taken out of the fleet unless scrap age is reached

TABLE 3.20

CYPRUS ANVIL

AUXILIARY EQUIPMENT REQUIREMENTS

D9 Dozer

Period	Reqd. shifts	Avail. shifts	Mech. avail.	Units required Oprg.	Util.	Fleet
1984 (NOV-DEC)	144	120	66%	1.20	60%	2
1985	850	710	66%	1.20	60%	2
1986	850	710	65%	1.20	60%	2
1987	850	710	65%	1.20	60%	2
1988	425	710	65%	0.60	30%	2
1989	100	355	65%	0.28	14%	2
1990	100	355	65%	0.28	14%	2
1991 (JAN-OCT)	50	291	65%	0.17	9%	2

No units are taken out of the fleet unless scrap age is reached

TABLE 3.21

CYPRUS ANVIL

AUXILIARY EQUIPMENT REQUIREMENTS

D8 Dozer

Period	Reqd. shifts	Avail. shifts	Mech. avail.	Units required Oprg.	Util.	Fleet
1984 (NOV-DEC)	72	120	67%	0.60	60%	1
1985	425	710	66%	0.60	60%	1
1986	425	710	65%	0.60	60%	1
1987	425	710	65%	0.60	60%	1
1988	425	710	65%	0.60	60%	1
1989	210	355	65%	0.59	59%	1
1990	210	355	65%	0.59	59%	1
1991 (JAN-OCT)	110	291	65%	0.38	38%	1

No units are taken out of the fleet unless scrap age is reached

TABLE 3.22

CYPRUS ANVIL

AUXILIARY EQUIPMENT REQUIREMENTS

Road Grader

Period	Reqd. shifts	Avail. shifts	Mech. avail.	Units required Oprg.	Util.	Fleet
1984 (NOV-DEC)	200	120	61%	1.67	56%	3
1985	1200	710	61%	1.69	56%	3
1986	1200	710	61%	1.69	56%	3
1987	1200	710	61%	1.69	56%	3
1988	600	710	61%	0.85	28%	3
1989	300	355	61%	0.85	28%	3
1990	300	355	60%	0.85	28%	3
1991 (JAN-OCT)	150	291	60%	0.52	17%	3

No units are taken out of the fleet unless scrap age is reached

TABLE 3.23

CYPRUS ANVIL

AUXILIARY EQUIPMENT REQUIREMENTS

Water Truck

Period	Reqd. shifts	Avail. shifts	Mech. avail.	Units required Oprg.	Util.	Fleet
1984 (NOV-DEC)	51	120	74%	0.43	43%	1
1985	300	710	74%	0.42	42%	1
1986	300	710	72%	0.42	42%	1
1987	300	710	72%	0.42	42%	1
1988	300	710	71%	0.42	42%	1
1989	150	355	71%	0.42	42%	1
1990	150	355	70%	0.42	42%	1
1991 (JAN-OCT)	80	291	70%	0.27	27%	1

No units are taken out of the fleet unless scrap age is reached

3.6 Manning

Following a calculation of equipment requirements a manning schedule has been prepared to project numbers of mine personnel needed on an annual basis through 1991.

Classifications of personnel have not been changed, but numbers within job class have been changed to conform to PAH computations.

It will be noted that the ratio of maintenance employees to operating personnel is somewhat higher than would normally be expected but this is due to the fact that the remoteness and extreme cold of the area makes it more difficult to hire and keep good maintenance personnel plus the fact that repairs performed in the sub-zero weather are understandably slower than in more favorable environments.

Another reason for the high ratio of maintenance employees to operators is because janitors, clerks and draftsmen and are not normally considered as effective repair personnel. Eliminating this type employee from the comparison would lower the ratio.

Table 3.24 indicates equipment shift requirements and Table 3.25 portrays the various necessary operators to man the required machines.

Tables 3.26 through 3.33 portray the salaried table of organization as well as individual salaries and cost for the period involved. It should be noted that operational geologists are included in general and administrative expense, and not in the mining cost.

Tables 3.34 through 3.41 illustrate the same information for hourly labor.

TABLE 3.24

CYPRUS ANVIL

EQUIPMENT SHIFT REQUIREMENTS

PERIOD	PRIMARY UNITS					AUXILIARY UNITS					
	Drill 9 7/8	Shovel 15 yd.	F.E.L Stpl.	Truck 120 ton	Truck 170 ton	F.E.L. 10 yd.	R.T.D.	Dozer D-9	Dozer D-8	Road Grader	Water Truck
1984 (NOV-DEC)	114	216	32	279	673	34	62	144	72	200	51
1985	804	1266	190	2092	3963	260	365	850	425	1200	300
1986	880	1277	182	3954	3966	210	365	850	425	1200	300
1987	855	1227	182	3084	3756	210	365	850	425	1200	300
1988	508	551	182	450	3005	110	300	425	425	600	300
1989	362	277	182	717	1873	100	150	100	210	300	150
1990	322	203	182	450	1612	100	150	100	210	300	150
1991 (JAN-OCT)	272	170	154	381	1441	60	60	50	110	150	80

TABLE 3.25

CYPRUS ANVIL

OPERATOR REQUIREMENTS

PERIOD	PRIMARY UNITS					AUXILIARY UNITS					
	Drill 9 7/8	Shovel 15 yd.	F.E.L Stpl.	Truck 120 ton	Truck 170 ton	F.E.L. 10 yd.	R.T.D.	Dozer D-9	Dozer D-8	Road Grader	Water Truck
1984 (NOV-DEC)	5	9	1	11	29	4	2	5	2	8	2
1985	6	9	1	14	28	4	2	5	2	8	2
1986	6	9	1	27	28	4	2	5	2	8	2
1987	6	9	1	22	26	4	2	5	2	8	2
1988	4	4	1	3	27	2	2	2	2	4	2
1989	3	2	1	5	13	2	1	1	1	2	1
1990	2	2	1	3	12	2	1	1	1	1	1
1991 (JAN-OCT)	2	2	1	3	12	2	1	1	1	1	1

TABLE 3.26

CYPRUS ANVIL

MINE SALARIED LABOR COST

1984 (NOV-DEC)

(0.17 Years)

JOB TITLE		ANNUAL COST ($/Man)	PERIOD COST ($x1000)
Operations Manager	1	50760	8.6
MINE OPERATION			
General Foreman	1	46200	7.9
Senior Mine Foreman	1	46200	7.9
Drilling&Blasting Foreman	1	46200	7.9
Shift Foreman	6	46200	47.1
Subtotal	9		70.8
MINE MAINTENANCE			
General Foreman Mech.	1	46200	7.9
Foreman Mech.	7	46200	55.0
General Foreman El.	1	46200	7.9
Foreman El.	1	46200	7.9
Design Engineer	1	46200	7.9
Senior Planning Foreman	1	46200	7.9
Subtotal	12		94.5
ENGINEERING			
Senior Mine Engineer	1	46200	7.9
Computer Appl./ Mine Eng.	2	46200	15.7
Surveyor	3	46200	23.6
Subtotal	6		47.2
TOTAL	28		221.1
OVT 5.0%			11.1
Subtotal			232.2
Fringes 30.0%			66.3
GRAND TOTAL	28		298.5

TABLE 3.27

CYPRUS ANVIL

MINE SALARIED LABOR COST

1985

(1.00 Year)

JOB TITLE		ANNUAL COST ($/Man)	PERIOD COST ($x1000)
Operations Manager	1	50760	50.8
MINE OPERATION			
General Foreman	1	46200	46.2
Senior Mine Foreman	1	46200	46.2
Drilling&Blasting Foreman	1	46200	46.2
Shift Foreman	8	46200	369.6
Subtotal	11		508.2
MINE MAINTENANCE			
General Foreman Mech.	1	46200	46.2
Foreman Mech.	7	46200	323.4
General Foreman El.	1	46200	46.2
Foreman El.	1	46200	46.2
Design Engineer	1	46200	46.2
Senior Planning Foreman	1	46200	46.2
Subtotal	12		554.4
ENGINEERING			
Senior Mine Engineer	1	46200	46.2
Computer Appl./ Mine Eng.	2	46200	92.4
Surveyor	3	46200	138.6
Subtotal	6		277.2
TOTAL	30		1390.6
OVT 5.0%			69.5
Subtotal			1460.1
Fringes 30.0%			417.2
GRAND TOTAL	30		1877.3

TABLE 3.28

CYPRUS ANVIL

MINE SALARIED LABOR COST

1986

(1.00 Year)

JOB TITLE		ANNUAL COST ($/Man)	PERIOD COST ($x1000)
Operations Manager	1	50760	50.8
MINE OPERATION			
General Foreman	1	46200	46.2
Senior Mine Foreman	1	46200	46.2
Drilling&Blasting Foreman	1	46200	46.2
Shift Foreman	8	46200	369.6
Subtotal	11		508.2
MINE MAINTENANCE			
General Foreman Mech.	1	46200	46.2
Foreman Mech.	7	46200	323.4
General Foreman El.	1	46200	46.2
Foreman El.	1	46200	46.2
Design Engineer	1	46200	46.2
Senior Planning Foreman	1	46200	46.2
Subtotal	12		554.4
ENGINEERING			
Senior Mine Engineer	1	46200	46.2
Computer Appl./ Mine Eng.	2	46200	92.4
Surveyor	3	46200	138.6
Subtotal	6		277.2
TOTAL	30		1390.6
OVT 5.0%			69.5
Subtotal			1460.1
Fringes 30.0%			417.2
GRAND TOTAL	30		1877.3

TABLE 3.29

CYPRUS ANVIL

MINE SALARIED LABOR COST

1987

(1.00 Year)

JOB TITLE		ANNUAL COST ($/Man)	PERIOD COST ($x1000)
Operations Manager	1	50760	50.8
MINE OPERATION			
General Foreman	1	46200	46.2
Senior Mine Foreman	1	46200	46.2
Drilling&Blasting Foreman	1	46200	46.2
Shift Foreman	8	46200	369.6
Subtotal	11		508.2
MINE MAINTENANCE			
General Foreman Mech.	1	46200	46.2
Foreman Mech.	7	46200	323.4
General Foreman El.	1	46200	46.2
Foreman El.	1	46200	46.2
Design Engineer	1	46200	46.2
Senior Planning Foreman	1	46200	46.2
Subtotal	12		554.4
ENGINEERING			
Senior Mine Engineer	1	46200	46.2
Computer Appl./ Mine Eng.	2	46200	92.4
Surveyor	3	46200	138.6
Subtotal	6		277.2
TOTAL	30		1390.6
OVT 5.0%			69.5
Subtotal			1460.1
Fringes 30.0%			417.2
GRAND TOTAL	30		1877.3

TABLE 3.30

CYPRUS ANVIL

MINE SALARIED LABOR COST

1988

(1.00 Year)

JOB TITLE		ANNUAL COST ($/Man)	PERIOD COST ($x1000)
Operations Manager	1	50760	50.8
MINE OPERATION			
General Foreman	1	46200	46.2
Senior Mine Foreman	1	46200	46.2
Drilling&Blasting Foreman	1	46200	46.2
Shift Foreman	8	46200	369.6
Subtotal	11		508.2
MINE MAINTENANCE			
General Foreman Mech.	1	46200	46.2
Foreman Mech.	5	46200	231.0
General Foreman El.	1	46200	46.2
Foreman El.	1	46200	46.2
Design Engineer	1	46200	46.2
Senior Planning Foreman	1	46200	46.2
Subtotal	10		462.0
ENGINEERING			
Senior Mine Engineer	1	46200	46.2
Computer Appl./ Mine Eng.	2	46200	92.4
Surveyor	3	46200	138.6
Subtotal	6		277.2
TOTAL	28		1298.2
OVT 5.0%			64.9
Subtotal			1363.1
Fringes 30.0%			389.5
GRAND TOTAL	28		1752.6

TABLE 3.31

CYPRUS ANVIL

MINE SALARIED LABOR COST

1989

(1.00 Year)

JOB TITLE		ANNUAL COST ($/Man)	PERIOD COST ($x1000)
Operations Manager	1	50760	50.8
MINE OPERATION			
General Foreman	1	46200	46.2
Senior Mine Foreman	1	46200	46.2
Drilling&Blasting Foreman	1	46200	46.2
Shift Foreman	2	46200	92.4
Subtotal	5		231.0
MINE MAINTENANCE			
General Foreman Mech.	1	46200	46.2
Foreman Mech.	4	46200	184.8
General Foreman El.	1	46200	46.2
Foreman El.	1	46200	46.2
Design Engineer	1	46200	46.2
Senior Planning Foreman	1	46200	46.2
Subtotal	9		415.8
ENGINEERING			
Senior Mine Engineer	1	46200	46.2
Computer Appl./ Mine Eng.	1	46200	46.2
Surveyor	2	46200	92.4
Subtotal	4		184.8
TOTAL	19		882.4
OVT 5.0%			44.1
Subtotal			926.5
Fringes 30.0%			264.7
GRAND TOTAL	19		1191.2

TABLE 3. 32

CYPRUS ANVIL

MINE SALARIED LABOR COST

1990

(1.00 Year)

JOB TITLE		ANNUAL COST ($/Man)	PERIOD COST ($x1000)
Operations Manager	1	50760	50.8
MINE OPERATION			
General Foreman	1	46200	46.2
Drilling&Blasting Foreman	1	46200	46.2
Shift Foreman	2	46200	92.4
Subtotal	4		184.8
MINE MAINTENANCE			
General Foreman Mech.	1	46200	46.2
Foreman Mech.	4	46200	184.8
General Foreman El.	1	46200	46.2
Foreman El.	1	46200	46.2
Design Engineer	1	46200	46.2
Senior Planning Foreman	1	46200	46.2
Subtotal	9		415.8
ENGINEERING			
Senior Mine Engineer	1	46200	46.2
Computer Appl./ Mine Eng.	1	46200	46.2
Surveyor	2	46200	92.4
Subtotal	4		184.8
TOTAL	18		836.2
OVT 5.0%			41.8
Subtotal			878.0
Fringes 30.0%			250.9
GRAND TOTAL	18		1128.9

TABLE 3.33

CYPRUS ANVIL

MINE SALARIED LABOR COST

1991 (JAN-OCT)

(0.82 Years)

JOB TITLE		ANNUAL COST ($/Man)	PERIOD COST ($x1000)
Operations Manager	1	50760	41.6
MINE OPERATION			
General Foreman	1	46200	37.9
Drilling&Blasting Foreman	1	46200	37.9
Shift Foreman	2	46200	75.8
Subtotal	4		151.6
MINE MAINTENANCE			
General Foreman Mech.	1	46200	37.9
Foreman Mech.	3	46200	113.7
General Foreman El.	1	46200	37.9
Foreman El.	1	46200	37.9
Senior Planning Foreman	1	46200	37.9
Subtotal	7		265.3
ENGINEERING			
Senior Mine Engineer	1	46200	37.9
Computer Appl./ Mine Eng.	1	46200	37.9
Surveyor	2	46200	75.8
Subtotal	4		151.6
TOTAL	16		610.1
OVT 5.0%			30.5
Subtotal			640.6
Fringes 30.0%			183.0
GRAND TOTAL	16		823.6

TABLE 3.34

CYPRUS ANVIL

MINE OPERATING AND MAINTENANCE HOURLY LABOR COST

1984 (NOV-DEC)

(0.17 Years)

JOB TITLE		PAY RATE $/hr	ANNUAL COST ($/Man)	PERIOD COST ($x1000)
MINE OPERATION				
Drill Operator	5	15.55	36946	31.4
Blaster	2	16.27	38657	13.1
Blaster Helper	3	13.87	32955	16.8
Shovel Operator	9	16.75	39798	60.9
F.E.L. Operator	1	16.03	38087	6.5
120 Ton truck driver	11	15.31	36376	68.0
170 Ton truck driver	29	15.31	36376	179.3
F.E.L. Operator	4	16.03	38087	25.9
R.T.D.	2	16.03	38087	12.9
Dozer D-9	5	16.03	38087	32.4
Dozer D-8	2	16.03	38087	12.9
Road Grader	8	16.03	38087	51.8
Water Truck	2	16.03	38087	12.9
Utilityman	15	13.39	31814	81.1
Subtotal	98			605.9
MINE MAINTENANCE				
H.D. Mechanic	49	17.47	41508	345.8
Automotive Mechanic	3	18.59	44169	22.5
Welder	16	17.23	40938	111.4
Machinist	3	17.47	41508	21.2
Lineman	1	17.47	41508	7.1
Electrician	6	17.47	41508	42.3
Electronic repair	1	17.95	42649	7.3
Crane Operator	3	15.79	37517	19.1
Utilityman	8	13.39	31814	43.3
Janitor	4	12.91	30674	20.9
Stat. Clerk	2	16.00	38016	12.9
Analysis Clerk	1	16.00	38016	6.5
Draftsman	1	16.00	38016	6.5
Subtotal	98			666.8
TOTAL	196			1272.7
Fringes 30 % on 2190 hours				351.9
GRAND TOTAL	196			1624.6

TABLE 3.35

CYPRUS ANVIL

MINE OPERATING AND MAINTENANCE HOURLY LABOR COST

1985

(1.00 Year)

JOB TITLE		PAY RATE $/hr	ANNUAL COST ($/Man)	PERIOD COST ($x1000)
MINE OPERATION				
Drill Operator	6	15.55	36946	221.7
Blaster	2	16.27	38657	77.3
Blaster Helper	3	13.87	32955	98.9
Shovel Operator	9	16.75	39798	358.2
F.E.L. Operator	1	16.03	38087	38.1
120 Ton truck driver	14	15.31	36376	509.3
170 Ton truck driver	28	15.31	36376	1018.5
F.E.L. Operator	4	16.03	38087	152.3
R.T.D.	2	16.03	38087	76.2
Dozer D-9	5	16.03	38087	190.4
Dozer D-8	2	16.03	38087	76.2
Road Grader	8	16.03	38087	304.7
Water Truck	2	16.03	38087	76.2
Utilityman	15	13.39	31814	477.2
Subtotal	101			3675.2
MINE MAINTENANCE				
H.D. Mechanic	49	17.47	41508	2033.9
Automotive Mechanic	3	18.59	44169	132.5
Welder	16	17.23	40938	655.0
Machinist	3	17.47	41508	124.5
Lineman	1	17.47	41508	41.5
Electrician	8	17.47	41508	332.1
Electronic repair	1	17.95	42649	42.6
Crane Operator	3	15.79	37517	112.6
Utilityman	8	13.39	31814	254.5
Janitor	4	12.91	30674	122.7
Stat. Clerk	2	16.00	38016	76.0
Analysis Clerk	1	16.00	38016	38.0
Draftsman	1	16.00	38016	38.0
Subtotal	100			4003.9
TOTAL	201			7679.1
Fringes 30 % on 2190 hours				2123.4
GRAND TOTAL	201			9802.5

TABLE 3.36

CYPRUS ANVIL

MINE OPERATING AND MAINTENANCE HOURLY LABOR COST

1986

(1.00 Year)

JOB TITLE		PAY RATE $/hr	ANNUAL COST ($/Man)	PERIOD COST ($x1000)
MINE OPERATION				
Drill Operator	6	15.55	36946	221.7
Blaster	2	16.27	38657	77.3
Blaster Helper	3	13.87	32955	98.9
Shovel Operator	9	16.75	39798	358.2
F.E.L. Operator	1	16.03	38087	38.1
120 Ton truck driver	27	15.31	36376	982.2
170 Ton truck driver	28	15.31	36376	1018.5
F.E.L. Operator	4	16.03	38087	152.3
R.T.D.	2	16.03	38087	76.2
Dozer D-9	5	16.03	38087	190.4
Dozer D-8	2	16.03	38087	76.2
Road Grader	8	16.03	38087	304.7
Water Truck	2	16.03	38087	76.2
Utilityman	15	13.39	31814	477.2
Subtotal	114			4148.1
MINE MAINTENANCE				
H.D. Mechanic	60	17.47	41508	2490.5
Automotive Mechanic	3	18.59	44169	132.5
Welder	21	17.23	40938	859.7
Machinist	3	17.47	41508	124.5
Lineman	1	17.47	41508	41.5
Electrician	10	17.47	41508	415.1
Electronic repair	1	17.95	42649	42.6
Crane Operator	3	15.79	37517	112.6
Utilityman	8	13.39	31814	254.5
Janitor	4	12.91	30674	122.7
Stat. Clerk	2	16.00	38016	76.0
Analysis Clerk	1	16.00	38016	38.0
Draftsman	1	16.00	38016	38.0
Subtotal	118			4748.2
TOTAL	232			8896.3
Fringes 30 % on 2190 hours				2460.0
GRAND TOTAL	232			11356.3

TABLE 3.37

CYPRUS ANVIL

MINE OPERATING AND MAINTENANCE HOURLY LABOR COST

1987

(1.00 Year)

JOB TITLE		PAY RATE $/hr	ANNUAL COST ($/Man)	PERIOD COST ($x1000)
MINE OPERATION				
Drill Operator	6	15.55	36946	221.7
Blaster	2	16.27	38657	77.3
Blaster Helper	3	13.87	32955	98.9
Shovel Operator	9	16.75	39798	358.2
F.E.L. Operator	1	16.03	38087	38.1
120 Ton truck driver	22	15.31	36376	800.3
170 Ton truck driver	26	15.31	36376	945.8
F.E.L. Operator	4	16.03	38087	152.3
R.T.D.	2	16.03	38087	76.2
Dozer D-9	5	16.03	38087	190.4
Dozer D-8	2	16.03	38087	76.2
Road Grader	8	16.03	38087	304.7
Water Truck	2	16.03	38087	76.2
Utilityman	15	13.39	31814	477.2
Subtotal	107			3893.5
MINE MAINTENANCE				
H.D. Mechanic	63	17.47	41508	2615.0
Automotive Mechanic	3	18.59	44169	132.5
Welder	21	17.23	40938	859.7
Machinist	3	17.47	41508	124.5
Lineman	1	17.47	41508	41.5
Electrician	10	17.47	41508	415.1
Electronic repair	1	17.95	42649	42.6
Crane Operator	3	15.79	37517	112.6
Utilityman	8	13.39	31814	254.5
Janitor	4	12.91	30674	122.7
Stat. Clerk	2	16.00	38016	76.0
Analysis Clerk	1	16.00	38016	38.0
Draftsman	1	16.00	38016	38.0
Subtotal	121			4872.7
TOTAL	228			8766.2
Fringes 30 % on 2190 hours				2424.0
GRAND TOTAL	228			11190.2

TABLE 3.38

CYPRUS ANVIL

MINE OPERATING AND MAINTENANCE HOURLY LABOR COST

1988

(1.00 Year)

JOB TITLE		PAY RATE $/hr	ANNUAL COST ($/Man)	PERIOD COST ($x1000)
MINE OPERATION				
Drill Operator	4	15.55	36946	147.8
Blaster	2	16.27	38657	77.3
Blaster Helper	2	13.87	32955	65.9
Shovel Operator	4	16.75	39798	159.2
F.E.L. Operator	1	16.03	38087	38.1
120 Ton truck driver	3	15.31	36376	109.1
170 Ton truck driver	27	15.31	36376	982.2
F.E.L. Operator	2	16.03	38087	76.2
R.T.D.	2	16.03	38087	76.2
Dozer D-9	2	16.03	38087	76.2
Dozer D-8	2	16.03	38087	76.2
Road Grader	4	16.03	38087	152.3
Water Truck	2	16.03	38087	76.2
Utilityman	10	13.39	31814	318.1
Subtotal	67			2431.0
MINE MAINTENANCE				
H.D. Mechanic	41	17.47	41508	1701.9
Automotive Mechanic	2	18.59	44169	88.3
Welder	15	17.23	40938	614.1
Machinist	3	17.47	41508	124.5
Lineman	1	17.47	41508	41.5
Electrician	7	17.47	41508	290.6
Electronic repair	1	17.95	42649	42.6
Crane Operator	2	15.79	37517	75.0
Utilityman	5	13.39	31814	159.1
Janitor	4	12.91	30674	122.7
Stat. Clerk	1	16.00	38016	38.0
Analysis Clerk	1	16.00	38016	38.0
Draftsman	1	16.00	38016	38.0
Subtotal	84			3374.3
TOTAL	151			5805.3
Fringes 30 % on 2190 hours				1605.3
GRAND TOTAL	151			7410.6

TABLE 3.39

CYPRUS ANVIL

MINE OPERATING AND MAINTENANCE HOURLY LABOR COST

1989

(1.00 Year)

JOB TITLE		PAY RATE $/hr	ANNUAL COST ($/Man)	PERIOD COST ($x1000)
MINE OPERATION				
Drill Operator	3	15.55	36946	110.8
Blaster	2	16.27	38657	77.3
Blaster Helper	1	13.87	32955	33.0
Shovel Operator	2	16.75	39798	79.6
F.E.L. Operator	1	16.03	38087	38.1
120 Ton truck driver	5	15.31	36376	181.9
170 Ton truck driver	13	15.31	36376	472.9
F.E.L. Operator	2	16.03	38087	76.2
R.T.D.	1	16.03	38087	38.1
Dozer D-9	1	16.03	38087	38.1
Dozer D-8	1	16.03	38087	38.1
Road Grader	2	16.03	38087	76.2
Water Truck	1	16.03	38087	38.1
Utilityman	10	13.39	31814	318.1
Subtotal	45			1616.5
MINE MAINTENANCE				
H.D. Mechanic	24	17.47	41508	996.2
Automotive Mechanic	2	18.59	44169	88.3
Welder	10	17.23	40938	409.4
Machinist	3	17.47	41508	124.5
Lineman	1	17.47	41508	41.5
Electrician	4	17.47	41508	166.0
Electronic repair	1	17.95	42649	42.6
Crane Operator	2	15.79	37517	75.0
Utilityman	5	13.39	31814	159.1
Janitor	2	12.91	30674	61.3
Stat. Clerk	1	16.00	38016	38.0
Analysis Clerk	1	16.00	38016	38.0
Draftsman	1	16.00	38016	38.0
Subtotal	57			2277.9
TOTAL	102			3894.4
Fringes 30 % on 2190 hours				1076.9
GRAND TOTAL	102			4971.3

TABLE 3.40

CYPRUS ANVIL

MINE OPERATING AND MAINTENANCE HOURLY LABOR COST

1990

(1.00 Year)

JOB TITLE		PAY RATE $/hr	ANNUAL COST ($/Man)	PERIOD COST ($x1000)
MINE OPERATION				
Drill Operator	2	15.55	36946	73.9
Blaster	2	16.27	38657	77.3
Blaster Helper	1	13.87	32955	33.0
Shovel Operator	2	16.75	39798	79.6
F.E.L. Operator	1	16.03	38087	38.1
120 Ton truck driver	3	15.31	36376	109.1
170 Ton truck driver	12	15.31	36376	436.5
F.E.L. Operator	2	16.03	38087	76.2
R.T.D.	1	16.03	38087	38.1
Dozer D-9	1	16.03	38087	38.1
Dozer D-8	1	16.03	38087	38.1
Road Grader	2	16.03	38087	76.2
Water Truck	1	16.03	38087	38.1
Utilityman	8	13.39	31814	254.5
Subtotal	39			1406.8
MINE MAINTENANCE				
H.D. Mechanic	22	17.47	41508	913.2
Automotive Mechanic	2	18.59	44169	88.3
Welder	9	17.23	40938	368.4
Machinist	3	17.47	41508	124.5
Lineman	1	17.47	41508	41.5
Electrician	4	17.47	41508	166.0
Electronic repair	1	17.95	42649	42.6
Crane Operator	2	15.79	37517	75.0
Utilityman	4	13.39	31814	127.3
Janitor	2	12.91	30674	61.3
Stat. Clerk	1	16.00	38016	38.0
Analysis Clerk	1	16.00	38016	38.0
Draftsman	1	16.00	38016	38.0
Subtotal	53			2122.1
TOTAL	92			3528.9
Fringes 30 % on 2190 hours				975.8
GRAND TOTAL	92			4504.7

TABLE 3.41

CYPRUS ANVIL

MINE OPERATING AND MAINTENANCE HOURLY LABOR COST

1991 (JAN-OCT)

(0.82 Years)

JOB TITLE		PAY RATE $/hr	ANNUAL COST ($/Man)	PERIOD COST ($x1000)
MINE OPERATION				
Drill Operator	2	15.55	36946	60.6
Blaster	2	16.27	38657	63.4
Blaster Helper	1	13.87	32955	27.0
Shovel Operator	2	16.75	39798	65.3
F.E.L. Operator	1	16.03	38087	31.2
120 Ton truck driver	3	15.31	36376	89.5
170 Ton truck driver	12	15.31	36376	357.9
F.E.L. Operator	2	16.03	38087	62.5
R.T.D.	1	16.03	38087	31.2
Dozer D-9	1	16.03	38087	31.2
Dozer D-8	1	16.03	38087	31.2
Road Grader	1	16.03	38087	31.2
Water Truck	1	16.03	38087	31.2
Utilityman	7	13.39	31814	182.6
Subtotal	37			1096.0
MINE MAINTENANCE				
H.D. Mechanic	19	17.47	41508	646.7
Automotive Mechanic	2	18.59	44169	72.4
Welder	7	17.23	40938	235.0
Machinist	3	17.47	41508	102.1
Lineman	1	17.47	41508	34.0
Electrician	4	17.47	41508	136.1
Electronic repair	1	17.95	42649	35.0
Crane Operator	2	15.79	37517	61.5
Utilityman	4	13.39	31814	104.4
Janitor	2	12.91	30674	50.3
Stat. Clerk	1	16.00	38016	31.2
Subtotal	46			1508.7
TOTAL	83			2604.7
Fringes 30 % on 2190 hours				720.2
GRAND TOTAL	83			3324.9

3.7 Costs

Mining costs have been classified to represent costs of a typical open pit mine of a size similar to the Cyprus Anvil property. Functional costs include the following:

1) Drilling, loading, haulage and auxiliary equipment
   - Operating labor
   - Maintenance labor
   - Parts and consumables (includes power & fuel)
2) Blasting
   - Operating labor
   - Parts and consumables (primarily explosives)
3) General Mine
   - Any operating labor not chargeable to the five primary functions
   - Parts and consumables
4) General Maintenance
   - Any maintenance labor not chargeable to the five primary functions
   - Parts and consumables
5) General Administration
   - All salaried personnel
   - Fringe benefits for salaried and day's pay personnel

A direct comparison with CAMC mining cost is somewhat difficult because CAMC allocation of maintenance (mechanical) costs are primarily made as a general support function rather than being charged directly to a specific mine activity. For example, welding is listed as welding only; planning, design and training are listed as mechanical labor, but would be classified as general maintenance with the PAH breakdown; pit mechanical could be charged to any of the operation functions. It is suggested that when mine and mill operations are started again at Faro that mine accounting be revised to provide improved information for management and operational personnel rather than for the convenience of the Accounting Department. Proper mining and milling functional control, to be most effective, must have detailed cost presentation. The work order and other reporting systems at Faro are quite good, but lumping of accounts masks the actual functional costs.

All costs developed by PAH contain the basic information supplied by CAMC, for example:

Labor cost - at actual

Fuel - $0.41/liter

Explosives - from budget information

Power - base case $0.07/kwh; higher cost case $0.10/kwh

Thirty percent fringe benefit

Conversion factors metric tonnes per bank cubic yard:

Waste 2.10 - Ore 2.90

Haul Cycles - From CAMC information

Powder Factors - 0.44 kg/BCY for ore

0.40 kg/BCY for waste

The PAH costs were developed by the determination of equipment standard productivity followed by the use of CAMC base information. It should be noted that PAH drilling and blasting costs in 1984 and early in 1985 benefit from the inclusion of stockpile ore tonnage into the divisor. Both of these cost centers show more than straight line increases between 1984-1985 and 1986.

Table 3.42 shows the mine operating costs estimated by PAH for future years at Anvil. Costs are shown by function. CAMC has estimated that reductions in fuel, explosives, and tire costs can be achieved by backhauling these items on the trucks used to transport concentrates to the coast. CAMC's estimates of savings were based on backhaul from Skagway. Very recent information indicates that the Alaskan government will not permit trucking of concentrates to Skagway at least in 1985. Backhaul from Haines, Alaska could result in some savings over the old combination rail and truck haulage. To allow for such possible savings PAH has used a ten percent reduction of delivered cost of explosives, a seven percent reduction of delivered cost of fuel, and a reduction of delivered costs of tires of $300 per tire, the last applying to 1986 and subsequent years. PAH has used the present contract delivered price of tires for 1984 and 1985. The costs in Table 3.42 also assume a \$0.07/kwh power cost.

Table 3.43 shows PAH's mine operating costs assuming there is no truck backhaul of supplies, and using the previously assumed (by CAMC) power cost of \$0.10/kwh. Costs are higher by a few cents per tonne of rock mined.

The costs in Tables 3.42 and 3.43 include some changes from previous CAMC budgets. PAH has added some additional mine supervision and has included survey personnel in the mining cost which were previously carried in general and administrative cost.

For costing purposes we have used 10.5 equipment operating hours per 12-hour shift, which takes into account normal operating delays. Effective productive time is 8.75 hours (525 minutes) per 12-hour shift.

The costs in Tables 3.42 and 3.43 both include the planned reduction of the annual mine operating period to 355 rather than 365 days plus the rehandle cost for the 10 days of holiday when the mill would require the normal daily feed. All rehandle costs reflect a front-end loader operation. The costs in both tables also include the effects of a single shift operation during the last three

TABLE 3.42

CYPRUS ANVIL

OPERATING COST SUMMARY BY FUNCTION
Reduced costs on power, fuel, explosives and tires
Canadian dollars

UNIT COST ($/Total tonne)

PERIOD	TOTAL Kt.	ORE Kt.	Drlng.*	Blstg.*	Loadg.	Haulg.	Aux. Equip.	Genl. Mine	Genl. Mntce.	Genl. Admin	Total	TOTAL COST ($x1000)	COST/ TONNE ORE
1984 (NOV-DEC)	4782	427	0.0360	0.1418	0.1230	0.3064	0.1201	0.0490	0.0743	0.1360	0.9866	4718	11.05
1985	27046	3604	0.0386	0.1476	0.1286	0.3383	0.1291	0.0496	0.0734	0.1479	1.0531	28482	7.90
1986	26471	4073	0.0494	0.1518	0.1370	0.4656	0.1409	0.0500	0.0730	0.1639	1.2316	32602	8.00
1987	25539	4073	0.0571	0.1519	0.1439	0.4498	0.1461	0.0507	0.0761	0.1684	1.2440	31771	7.80
1988	12306	4073	0.0717	0.1526	0.1655	0.5587	0.1901	0.0578	0.1177	0.2729	1.5870	19530	4.79
1989	6715	4073	0.0944	0.1509	0.1818	0.7094	0.1766	0.0794	0.1428	0.3378	1.8731	12578	3.09
1990	5224	4073	0.1051	0.1515	0.1981	0.7446	0.2325	0.0807	0.1828	0.4029	2.0982	10961	2.69
1991 (JAN-OCT)	4399	3436	0.1097	0.1510	0.1988	0.7787	0.1582	0.0735	0.1270	0.3509	1.9478	8568	2.49
TOTAL	112482	27832	0.0581	0.1504	0.1470	0.4746	0.1508	0.0550	0.0901	0.2006	1.3265	149210	5.36

Distribution within Total:

	Drlng.*	Blstg.*	Loadg.	Haulg.	Aux. Equip.	Genl. Mine	Genl. Mntce.	Genl. Admin	Total
Operating Labor	0.0097	0.0090	0.0158	0.0734	0.0368	0.0230	0.0000	0.0000	0.1677
Maintenance Labor	0.0126	0.0000	0.0364	0.0950	0.0334	0.0000	0.0321	0.0000	0.2095
Parts & Consumables	0.0357	0.1414	0.0947	0.3061	0.0806	0.0320	0.0580	0.0000	0.7485
Mine Overhead	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.2006	0.2006

* No costs for stockpile ore

TABLE 3.43

CYPRUS ANVIL

OPERATING COST SUMMARY BY FUNCTION

Canadian dollars

PERIOD	TOTAL Kt.	ORE Kt.	UNIT COST ($/Total tonne) Drlng.*	Blstg.*	Loadg.	Haulg.	Aux. Equip.	Genl. Mine	Genl. Mntce.	Genl. Admin	Total	TOTAL COST ($x1000)	COST/ TONNE ORE
1984 (NOV-DEC)	4782	427	0.0370	0.1568	0.1310	0.3128	0.1217	0.0490	0.0745	0.1360	1.0188	4872	11.41
1985	27046	3604	0.0398	0.1633	0.1369	0.3452	0.1303	0.0496	0.0735	0.1479	1.0865	29385	8.15
1986	26471	4073	0.0507	0.1680	0.1456	0.4742	0.1425	0.0500	0.0732	0.1639	1.2681	33568	8.24
1987	25539	4073	0.0584	0.1680	0.1524	0.4575	0.1477	0.0507	0.0763	0.1684	1.2794	32675	8.02
1988	12306	4073	0.0733	0.1682	0.1737	0.5681	0.1921	0.0578	0.1179	0.2729	1.6240	19985	4.91
1989	6715	4073	0.0966	0.1658	0.1898	0.7217	0.1785	0.0794	0.1430	0.3378	1.9126	12843	3.15
1990	5224	4073	0.1075	0.1660	0.2057	0.7574	0.2350	0.0807	0.1831	0.4029	2.1383	11170	2.74
1991 (JAN-OCT)	4399	3436	0.1121	0.1655	0.2065	0.7920	0.1598	0.0735	0.1273	0.3509	1.9876	8743	2.54
TOTAL	112482	27832	0.0595	0.1661	0.1553	0.4832	0.1524	0.0550	0.0903	0.2006	1.3624	153241	5.51

Distribution within Total:

	Drlng.*	Blstg.*	Loadg.	Haulg.	Aux. Equip.	Genl. Mine	Genl. Mntce.	Genl. Admin	Total
Operating Labor	0.0097	0.0090	0.0158	0.0734	0.0368	0.0230	0.0000	0.0000	0.1677
Maintenance Labor	0.0126	0.0000	0.0364	0.0949	0.0333	0.0000	0.0323	0.0000	0.2095
Parts & Consumables	0.0372	0.1571	0.1030	0.3148	0.0822	0.0320	0.0580	0.0000	0.7843
Mine Overhead	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.2006	0.2006

* No costs for Stockpile Ore

years of mine life. The projected rise in cost through the seven years reflects the effect of increased haul distances, deterioration of equipment and disproportionate overhead costs as total tonnage drops after 1987.

In summary, the mining costs at CAMC are within acceptable limits for good mining cost control. With labor costs representing approximately 41 percent of the total, it is obvious that the September labor negotiations are another area for cost improvement. Another possible area of cost improvement is tire life. As pointed out above, tire life for the Euclid trucks has been averaging about 2,800 hours, PAH has assumed this will be improved to 3,500 hours. Further improvement to 4,000 to 4,500 hours would obviously have a beneficial effect on costs. PAH has used a 170 ton truck fuel consumption rate of 85 liters/hr in 1984 and 1985 and raised this to 100 liters therafter to reflect the increasing haulage distances and lifts. It is likely that the drill penetration rate may improve in later years when anticipated higher percentages of softer rock are encountered. Drill operating costs were calculated using a weighted average of the costs for one diesel powered drill and two electric drills. Blasting costs will also improve as softer rock allows a spread of the blasting pattern and explosive prices are reduced by a recently negotiated contract.

Tables 3.44 through 3.88 show backup calculations for the costs in Table 3.42.

Table No.	Contents
3.44	Mine Overhead Cost/Tonne
3.45-3.52	Cost/Total Tonne By Function
3.53-3.60	Operating Labor Cost/Tonne
3.61-3.68	Maintenance Labor Cost/Tonne
3.69-3.76	Repair Parts & Consumables Cost/Tonne
3.77-3.88	Equipment Operating Cost/Shift

TABLE 3.44

CYPRUS ANVIL

MINE OVERHEAD COST/Tonne

PERIOD	SALARIED Salary	SALARIED Fringes	HOURLY Fringes	TOTAL ($x1000)	TOTAL Kt.	Cost/ Tonne
1984 (NOV-DEC)	221.1	77.4	351.9	650.4	4782	0.1360
1985	1390.6	486.7	2123.4	4000.7	27046	0.1479
1986	1390.6	486.7	2460.0	4337.3	26471	0.1639
1987	1390.6	486.7	2424.0	4301.3	25539	0.1684
1988	1298.2	454.4	1605.3	3357.9	12306	0.2729
1989	882.4	308.8	1076.9	2268.1	6715	0.3378
1990	836.2	292.7	975.8	2104.7	5224	0.4029
1991 (JAN-OCT)	610.1	213.5	720.2	1543.8	4399	0.3509
TOTAL	8019.8	2806.9	11737.5	22564.2	112482	0.2006

TABLE 3.45

CYPRUS ANVIL

COST PER TOTAL TONNE BY FUNCTION
1984 (NOV-DEC)

FUNCTION	OPERATING LABOR	MAINT. LABOR	PARTS & CONSUMABLES	MINE OVERHEAD	TOTAL
DRILLING	0.0066	0.0075	0.0219	0.0000	0.0360
BLASTING	0.0063	0.0000	0.1355	0.0000	0.1418
LOADING	0.0141	0.0295	0.0794	0.0000	0.1230
HAULING	0.0517	0.0604	0.1943	0.0000	0.3064
AUXILIARY	0.0311	0.0257	0.0633	0.0000	0.1201
GENERAL MINE	0.0170	0.0000	0.0320	0.0000	0.0490
GEN. MAINTENANCE	0.0000	0.0163	0.0580	0.0000	0.0743
GEN. ADMINISTRTN	0.0000	0.0000	0.0000	0.1360	0.1360
TOTAL	0.1268	0.1394	0.5844	0.1360	0.9866

TABLE 3.46

CYPRUS ANVIL

COST PER TOTAL TONNE BY FUNCTION
1985

FUNCTION	OPERATING LABOR	MAINT. LABOR	PARTS & CONSUMABLES	MINE OVERHEAD	TOTAL
DRILLING	0.0082	0.0070	0.0234	0.0000	0.0386
BLASTING	0.0065	0.0000	0.1411	0.0000	0.1476
LOADING	0.0146	0.0309	0.0831	0.0000	0.1286
HAULING	0.0565	0.0669	0.2149	0.0000	0.3383
AUXILIARY	0.0323	0.0280	0.0688	0.0000	0.1291
GENERAL MINE	0.0176	0.0000	0.0320	0.0000	0.0496
GEN. MAINTENANCE	0.0000	0.0154	0.0580	0.0000	0.0734
GEN. ADMINISTRTN	0.0000	0.0000	0.0000	0.1479	0.1479
TOTAL	0.1357	0.1482	0.6213	0.1479	1.0531

TABLE 3.47

CYPRUS ANVIL

COST PER TOTAL TONNE BY FUNCTION
1986

FUNCTION	OPERATING LABOR	MAINT. LABOR	PARTS & CONSUMABLES	MINE OVERHEAD	TOTAL
DRILLING	0.0084	0.0104	0.0306	0.0000	0.0494
BLASTING	0.0067	0.0000	0.1451	0.0000	0.1518
LOADING	0.0149	0.0334	0.0887	0.0000	0.1370
HAULING	0.0756	0.0886	0.3014	0.0000	0.4656
AUXILIARY	0.0332	0.0319	0.0758	0.0000	0.1409
GENERAL MINE	0.0180	0.0000	0.0320	0.0000	0.0500
GEN. MAINTENANCE	0.0000	0.0150	0.0580	0.0000	0.0730
GEN. ADMINISTRTN	0.0000	0.0000	0.0000	0.1639	0.1639
TOTAL	0.1568	0.1793	0.7316	0.1639	1.2316

TABLE 3. 48

CYPRUS ANVIL

COST PER TOTAL TONNE BY FUNCTION
1987

FUNCTION	OPERATING LABOR	MAINT. LABOR	PARTS & CONSUMABLES	MINE OVERHEAD	TOTAL
DRILLING	0.0087	0.0131	0.0353	0.0000	0.0571
BLASTING	0.0069	0.0000	0.1450	0.0000	0.1519
LOADING	0.0155	0.0356	0.0928	0.0000	0.1439
HAULING	0.0683	0.0909	0.2906	0.0000	0.4498
AUXILIARY	0.0344	0.0331	0.0786	0.0000	0.1461
GENERAL MINE	0.0187	0.0000	0.0320	0.0000	0.0507
GEN. MAINTENANCE	0.0000	0.0181	0.0580	0.0000	0.0761
GEN. ADMINISTRTN	0.0000	0.0000	0.0000	0.1684	0.1684
TOTAL	0.1525	0.1908	0.7323	0.1684	1.2440

TABLE 3.49

CYPRUS ANVIL

COST PER TOTAL TONNE BY FUNCTION
1988

FUNCTION	OPERATING LABOR	MAINT. LABOR	PARTS & CONSUMABLES	MINE OVERHEAD	TOTAL
DRILLING	0.0120	0.0162	0.0435	0.0000	0.0717
BLASTING	0.0116	0.0000	0.1410	0.0000	0.1526
LOADING	0.0160	0.0424	0.1071	0.0000	0.1655
HAULING	0.0887	0.1134	0.3566	0.0000	0.5587
AUXILIARY	0.0434	0.0427	0.1040	0.0000	0.1901
GENERAL MINE	0.0258	0.0000	0.0320	0.0000	0.0578
GEN. MAINTENANCE	0.0000	0.0597	0.0580	0.0000	0.1177
GEN. ADMINISTRTN	0.0000	0.0000	0.0000	0.2729	0.2729
TOTAL	0.1975	0.2744	0.8422	0.2729	1.5870

TABLE 3.50

CYPRUS ANVIL

COST PER TOTAL TONNE BY FUNCTION
1989

FUNCTION	OPERATING LABOR	MAINT. LABOR	PARTS & CONSUMABLES	MINE OVERHEAD	TOTAL
DRILLING	0.0165	0.0211	0.0568	0.0000	0.0944
BLASTING	0.0164	0.0000	0.1345	0.0000	0.1509
LOADING	0.0176	0.0470	0.1172	0.0000	0.1818
HAULING	0.0975	0.1485	0.4634	0.0000	0.7094
AUXILIARY	0.0454	0.0378	0.0934	0.0000	0.1766
GENERAL MINE	0.0474	0.0000	0.0320	0.0000	0.0794
GEN. MAINTENANCE	0.0000	0.0848	0.0580	0.0000	0.1428
GEN. ADMINISTRTN	0.0000	0.0000	0.0000	0.3378	0.3378
TOTAL	0.2408	0.3392	0.9553	0.3378	1.8731

TABLE 3.51

CYPRUS ANVIL

COST PER TOTAL TONNE BY FUNCTION
1990

FUNCTION	OPERATING LABOR	MAINT. LABOR	PARTS & CONSUMABLES	MINE OVERHEAD	TOTAL
DRILLING	0.0141	0.0248	0.0662	0.0000	0.1051
BLASTING	0.0211	0.0000	0.1304	0.0000	0.1515
LOADING	0.0225	0.0508	0.1248	0.0000	0.1981
HAULING	0.1045	0.1550	0.4851	0.0000	0.7446
AUXILIARY	0.0584	0.0508	0.1233	0.0000	0.2325
GENERAL MINE	0.0487	0.0000	0.0320	0.0000	0.0807
GEN. MAINTENANCE	0.0000	0.1248	0.0580	0.0000	0.1828
GEN. ADMINISTRTN	0.0000	0.0000	0.0000	0.4029	0.4029
TOTAL	0.2693	0.4062	1.0198	0.4029	2.0982

TABLE 3.52

CYPRUS ANVIL

COST PER TOTAL TONNE BY FUNCTION
1991 (JAN-OCT)

FUNCTION	OPERATING LABOR	MAINT. LABOR	PARTS & CONSUMABLES	MINE OVERHEAD	TOTAL
DRILLING	0.0138	0.0266	0.0693	0.0000	0.1097
BLASTING	0.0206	0.0000	0.1304	0.0000	0.1510
LOADING	0.0219	0.0514	0.1255	0.0000	0.1988
HAULING	0.1017	0.1643	0.5127	0.0000	0.7787
AUXILIARY	0.0497	0.0318	0.0767	0.0000	0.1582
GENERAL MINE	0.0415	0.0000	0.0320	0.0000	0.0735
GEN. MAINTENANCE	0.0000	0.0690	0.0580	0.0000	0.1270
GEN. ADMINISTRTN	0.0000	0.0000	0.0000	0.3509	0.3509
TOTAL	0.2492	0.3431	1.0046	0.3509	1.9478

TABLE 3.53

CYPRUS ANVIL

OPERATING LABOR COST/Tonne

1984 (NOV-DEC)

Total Material Mined = 4782 Ktonnes

EQUIPMENT	TOTAL COST ($x1000)	COST/Tonne ($)
DRILLING		
Drill 9 7/8 inch	31.4	0.0066
BLASTING	29.9	0.0063
LOADING		
Shovel 15 yd	60.9	0.0127
F.E.L.	6.5	0.0014
HAULING		
120 - Ton truck	68.0	0.0142
170 - Ton truck	179.3	0.0375
AUXILIARY		
F.E.L.	25.9	0.0054
R.T.D.	12.9	0.0027
D9 Dozer	32.4	0.0068
D8 Dozer	12.9	0.0027
Road Grader	51.8	0.0108
Water Truck	12.9	0.0027
GENERAL MINE	81.1	0.0170
TOTAL	605.9	0.1268

TABLE 3.54

CYPRUS ANVIL

OPERATING LABOR COST/Tonne

1985

Total Material Mined = 27046 Ktonnes

EQUIPMENT	TOTAL COST ($x1000)	COST/Tonne ($)
DRILLING		
Drill 9 7/8 inch	221.7	0.0082
BLASTING	176.2	0.0065
LOADING		
Shovel 15 yd	358.2	0.0132
F.E.L.	38.1	0.0014
HAULING		
120 - Ton truck	509.3	0.0188
170 - Ton truck	1018.5	0.0377
AUXILIARY		
F.E.L.	152.3	0.0056
R.T.D.	76.2	0.0028
D9 Dozer	190.4	0.0070
D8 Dozer	76.2	0.0028
Road Grader	304.7	0.0113
Water Truck	76.2	0.0028
GENERAL MINE	477.2	0.0176
TOTAL	3675.2	0.1357

TABLE 3.55

CYPRUS ANVIL

OPERATING LABOR COST/Tonne

1986

Total Material Mined = 26471 Ktonnes

EQUIPMENT	TOTAL COST ($x1000)	COST/Tonne ($)
DRILLING		
Drill 9 7/8 inch	221.7	0.0084
BLASTING	176.2	0.0067
LOADING		
Shovel 15 yd	358.2	0.0135
F.E.L.	38.1	0.0014
HAULING		
120 - Ton truck	982.2	0.0371
170 - Ton truck	1018.5	0.0385
AUXILIARY		
F.E.L.	152.3	0.0058
R.T.D.	76.2	0.0029
D9 Dozer	190.4	0.0072
D8 Dozer	76.2	0.0029
Road Grader	304.7	0.0115
Water Truck	76.2	0.0029
GENERAL MINE	477.2	0.0180
TOTAL	4148.1	0.1568

TABLE 3.56

CYPRUS ANVIL

OPERATING LABOR COST/Tonne

1987

Total Material Mined = 25539 Ktonnes

EQUIPMENT	TOTAL COST ($x1000)	COST/Tonne ($)
DRILLING		
Drill 9 7/8 inch	221.7	0.0087
BLASTING	176.2	0.0069
LOADING		
Shovel 15 yd	358.2	0.0140
F.E.L.	38.1	0.0015
HAULING		
120 - Ton truck	800.3	0.0313
170 - Ton truck	945.8	0.0370
AUXILIARY		
F.E.L.	152.3	0.0060
R.T.D.	76.2	0.0030
D9 Dozer	190.4	0.0075
D8 Dozer	76.2	0.0030
Road Grader	304.7	0.0119
Water Truck	76.2	0.0030
GENERAL MINE	477.2	0.0187
TOTAL	3893.5	0.1525

TABLE 3.57

CYPRUS ANVIL

OPERATING LABOR COST/Tonne

1988

Total Material Mined = 12306 Ktonnes

EQUIPMENT	TOTAL COST ($x1000)	COST/Tonne ($)
DRILLING		
Drill 9 7/8 inch	147.8	0.0120
BLASTING	143.2	0.0116
LOADING		
Shovel 15 yd	159.2	0.0129
F.E.L.	38.1	0.0031
HAULING		
120 - Ton truck	109.1	0.0089
170 - Ton truck	982.2	0.0798
AUXILIARY		
F.E.L.	76.2	0.0062
R.T.D.	76.2	0.0062
D9 Dozer	76.2	0.0062
D8 Dozer	76.2	0.0062
Road Grader	152.3	0.0124
Water Truck	76.2	0.0062
GENERAL MINE	318.1	0.0258
TOTAL	2431.0	0.1975

TABLE 3.58

CYPRUS ANVIL

OPERATING LABOR COST/Tonne

1989

Total Material Mined = 6715 Ktonnes

EQUIPMENT	TOTAL COST ($x1000)	COST/Tonne ($)
DRILLING		
Drill 9 7/8 inch	110.8	0.0165
BLASTING	110.3	0.0164
LOADING		
Shovel 15 yd	79.6	0.0119
F.E.L.	38.1	0.0057
HAULING		
120 - Ton truck	181.9	0.0271
170 - Ton truck	472.9	0.0704
AUXILIARY		
F.E.L.	76.2	0.0113
R.T.D.	38.1	0.0057
D9 Dozer	38.1	0.0057
D8 Dozer	38.1	0.0057
Road Grader	76.2	0.0113
Water Truck	38.1	0.0057
GENERAL MINE	318.1	0.0474
TOTAL	1616.5	0.2408

TABLE 3.59

CYPRUS ANVIL

OPERATING LABOR COST/Tonne

1990

Total Material Mined = 5224 Ktonnes

EQUIPMENT	TOTAL COST ($x1000)	COST/Tonne ($)
DRILLING		
Drill 9 7/8 inch	73.9	0.0141
BLASTING	110.3	0.0211
LOADING		
Shovel 15 yd	79.6	0.0152
F.E.L.	38.1	0.0073
HAULING		
120 - Ton truck	109.1	0.0209
170 - Ton truck	436.5	0.0836
AUXILIARY		
F.E.L.	76.2	0.0146
R.T.D.	38.1	0.0073
D9 Dozer	38.1	0.0073
D8 Dozer	38.1	0.0073
Road Grader	76.2	0.0146
Water Truck	38.1	0.0073
GENERAL MINE	254.5	0.0487
TOTAL	1406.8	0.2693

TABLE 3.60

CYPRUS ANVIL

OPERATING LABOR COST/Tonne

1991 (JAN-OCT)

Total Material Mined = 4399 Ktonnes

EQUIPMENT	TOTAL COST ($x1000)	COST/Tonne ($)
DRILLING		
Drill 9 7/8 inch	60.6	0.0138
BLASTING	90.4	0.0206
LOADING		
Shovel 15 yd	65.3	0.0148
F.E.L.	31.2	0.0071
HAULING		
120 - Ton truck	89.5	0.0203
170 - Ton truck	357.9	0.0814
AUXILIARY		
F.E.L.	62.5	0.0142
R.T.D.	31.2	0.0071
D9 Dozer	31.2	0.0071
D8 Dozer	31.2	0.0071
Road Grader	31.2	0.0071
Water Truck	31.2	0.0071
GENERAL MINE	182.6	0.0415
TOTAL	1096.0	0.2492

TABLE 3.61

CYPRUS ANVIL

MAINTENANCE LABOR COST/Tonne

1984 (NOV-DEC)

Total Material Mined = 4782 Ktonnes

EQUIPMENT	OPERATING SHIFTS	MAINTENANCE LABOR COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	114	313.24	35.70	0.0075
LOADING				
Shovel 15 yd	216	612.36	132.30	0.0277
F.E.L.	32	273.50	8.80	0.0018
HAULING				
120 - Ton truck	279	240.70	67.20	0.0141
170 - Ton truck	673	329.34	221.60	0.0463
AUXILIARY				
F.E.L.	34	273.50	9.30	0.0019
R.T.D.	62	108.39	6.70	0.0014
D9 Dozer	144	307.14	44.20	0.0092
D8 Dozer	72	202.17	14.60	0.0031
Road Grader	200	189.62	37.90	0.0079
Water Truck	51	202.98	10.40	0.0022
GENERAL MAINTENANCE			78.10	0.0163
TOTAL			666.80	0.1394

TABLE 3.62

CYPRUS ANVIL

MAINTENANCE LABOR COST/Tonne

1985

Total Material Mined = 27046 Ktonnes

EQUIPMENT	OPERATING SHIFTS	MAINTENANCE LABOR COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	804	234.93	188.90	0.0070
LOADING				
Shovel 15 yd	1266	619.87	784.80	0.0290
F.E.L.	190	273.50	52.00	0.0019
HAULING				
120 - Ton truck	2092	240.70	503.50	0.0186
170 - Ton truck	3963	329.34	1305.20	0.0483
AUXILIARY				
F.E.L.	260	319.80	83.10	0.0031
R.T.D.	365	107.63	39.30	0.0015
D9 Dozer	850	304.62	258.90	0.0096
D8 Dozer	425	200.63	85.30	0.0032
Road Grader	1200	188.46	226.20	0.0084
Water Truck	300	201.61	60.50	0.0022
GENERAL MAINTENANCE			416.20	0.0154
TOTAL			4003.90	0.1482

TABLE 3.63

CYPRUS ANVIL

MAINTENANCE LABOR COST/Tonne

1986

Total Material Mined = 26471 Ktonnes

EQUIPMENT	OPERATING SHIFTS	MAINTENANCE LABOR COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	880	313.24	275.70	0.0104
LOADING				
Shovel 15 yd	1277	653.18	834.10	0.0315
F.E.L.	182	273.50	49.80	0.0019
HAULING				
120 - Ton truck	3954	245.22	969.60	0.0366
170 - Ton truck	3966	347.19	1377.00	0.0520
AUXILIARY				
F.E.L.	210	546.99	114.90	0.0043
R.T.D.	365	124.84	45.60	0.0017
D9 Dozer	850	329.97	280.50	0.0106
D8 Dozer	425	229.42	97.50	0.0037
Road Grader	1200	200.12	240.10	0.0091
Water Truck	300	218.70	65.60	0.0025
GENERAL MAINTENANCE			397.80	0.0150
TOTAL			4748.20	0.1793

TABLE 3.64

CYPRUS ANVIL

MAINTENANCE LABOR COST/Tonne

1987

Total Material Mined = 25539 Ktonnes

EQUIPMENT	OPERATING SHIFTS	MAINTENANCE LABOR COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	855	391.55	334.80	0.0131
LOADING				
Shovel 15 yd	1227	684.62	840.00	0.0329
F.E.L.	182	382.54	69.60	0.0027
HAULING				
120 - Ton truck	3084	264.87	816.90	0.0320
170 - Ton truck	3756	400.36	1503.80	0.0589
AUXILIARY				
F.E.L.	210	546.99	114.90	0.0045
R.T.D.	365	124.84	45.60	0.0018
D9 Dozer	850	329.97	280.50	0.0110
D8 Dozer	425	229.42	97.50	0.0038
Road Grader	1200	200.12	240.10	0.0094
Water Truck	300	218.70	65.60	0.0026
GENERAL MAINTENANCE			463.40	0.0181
TOTAL			4872.70	0.1908

TABLE 3.65

CYPRUS ANVIL

MAINTENANCE LABOR COST/Tonne

1988

Total Material Mined = 12306 Ktonnes

EQUIPMENT	OPERATING SHIFTS	MAINTENANCE LABOR COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	508	391.55	198.90	0.0162
LOADING				
Shovel 15 yd	551	816.48	449.90	0.0366
F.E.L.	182	390.71	71.10	0.0058
HAULING				
120 - Ton truck	450	290.07	130.50	0.0106
170 - Ton truck	3005	420.82	1264.60	0.1028
AUXILIARY				
F.E.L.	110	546.99	60.20	0.0049
R.T.D.	300	126.66	38.00	0.0031
D9 Dozer	425	329.60	140.10	0.0114
D8 Dozer	425	233.72	99.30	0.0081
Road Grader	600	199.94	120.00	0.0098
Water Truck	300	223.04	66.90	0.0054
GENERAL MAINTENANCE			734.80	0.0597
TOTAL			3374.30	0.2744

TABLE 3.66

CYPRUS ANVIL

MAINTENANCE LABOR COST/Tonne

1989

Total Material Mined = 6715 Ktonnes

EQUIPMENT	OPERATING SHIFTS	MAINTENANCE LABOR COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	362	391.55	141.70	0.0211
LOADING				
Shovel 15 yd	277	816.48	226.20	0.0337
F.E.L.	182	490.89	89.30	0.0133
HAULING				
120 - Ton truck	717	291.40	208.90	0.0311
170 - Ton truck	1873	420.82	788.20	0.1174
AUXILIARY				
F.E.L.	100	585.28	58.50	0.0087
R.T.D.	150	126.94	19.00	0.0028
D9 Dozer	100	323.22	32.30	0.0048
D8 Dozer	210	241.13	50.60	0.0075
Road Grader	300	201.21	60.40	0.0090
Water Truck	150	223.46	33.50	0.0050
GENERAL MAINTENANCE			569.30	0.0848
TOTAL			2277.90	0.3392

TABLE 3.67

CYPRUS ANVIL

MAINTENANCE LABOR COST/Tonne

1990

Total Material Mined = 5224 Ktonnes

EQUIPMENT	OPERATING SHIFTS	MAINTENANCE LABOR COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	322	403.06	129.80	0.0248
LOADING				
Shovel 15 yd	203	816.48	165.70	0.0317
F.E.L.	182	546.99	99.60	0.0191
HAULING				
120 - Ton truck	450	291.19	131.00	0.0251
170 - Ton truck	1612	420.82	678.40	0.1299
AUXILIARY				
F.E.L.	100	625.14	62.50	0.0120
R.T.D.	150	131.71	19.80	0.0038
D9 Dozer	100	339.57	34.00	0.0065
D8 Dozer	210	254.67	53.50	0.0102
Road Grader	300	204.47	61.30	0.0117
Water Truck	150	231.12	34.70	0.0066
GENERAL MAINTENANCE			651.80	0.1248
TOTAL			2122.10	0.4062

TABLE 3.68

CYPRUS ANVIL

MAINTENANCE LABOR COST/Tonne

1991 (JAN-OCT)

Total Material Mined = 4399 Ktonnes

EQUIPMENT	OPERATING SHIFTS	MAINTENANCE LABOR COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	272	430.59	117.10	0.0266
LOADING				
Shovel 15 yd	170	816.48	138.80	0.0316
F.E.L.	154	565.28	87.10	0.0198
HAULING				
120 - Ton truck	381	291.43	111.00	0.0252
170 - Ton truck	1441	424.48	611.70	0.1391
AUXILIARY				
F.E.L.	60	625.14	37.50	0.0085
R.T.D.	60	131.71	7.90	0.0018
D9 Dozer	50	339.57	17.00	0.0039
D8 Dozer	110	254.80	28.00	0.0064
Road Grader	150	204.47	30.70	0.0070
Water Truck	80	231.12	18.50	0.0042
GENERAL MAINTENANCE			303.40	0.0690
TOTAL			1508.70	0.3431

TABLE 3.69

CYPRUS ANVIL

REPAIR PARTS & CONSUMABLES COST/tonne

1984 (NOV-DEC)

Total Material Mined = 4782 Ktonnes

EQUIPMENT	OPERATING SHIFTS	PARTS & CONSUMABLES COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	114	920.54	104.90	0.0219
BLASTING			647.80	0.1355
LOADING				
Shovel 15 yd	216	1628.97	351.90	0.0736
F.E.L.	32	872.19	27.90	0.0058
HAULING				
120 - Ton truck	279	749.28	209.00	0.0437
170 - Ton truck	673	1070.44	720.40	0.1506
AUXILIARY				
F.E.L.	34	872.19	29.70	0.0062
R.T.D.	62	318.72	19.80	0.0041
D9 Dozer	144	662.58	95.40	0.0199
D8 Dozer	72	466.11	33.60	0.0070
Road Grader	200	445.96	89.20	0.0187
Water Truck	51	692.66	35.30	0.0074
GENERAL MINE			153.02	0.0320
GENERAL MAINTENANCE			277.36	0.0580
GENERAL ADMINISTRN.			0.00	0.0000
TOTAL			2795.28	0.5844

TABLE 3. 70

CYPRUS ANVIL

REPAIR PARTS & CONSUMABLES COST/tonne

1985

Total Material Mined = 27046 Ktonnes

EQUIPMENT	OPERATING SHIFTS	PARTS & CONSUMABLES COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	804	786.83	632.60	0.0234
BLASTING			3816.30	0.1411
LOADING				
Shovel 15 yd	1266	1643.99	2081.30	0.0770
F.E.L.	190	872.19	165.70	0.0061
HAULING				
120 - Ton truck	2092	749.28	1567.50	0.0580
170 - Ton truck	3963	1070.44	4242.20	0.1569
AUXILIARY				
F.E.L.	260	941.65	244.80	0.0091
R.T.D.	365	317.56	115.90	0.0043
D9 Dozer	850	658.38	559.60	0.0207
D8 Dozer	425	463.53	197.00	0.0073
Road Grader	1200	444.05	532.90	0.0197
Water Truck	300	690.61	207.20	0.0077
GENERAL MINE			865.47	0.0320
GENERAL MAINTENANCE			1568.67	0.0580
GENERAL ADMINISTRN.			0.00	0.0000
TOTAL			16797.14	0.6213

TABLE 3.71

CYPRUS ANVIL

REPAIR PARTS & CONSUMABLES COST/tonne

1986

Total Material Mined = 26471 Ktonnes

EQUIPMENT	OPERATING SHIFTS	PARTS & CONSUMABLES COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	880	920.54	810.10	0.0306
BLASTING			3841.60	0.1451
LOADING				
Shovel 15 yd	1277	1710.62	2184.50	0.0825
F.E.L.	182	899.27	163.70	0.0062
HAULING				
120 - Ton truck	3954	792.70	3134.30	0.1184
170 - Ton truck	3966	1221.19	4843.20	0.1830
AUXILIARY				
F.E.L.	210	1309.54	275.00	0.0104
R.T.D.	365	356.62	130.20	0.0049
D9 Dozer	850	700.62	595.50	0.0225
D8 Dozer	425	511.51	217.40	0.0082
Road Grader	1200	470.19	564.20	0.0213
Water Truck	300	752.87	225.90	0.0085
GENERAL MINE			847.07	0.0320
GENERAL MAINTENANCE			1535.32	0.0580
GENERAL ADMINISTRN.			0.00	0.0000
TOTAL			19367.99	0.7316

TABLE 3.72

CYPRUS ANVIL

REPAIR PARTS & CONSUMABLES COST/tonne

1987

Total Material Mined = 25539 Ktonnes

EQUIPMENT	OPERATING SHIFTS	PARTS & CONSUMABLES COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	855	1054.25	901.40	0.0353
BLASTING			3703.00	0.1450
LOADING				
Shovel 15 yd	1227	1773.49	2176.10	0.0852
F.E.L.	182	1062.85	193.40	0.0076
HAULING				
120 - Ton truck	3084	822.19	2535.60	0.0993
170 - Ton truck	3756	1300.94	4886.30	0.1913
AUXILIARY				
F.E.L.	210	1309.54	275.00	0.0108
R.T.D.	365	356.62	130.20	0.0051
D9 Dozer	850	700.62	595.50	0.0233
D8 Dozer	425	511.51	217.40	0.0085
Road Grader	1200	470.19	564.20	0.0221
Water Truck	300	752.87	225.90	0.0088
GENERAL MINE			817.25	0.0320
GENERAL MAINTENANCE			1481.26	0.0580
GENERAL ADMINISTRN.			0.00	0.0000
TOTAL			18702.51	0.7323

TABLE 3.73

CYPRUS ANVIL

REPAIR PARTS & CONSUMABLES COST/tonne

1988

Total Material Mined = 12306 Ktonnes

EQUIPMENT	OPERATING SHIFTS	PARTS & CONSUMABLES COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	508	1054.25	535.60	0.0435
BLASTING			1734.70	0.1410
LOADING				
Shovel 15 yd	551	2037.21	1122.50	0.0912
F.E.L.	182	1075.10	195.70	0.0159
HAULING				
120 - Ton truck	450	859.98	387.00	0.0314
170 - Ton truck	3005	1331.62	4001.50	0.3252
AUXILIARY				
F.E.L.	110	1309.54	144.00	0.0117
R.T.D.	300	359.40	107.80	0.0088
D9 Dozer	425	700.02	297.50	0.0242
D8 Dozer	425	518.68	220.40	0.0179
Road Grader	600	469.90	281.90	0.0229
Water Truck	300	759.38	227.80	0.0185
GENERAL MINE			393.79	0.0320
GENERAL MAINTENANCE			713.75	0.0580
GENERAL ADMINISTRN.			0.00	0.0000
TOTAL			10363.94	0.8422

TABLE 3.74

CYPRUS ANVIL

REPAIR PARTS & CONSUMABLES COST/tonne

1989

Total Material Mined = 6715 Ktonnes

EQUIPMENT	OPERATING SHIFTS	PARTS & CONSUMABLES COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	362	1054.25	381.60	0.0568
BLASTING			903.10	0.1345
LOADING				
Shovel 15 yd	277	2037.21	564.30	0.0840
F.E.L.	182	1225.38	223.00	0.0332
HAULING				
120 - Ton truck	717	861.99	618.00	0.0920
170 - Ton truck	1873	1331.62	2494.10	0.3714
AUXILIARY				
F.E.L.	100	1366.98	136.70	0.0204
R.T.D.	150	359.83	54.00	0.0080
D9 Dozer	100	689.38	68.90	0.0103
D8 Dozer	210	531.03	111.50	0.0166
Road Grader	300	472.00	141.60	0.0211
Water Truck	150	760.01	114.00	0.0170
GENERAL MINE			214.88	0.0320
GENERAL MAINTENANCE			389.47	0.0580
GENERAL ADMINISTRN.			0.00	0.0000
TOTAL			6415.15	0.9553

TABLE 3.75

CYPRUS ANVIL

REPAIR PARTS & CONSUMABLES COST/tonne

1990

Total Material Mined = 5224 Ktonnes

EQUIPMENT	OPERATING SHIFTS	PARTS & CONSUMABLES COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	322	1073.91	345.80	0.0662
BLASTING			681.40	0.1304
LOADING				
Shovel 15 yd	203	2037.21	413.60	0.0792
F.E.L.	182	1309.54	238.30	0.0456
HAULING				
120 - Ton truck	450	861.67	387.80	0.0742
170 - Ton truck	1612	1331.62	2146.60	0.4109
AUXILIARY				
F.E.L.	100	1426.77	142.70	0.0273
R.T.D.	150	367.15	55.10	0.0105
D9 Dozer	100	716.63	71.70	0.0137
D8 Dozer	210	553.59	116.30	0.0223
Road Grader	300	477.40	143.20	0.0274
Water Truck	150	771.50	115.70	0.0221
GENERAL MINE			167.17	0.0320
GENERAL MAINTENANCE			302.99	0.0580
GENERAL ADMINISTRN.			0.00	0.0000
TOTAL			5328.36	1.0198

TABLE 3.76

CYPRUS ANVIL

REPAIR PARTS & CONSUMABLES COST/tonne

1991 (JAN-OCT)

Total Material Mined = 4399 Ktonnes

EQUIPMENT	OPERATING SHIFTS	PARTS & CONSUMABLES COST/SHIFT	TOTAL COST ($x1000)	COST/ tonne
DRILLING				
Drill 9 7/8 inch	272	1120.90	304.90	0.0693
BLASTING			573.60	0.1304
LOADING				
Shovel 15 yd	170	2037.21	346.30	0.0787
F.E.L.	154	1336.97	205.90	0.0468
HAULING				
120 - Ton truck	381	862.03	328.40	0.0747
170 - Ton truck	1441	1337.10	1926.80	0.4380
AUXILIARY				
F.E.L.	60	1426.77	85.60	0.0195
R.T.D.	60	367.15	22.00	0.0050
D9 Dozer	50	716.63	35.80	0.0081
D8 Dozer	110	553.82	60.90	0.0138
Road Grader	150	477.40	71.60	0.0163
Water Truck	80	771.50	61.70	0.0140
GENERAL MINE			140.77	0.0320
GENERAL MAINTENANCE			255.14	0.0580
GENERAL ADMINISTRN.			0.00	0.0000
TOTAL			4419.41	1.0046

TABLE 3.77

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

Drill 9 7/8 inch
Electrical

LIST PRICE $x1000	$	1580.0
BIT COST	$	4600
BIT LIFE HRS		137
POWER CONSUMPTION (kWhr/hr)		125

COST BREAKDOWN ($/hr.)

Power cost $0.070/KWhr	$	8.75
Bit Cost	$	33.58
Drill Pipe & Stabilizer @ 1/4 Bit Cost	$	8.40
Repair Parts (0.0372 x Delivered Cost in $x1000)	$	58.78
Maintenance Labor (0.0204 x Delivered Cost in $x1000)	$	32.23
OPERATING HRS./SHIFT		10.5
DRILLING HRS./SHIFT		7.0

COST BREAKDOWN ($/Shift)

Bit Cost @ 7.00hrs./shift	$	235.06
Drill Pipe & Stabilizer @ 7.00hrs./shift	$	58.77
Repair Parts @10.50hrs/shft	$	617.19
Maintenance Labor @10.50hrs./shift	$	338.42
Power Cost @10.50 hrs.shift	$	91.88

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.78

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

Drill 9 7/8 inch
Diesel

LIST PRICE $x1000	$	1580.0
BIT COST	$	4600
BIT LIFE HRS		137
FUEL CONSUMPTION LIT./HR		82

COST BREAKDOWN ($/hr.)

Fuel cost $0.410/Litre	$	33.62
Bit Cost	$	33.58
Drill Pipe & Stabilizer @ 1/4 Bit Cost	$	8.40
Repair Parts (0.0468 x Delivered Cost in $x1000)	$	73.94
Maintenance Labor (0.0302 x Delivered Cost in $x1000)	$	47.72
OPERATING HRS./SHIFT		10.5
DRILLING HRS./SHIFT		7.0

COST BREAKDOWN ($/Shift)

Bit Cost @ 7.00hrs./shift	$	235.06
Drill Pipe & Stabilizer @ 7.00hrs./shift	$	58.77
Repair Parts @10.50hrs/shft	$	776.37
Maintenance Labor @10.50hrs./shift	$	501.06
Fuel Cost @10.50 hrs.shift	$	353.01

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.79

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

Shovel 15 yd

LIST PRICE $x1000	$	3240.0
POWER CONSUMPTION (kWhr/hr)		550
COST BREAKDOWN ($/hr.)		
Power cost $0.070/KWhr	$	38.50
Repair Parts (0.0480 x Delivered Cost in $x1000)	$	155.52
Maintenance Labor (0.0240 x Delivered Cost in $x1000)	$	77.76
OPERATING HRS./SHIFT		10.5
COST BREAKDOWN ($/Shift)		
Maintenance Labor	$	816.48
Parts & Consumables	$	2037.21
TOTAL OPERATING COST/SHIFT	$	2853.69

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.80

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

F.E.L.

LIST PRICE $x1000	$	920.0
LIST PRICE EXCLUDING TIRES	$	861.4
FUEL CONSUMPTION LIT./HR		85
COST PER TIRE	$	14650
NUMBER OF TIRES		4
TIRE LIFE HRS		5000
COST BREAKDOWN ($/hr.)		
Fuel cost $0.410/litre	$	34.85
Tire Cost	$	11.72
Repair Parts (0.0648 x Delivered Cost in $x1000)	$	55.82
Maintenance Labor (0.0432 x Delivered Cost in $x1000)	$	37.21
OPERATING HRS./SHIFT		10.5
COST BREAKDOWN ($/Shift)		
Maintenance Labor	$	390.71
Parts & Consumables	$	1075.10
TOTAL OPERATING COST/SHIFT	$	1465.81

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.81

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

120 - Ton truck

LIST PRICE $x1000	$	940.0
LIST PRICE EXCLUDING TIRES	$	884.5
FUEL CONSUMPTION LIT./HR		60
COST PER TIRE	$	9250
NUMBER OF TIRES		6
TIRE LIFE HRS		3500

COST BREAKDOWN ($/hr.)		
Fuel cost $0.410/litre	$	24.60
Tire Cost	$	15.86
Repair Parts (0.0432 x Delivered Cost in $x1000)	$	38.21
Maintenance Labor (0.0288 x Delivered Cost in $x1000)	$	25.47
OPERATING HRS./SHIFT		10.5

COST BREAKDOWN ($/Shift)		
Maintenance Labor	$	267.44
Parts & Consumables	$	826.04
TOTAL OPERATING COST/SHIFT	$	1093.48

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.82

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

170 - Ton truck

LIST PRICE $x1000	$	1300.0
LIST PRICE EXCLUDING TIRES	$	1210.0
FUEL CONSUMPTION LIT./HR		100
COST PER TIRE	$	15000
NUMBER OF TIRES		6
TIRE LIFE HRS		3500

<u>COST BREAKDOWN ($/hr.)</u>		
Fuel cost $0.410/litre	$	41.00
Tire Cost	$	25.71
Repair Parts (0.0432 x Delivered Cost in $x1000)	$	52.27
Maintenance Labor (0.0288 x Delivered Cost in $x1000)	$	34.85
OPERATING HRS./SHIFT		10.5
<u>COST BREAKDOWN ($/Shift)</u>		
Maintenance Labor	$	365.93
Parts & Consumables	$	1249.29
TOTAL OPERATING COST/SHIFT	$	1615.22

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

The fuel consumption for 1984-85 is 85 litre/whr.

The tire cost for 1984-85 is $ 11,700.

TABLE 3.83

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

F.E.L.

LIST PRICE $x1000	$	920.0
LIST PRICE EXCLUDING TIRES	$	861.4
FUEL CONSUMPTION LIT./HR		85
COST PER TIRE	$	14650
NUMBER OF TIRES		4
TIRE LIFE HRS		5000

<u>COST BREAKDOWN ($/hr.)</u>

Fuel cost $0.410/litre	$	34.85
Tire Cost	$	11.72
Repair Parts (0.0648 x Delivered Cost in $x1000)	$	55.82
Maintenance Labor (0.0432 x Delivered Cost in $x1000)	$	37.21
OPERATING HRS./SHIFT		10.5

<u>COST BREAKDOWN ($/Shift)</u>

Maintenance Labor	$	390.71
Parts & Consumables	$	1075.10

TOTAL OPERATING COST/SHIFT	$	1465.81

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.84

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

R.T.D.

LIST PRICE $x1000	$	315.0
LIST PRICE EXCLUDING TIRES	$	298.6
FUEL CONSUMPTION LIT./HR		25
COST PER TIRE	$	4110
NUMBER OF TIRES		4
TIRE LIFE HRS		3000

COST BREAKDOWN ($/hr.)

Fuel cost $0.410/litre	$	10.25
Tire Cost	$	5.48
Repair Parts (0.0460 x Delivered Cost in $x1000)	$	13.74
Maintenance Labor (0.0300 x Delivered Cost in $x1000)	$	8.96
OPERATING HRS./SHIFT		10.5

COST BREAKDOWN ($/Shift)

Maintenance Labor	$	94.08
Parts & Consumables	$	309.44
TOTAL OPERATING COST/SHIFT	$	403.52

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.85

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

D9 Dozer

LIST PRICE $x1000	$	550.0
FUEL CONSUMPTION LIT./HR		35
<u>COST BREAKDOWN ($/hr.)</u>		
Fuel cost $0.410/litre	$	14.35
Repair Parts (0.0700 x Delivered Cost in $x1000)	$	38.50
Maintenance Labor (0.0420 x Delivered Cost in $x1000)	$	23.10
OPERATING HRS./SHIFT		10.5
<u>COST BREAKDOWN ($/Shift)</u>		
Maintenance Labor	$	242.55
Parts & Consumables	$	554.93

TOTAL OPERATING COST/SHIFT	$	797.48

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.86

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

D8 Dozer

LIST PRICE $x1000	$	390.0
FUEL CONSUMPTION LIT./HR		30
COST BREAKDOWN ($/hr.)		
Fuel cost $0.410/litre	$	12.30
Repair Parts (0.0700 x Delivered Cost in $x1000)	$	27.30
Maintenance Labor (0.0420 x Delivered Cost in $x1000)	$	16.38
OPERATING HRS./SHIFT		10.5
COST BREAKDOWN ($/Shift)		
Maintenance Labor	$	171.99
Parts & Consumables	$	415.80
TOTAL OPERATING COST/SHIFT	$	587.79

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.87

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

Road Grader

LIST PRICE $x1000	$	340.0
LIST PRICE EXCLUDING TIRES	$	331.1
FUEL CONSUMPTION LIT./HR		25
COST PER TIRE	$	1480
NUMBER OF TIRES		6
TIRE LIFE HRS		3000

COST BREAKDOWN ($/hr.)

Fuel cost $0.410/litre	$	10.25
Tire Cost	$	2.96
Repair Parts (0.0696 x Delivered Cost in $x1000)	$	23.04
Maintenance Labor (0.0420 x Delivered Cost in $x1000)	$	13.91
OPERATING HRS./SHIFT		10.5

COST BREAKDOWN ($/Shift)

Maintenance Labor	$	146.05
Parts & Consumables	$	380.63
TOTAL OPERATING COST/SHIFT	$	526.68

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

TABLE 3.88

CYPRUS ANVIL

EQUIPMENT OPERATING COST/SHIFT

Water Truck

LIST PRICE $x1000	$	720.0
LIST PRICE EXCLUDING TIRES	$	664.5
FUEL CONSUMPTION LIT./HR		60
COST PER TIRE	$	9250
NUMBER OF TIRES		6
TIRE LIFE HRS		3500

COST BREAKDOWN ($/hr.)		
Fuel cost $0.410/litre	$	24.60
Tire Cost	$	15.86
Repair Parts (0.0432 x Delivered Cost in $x1000)	$	28.71
Maintenance Labor (0.0288 x Delivered Cost in $x1000)	$	19.14
OPERATING HRS./SHIFT		10.5
COST BREAKDOWN ($/Shift)		
Maintenance Labor	$	200.97
Parts & Consumables	$	726.28
TOTAL OPERATING COST/SHIFT	$	927.25

Above costs are average costs over the life of the equipment.
Costs for each year have been adjusted to reflect the age of the equipment.

4.0 MILLING

4.0 MILLING

4.1 Summary

A study of the data available shows that the plant can treat the projected 11,160 metric tons per day at a grind of 80 percent or more passing 70 microns if the plant modifications planned by CAMC and discussed below are made.

PAH projects recoveries that are the same as the CAMC predictions for 1984 and 1985. The PAH lead recoveries average 1.2 percent lower for the period 1986 to 1991 than the CAMC data. The PAH zinc recoveries average 2.2 percent lower for the period 1986 to 1991 than the CAMC data. PAH believes that lead and zinc concentrate grades will be 0.5 percent lower than the CAMC projections.

PAH basically agrees with the CAMC operating costs with slightly higher costs for periods of lower tonnages and for the treatment of oxide ore.

The water supply must be substantially increased by May of 1985 by the development of more fresh water (3.3 million Imperial gallons versus the 2.0 million Imperial gallons under the present water permit) or by the installation of a tailings water recycle system. Either alternative will cost about $1,800,000.

The mill modification plan and schedule appears to be adequate to produce the necessary results for plant operation under the proposed tonnages and manpower requirements. However, given the items to be completed prior to startup, it is doubtful if a November, 1984 startup can be achieved. Better definition of the modifications, the schedule, the costs, and the cost effectiveness of each individual modification is required.

4.2 Milling Capacity

For purposes of analyzing the mill capacity the plant was divided into four sections: crushing, grinding, flotation and concentrate handling. Each section is reviewed below. The general statement can be made that PAH estimates that the mill can process 11,160 mtpd at a grind of 80 percent passing 70 microns.

Crushing

The major limitation of the crushing circuit is in material handling, the most common restriction in any crushing plant. Since going to open circuit crushing, the tertiary cone crushers do not represent a limitation on plant throughput. The material handling limitation will be resolved by the installation of the mine run rock breaker, the double-deck screens between the primary crusher and the coarse ore pile and by modifying drop boxes and conveyor loading points. In addition, the 30-inch fine ore conveyors are to be replaced. Details concerning the proposed crusher modification are covered in Section 4.6 (Plant Modifications). While several means are available to eliminate bottlenecks, the end result will be a crushing circuit capable of supplying the mill at 11,160 mtpd.

The capacity to crush 11,160 mtpd has been attained by coarsening the rod mill feed (open circuit crushing). The design criteria for the 11,160 mtpd crushing plant are:

	Design Criteria for Belts 14, 15, 16 & 17	Stated Criteria for Crushing Plant
Daily Rate	11,160 mt	12,000
Operating Hours	12	12
Operating Shifts	2	--
Catch up Capacity	130%	--
Surge Capacity	120%	--
Design Rate (mtph)	1,437	1,000

The above belt design criteria therefore provide for a plant capable of crushing 17,244 tonnes in 12 hours during two shifts. This is a highly conservative design, and in fact, it allows for expansion to 1,750 mtph or 21,000 mtpd. Before expending the capital to increase the material handling capacity of the crushing plant an in-depth analysis should be made of the following:

1) Operate the crushing plant three shifts per day, 19 shifts per week (two continuous shifts down for maintenance).

2) Pull full power on crushers and attempt to produce a finer final product.

3) Evaluate going back to closed circuit crushing on the tertiary crushers.

If the above can be accomplished, the grinding section will be capable of producing a finer product at 11,160 mtpd and the crushing productivity can be increased. Studies have shown that the effect of closed circuit crushing rather than open circuit prior to rod milling is to reduce power consumption per ton in the rod mill by 10 to 20 percent or to produce a finer grind which almost always results in improved recoveries of metals.

Grinding

The existing grinding circuits can process 11,160 mtpd at a feed product size of 80 percent passing 70 microns. As noted is the plant modification program, provision for handling higher pulp volumes will be required. The basis for estimating the capacity of the grinding section is shown in Table 4.1.

TABLE 4.1

GRINDING DESIGN CRITERIA
CYPRUS ANVIL MINE

Daily Milling Rate (mtpd)	11,500
Operating Availability (%)	92
Rod Mill Feed (80 Percent Passing-Microns)	15,000
Final Product (80 Percent Passing-Microns)	70
Bond Work Index (kwh/mt)	13.2
Mill Sizes and Horsepower Draw	
Three Rod Mills, 9-ft dia by 12-ft	398 hp
Three Ball Mills, 9-ft dia by 12-ft	427 hp
One Ball Mill, 13-1/2-ft dia by 22-ft	2,402 hp
Rod Mill, 12-1/2-ft dia by 16-ft	1,270 hp
Ball Mill, 13-1/2-ft dia by 22- ft	2,430 hp
Ball Mill, 13-1/2 ft dia by 22-ft	2,500 hp
Energy Transmission Loss for all Mills (%)	5

If the above criteria are met, in particular the maximum power draft, the final product size may approach 65 microns. However, 70 microns should be utilized for all estimates since it is nearly impossible to utilize full power at all times. A mill with new liners is an example of less than full power utilization.

Flotation/Regrind

The important parameter in this section is residence time in each flotation section. Based on previous operating experience, CAMC has stated that the residence times at 11,150 mtpd are adequate. Because previous plant operation suffered from lack of adequate water supply and other process controls, it is difficult for PAH to judge this conclusion, although the residence times do not look unreasonable when compared to industry practice. The only caution in accepting these times is that the reduction in residence is to be accomplished while producing higher recoveries and better concentrate grades than in the past. Given improvements in density and reagent control, and the fine grind, it is not unreasonable to expect the above improvements at lower residence times, but it will undoubtedly take some time to optimize the flotation section.

Concentrate Handling

The instrumentation, equipment modifications and installations projected for the dewatering and drying sections of the plant should provide more than adequate facilities for concentrate dewatering filtration and drying. Some of these changes are discussed in the memo to D. Gregoire from F. Meyer and R. Hall of March 10, 1984 regarding the vacuum system, and in the memo of June 6, 1984 to John Maissan from Trevor Arbuckle regarding the zinc loadout belt, and modifications and repairs required to improve the efficiency of the dewatering area. The modifications include a complete revamping of the vacuum system for the concentrate filters, the installation of four new slurry pumps to handle concentrates, instrumentation and control improvements costing $420,000 including indirect costs, general maintenance and streamlining plant modifications, and the conversion of the concentrate dryers to oil firing.

The estimated base cost of the changes to the dewatering section is $775,000. These costs are shown in Table 4.11, Mill Modifications and Costs, based on data developed by Rick Visagie and John Maissan. With a 20 percent allowance for indirect costs, this cost becomes $930,000.

The conversion of the concentrate dryers to firing with oil is estimated to cost $700,000. With a 20 percent allowance for indirect costs this figure becomes $840,000. Total expenditures in this section of the plant are then $1,770,000.

4.3 Recovery and Concentrate Grade

The most difficult part of the mill evaluation has been the estimation of future recoveries and concentrate grades. The difficulty in the analysis is caused by:

1) Lack of information on how the projected recoveries were developed.

2) No quantification of improvements from proposed mill modifications.

3) The lower plant recoveries and concentrate grades during earlier years.

The current CAMC recovery estimates for each ore type are based upon the analysis of laboratory and pilot plant data by P. Taggart. This analysis provides for estimating recovery and concentrate grades for each ore type at various grinds. PAH broke the analysis into three parts: effect of grind, effect of feed grade on recovery and an estimate of a base recovery and concentrate grades.

The effect of grind that has been utilized in the CAMC analysis is approximately a 0.05 percent improvement in lead recovery and a 0.1 percent improvement in zinc recovery for each one micron decrease in flotation feed size (80 percent passing size). Based upon test work completed to date, this effect of grind on recovery appears correct. Earlier estimates of CAMC provided for no improvement in concentrate grade at finer grinds.

The second part of the recovery analysis involves determining the effect of feed grade upon recovery. In the CAMC analysis no effect of feed grade upon recovery is considered. For each ore type, recovery is kept constant (for each grind) despite large variations in feed grade. Table 4.2 illustrates the range in feed grades for the major ore types in the plant feed during 1985 to 1991. The range

TABLE 4.2

FEED GRADES (%) BY ORE TYPE
CYPRUS ANVIL MINE

Period (Qt/Yr)	2a		2bcd		2ec		2ef		2g		2h	
	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn
2/85	3.21	5.32	3.17	5.24	3.40	6.00	3.15	5.25	3.00	5.00	3.21	5.40
3/85	2.52	4.22	2.67	4.39	3.16	3.97	2.54	4.27	2.60	4.30	2.48	4.26
4/85	2.39	3.11	2.26	3.47	2.34	3.45	2.39	3.45	2.50	4.00	2.26	3.51
1/86	2.99	4.76	2.77	4.50	2.85	4.50	2.90	4.61	3.17	5.16	2.71	4.31
2/86	2.60	4.06	2.80	4.43	2.59	4.32	2.64	4.19	2.89	4.28	2.50	3.90
3/86	2.73	4.72	2.54	4.86	2.66	4.85	3.32	4.93	3.20	4.78	3.19	5.06
4/86	3.03	4.11	2.55	3.69	2.49	3.64	2.82	3.82	2.92	3.93	2.75	3.75
1/87	2.58	3.62	2.55	3.64	2.66	3.73	2.62	3.68	2.56	3.59	2.63	3.69
2/87	2.79	4.04	2.78	3.99	2.83	4.12	2.84	4.15	2.82	4.10	2.85	4.16
3/87	2.93	4.03	2.99	4.06	3.00	4.17	3.00	4.12	3.04	4.10	2.95	4.25
4/87	2.76	3.96	2.74	3.86	2.75	3.90	2.75	3.88	2.70	3.70	--	--
1/88	3.24	4.06	2.82	4.09	2.92	4.23	2.92	4.46	2.84	4.19	2.81	4.14
2/88	3.43	4.31	3.41	4.32	3.31	4.16	3.38	4.27	3.31	4.17	3.42	4.32
3/88	3.79	4.98	3.43	4.30	3.35	4.19	3.44	4.32	3.43	4.30	3.46	4.35
4/88	3.15	4.96	3.76	4.92	3.80	4.99	3.78	4.96	3.76	4.92	3.79	4.98
1/89	3.26	4.73	3.16	4.68	3.33	4.77	3.09	4.65	3.42	4.81	3.37	4.79
2/89	3.01	4.60	3.01	4.61	3.01	4.60	3.01	4.61	3.00	4.60	3.01	4.60
3/89	3.30	4.80	3.30	4.80	3.30	4.80	3.30	4.80	3.30	4.80	3.30	4.80
1/90	2.90	4.40	2.90	4.40	2.90	4.40	2.90	4.40	2.90	4.40	2.90	4.40
2/90	2.90	4.40	2.90	4.40	2.90	4.40	2.90	4.40	--	--	--	--
3/90	3.35	5.00	3.35	5.02	3.27	4.90	3.24	4.85	--	--	3.44	5.13
4/90	3.28	4.90	3.16	4.74	3.13	4.73	3.84	4.33	--	--	3.40	5.06
1/91	2.50	3.90	2.50	3.90	2.50	3.90	2.50	3.90	--	--	--	--
2/91	2.59	4.09	2.59	4.10	2.59	4.08	2.58	4.08	--	--	--	--
3/91	2.23	4.35	2.28	4.42	2.32	4.39	2.34	4.32	--	--	--	--
Average	2.94	4.38	2.90	4.35	2.89	4.37	2.97	4.35	3.02	4.38	3.02	4.44
High	3.79	5.32	3.76	5.24	3.80	6.00	3.78	5.25	3.76	5.16	3.79	5.40
Low	2.23	3.11	2.26	3.47	2.32	3.45	2.34	3.45	2.50	3.59	2.26	3.51

of lead assays is typically 1.5 percent and the range of zinc assays is 2.0 percent. Over a range of assays this wide the recoveries cannot remain constant. In telephone conversations with the CAMC staff it became apparent that insufficient data are available at this time to evaluate the effect of heads upon recovery. However, despite this shortcoming, some measure of the effect must be estimated. PAH evaluated a constant tail approach for estimating the effect of head grades on recovery. The basic assumption was made that a constant tail would be produced no matter what the feed grade. This is a somewhat conservative approach, and the effect of this conservatism is discussed later. The basis for arriving at the constant tail was the following data, essentially derived from the operating data of December, 1981 (all sulfide ore).

Weight of Concentrate from CAMC estimates
Feed Assay of 3.1% lead, 5.0% zinc
Lead Recovery of 81.6% @ 70 micron grind
Zinc Recovery of 84.0% @ 70 micron grind
Lead Tail of 0.64% @ 70 micron grind
Zinc Tail of 0.89% @ 70 micron grind

The CAMC estimation assumes a constant recovery with varying head grade, therefore producing a sharply varying tail. A comparison of the two approaches is shown in Figure 4-1. The figure is for lead. A similar curve can be developed for zinc.

While it is true that for each ore type the relationship between head grade variation and recovery should be developed, the above criteria were applied against the yearly head grade. Data utilized in the constant tail analysis are shown in Tables 4.3 and 4.4, and a comparison to CAMC predictions is shown in Table 4.5. A review of Table 4.5 shows that the lower heads, at a constant tail, do result in lower than predicted recoveries and show that the current CAMC prediction must be adjusted for the effect of head grade.

As a further check on the current CAMC predictions, a comparison was made between these predictions for each ore type and past plant performance. For the months of December, 1981 through May, 1982, predicted recoveries and grades were calculated taking into account the effect of

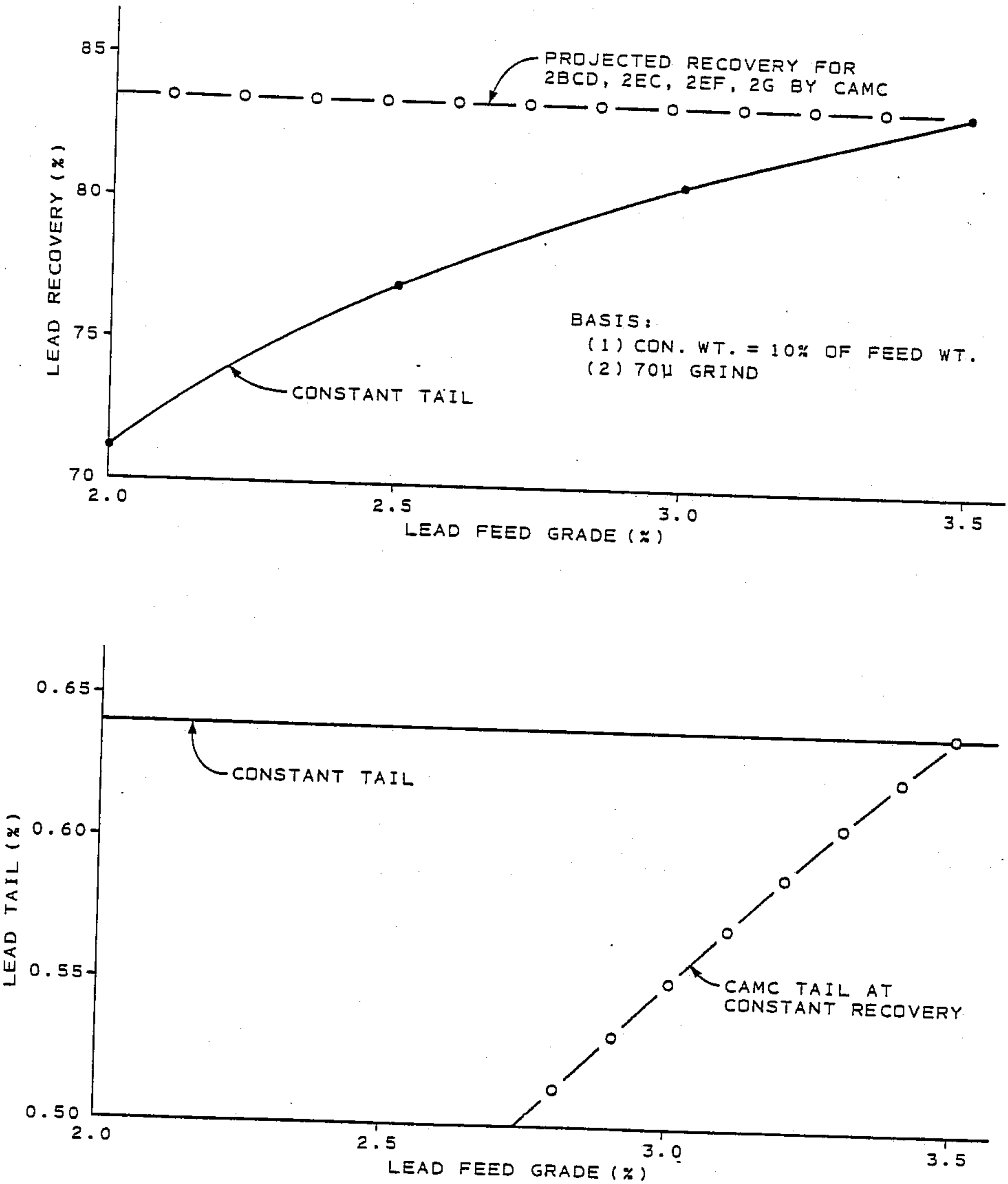


FIGURE 4-1

Illustration of Effect
of Tail Assay on Recovery
CYPRUS ANVIL MINE

TABLE 4.3

AVERAGE HEADS BY CALENDAR QUARTER
CYPRUS ANVIL MINE

Quarter	Lead (%)	Zinc (%)
1st, '86	2.88	4.60
2nd, '86	2.68	4.25
3rd, '86	3.04	4.89
4th, '86	2.75	3.82
Average	2.84	4.39
1st, '87	2.61	3.67
2nd, '87	2.83	4.12
3rd, '87	3.00	4.13
4th, '87	2.74	3.88
Average	2.80	3.95
1st, '88	2.87	4.29
2nd, '88	3.37	4.26
3rd, '88	3.43	4.31
4th, '88	3.78	4.96
Average	3.36	4.46
1st, '89	3.21	4.71
2nd, '89	3.01	4.61
3rd, '89	3.30	4.80
Average	3.16	4.71
1st, '90	2.90	4.40
2nd, '90	2.90	4.40
3rd, '90	3.31	4.97
4th, '90	3.09	4.65
Average	3.05	4.61
1st, '91	2.50	3.90
2nd, '91	2.59	4.08
3rd, '91	2.28	4.38
Average	2.46	4.12

TABLE 4.4

EFFECT OF CONSTANT TAIL
CYPRUS ANVIL MINE

Quarter	Tons Ore	Tons Tailing	Tons Lead Feed	Tons Lead Tail[1] (Fix)	Tons Zinc Feed	Tons Zinc Tail[1] (Fix)	Lead Recovery Fix Tail	Lead Recovery Pred CAMC	Zinc Recovery Fix Tail	Zinc Recovery Pred CAMC
1st, '86	1,004,400	890,730	28,898.0	5,789.7	46,193.1	8,105.6	80.0	82.3	82.5	83.7
2nd, '86	1,015,360	908,806	27,205.4	5,907.2	43,208.4	8,270.1	78.3	81.8	80.9	83.9
3rd, '86	1,026,720	903,811	31,180.8	5,874.8	50,217.6	8,224.7	81.2	82.6	83.6	83.9
4th, '86	1,026,720	924,810	28,214.0	6,011.3	39,199.2	8,415.8	78.7	81.7	78.5	83.9
						Average	79.6	82.1	81.4	83.8
1st, '87	1,004,400	908,519	26,240.2	5,814.5	36,849.3	8,085.8	77.8	83.1	78.1	84.6
2nd, '87	1,015,560	908,865	28,721.4	5,816.7	41,844.6	8,088.9	79.7	83.2	80.7	84.4
3rd, '87	1,026,720	916,411	30,798.0	5,865.0	42,362.4	8,156.1	81.0	83.3	80.7	84.5
4th, '87	1,026,720	923,497	28,175.6	5,910.4	39,805.1	8,219.0	79.0	83.0	79.4	84.6
						Average	79.4	83.2	79.7	84.5
1st, '88	1,015,560	904,536	29,125.3	5,789.0	43,517.2	8,050.4	80.1	82.5	81.5	84.5
2nd, '88	1,015,560	897,537	34,229.2	5,744.2	43,219.3	7,988.0	83.2	83.0	81.5	84.3
3rd, '88	1,026,720	906,008	35,251.7	5,998.5	44,218.8	8,063.5	83.1	82.7	81.8	84.2
4th, '88	1,026,720	891,276	38,835.4	5,704.2	50,976.0	7,932.4	85.3	82.1	84.4	83.6
						Average	83.0	82.5	82.3	84.2
1st, '89	1,004,400	883,592	32,286.4	5,655.0	47,279.6	7,864.0	82.5	82.5	83.4	84.0
2nd, '89	1,015,560	897,792	30,552.8	3,745.9	46,773.1	7,900.6	81.2	82.7	83.1	84.2
3rd, '89	1,026,720	899,875	33,881.8	5,759.2	49,282.6	8,008.9	83.0	82.9	83.7	84.4
						Average	82.2	82.7	83.4	84.2
1st, '90	1,004,400	892,701	29,127.6	5,713.3	44,193.6	7,945.0	80.4	82.5	82.0	84.3
2nd, '90	1,015,560	902,435	29,451.2	5,775.6	44,684.6	8,031.7	80.4	83.2	82.0	84.6
3rd, '90	1,026,720	896,783	32,961.0	5,739.4	50,981.6	7,981.4	82.6	81.2	84.3	84.2
4th, '90	1,026,720	905,787	31,677.1	5,797.0	47,724.8	8,061.5	81.7	80.4	83.1	83.8
						Average	81.3	81.8	82.9	84.2
1st, '91	1,004,400	907,036	25,110.0	5,805.0	39,171.6	8,072.6	76.9	82.2	79.4	84.3
2nd, '91	1,015,560	912,186	26,272.6	5,838.0	41,465.9	8,118.5	77.8	81.7	80.4	84.3
3rd, '91	1,026,720	921,671	23,404.7	5,898.7	44,704.0	8,202.9	74.8	79.8	81.7	83.9
						Average	76.5	81.3	80.5	84.2

[1] For 1986 Tail is 0.65% Pb, 0.91% Zn at a 75-micron grind.

For 1987 Onward Tail is 0.64% Pb, 0.89% Zn at a 70-micron grind.

TABLE 4.5

SUMMARY
FIXED TAIL RECOVERY ANALYSIS
CYPRUS ANVIL MINE

Year	Lead Recovery(%) Fixed Tail	Lead Recovery(%) CAMC Predicted	Lead Recovery(%) Diff.[1]	Zinc Recovery(%) Fixed Tail	Zinc Recovery(%) CAMC Predicted	Zinc Recovery(%) Diff.[1]
1986	79.6	82.1	-2.5	81.4	83.8	-2.4
1987	79.4	83.2	-3.8	79.7	84.5	-4.8
1988	83.0	82.5	+0.5	82.3	84.2	-0.9
1989	82.2	82.7	-0.5	83.4	84.2	-0.8
1990	81.3	81.8	-0.5	82.9	84.2	-1.3
1991	76.5	81.3	-4.8	80.5	84.2	-3.7

[1] Fixed Tail Minus Predicted.

grind, types of ores and current CAMC projected recoveries for each ore type. A comparison of the predictions based on the current CAMC approach versus actual results is shown in Table 4.6. It is no surprise that the CAMC predictions give higher recoveries and better grades. The improvements represent the anticipated better results from a better operated mill.

Given the above analysis, a conclusion must be reached regarding recoveries and concentrate grades. Because of the lack of precise data, this conclusion is somewhat subjective. However, PAH feels that our analysis is more inclusive of the pertinent parameters than the current CAMC model. PAH agrees that significant improvements can be made in the mill operation, but that insufficient evidence has been developed to support the predicted recoveries. Likewise, an estimate using all the adjustments of PAH's analysis may be too conservative. For this reason, PAH has adopted a correction to CAMC recovery predictions that includes 50 percent of the reductions in recoveries estimated by the PAH study. The combined effects of the PAH analysis are shown in Table 4.7 where the recoveries for lead and zinc labelled PAH economics are those used in the PAH calculations of the cash flows by calendar year. For concentrate grades, PAH feels that there are less data available than for recoveries and recommends that all concentrate grades be reduced by 0.5 percent.

TABLE 4.6

ACTUAL VS. CMAC PREDICTED RECOVERIES AND CONCENTRATE GRADES
CYPRUS ANVIL MINE

Month	Lead Recovery (%) Actual	Lead Recovery (%) Predicted	Zinc Recovery (%) Actual	Zinc Recovery (%) Predicted	Lead Conc. Grade (%) Actual	Lead Conc. Grade (%) Predicted	Zinc Conc. Grade (%) Actual	Zinc Conc. Grade (%) Predicted
12/81	80.6	81.4	82.5	82.2	59.1	61.5	50.6	51.2
1/82	79.7	81.6	82.4	82.8	60.9	62.3	50.3	51.5
2/82	78.2	77.8	73.0	81.6	59.8	59.3	49.8	50.6
3/82	71.9	72.0	75.3	75.7	58.1	58.2	48.8	48.7
4/82	70.3	70.2	76.3	74.0	59.6	58.0	47.8	48.0
5/82	71.6	72.7	73.0	76.4	55.7	58.4	49.6	49.2
AVERAGE	75.4	76.0	77.1	78.8	58.9	59.6	49.5	49.9
DIFFERENCE[1]	-0.6		-1.7		-0.7		-0.4	

[1] Actual Minus Predicted

TABLE 4.7

SUMMARY OF RECOVERY REVIEW
CYPRUS ANVIL MINE

	YEAR							
	1984	1985	1986	1987	1988	1989	1990	1991
Lead Recovery								
Difference from Table 4.6	0.0	0.0	-0.6	-0.6	-0.6	-0.6	-0.6	-0.6
Feed Grade Correction	0.0	0.0	-2.5	-3.8	+0.5	-0.5	-0.5	-6.8
Sum	0.0	0.0	-3.1	-4.4	-0.1	-1.1	-1.1	-7.4
CAMC Prediction	65.2	77.4	82.1	83.2	82.6	82.7	81.8	83.3
PAH Economics	65.2	77.4	80.6	81.0	82.6	82.1	81.3	79.6
Zinc Recovery								
Difference from Table 4.6	0.0	0.0	-1.7	-1.7	-1.7	-1.7	-1.7	-1.7
Feed Grade Correction	0.0	0.0	-2.4	-4.8	-0.9	-0.8	-1.3	-3.7
Sum	0.0	0.0	-4.1	-6.5	-2.6	-2.5	-3.0	-5.4
CAMC Prediction	67.9	79.4	83.8	84.5	84.2	84.2	84.2	84.2
PAH Economics	67.9	79.4	81.8	81.3	82.9	82.0	82.7	81.5

4.4 Operating Costs

The basis for the PAH review of projected operating costs is the "Summary of Mill Cost Estimates - July, 1984" prepared by H. M. Visagie. A summary of the operating costs from this report is contained in Table 4.9. The PAH estimates by year of operating costs are presented in Table 4.10.

TABLE 4.8

HISTORICAL RECOVERY DATA
CYPRUS ANVIL MINE

Month	Grind[1] P_{80}	Lead[1] Rec/Grade (%)	Zinc[1] Rec/Grade (%)	Milling[1] Rate	Ag Rec[1] (%)	Head Lead (%)	Head Zinc (%)	Head Silver gm/Mt
12/81	85	80.6/59.1	82.5/50.6	11,442	57.8	3.1	5.0	38.4
1/82	--	79.7/60.9	82.8/50.3	9,925	57.4	2.9	5.3	32.3
2/82	70	78.2/54.8	73.0/49.8	10,592	55.2	2.7	4.4	33.6
3/82	72	71.9/58.1	75.3/48.8	11,219	47.3	2.9	4.9	30.4
4/82	65	70.3/59.6	76.3/47.8	10,296	49.3	2.0	5.2	37.1
5/82	81	71.6/55.7	73.0/49.6	10,948	49.4	2.6	4.4	36.1

[1] From CAMC Summary of Operating Costs

TABLE 4.9

CAMC ESTIMATED OPERATING COSTS[1]
CYPRUS ANVIL MINE

	Cost($/mt)
Operations	
Personnel	1.13
Flotation	2.11
Steel	1.50
Crusher/Grinding	0.17
Other	0.27
Repair	
Personnel	0.76
Supplies	0.88
Power	2.90
Heating/Drying	0.47
	10.18

1 From "Cyprus Anvil Mining Corporation, Summary of Mill Cost Estimates"

11,160 mtpd
Zone 3 Ore

TABLE 4.10

PAH ESTIMATED OPERATING COSTS
BY YEARS
CYPRUS ANVIL MINE

Year	Milling Rate (MTPD)	Percent Oxide Ore in Feed	Mill Operating Cost
1984	7,000	100	13.15
1985	9,870	23	10.72
1986	11,160	0	10.09
1987	11,160	0	10.09
1988	11,160	0	10.09
1989	11,160	0	10.09
1990	11,160	0	10.09
1991	11,160	0	10.09

PAH agrees with the CAMC projected operating cost with the following exceptions:

- The cost of mine power must be removed.
- There is a potential for higher labor requirements in the early years.
- Freight savings must be guaranteed from back haul credits.
- There will be increased costs for processing tonnages below 11,160 mtpd.
- Adjustment of operating costs based on percent of oxide ore in feed is necessary.

The power cost of $2.90 per tonne includes power used in the mine, maintenance, mill and G&A. The plant operating cost was reduced by $0.09 per tonne by removing the power cost for the mine. This cost is included in the PAH mining cost estimate.

In the early years, 1984 and 1985, it may be necessary to increase the number of maintenance personnel, and in particular the electrician/instrumentation personnel, to allow for the in-plant modifications. While the major modifications may be accomplished by outside contractors, the CAMC maintenance staff will undoubtedly feel an extra burden. However, PAH has not altered the personnel represented based upon the above. Additionally, the acceptance by PAH of the projected mill manpower is dependent upon the satisfaction of the assumptions listed in Table 4.12 (Section 4.7). Critical among these assumptions is the completion of plant modifications.

The operating costs in Tables 4.9 and 4.10 assume a $100 per tonne reduction in freight charges via a back haul credit. This adjustment must be confirmed at a later date.

The mill operating cost varies with throughput rate because some cost items remain constant. The items that remain constant are labor costs and fixed power charges. To compensate for the lower milling rates in 1984 and 1985, the operating cost (Table 4.9) was modified to reflect constant labor costs and the influence of the 6,015,000 kwh per month of power that does not change with milling rate. The increase in milling costs for 1984 and 1985 are as follows:

	1984	1985
Milling Rate (mtpd)	7,000	9,870
Increased Power Cost ($/mt)	0.65	0.09
Increased Labor Cost ($/mt)	1.12	0.25

The treatment of oxide ore has a higher reagent cost, \$3.40 per tonne versus \$2.11 per tonne for Zone 3 ore. The yearly operating costs therefore have to be adjusted to compensate for the percent of oxide ore milled. The correction for the added cost of processing oxide ore is as follows:

	1984	1985
Added reagent cost (\$/mt)	1.29	0.24

4.5 Water Supply

The interoffice correspondence of May 10, 1984 from B. A. Arsenault to John Maissan, titled Mill Water Requirement from External Sources, indicates that 7,500 USGPM must be supplied to the plant for a proposed ore tonnage of 11,500 metric tons per day. The memorandum presents a realistic analysis of the plant water requirements. The data indicate that 0.652 USGPM are required per metric ton of ore treated per day.

During operations in November and December of 1984, the ore tonnage is projected to be an average of 7,000 metric tons per day. The water requirements would then average 4,564 USGPM. On an annual basis this is equal to the present allowed water usage under the license agreement of two billion Imperial gallons per annum. The existing water reservoir dam and water system are capable of delivering this average rate to the plant. Projections of water usage indicate that the license must be modified to allow a usage of 7,500 USGPM or 3.3 billion Imperial gallons per year, and an increase in the volume of water storage must be provided. An alternative to an increased fresh water supply is the installation of the tailings water recycle system described in the memo of May 31, 1984 to D. Gregoire from F. J. Meyer, titled Reclaim Water Pump Station. One of the alternatives should be selected and work should start on the system so that it will be in place by the summer of 1985. Based on the projected ore tonnage of 135,000 metric tons for November and 292,000 tons for December of 1984, the average water rate in December will be 6,141 US gallons per minute. The allowed instantaneous water rate under the license is 7.21 million Imperial gallons per day or 6,000 US gallons per minute. Presumably, the difference between the water delivered from the reservoir and the plant requirement will be made up from the $200,000 system to be installed to deliver water from the 150 million Imperial gallons stored in the bottom of the mine pit. This system is to deliver 750 USGPM to the plant. During December, then, 750 USGPM will be withdrawn from the mine pit and 5,391 USGPM from the reservoir.

During the period from January 1 to May 31, 1985, an average of 6,000 USGPM will be required for plant operations, 750 USGPM from the mine pit and 5,250 USGPM from the reservoir. At the projected withdrawal rate of 750

USGPM from the mine pit the 150 million Imperial gallons stored there will last until the last half of May. The selected new water system must be installed and operating by that time. Either the increase in the size of the existing reservoir system or the tailings water recycle system are estimated to cost $1,800,000 by CAMC.

4.6 Mill Modification Schedule

At the time of the PAH review, there was no unifying document describing the modifications to be made, the time schedule for each and a definitive cost estimate. In addition to the reports supplied to PAH on proposed modifications (Appendix A) a handwritten four-page outline was prepared by H. M. Visagie and John Maissan listing major items (Table 4.11). The analysis of the proposed modifications and schedule is based upon the above documents. No effort has been made by PAH to check the accuracy of the CAMC capital cost estimates.

Utilizing the cited reports, PAH attempted to develop a schedule for performing the proposed modifications. The effort proved futile because of a lack of detailed information. Certain sections of the plant modification program, such as electrical and instrumentation, are well documented. For other sections, like the approximately $1,000,000 of routine maintenance, PAH has little or no backup. In the following text PAH comments generally on how the program can be scheduled and on areas where some concern exists as to the proposed modifications.

Schedule

The current, preliminary schedule calls for the following to be performed prior to startup:

General Maintenance			$1,060,000
Modifications -	Chutes	$ 20,000	
	Screens	40,000	
	Pumps	50,000	
	Vacuum/Filtrate	100,000	
	Lime	100,000	
	Water	100,000	
	Tailing	200,000	
			610,000
Electrical/Instr. -	Crusher	25,000	
	Dewatering	200,000	
			225,000
TOTAL (w/o Contingencies or Indirect Costs)			$1,895,000

TABLE 4.11

PLANT MODIFICATIONS AND COSTS ($\$X10^3$)
CYPRUS ANVIL MINE

Area	To Be Performed Prior to Startup	To Be Performed During Operation	To Be Performed During a Shutdown
Crushing	Chute modifications ($20) Double-deck screens ($40) Instrumentation ($25) Maintenance ($360) Contingency ($175)	Rock breaker ($200) Electrical/Instr. ($300)	Apron feeder pocket ($50) Replace 30-In. conv. ($320) Feed chute 2008 ($50) Instrument. ($175) Floor ($100)
Grinding	Maintenance ($200) Contingency ($75)	Variable speed pumps ($200) Chutes ($150) Instrum. ($75) Piping ($50)	Final instrum. ($50)
Flotation	Maintenance and Launders ($200) Contingency ($100)	Zinc conditioners ($250) Instrument. ($100)	New regrind cyclones ($150) Instrument. ($100)
Dewatering	Instrument. ($200) Pumps ($50) Vac./Filtrate System ($100) Maintenance ($100) Loadout Maint. ($100) Contingency ($75)	Instrument. ($50)	Instrument. ($100)
Oil Conversion		Complete system ($600) Contingency ($100)	
Water Supply		Complete system ($1,800)	
Internal Water Reagents	Maintenance ($100) Lime system ($100) Water system ($100) Contingency ($100)	Water system ($100)	
Tailings Disposal	New dump point ($200)		

Indirect Costs are estimated to be $1,010,000 in addition to the above costs. All costs are CAMC Estimates

Assuming that the general maintenance cost is 50 percent labor and that the capital improvement costs are 25 percent labor, the required man months of labor to perform the above are approximately 150. Therefore, the proposed maintenance staff of 57 personnel would require three months to perform the necessary work. Obviously, outside contractors must be utilized, pending union approval, to accomplish the desired work in a reasonable period of time.

In addition to the proposed work to be accomplished prior to startup, PAH recommends that the following also be completed:

Installation of variable speed pumps in grinding area	$ 200,000
Zinc conditioners	250,000
Fuel oil conversion on dryers	100,000
Mine water system	100,000
Chute modification in grinding area	150,000
TOTAL (w/o Contingencies or Indirect Costs)	$ 900,000

The above extra items would add 48 man months of effort, for a total of nearly 200 man months. This effort, plus the time to marshal the manpower, co-ordinate delivery of materials and staff for restart would indicate a startup is technically feasible in December, 1984. Critical to meeting this time frame would be the development of an accurate, detailed project schedule.

The remainder of the modifications can easily be spread through the initial six to eight months of 1985. ?

Modifications

PAH has examined the modification reports for the individual plant areas. The reports demonstrate that considerable effort has been spent in identifying the means of improving the plant. The following item delineates those areas where PAH feel additional study is required.

Replacement of 30-Inch Conveyor Belts

It is proposed to replace the 30-inch fine ore conveyors with 42-inch conveyors. This is deemed necessary to prevent spilling at higher throughput rates. As mentioned earlier (Section 4.2) the design criteria for these conveyors seem unduly conservative. The speed of the 30-inch conveyors could be increased to 557 fpm at a reduced capital expense, while supply 1,310 mtpd capacity (including 20 percent surge). While this speed is slightly higher than recommended by CEMA, it is not unreasonable, particularly for a plant which could have a short life or may be sold. If the speed increase for the 30-inch belt is rejected, and we believe 30-inch belts should be used, 36-inch belts could be used with an accompanying 15 percent capital savings over 42-inch belts.

Instrumentation

The following costs are allocated for instrumentation/electrical improvements in the concentrator:

Crushing	$ 500,000
Grinding	125,000
Flotation	200,000
Dewatering	400,000
TOTAL (w/o Contingencies or Indirect Costs)	$ 1,225,000

This is a significant cost for modification of the instruments. We believe that the necessity for the instrumentation package is not fully justified. In the crushing circuit the new instrumentation is to allow for a two-man operation (one central control room). The PAH staff experience in similar crushing facilities (with and without central control rooms) is that is is difficult to eliminate the third person simply with instrumentation. Before initiating the current mill instrumentation program, each item should be economically justified.

Regrind Hydrocyclone

The volume of pulp generated at higher milling rates will exceed the capacity of the existing regrind cyclones. CAMC has suggested that the next larger cyclones be installed. Prior to this change it must be ascertained that the larger cyclones can give the desired product size distribution. It is more difficult to make a fine split with a 15-inch diameter cyclone than with a 10-inch.

4.7 Mill Operation

The Mill manpower requirements covered in the memo to D. Gregoire from J. M. Maissan of May 8, 1984, as revised, provide for a total mill operating crew of 145 people, including staff. This memo accurately represents the requirements for a steady plant operation, if the underlying assumptions, included as part of that memo, are met. A copy of those assumptions are included here as Table 4.12.

Among the more important of the assumptions used to estimate the mill manpower are the completion of the plant modifications to allow for reduced labor requirements in the operating crew. If the necessary plant modifications are not completed by plant startup, additional operators will be added. As noted in the assumptions, five people should be allowed for vacation relief and for training resulting from labor turnover has not been allowed. For the purposes of the operating cost estimate, PAH has allowed the 145-person base case presented in the Summary of Mill Operating Cost Estimates prepared by H. M. Visagie in July, 1984.

TABLE 4.12

UNDERLYING ASSUMPTIONS TO
MILL MANPOWER REQUIREMENTS
CYPRUS ANVIL MINE

1) A maximum instantaneous mill feed rate of 12,065 mtpd (11,160 mtpd at 92.5 percent availability) was considered.

2) Vacation relief was not specifically allowed for.

3) Apprentices are not included in the numbers.[1]

4) All maintenance - mechanical and electrical was assumed to have been completed.

5) Projects of significant size would/may require additional people.

6) Operators and attendants are responsible for clean-up in addition to their other duties.

7) A new control room and control system would be required for the crushers so that they can be operated by two men.

8) Installation of a rock breaker, a change in conveyors from 30 inch to 42 inch, a redesign of chutes to minimize blockages and wear, and reuse of double-deck screens to increase screening efficiencies would all be required.

9) Components of the fine ore feeding system (slot feeders, chutes) must be redesigned, and adequate supply and pressure of water for the grinding mills ensured to reduce spillage at feed and discharge ends.

10) A cyclone feed control system including variable speed pumps to minimize pump box spillage and improve cyclone performance is required.

11) Flotation changes required to reduce spillage and water consumption include splitter boxes, O.K. cell feed and intermediate boxes, and all concentrate launders (pipes).

12) Regrind cyclone clusters have to be changed to handle the larger (actual) feed rate they must handle and the grinding steel addition method must be altered to reduce ball spillage.

[1] Four apprentices have been added since the original manning table was prepared.

TABLE 4.12 (Cont.)

UNDERLYING ASSUMPTION TO
MILL MANPOWER REQUIREMENTS
CYPRUS ANVIL MINE

13) A complete revision of the dewatering system is required - including automatic density controls, installation of vertical pumps, automatic filter boot level control, and an improved filtrate/vacuum system.

14) Conversion from coal to oil with controls, in the heat generators and concentrate driers is required.

15) No major changes in the loadout to reduce the necessary manpower were assumed.

16) A conventional pressure lime unloading system would be required in addition to an improved soda ash loop for better pH control.

17) Also required is an augmented - increased volume and pressure - water supply to reduce the spills and blockages caused by an inadequate water supply volume or pressure. Internal recycle must be nil or minimal.

OLD CREW ?, NFG.

?

PAH regards the capabilities of the people in the present mill operating staff as more than adequate to operate the plant, assuming that the key positions not now filled will be staffed with people whose level of competence is similar to the staff members now on the site. It is also our understanding that a substantial number of the better people at the shift foreman and operator levels are available for rehire to provide a good base for an operating crew.

It must be noted that the manpower requirements of 145 will not include people working on the continuing modifications that will be taking place between the initial plant startup and the completion of the modifications. These people must be regarded as a separate group and can be included as part of the capital cost of the modifications.

4.8 Recommendations

If CAMC decides to proceed with the proposed startup, PAH recommends that:

1) A detailed schedule of the plant modifications be produced, including capital costs by item, job sequencing, manpower requirements, with a cost rationale for each proposed modification so that the efficacy of each modification can be assessed. It is our understanding that F. Meyer intends to produce a document by August 1 that will address at least some of these problems.

2) An evaluation of the effect of ore grade, grind and other variables on plant recoveries be made by ore type with sophisticated computer analysis utilizing plant, pilot plant and laboratory data.

These more precise evaluation of capital costs and recoveries will better define the economics of plant startup and operation, should lead to capital cost savings, and still achieve the tonnage rates currently projected.

5.0 CONCENTRATE MARKETING

5.0 CONCENTRATE MARKETING

5.1 Marketability

We have reviewed detailed concentrate analyses for the selective lead concentrates and selective zinc concentrates produced by CAMC in the past. We have reviewed correspondence in CAMC's files with potential customers for the concentrates and with consultants commenting on concentrate markets. We have discussed marketability with merchants and custom smelters, and reviewed pertinent material in our own files. It is our opinion that the concentrates are marketable to East Asian and western European smelting companies, but the marketing effort must be very carefully planned and executed.

The tonnages of concentrates that would be produced are very large. The annual tonnage of zinc concentrates is in the 250,000 to 310,000 DMT range, and the annual tonnage of lead concentrates is in the 125,000 to 190,000 DMT range. Such large tonnages coming on the market, if not handled carefully, could result in less favorable terms (treatment charges) than might otherwise be obtained.

The CAMC zinc concentrate is not a high quality concentrate because of its relatively low zinc content, roughly 50 percent to 51 percent zinc, and its high mercury content, over 300 ppm. Certain treatment plants can handle large quantities of high mercury material because they have installed circuits in their plants to get rid of it. Other plants must blend this material with low mercury concentrates to achieve an acceptable mercury content. Some penalty for mercury can be expected. The amount of the penalty could depend to some extent on the buying smelters' perception of the difficulty CAMC will have in selling very large tonnages of concentrates.

Some smelters charge a penalty of US$1.50 or $2.00 per tonne of zinc concentrate smelted for iron content exceeding 10 percent, and in some cases, 8 percent. One plant is allowing 10 percent iron plus manganese free, and charging $5.00 per unit (1 percent) over 10 percent.

The CAMC lead concentrate is a relatively clean concentrate and should pose no marketing problem from the point of view of its composition. The mercury content is a little on the high side at 45 ppm. The low bismuth content (about 0.007 percent) is favorable.

It is our recommendation that if Dome decides to start up the mine, negotiations for smelting contracts should begin at once. It is not too early to start negotiations for 1985 tonnages. Substantive and continual contacts with the market will be required. We recommend that CAMC engage a consultant who has recent hands-on experience with marketing lead and zinc concentrates to assist with the marketing effort.

We have not considered marketing of the concentrates in the United States since costs of doing so would probably be very high compared to Asian and European costs. For one thing, shipping costs would be higher since the Jones Act would apply on ocean shipping from one U.S. port (Skagway or Haines) to another. In addition, teatment charges in the United States would be higher. Terms at one major U.S. zinc plant might be as follows:

1. Pay for 85 percent of the zinc, with a minimum deduction of 8 units, at the High Grade zinc price less US$0.02/lb of payable metal.

2. Treatment charge US$160-$165/DST ($176.37 - $181.88/DMT) based on a US settlement price of $0.51. Upscale $3.50/$0.01 price increase. Downscale $2.50/$0.01 price decrease.

5.2 Treatment Contract Terms

Zinc Concentrates

Payable Metal

Standard terms in zinc smelting contracts call for payment for 85 percent of the contained zinc, with a minimum deduction of eight units (8 percent) from the zinc assay. The smelter pays for 100 percent of the metal remaining after the deduction. We have used these standard metal deduction terms in our cash flow projections.

The zinc concentrates do not contain payable metals other than zinc.

Treatment Charges

940

At the present European Producer Price of US$990/MT for zinc, the range of teatment charges is in the range of US$150 to $160/DMT cif Japanese or western European ports. Most contracts call for an increase or decrease of $3.50 or $3.00 per DMT in the treatment charge for each US$0.01 increase or decrease in the European Producer Price. There will be a floor treatment charge on the downside.

As noted above, it is most likely that some penalty will be charged for mercury. The amount of the charge, like the rest of the terms, will be subject to negotiation.

No iron penalty will be incurred if the iron content is below 8 percent. If it exceeds 10 percent a penalty of US$1.50 or $2.00 per unit in excess of 10 percent could be incurred.

<u>Lead Concentrates</u>

<u>Payable Metal</u>

We have used standard terms of payment for 95 percent of the contained lead, with a minimum deduction of three units (3 percent) from the lead assay. The smelter pays for 100 percent of the lead remaining after the deduction.

Normal 50 grs./tonne

The lead concentrates contain payable silver. We have used a deduction of one troy ounce per DMT of concentrate, with payment for 100 percent of the remaining silver. The silver deduction varies considerably. Examples are:

deduct 30 grams, pay for 100 percent of the balance

deduct 50 grams, pay for 100 percent of the balance

pay for 95 percent

deduct 1 troy oz, pay for 95 percent of the balance

Treatment Charges

33-35%

We have used a treatment charge of US$150/DMT cif Japanese or western European ports in our cash flow projections. The market varies from about $145 to $155, and might touch as low as $140. There will typically be a clause calling for an increase of US$2.50 or $3.00/DMT for US$0.01 increase in the lead price above a base LME price stated in the contract. The clause will also call for a decrease of US$2.00 or $2.50/DMT for each US$0.01 decrease in the lead price below the base price.

The experience of CAMC has been that certain smelters levied an explicit silver refining charge, while others did not. We have used a silver refining charge of US$0.25 per payable troy ounce of silver, but have only charged it against the payable silver content of half of the lead concentrates in our cash flow projections.

Agent Fee 0.5 to 0.7% of NSR

5.3 Other Marketing Costs

Certain other costs will be incurred for such items as draft surveying of vessels, representation at destination ports, umpire sampling and assaying, etc. We have used a cost of US$1.50/DMT for this cost, based on CAMC's past experience.

5.4 Losses in Transport

Low

We have assumed that 0.50 percent of the concentrates produced will be lost in transportation and handling. This is based on past experience of CAMC.

5.5 Moisture Content

We have assumed moisture contents of 6.5 percent for the zinc concentrate and 6.0 percent for the lead concentrate based on CAMC's past experience.

6.0 CONCENTRATE TRANSPORTATION

6.0 CONCENTRATE TRANSPORTATION

6.1 Overland Transportation

PAH has reviewed three alternative methods of transporting concentrates to ports for loading on ships for transport to Europe and Asia. These are:

1. truck haulage from Faro to Skagway, Alaska;
2. truck haulage from Faro to Haines, Alaska;
3. truck haulage from Faro to Whitehorse, Yukon, and rail haulage from Whitehorse to Skagway.

Method 3 is the method which has been used in the past and is the method which CAMC would be obliged to use unless permission can be obtained from the Canadian Transport Commission and the Alaskan state government to truck over the Faro-to-Skagway (South Klondike) road or the Faro-to-Haines road. CAMC is actively seeking to obtain such permission, but has been advised recently by the Alaskan state government that it does not favor use of the Skagway road. In our base case cash flow projections, we have assumed trucking to Haines.

As part of our evaluation of the Anvil mine we engaged Mr. Robert H. Hurst, a trucking consultant familiar with the roads in question, to drive the roads and prepare a brief report on the feasibility of using them. His report is included as Appendix B. His findings are that it is technically feasible to use either road. If CAMC has a choice, he recommends the Skagway route because of the shorter distance (288 miles shorter), better road conditions, fewer road closures during winter months, and less tourist traffic.

The concentrate transportation costs for each alternative are estimated to be as shown here:

trucking Faro to Skagway	$45.89/WMT
trucking Faro to Haines	52.00/WMT
trucking & rail to Skagway	58.00/WMT

The cost of trucking to Skagway is based on a cost of $0.13/tonne-mile for a distance of 353 miles, and was provided by CAMC based on bids and CAMC estimates. The cost of trucking to Haines is based on a bid received by CAMC, and amounts to about $0.11 tonne-mile for a distance of 470 miles. The cost of combined truck and rail haulage is based on a recent tentative estimate by the White Pass and Yukon Railway for the first quarter of 1985.

The cost savings which could be achieved by trucking all the way from Faro to Skagway instead of trucking to Haines, and the additional cost of trucking and rail to Skagway instead of trucking to Haines are shown here.

	Cost Reduction by Trucking to Skagway			Additional Cost of Truck and Rail to Skagway		
	Annual	Cost per Pound of Payable Metal		Annual	Cost per Pound of Payable Metal	
	(Million)	Zinc	Lead	(Million)	Zinc	Lead
1985	$2.4	$0.007	$0.005	$ 2.4	$0.007	$0.005
1986	2.8	0.007	0.005	2.8	0.007	0.005
1987	2.7	0.007	0.005	2.6	0.007	0.005
1988	3.2	0.007	0.005	3.1	0.007	0.005
1989	3.1	0.007	0.005	3.1	0.007	0.005
1990	3.1	0.007	0.005	3.0	0.007	0.005
1991	2.3	0.007	0.005	2.3	0.007	0.005

The cost differentials shown above for trucking to Skagway instead of Haines are based strictly on trucking cost differentials. As discussed below, port facilities would have to be constructed at Haines, while storage and shiploading facilities already exist at Skagway.

6.2 Port Facilities

In the past CAMC's concentrates were stored and loaded onto ships at the White Pass and Yukon terminal at Skagway. Based on actual costs in 1982, CAMC has estimated that the annual cost of using the terminal at Skagway would be $3,120,000. In 1982 CAMC concentrates were arriving at the terminal in rail cars. If trucking is permitted, minor modifications to the terminal would be required for receipt of concentrates by truck. CAMC has assumed that no front-end capital charge would be made by White Pass and Yukon for these modifications.

In the case of trucking to Haines, port facilities would have to be constructed at Haines. No detailed estimate has been made of the cost of constructing the terminal, but CAMC and PAH concur that a figure of $5 million is reasonable. CAMC has assumed that a contract would be made with an outside party to construct and operate the terminal, and that such party would recover their investment in the terminal through charges to CAMC over some amortization period. There is no operating cost estimate for the Haines port facility. In our base case cash flow projections we have used the same annual figure as was estimated for the Skagway terminal, and that this amount would be ample to cover amortization of the terminal construction costs.

6.3 Ocean Freight

We have used an average ocean freight cost of US$18.00/WMT (Cdn$24.00) of concentrate for ocean freight. The actual cost should be somewhat less for shipment to East Asian ports and somewhat higher for shipment to western European ports.

7.0 FINANCIAL EVALUATION

7.0 FINANCIAL EVALUATION

7.1 Ore Reserve and Production Schedule

The ore reserves used for this evaluation are:

		Zn	Pb	Ag
Pit ore	26,432,000 DMT	4.3%	2.9%	36.1 g/t
Stockpiles	1,400,000 DMT	4.7%	2.9%	37.6 g/t
Total	27,832,000 DMT	4.3%	2.9%	36.2 g/t

Table 7.1 shows the production schedule used in this evaluation.

There is no comprehensive current reserve estimate for the Faro deposit, nor is there a detailed life-of-mine mining plan. PAH has based its evaluation on the best reserve information available. We believe that the reserve figures we have used are reasonable for this evaluation.

There were minor discrepancies between the ore production schedule developed by CAMC's mining staff and the ore milling schedule used by the CAMC metallurgical staff. These were primarily in 1985 and 1991. In order to make the numbers match, PAH has made some necessary adjustments to these numbers, none of which in themselves have a

TABLE 7.1

PRO FORMA PRODUCTION STATEMENT
CYPRUS ANVIL MINE

PRODUCTION YEAR	1984	1985	1986	1987	1988	1989	1990	1991
ORE MILLED, DMT	427000	3604000	4073000	4073000	4073000	4073000	4073000	3436000
WASTE MINED, DMT	4355000	23442000	22398000	21466000	8233000	2642000	1151000	963000
STRIP RATIO	10.20	6.50	5.50	5.27	2.02	0.65	0.28	0.28
ZN ASSAY OF ORE MILLED, %	4.7	4.4	4.4	4.0	4.5	4.7	4.6	4.1
MILL RECOVERY OF ZN, %	67.90	79.40	81.80	81.30	82.90	82.00	82.70	81.50
ZINC CONCENTRATE, WMT	30235.	267118.	306125.	276054.	319801.	329087.	324835.	240230.
ZINC CONCENTRATE, DMT	28390.	250815.	287441.	259205.	300283.	309002.	305010.	225568.
ZN ASSAY OF ZN CONC, %	48.0	50.2	51.0	51.1	50.6	50.8	50.8	50.9
TRANSPORT & HANDLING LOSS FOR ZINC, %	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
SMELTER PAY FOR ZN, %	83.3	84.1	84.3	84.3	84.2	84.3	84.3	84.3
COMPOSITE RECOVERY FOR ZN, %	56.30	66.41	68.62	68.23	69.44	68.74	69.33	68.35
ZN PAID FOR/TON ORE MILLED, LBS	58.34	64.42	66.57	60.17	68.89	71.23	70.31	61.78
PAYABLE ZINC, LBS	24909895.	232176571.	271125472.	245061074.	280603228.	290106794.	286358782.	212269326.
PB ASSAY OF ORE MILLED	2.9	2.7	2.8	2.8	3.4	3.2	3.0	2.4
MILL RECOVERY OF PB, %	65.20	77.40	80.60	81.00	82.60	82.10	81.30	79.60
LEAD CONCENTRATE, WMT	14756.	132177.	159467.	158701.	199751.	185945.	175501.	116159.
LEAD CONCENTRATE, DMT	13921.	124695.	150440.	149718.	188445.	175420.	165567.	109584.
PB ASSAY OF PB CONC, %	58.0	60.4	61.1	61.7	60.7	61.0	60.0	59.9
TRANSPORT & HANDLING LOSS FOR LEAD, %	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
SMELTER PAY FOR PB, %	94.8	95.0	95.0	95.0	95.0	95.0	95.0	95.0
COMPOSITE RECOVERY FOR PB, %	61.52	73.16	76.19	76.57	78.08	77.61	76.85	75.24
PB PAID FOR/TON ORE MILLED, LBS	39.33	43.55	47.03	47.26	58.52	54.75	50.83	39.81
PAYABLE LEAD, LBS	16794858.	156950873.	191549834.	192502176.	238368774.	222989605.	207014442.	136777139.
AG ASSAY OF PB CONCENTRATE, TR OZ/DMT	16.9	18.2	16.6	18.3	18.8	17.9	16.3	15.0
PAYABLE AG, TR OZ/DMT CONC, TROY OUNCES	15.9	17.2	15.6	17.3	17.8	16.9	15.3	14.0
PAYABLE SILVER, TR OZ	220232.	2134036.	2335134.	2577171.	3337546.	2949769.	2520504.	1526509.
TR OZ AG PAID FOR/LB PAYABLE PB	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
LBS ZN & PB/TON ORE MILLED	97.7	108.0	113.6	107.4	127.4	126.0	121.1	101.6
TOTAL PAYABLE LBS OF LEAD AND ZINC	41704752.	389127444.	462675306.	437563249.	518972002.	513096399.	493373224.	349046466.

significant impact on the project economics. For example, in 1985 we have assumed milling of 3,604,000 DMT of ore. This figure is taken directly from a CAMC milling schedule. It was about 94,000 DMT more than the mining schedule showed. The ore tonnage mined in 1985 was adjusted upward to reflect the milling tonnage requirement. Naturally there is some uncertainty associated with the waste tonnage that will be mined in the last year or two. We believe, however, that the figures used in this evaluation are reasonable in light of everything known about the ore body at this time.

The production schedule is based on the mining schedule discussed in Section 3.0, mill recoveries discussed in Section 4.0, smelter payable metals, and other factors discussed in Sections 5.0 and 6.0, and summarized again briefly in this Section 7.0.

7.2 Metal Prices

PAH has projected cash flows from operations for current zinc, lead, and silver prices. These are as follows:

		US$	Cdn$
Zinc/lb., EPP	43	0.45	0.60
Lead/lb., LME	18	0.22	0.29
Silver/troy oz., LME		7.50	10.00

7.3 Sales of Concentrator Products

All metals are assumed to be paid for in the year in which they are produced. In fact, there would be a time lag between production of concentrates and partial and final payments by the smelter. We have allowed for $10 million working capital at the outset of milling to reflect the time lag between concentrate production and payment.

7.4 Treatment and Freight Costs

Smelter Deductions

Zinc Concentrates

The only payable metal in the zinc concentrates is zinc. We have used standard terms of a deduction of eight units from the zinc assay, or 15 percent of the zinc content, whichever is greater.

Lead Concentrates

In addition to the lead, the lead concentrates contain payable silver. We have used standard terms of payment for 95 percent of the lead, with a minimum deduction of three units. We have used a deduction of one troy ounce of silver per DMT of concentrate to estimate payable silver.

Treatment Costs

Zinc Concentrates

We have used a base treatment charge of US$155/DMT of concentrate CIF Japanese or western European ports at a European Produer Price of US$990/MT (US$0.45 per pound) of payable zinc. A mercury penalty of US$5.00/DMT has been used. No other penalties, e.g., iron, have been taken into account.

Lead Concentrates

We have used a treatment charge of US$150/DMT of concentrate CIF Japanese or western European ports. For the silver in the concentrate we have used a refining charge of US$0.25 per payable ounce for one-half of the payable silver. This reflects an assumption that half of the lead concentrates would go to smelters who levy a refining charge for the silver, and half would go to smelters which do not.

Land Freight

We have assumed trucking all the way from the concentrator to the port at Haines, Alaska. We have used a cost of $52.00/WMT of zinc or lead concentrate. This is based on a bid received by CAMC. We have assumed a moisture content of 6.5 percent for the zinc concentrate and 6.0 percent for the lead concentrate.

CAMC is attempting to obtain permission to truck concentrates from Faro to Skagway, Alaska. The Alaskan government has recently denied permission to do this at least for the immediate future. Should CAMC be successful in obtaining permission to truck over the Skagway route, the cost is estimated at $45.89/WMT of concentrate.

Port Charge

CAMC has estimated that the annual cost of using the existing terminal at Skagway would be $3,120,000. This is based on actual costs prior to shutdown of the Anvil mill in 1982. Our cash flow projection assumes trucking to Haines. A terminal will have to be built there. There is no operating cost estimate for a terminal at Haines. After discussions with CAMC we have agreed that it is reasonable for this evaluation to use the same annual cost for a terminal at Haines as the cost estimated for the Skagway terminal.

Ocean Freight

We have used US$18.00/WMT of zinc or lead concentrate for ocean freight. The actual cost should be somewhat less for shipment to Japan and somewhat higher for shipment to Europe.

Agent Fee

Other Costs

A cost of US$1.50/DMT of concentrate smelted has been used to cover representation at destination ports, draft surveying, umpire sampling and assaying, etc.

7.5 Cash Production Costs of Concentrate Production

Cash production costs of concentrate production include costs of mining, milling, general and administrative costs, townsite expenses, and Vancouver and Calgary overhead. Mining and milling cost estimates are discussed in more detail in Sections 3.0 and 4.0 of this report.

Mining Cost

The mine operating costs are shown in Table 7.2 These costs assume an electric power cost of $0.07/kwh and backhauling by truck of certain operating supplies. The

TABLE 7.2

MINING COST PER*
CYPRUS ANVIL MINE

Year	Cost Per DMT of Rock Mined	Total Tonnes Mined (Millions)	Annual Cost (Millions)
1984 (Nov-Dec)	$0.99/DMT**	4.8	$ 4.7
1985	1.05	27.0	28.5
1986	1.23	26.5	32.6
1987	1.24	25.6	31.8
1988	1.59	12.3	19.5
1989	1.87	6.7	12.6
1990	2.10	5.2	10.9
1991	1.95	4.4	8.6
Avg.	$1.33	Total 112.5	Total $149.2

*Includes rehandling of stockpiled ore.
**Rounded from Table 3.42.

savings due to backhaul were estimated based on use of the port at Skagway. If the port at Haines is used, the full reduction in tire, fuel, and explosives costs may not be achieved.

Milling Cost

The mill operating costs are shown in Table 7.3. The costs in 1984 and 1985 reflect the lower milling rate and the milling of the oxide ores.

General and Administrative and Townsite Expense

We have used for this evaluation an estimated annual amount of $10,331,000. The estimate on which this is based was provided by CAMC. PAH has made a slight modification of the CAMC estimate. The modified estimate is shown in Table 7.4. The estimate takes into account a federal credit of $364,000 for an apprenticeship program. CAMC has explained that if certain objectives are successfully negotiated in the new union contract, including payment by employees of a higher share of housing and utility costs, townsite expense could be reduced. Until probable results of labor contract negotiations become more clear, however, it has been thought best by PAH to leave the figure at the present level. This opinion is concurred in by Mr. Thurmond.

TABLE 7.3

MILLING COST PER TONNE OF ORE MILLED
CYPRUS ANVIL MINE

1984	$13.15/DMT
1985	10.72
1986-91	10.09

TABLE 7.4

GENERAL AND ADMINISTRATIVE AND TOWNSITE COST[1]
CYPRUS ANVIL MINE

Personnnel	Number of Personnel		Cost
Administration	2		$ 144,000
Engineering	5		300,000
Safety & security	7		354,000
Environment	3		169,000
Personnel	8		470,000
Townsite	13		776,000
Accounting	13		692,000
Purchasing	14		760,000
	65	Subtotal	$3,665,000
Supplies and Other			
Administration			$ 2,713,000
Engineering			480,000
Safety & security			118,000
Environment			107,000
Personnel[2]			563,000
Townsite			2,517,000
Accounting			146,000
Purchasing			22,000
		Subtotal	$7,030,000
	Total		$10,331,000

[1]Provided by CAMC, except PAH has added two persons at $55,000 each to personnel.

[2]$927,000 cost net of a credit of $364,000 for government reimbursement of costs of an apprenticeship program.

The estimate figure of $10,331,000 includes municipal taxes paid for maintenance of roads in the townsite and similar services.

We have used a full year's cost for 1991 although the mill is only projected to run a little over ten months of that year.

Vancouver and Calgary Overhead

CAMC has budgeted an amount of $1,800,000 per year for this cost. PAH has used this figure.

7.6 Capital Expenditures

CAMC has budgeted capital expenditures for 1984 and 1985 for the Anvil mine, mill and townsite as shown in Table 7.5. PAH has reviewed these estimates and has used them in the cash flow projection.

PAH also briefly reviewed the replacement capital expenditures which had been budgeted by CAMC for 1986-1991. After review with CAMC, replacement capital expenditures are assumed to be as shown here for years after 1985.

TABLE 7.5*

ANVIL MINE CAPITAL EXPENDITURES
CYPRUS ANVIL MINE

	1984	1985
Mine		
1. drill		$1,800,000
2. pit dewatering		100,000
3. remove unstable mat'l threatening east wall of pit, ± 1,000,000 yds	$1,500,000	
4. exploration drilling		500,000
5. light vehicle fleet		200,000
6. crusher for gravel		225,000
7. computer system/engineering	150,000	
8. computer software	100,000	
subtotal	$1,750,000	$2,825,000
Mill		
1. expansion/modifications	$4,956,000	$3,398,000
2. replacement capital		400,000
subtotal	$4,956,000	$3,798,000
Townsite		
1. Chateau Jomini		$ 900,000
2. repair houses		274,000
3. demothball houses		100,000
subtotal		$1,274,000
Recruiting	$1,200,000	$ <600,000>
Environmental		96,000
Total	$7,906,000	$7,393,000

*As budgeted by CAMC. Mine and mill areas reviewed by PAH.

	1986-1989 per year	1990 per year	1991 per year
mine	$200,000	$100,000	$ -
townsite	100,000	100,000	100,000
environmental	200,000	200,000	200,000
miscellaneous	100,000	100,000	100,000
	$600,000	$500,000	$400,000

The above figures do not include any amounts for replacement capital for the mill since it is not possible at this time to assign benefits to such capital expenditures. No major tailings project is needed unless the Vangorda Plateau deposits are developed. There should be no need to purchase major mining equipment with the exception of the one blasthole drill shown in Table 7.5 unless the Vangorda Plateau deposits are developed.

7.7 Cash Flow from Operations

Table 7.6 shows the cash flow from operations. It will be noted that the table begins with 1984. Only the last two months of 1984 are included. The reason for beginning the schedule at November 1, 1984 is that all mining schedules, milling schedules, etc., prepared by CAMC have been built around that date. For the purpose of this evaluation it is not necessary to develop an entirely new schedule. It would have been far too time-consuming. The impact of a later startup is primarily to incur additional holding costs and to slip cash flows back in time. For

TABLE 7.6

PRO FORMA CASH FLOW STATEMENTS
CYPRUS ANVIL MINE
ZINC PRICE = $ 0.60/POUND
LEAD PRICE = $ 0.29/POUND
SILVER PRICE = $ 10.00/TR OZ
ALL DOLLAR AMOUNTS ARE SHOWN IN CANADIAN DOLLARS

CASH FLOW YEAR	1984	1985	1986	1987	1988	1989	1990	1991
SALES OF ZINC	14945937.	139305942.	162675283.	147036644.	168361937.	174064076.	171815269.	127361596.
SALES OF LEAD	4870509.	45515753.	55549452.	55825631.	69126944.	64666986.	60034188.	39665370.
SALES OF SILVER	2202323.	21340365.	23351337.	25771706.	33375460.	29497695.	25205041.	15265094.
TOTAL SALES	22018768.	206162060.	241576073.	228633981.	270864342.	268228757.	257054499.	182292060.
-ZINC SMELTING	5994996.	52964260.	60698623.	54736123.	63410333.	65251590.	64408578.	47632889.
-LEAD SMELTING	2770217.	24814377.	29937612.	29793880.	37500517.	34908515.	32947767.	21807277.
-SILVER REFINING	36338.	352116.	385297.	425233.	550695.	486712.	415883.	251874.
-CONCENTRATE FREIGHT, OCEAN	1079780.	9583075.	11174197.	10434118.	12469259.	12360767.	12008068.	8553337.
-CONCENTRATE FREIGHT, LAND	2339524.	20763329.	24210760.	22607255.	27016729.	26781661.	26017480.	18532231.
-PORT CHARGES	520000.	3120000.	3120000.	3120000.	3120000.	3120000.	3120000.	3120000.
-OTHER CONCENTRATE COSTS	84197.	747265.	871304.	813758.	972568.	963999.	936447.	666953.
-MINING COSTS	4717921.	28402143.	32601684.	31770516.	19529622.	12577867.	10960997.	8568372.
-MILLING COSTS	5615050.	38634880.	41096570.	41096570.	41096570.	41096570.	41096570.	34669240.
-GENERAL AND ADMINISTRATIVE	1721800.	10331000.	10331000.	10331000.	10331000.	10331000.	10331000.	10331000.
-CALGARY,VANCOUVER OVERHEAD	300000.	1800000.	1800000.	1800000.	1800000.	1800000.	1800000.	1500000.
-CAPITAL EXPENDITURES	7906000.	7393000.	600000.	600000.	600000.	600000.	500000.	400000.
(-/+)WORKING CAPITAL	-10000000.	0.	0.	0.	0.	0.	0.	10000000.
OPERATING CASH FLOW	-21067055.	7176615.	24748947.	21105529.	52467048.	57950076.	52511710.	36258887.

COLUMNS MAY NOT FOOT IN LAST DIGIT DUE TO COMPUTER ROUNDING.

example, if a startup were scheduled for the second quarter of 1985, there would not be a great deal of difference in the time pattern of capital expenditure outlays. The working capital would shift into 1985. Cash receipts would be set back several months as would cash operating costs.

The costs in the table do not include depreciation of fixed asset balances presently on CAMC's books, depreciation of pre-startup capital outlays, interest on present or future debt, Yukon royalty, or federal income taxes.

While the table is largely self-explanatory, a few comments are in order. First we have estimated working capital requirements at between $9 and $10 million. Based on our discussions with CAMC staff, CAMC estimates $10 million would be required. We have shown $10 million as working capital required in 1984, and recovered in 1991.

The cost of stripping 4,355,000 DMT of waste in November and December 1984 is included as a mining cost in 1984.

Finally, the mining costs used to calculate the annual costs are taken from Table 3.42. They are not the rounded costs shown in Table 7.2.

7.8 Cash Costs per Pound of Payable Metal

Table 7.7 shows the cash cost per pound of payable zinc and lead for each year. The method of coproduct cost accounting we have used is to sum the yearly cash costs incurred up to the point at which the two concentrates, zinc and lead, are produced. These costs include mining, milling, general and administrative, townsite, and corporate overhead costs. That sum is then divided by the total pounds of payable zinc and lead produced in that year, to arrive at the cost per pound of zinc or lead. Silver is a byproduct of the lead concentrate and the silver credit is applied against the cost of lead.

7.9 Discussion of Results

As can be seen from inspection of Table 7.6 at present zinc, lead, and silver prices, and assuming that the power cost of $0.07/kwh is achieved, and truck haulage of concentrates from Faro to Haines is allowed with truck backhaul of operating supplies, the project could begin generating cash in 1985. Recovery of the incremental investment, including working capital, in the project could take place in about two years from startup. In 1988-91, as a result of a large drop in the stripping ratio and a significant increase in the zinc and lead grades of ore milled, the project has the potential to generate significant cash flows.

TABLE 7.7

CASH COST PER POUND OF PAYABLE METAL*
BASE CASE
CYPRUS ANVIL MINE

	1985	1986	1987	1988	1989	1990	1991
ZINC							
MINING, MILLING, G&A, TOWNSITE	0.204	0.186	0.194	0.140	0.128	0.130	0.158
LAND TRANSPORT	0.060	0.059	0.059	0.059	0.059	0.059	0.059
OCEAN TRANSPORT	0.037	0.035	0.035	0.034	0.034	0.034	0.037
SMELTING	0.230	0.226	0.225	0.228	0.227	0.227	0.227
	0.531	0.506	0.513	0.461	0.448	0.450	0.481
CAPITAL	0.019	0.001	0.001	0.001	0.001	0.001	0.001
TOTAL (CANADIAN $)	0.550	0.507	0.514	0.462	0.449	0.451	0.482
TOTAL (U.S. $)	0.413	0.380	0.386	0.347	0.337	0.338	0.362
LEAD							
MINING, MILLING, G&A, TOWNSITE	0.204	0.186	0.194	0.140	0.128	0.130	0.158
LAND TRANSPORT	0.044	0.043	0.043	0.044	0.043	0.044	0.044
OCEAN TRANSPORT	0.027	0.026	0.026	0.025	0.025	0.026	0.028
SMELTING	0.160	0.158	0.156	0.159	0.158	0.161	0.161
	0.435	0.413	0.419	0.368	0.354	0.361	0.391
SILVER CREDIT	0.134	0.120	0.132	0.138	0.130	0.120	0.110
	0.301	0.293	0.287	0.230	0.224	0.241	0.281
CAPITAL	0.019	0.001	0.001	0.001	0.001	0.001	0.001
TOTAL (CANADIAN $)	0.320	0.294	0.288	0.231	0.225	0.242	0.282
TOTAL (U.S. $)	0.240	0.221	0.216	0.173	0.169	0.182	0.212

The discounted cash flow return on investment (ROI) represented by the pre-tax cash flows in Table 7.6 is 78.7 percent. The cumulative net cash flow is $231 million. The net present value at a 15 percent discount rate is $114 million. These are pre-tax numbers. In view of the tax position of CAMC, it is unlikely that any taxes would be payable for some years, according to Dome management. That being the case the after-tax ROI would also be very high.

APPENDIX A

LIST OF DATA REVIEWED

FOR

MINING AND MILLING

APPENDIX A

Reference Documents

Mining

Mine basic information data has been obtained from the following:

1) Cost Detail General Ledger

2) Cyprus Anvil Mining Corporation 1984 Budget, 20,500 bcy/day, revised May, 1984.

3) Reports to D. Gregoire from R. Tolbert April 18, 1984, June 13, 1983.

4) Reports to J. Carrington from J. Purkis, Nov. 1982.

5) Cyprus Anvil Mining Corporation, "Information For Prospective Buyers", March 1984.

6) Budget proposal to support operations group start-up plan commencing July 1, 1984.

7) Cyprus Anvil Mining Corporation Equipment Numbering System.

8) Cyprus Anvil Faro Zone 3, Reserve Estimate, Hand Calculated, G. I. Hall, D. J. Slack, May 1, 1984.

9) Euclid Acceptance Trials, October 3, 1983.

10) Monthly Tire Report, June 4, 1984.

11) Description of Modeling Method and Model E-3, Letter Peter Clarke to Dr. A. G. Journer, November 9, 1981.

12) Combined Mechanical/Operations Group Schedule and Budget For Start-Up - to D. Gregoire from M. Nicholson, June 7, 1984.

13) Operations Department Monthly Report, May 1984.

14) "No. 4 Drill Reframe", D. Gregoire from M. Nicholson, June 6, 1984.

Reference Documents

Milling

The following documents were reviewed as part of the mill study.

1) Mill Monthly Reports for February and May, 1982. Also recoveries, concentrate grades, ore types and head assays for all of 1982 (supplied by telephone).

2) Metallurgical Forecasts (computer output) for 1984 through 1991. Program utilizes mine production data and metallurgical estimate of P. Taggart.

3) Summary of Mill Cost Estimates, July 1984 by H. M. Visagie.

4) Results of Mill Oxide Ore Test, March 11 through 15, 1982.

5) Mill Department Budget and Production Forecast, 1982.

6) Faro Zone 3 Metallurgy, pages 5 through 85 of report dealing with recovery and milling rate.

7) "Sensitivity Analysis of March 20, 1984" (Memo), April 11, 1984 by J. Maissan.

8) "P. Taggart Review Memo on Toronto Dominion Bank Report" (Memo), April 28, 1982 by P. Taggart.

9) "Metallurgical Test Work Update" (Memo), October 6, 1983 by J. Levanaho.

10) "Metallurgical Test Work" (Memo), November 16, 1982 by P. Taggart.

11) "Metallurgical Performance" (Memo), February 11, 1983 by P. Taggart.

12) "Ore-Types Locked-Cycle Tests: Discussion of Results" (Memo), April 15, 1983 by R. Murarka.

13) "Predictions of Recoveries at Fine Grinds" (Memo), January 10, 1983 by R. Murarka.

14) "Annual Cash Flow Inprovement with Fine Grind" (Memo), February 21, 1983 by R. Murarka.

15) "Reagents MATRIX Tests: S1 Blend" (Memo), June 22, 1983 by R. Murarka.

16) "Fine Grinding" (Memo), April 14, 1983 by P. Taggart.

17) Two page summary of laboratory pilot plant and mill results on different ore types. No date. No author.

18) "Heat Loss in the Dryer System" (Memo), December 30, 1982 by R. Murarka.

19) Process Group Report on the Grinding Circuit (no date).

20) Control System Concept - Grinding (no date).

21) Variable Speed Drives by B. McCaffery (no date).

22) "Summary of Decisions Reached at PID Meeting May 16, 1984" (Memo), May 16, 1984 by H. Gauthier.

23) "Vacuum System" (Memo), March 20, 1984 by F. Meyer.

24) "Zinc Conditoning" (Memo), May 31, 1984 by B. Arsenault.

25) "Grinding Circuit, Mill By-pass Arrangements" (Memo), April 20, 1984 by F. Meyer.

26) "Process Group Report on the Flotation Section" (Memo), April 10, 1984 by B. Arsenault.

27) "Final Grinding Reports from Unit Operation", (Memo), March 12, 1984 by F. Meyer.

28) "Upgrading of Plant Instrumentation" (Memo), March 20, 1984 by F. Meyer.

29) "CAML Concentrator-Grinding Drive" (Memo), May 17, 1984 by F. Meyer.

30) "Primary and Secondary Crushing Plant Report - Mr. M. Kay" (Memo), May 17, 1984 by F. Meyer.

31) "Reagents and Grinding Media" (Memo), November 28, 1983 by P. Cowell.

32) "Mill Manpower Requirements" (Memo), May 8, 1984 by J. Maissan.

33) "Design Calculations for Grinding Hydrocyclones" (Memo), March 22, 1984 by B. Arsenault.

34) Cyprus Anvil Mining Corporation - Mill Data Book prepared by Parsons-Jurden.

APPENDIX B

LETTER ON CONCENTRATE TRUCKING

ROBERT H. HURST

July 19, 1984

ROBERT H. HURST
2301 Broadway, #203
San Francisco, California
94115

19 July 1984

Mr. Robert Winkle
Pincock, Allen and Holt
1750 East Benson Highway
Tucson, Arizona

Dear Mr. Winkle:

You asked me to make an evaluation of the Whitehorse, Y.T. to Skagway, Alaska road as it would apply to the movement of concentrate from Faro. I made two roundtrips over this road in the course of this phase of the assignment.

Subsequently, you asked me to go to Haines, Alaska with the same objective. Only one roundtrip was involved in this phase.

Discussions were held with various people with regard to developing information as it might apply to the overall movement of the concentrate to ports mentioned above. This information includes the following -

- Description and condition of Whitehorse to Skagway highway;
- Description and condition of Whitehorse to Haines highway;
- Labor agreement changes;
- Equipment; and
- Discussions with White Pass & Yukon personnel and Lynden Transport personnel.

Whitehorse, Y.T. to Skagway, Alaska - 113 miles

The highway from Whitehorse to the Carcross turnoff is of a very high quality. From the turnoff to Carcross the road is good, by Yukon standards. Presently, there is construction going on for four kilometres, at a cost of $130,000 per Km. The surface is compacted dirt. There are several steep, over 7%, grades with some blind curves evident. This is especially true in the Spirit Lake region. From Carcross to the Canadian-U.S. border the road is of a very high quality. There are, of course, a few potholes but they are being repaired constantly. Deterioration of the road, at this time of year, is at a minimum. Most of the damage to the road is being caused by the buses. The present weight limit is 9,000 Kgs. The buses are grossing out something over 16,000 Kgs and are exempted from the weight limit.

Seasonal snow conditions would require equipment to keep the road open. I believe that financial arrangements could be made with the Territorial Government to accomplish the necessary maintenance and snow removal. Cassiar Asbestos had such an arrangement in order to keep the road open into the Clinton Creek, Y.T. mine.

Visibility becomes a problem when approaching the summit of the White Pass. There is a large vista point where chains could be put on and taken off without blocking the highway.

The 13 mile stretch from the border to Skagway must be approached with extreme caution. The precautions that a prudent operator would take before starting up the operation would include specifying engine brakes, over-size brakes and over-size air tanks. He also would hold comprehensive Driver Education courses.

There are two escape runouts that appear to be very steep with a difficult approach angle from the highway. The chance of a rollover, especially with the high center of gravity of the trucks, are very high. These runouts would certainly do much damage to the equipment.

On this 13 mile stretch, the gradibility of 8% for almost 6 miles is probably the worst section of the entire route. The surface is good and with sufficient width, up to 28 feet, but the length of this section is where the problem arises. Again, equipment to do the job and having drivers approach this area with safety is paramount.

Skagway has a very thin pavement within the city limits. The surface will not stand up for long with the intended weights being carried. Access to the terminal is good and the present transfer and dumping equipment could be utilized. Estimated time factor should be based on a 35 mile per hour average one-way or approximately 3 hours. Buses are doing this run in 2 hours 20 minutes. This, of course, would lengthen out during the winter months.

Whitehorse, Y.T. to Haines, Alaska - 257 miles

Whitehorse to Haines Junction - 98 miles

There are at present two major road building contracts on this portion of the Alcan Highway. Every year they are extending the paved portion of the highway westward from Whitehorse. This road is in satisfactory condition.

Haines Junction to Dezadeash Lodge - 34 miles

The condition of this portion of the road is excellent with routine maintenance being done.

Dezadeash Lodge to Chilkat Pass - 60 miles

This is the worst section of the road, winter and summer. The snowfall in this area is such that the road can be closed for days at a time. As this is flat country, for the most part, the drifts cause all kinds of trouble and avalanche conditions can exist on both ends.

Chilkat Pass to Haines - 65 miles

This is an excellent road, paved almost all the way. In the winter this section receives a deep snow pack. Avalanches in this area are not uncommon.

The round trip, loaded one-way, takes approximately 11 hours in the summer under the best of conditions. Road width varies from 20 to 28 feet.

Recommendations

If there is to be a choice of which road the trucks are to use, I would strongly state that the Whitehorse-Skagway portion be utilized rather than the Whitehorse-Haines route for the following reasons -

- Less mileage, 288 fewer miles on the roundtrip,
- Better road conditions,
- Less road closures during winter months, and
- Less bus traffic, recreational vehicles and generally less tourist traffic.

As a final observation, any prudent operator would welcome the chance to bid, and if successful, the opportunity to utilize this route.