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CYPRUS ANVIL MINING CORPORATION

ORE RESERVE AND PIT DESIGN
DOCUMENTS

COMPILED DECEMBER 4, 1975

CYPRUS

Chronology

Cyprus Anvil Mining Corporation — Interoffice Correspondence

To: R. H. Marshall

Date: December 5, 1974

From: J. C. Devitt

Subject: ORE RESERVES - CHRONOLOGY

On October 30 I met with Dave Jennings and Daryl Hanson to discuss the history and development of the present ore reserve interpretations for Zones 1, 2 and 3. The following includes information gathered from that meeting in conjunction with material taken from file information.

The Ralph M. Parsons Co. submitted a report in February 1967 outlining a mining study for the Faro orebody based on then available information. They estimated 47,000,000 tons of ore mineable by open pit methods. In designing a pit, they laid out a pit to extract 41,000,000 tons of ore from Zone 1 at a 4.81 to 1 stripping ratio. An option was left to either extract 5,000,000 additional tons from Zone 1 at an increased 13.78 to 1 stripping ratio or to extract 5,000,000 tons from Zones 2 and 3 at an assumed 6.36 to 1 stripping ratio. Parsons was instructed to assume that this ore in Zones 2 and 3 existed. Their pit design included only an "ultimate pit" in Zone 1 which would extract 46,000,000 tons of ore, and a "recommended pit" in Zone 1 which would extract 41,000,000 tons of ore. Drilling was inconclusive in Zones 2 and 3, and no pits were designed.

J. F. Olk submitted a mining study in April 1967 indicating a geological reserve, exclusive of dilution, of 51,166,000 ore tons in the following breakdown:

Zone 1	41,617,000
Zone 2	3,916,000
Zone 3	5,633,000
Total	51,166,000

Mr. Olk designed a pit to extract 33,362,000 ore tons from Zone 1 at a 6.30 to 1 stripping ratio and 3,916,000 ore tons from Zone 2 at a 7.00 to 1 stripping ratio. Ore delineation in Zone 3 was considered inconclusive, so no pit design was attempted for it.

E. N. Pennebaker submitted his geological reserve on October 3, 1967. He tabulated the following tonnages of undiluted, assured ore:

Zone 1	33,685,529
Zone 2	5,159,335
Zone 3	23,301,661
Total	62,146,525

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By assuming a 30% ore dilution by internal waste in Zones 1 and 3, the following tonnages were derived:

Zone 1	34,206,995
Zone 2	5,159,335
Zone 3	24,106,611
Total	63,472,941

19,960,044

This total has been used to date as the official ore reserve tonnage for the Faro orebody.

In the summer of 1970, J. Gondi, Mine Geologist, compiled a set of revised bench plans for Zones 1 and 3. This was the first major revision of bench plans and ore reserves since the pre-production bench plans were compiled and included fill-in diamond drill holes which had been added and historical information gathered during the first year of mining. An ultimate pit was designed to include all material which was interpreted as having a grade of higher than 5% combined Pb-Zn. No attempt was made to base the pit design on economic limits. A summary of ore contained within that pit prior to the start of mining follows:

Zone 1	35,487,019 tons	
Zone 3	22,271,792 tons	
Total	57,758,811 tons	(18,163,145 cu. yd.)

A total of 121,616,155 cu. yd. of waste was included in this pit, resulting in a stripping ratio of 7.7 to 1. Zone 2 was not considered in this study. These tonnages very closely approximate Mr. Pennebaker's ore estimate as outlined previously. Dave Jennings' comments on this revision were that Gondi's efforts were not aimed at producing a more detailed orebody study, but that he simply added the new drilling information on the existing interpretative basis and made the bench plans more closely fit the geology by removing fault structures which had been erroneously placed in the original model.

In the winter of 1971-72, J. Keily and D. Jennings independently initiated studies to determine the validity of the 5% combined cut-off grade. D. Jennings' purpose was to establish potential anomalies of economic ore from which to base a diamond drilling program. Dave indicated that it became increasingly obvious to both himself and Keily as they studied the reserve model that a significant percentage of material which had been previously projected as ore simply did not exist as ore grade material, but instead could be identified as thin horizontal stratifications of low grade massive

pyritic sulfides. It was seen that many drill hole intercepts with this massive pyrite material had been ignored in prior studies, mostly because the drilling information, especially in Zone 3, was not complete enough to bring to light the significance of the occurrence of the massive pyrite layers.

The above indications led to additional drilling in 1972 and the initiation of a massive detailed study of the reserve situation under the direction of W. Krats, Chief Engineer. Evaluation of the reserve after completion of the 1972 drilling led to the compilation of:

1. Geological Reserve Estimate - Zone 1 - January 1973

This estimate was done as a basis for five year planning, and included only benches 3910-3590, both mined and remaining. No ore above 3910 bench was calculated, as it had been already mined out. This estimate utilized:

a) Review and update of all diamond drill logs

b) New sections and bench maps

c) Editing of cross-sections to remove 5' intersections of less than 5% combined, some of which had been incorporated into the 1970 reserve as ore without a systematic basis

d) Adoption of 40' bench composite grades instead of the former approach of averaging only the ore within the bench to determine the ore block grade for the bench

e) Reweighting of areas of influence using holes surrounding the ore as well as those within

A comparison of the total material for benches 3910-3590 for the 1970 reserve and the above reserve follows:

1970 Estimate		1973 Estimate	
Tons	Grade	Tons	Grade
28,542,435	9.5	27,749,824	9.8

The decrease of 769,055 tons and increase in grade by 0.3% is largely a result of the elimination of material formerly included which had grades of less than 5%.

2. Geological Reserve Estimate - Zones 1, 2, 3 - June 1973

This estimate added the completed Zone 3 estimate to the Zone 1 estimate outlined above. The same revision criteria applicable to the January revision of Zone 1 also applied to this estimate. The ore tonnage decrease was of a much greater magnitude, however, because of the much greater concentration of massive pyrite lenses in this zone (see attached memo "The Occurrence of Massive Pyrite Within Zones 1-3, Sept. 26, 1972). A comparison of the Zone 3 ore estimate from 1970 to the revised 1973 estimate follows:

1970 Estimate		1973 Estimate	
Tons	Grade	Tons	Grade
22,271,792	8.2%	17,967,396	8.3%

This loss of 4,304,396 tons is directly attributable to the delineation of the low grade pyritic sulfide lenses.

Zone 2 ore was estimated on a trial basis, resulting in an estimate of 2,928,188 tons @ 7.5%. However, it was not felt at this time to be a reliable estimate because:

a) It had not been checked in detail

b) The estimate was based on 40 ft. benches, which were not optimum for the nature of the deposit

c) Diamond drilling had been completed in the summer of 1973, but was not fully assessed

d) The orientation of the deposit is skewed with respect to Zones 1 and 3. This estimate and those in the past had been based on an extension of the Zone 1-3 cross-sectional grid, whereas a grid properly oriented to the trend of the deposit, with corresponding sections, should be properly utilized to calculate reserve estimates.

Also, all earlier estimates for Zone 2 utilized rotary holes which had unsatisfactory chip recovery and diamond drill holes with poor core recovery. The 1973 drilling program was laid out on the new reoriented grid pattern and was successfully designed to eliminate the usage of these unsatisfactory holes for reserve information.

One item which has relevance to all three zones in this analysis involves the projection of drill holes onto cross-sections. It was discovered that prior cross-sections had projected holes located as much as 50 feet from a cross-section directly onto that section without consideration for the dipping of rock units or the surface. This resulted in distortions in projection throughout the orebody and was especially significant in Zones 2 and 3. All necessary compensations for the projection of holes onto sections were accounted for in the present computation.

3. Ultimate Pit Design - Zones 1 and 3 - April 1973

This design was based on the above geological reserve estimate, but was designed to eliminate certain ore quantities because of excessive stripping ratios or due to smoothing of pit walls. Quantities deleted are:

	Tons	Grade
Zone 1	145,810	10.8
Zone 3	1,487,060	8.9
	1,632,870	9.1

The total measured quantities within the ultimate pit were calculated from the pit status as of January 1, 1973, and are as follows:

	Tons	Grade
Zone 1	23,950,410	8.7
Zone 3	16,480,360	8.2
	40,430,770	8.5

Some 90,252,560 cu. yd. of waste were measured as remaining at the time, giving a stripping ratio of 7.1 to 1.

In my conversations with Dave Jennings, he stated that he had closely followed and assisted with the preparation of the geologic reserves as above outlined. He indicated that all work done on both the geologic reserves and ultimate pit reserves had been painstakingly calculated and double checked. The results obtained, he felt, were absolutely top quality and reliable within the context of the drilling information obtained to date.

4. Geologic Reserve Estimate - Zones 1, 2, 3 - December 1973

In the summer of 1973, six diamond drill holes were drilled in Zones 1 and 3 to define the east and west pit walls and to confirm the presence of a diorite dike system in the vicinity of the "Faro Fault." The presence of the dike system was confirmed. The major segment of the dike separates Zones 1 and 3 with approximately 100 feet of barren diorite dipping at about 80° to the southeast. Previously, this structure was thought to be a fault, and ore was projected up to the line of the hypothetical fault. The replacement of the fault with the dike delineation on the bench maps resulted in a substantial ore loss in both Zones 1 and 3. An ore reserve estimate was calculated from these bench maps, utilizing the new drilling information. The basis for interpretation, calculation, and projection remained the same as in the previous estimate.

A comparison of ore tonnages is shown as follows:

	June 1973		December 1973		
	Tons	Grade	Tons	Grade	Tonnage Loss
Zone 1	27,749,824	9.8	26,379,864	9.9	1,369,960
Zone 3	17,967,396	8.3	17,176,406	8.3	790,990
	45,717,220		43,556,270		2,160,950

Zone 2, which suffered considerably from poor recovery in previous analyses, received 23 diamond drill holes in 1973 to complete the drilling pattern based on the newly revised and oriented grid system. A new geologic reserve was calculated, based on the new grid orientation, new sections, new bench maps, and completed drill information. The estimate was calculated using a 20 foot bench interval to allow more selective mining of the much thinner deposit. The total ore thus calculated is:

	December 1973	
	Tons	Grade
Zone 2	3,424,003	8.5

The total ore for Zone 1, 2 and 3 in this estimate is:

		December 1973	
		Tons	Grade
Zones 1, 2 and 3	Total	46,980,273	9.2

As was the case for the June 1973 estimate, this calculation included only benches from 3910 downward. All ore, both mined and remaining, was calculated for these benches. For comparative purposes, the ore actually mined from the 4275 to 3950 benches could be added to the above estimate to get an accurate conception of the total deposit before mining. This is shown as follows: ——

	Tons	
Zone 1, mined, 4275 to 3950 benches	6,286,738	
Zone 1, reserve, 3910 to 3590 benches	26,379,864	
Zone 3, reserve, 3870 to 3350 benches	17,176,406	
Zone 2, reserve, 3930 to 3710 benches	3,424,003	
Total, before mining	53,267,011	- Geological Reserve

No additional pit design was done on Zones 1 and 3 to accommodate the new reserve. The ultimate pit designed in April 1973 was retained, but bench maps were modified to include all diamond drilling updates based on the December 1973 estimate. These maps were provided to Laurich-Kennedy Associates in mid-1974 to use as a base for mine planning. In the course of planning, they re-planimetered the ore blocks and waste remaining from July 1, 1974 to exhaustion for all zones. As stated, the April 1973 pit outline was used for Zones 1 and 3, and Laurich-Kennedy designed a pit to mine essentially all ore from Zone 2. A summation of their measured volumes follows:

	As of July 1, 1974		
	Ore Tons	Waste bcy	S.R.
Zones 1 and 3	34,660,508	90,597,642	8.31/1
Zone 2	3,392,236	5,609,778	5.26/1
Total Reserves as of July 1, 1974	38,052,744	96,207,420	8.04/1
Adding production to July 1, 1974 +	14,433,597	35,037,333	7.72/1
In Pit Reserves at Start of Mining	52,486,341	131,244,753	7.95/1

A tabulation of variances follows together with explanations starting from the official reserve figure prior to mining and ending at the present estimate given above, projected back to the start of mining.

	Ore Tons
E. N. Pennebaker geological reserves, Zones 1, 2, 3	63,472,941
Production losses, benches 4275-3950	- 320,733
Outside ultimate pit (Zones 1 & 3)	- 1,632,870
Diorite dike delineation (Zones 1 & 3, from 1973 drilling)	- 2,160,950
Zone 2 pit-reanalysis after drilling completion	- 1,767,099
Zone 1 - definition of -5% pyritic sulfides	- 769,055
Zone 3 - definition of -5% pyritic sulfides	- 4,304,396
Before Mining, After Adjustments	52,517,838

This reserve estimate total checks very closely with the in-pit reserve estimate calculated by Laurich-Kennedy as outlined previously and projected back to the start of mining. Several spot checks were made of Laurich-Kennedy's waste calculations by re-planimetering benches, and were found to be satisfactory.

J. C. Devitt
Chief Mining Engineer

JCD/mm

cc. J. F. Olk

Recommendations:

1. Await results of DDH resampling programme and follow up as seems appropriate.

2. Tie in as much of the above discussion with a mineragraphic study and some broad ore-typing and metallurgical testing.

P.M. Pettigrew

P. M. Pettigrew
Geologist

PMP/mm

E. N. PENNEBAKER
CONSULTANT GEOLOGIST
P.O. BOX 817
SCOTTSDALE, ARIZONA

JAN 27 1966

TO		FOR REPLY / COMMENT
RMA	GGK	FAR
EAB	KL XC	WJR
WKB	HEM	DTS
RHC XC	PJM	RET ✓
RRG	HTM	PRW
JOH	JDM	
WEH	DPM	

TELEPHONE 945-8352 CODE 602
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cc KL RHC

Office: 23 East First Street

January 25, 1966

Mr. Robert E. Thurmond
Project Manager
Anvil Mining Corporation Limited
523 West Sixth Street
Los Angeles, California 90014

Dear Bob:

The following are my preliminary comments on the Anvil Mining Project, based on my recent trip to Whitehorse and Faro Camp and on my study of the project to date. The following items are listed in the same order as in your letter to me dated January 9, 1966.

1. Ore Reserve Evaluation

   (a) Preliminary estimate based on current limited information.

      (i) Now in progress.

   (b) Refined estimate as sufficient information is available.

      (i) To be done as the results of current and future drilling come in.

2. Critical Evaluation of Basic Data

   (a) Drilling Techniques.

      (i) The use of wire-line and drilling to cut NQ core are heartily approved. This should give better core recovery and will provide a larger sample. Core recovery in the past with AXF bit size was not satisfactory in many holes.

   (b) Sampling Techniques.

      (i) Core recovery and sample intervals need to be measured more precisely, say to 0.10 foot.

      (ii) ~~Mr. Adamson will~~ provide a better classification of the local rock types, and this terminology should be followed in logging.

      (iii) Mr. Adamson also suggests that a graphic log be

combined with the written log, and I am in favor of this suggestion.

(iv) When time permits, the holes now drilled should be re-logged, following the foregoing suggestions.

(v) It is expected that drilling with NQ size bit will give satisfactory core recovery and obviate the need for taking sludge samples. Should this not be the case, then sludge samples must be taken much more carefully than at present. This will require larger sludge boxes equipped with baffles.

(c) Assaying and Check Assaying

Check assaying should be continued, with a critical analyses to be made when more results become available. At present it appears from checking some of the limited data now at hand that composites have not always been made with weighed precision. If this be true, then procedures should be improved.

3. General Evaluation of Drilling Program

(a) Hole spacing and depth.

(i) The spacing of holes 200 feet apart in both directions is approved. A hole at the centers of one or two selected squares may be worthwhile to check the average grade of the block as determination by calculations based on the four holes at the corners of these squares. I still am inclined to recommend that a shaft be sunk on a selected drill hole to check the grade, to provide a bulk metallurgical sample, and, as Mr. E. S. Allen points out, to examine the occurrence and migration of ground water.

(ii) A few deeper holes will definitely be required in the Faro No. 1 and No. 2 zones to search for deeper horizons of ore. This should be done this summer, because, should such discoveries be made, they would influence plans now being evolved. Definite recommendations will be made when sections have been completed and studied.

(b) Location of drilling outside of but in the immediate area of known Faro mineralization.

(i) Such recommendations will be made when the study just mentioned has been further advanced.

4. Review of Geological Data

(a) Interpretation of available geological data.

(i) Study is now in progress.

(b) Recommendation as to additional requirements.

(i) Such will be made in due course.

5. Review of exploration plans for targets other than Faro.

(a) Geological reconnaissance.

(i) I plan on doing this about May, 1966.

(b) Ground geophysics.

(i) This will be considered when the geophysical studies already completed become available to me.

(c) Geochemistry.

(i) Same comment as above.

(d) Drilling.

(i) In general I believe that ground geophysics, geochemistry and drilling (outside of Faro) should be distributed in two zones of somewhat different geologic characteristics. In the Faro-Vangorda zone there have been several significant discoveries to date (Faro, Van Gorda, Swim, and B.S. deposits). The host rock here is phyllite and schist. Certainly one or two showings should be investigated and drilled in this same zone during the coming season, and the DEA and PEA localities appear to be promising. The ACE locality, although farther north, contains the same phyllite and schist as bedrock and also appears promising.

(ii) A second zone is farther north and also trends NW-SE. The bedrock here is a somewhat older formation consisting of chert, black slate, etc. Promising airborne geophysical results and reported gossan made this zone of much interest. Localities that now appear favorable are ZEUS, YETI, IVAN and TEDDY.

(iii) I plan to study this matter of targets other than Faro in the very near future and am hoping that the geophysical and geochemical data reported to be filed in Vancouver will soon be made available.

(iv) To me the Beta area is also of interest because of its situation near the anticlinal crest shown on Campbell's sketch (Fig. 2), and the relations here will be studied with much interest.

6. General Evaluation of the Faro deposit.

(a) Comparison with other economic ore deposits as to grade, size, location, etc.

(i) This will be done at a later date.

Rubidium

The occurrence of the rare and valuable metal, rubidium, in appreciable amounts in several samples (1 to 2 pounds per ton) makes it necessary to learn more about its occurrence in the Faro ore body. The writer recommends that two composite samples be accurately made up from assay pulps for each drill hole in the massive sulfide zone (one for the upper half of the zone; the other for the lower half) and that another be made up for any zones of disseminated sulfides encountered above or below the massive sulfide zone. These should be accurately analyzed for rubidium by a chemist familiar with such procedures.

GENERAL OBSERVATION

Much of the activity now under way at Faro is for the purpose of obtaining accurate samples of the ore body, upon which much critical analysis and many plans will be based. Therefore proper sampling and logging of diamond drill core should be a paramount, not an incidental, activity, and attention should not be diverted from this task by the other numerous and pressing activities that are going on.

Yours very truly

E. N. Pennebaker

ENP:mc

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E. N. PENNEBAKER
CONSULTANT GEOLOGIST
P.O. BOX 817
SCOTTSDALE, ARIZONA

Office: 23 East First Street

March 2, 1966

Mr. R. E. Thurmond
Project Manager
Anvil Mining Corporation Limited
P.O. Box 2470
Whitehorse, Y. T.
CANADA

Dear Bob:

In accordance with your instructions I am submitting herewith an interim report on zinc-lead-silver ore reserves for the No. 1 ore body at Faro camp, Yukon Territory. This reserve is based on drilling completed by December 31, 1965, although in some cases the assays only became available after January 1, 1966.

Also included are remarks concerning the exploration program proposed for the coming season.

Much more drilling must be done before a final reserve can be estimated, and hence the figures presented in this report are tentative. The holes at present are irregularly spaced and some are as far as 800 feet apart, whereas the completed grid will have holes at 200-foot intervals, both north-south and east-west. An estimate of this ore deposit is only warranted at this time because of the apparent regularity in its shape. Nevertheless it is not yet proper to call it "proved ore" because of the drilling that remains to be done. In my opinion most of the body as now known can be termed "probable ore", and a smaller amount is designated as "possible ore".

Core recovery from the nine diamond drill holes cutting the ore body ranges from 66 to 100%, the arithmetical average being 79%. In this estimate it is assumed that the core lost from an assay interval (generally of five feet) is of the same tenor as the core recovered from that interval. Also used in the estimate are the assay results from four rotary drill holes, their sampling being of unknown reliability.

A conversion factor of nine cubic feet of ore equals one short ton was employed. Although a few recent determinations of specific gravity indicate that the ore may be heavier, the use of the conservative figure of nine cubic feet is judged to be proper for the current estimate.

On January 1, 1966, in my opinion there were not enough drill holes through the ore body to determine its average grade by a weighted average of the grades of individual blocks constructed on cross-sections. Consequently an average grade was taken of the ore interval cut by each hole and weighted according to the lengths of the various intervals. The following results were obtained:

Weighted average, as above:	4.07% Pb
	6.30% Zn
	1.16 oz. Ag.
	0.12% Cu

The short tons of one, derived from cross-sections at 400-foot intervals from west to east, are as follows:

PROBABLE ORE

Section No.	Short Tons
12,400 E	1,779,982
12,800 E	1,666,650
13,200 E	4,615,509
13,600 E	7,435,481
14,000 E	6,844,376
14,400 E	6,893,264
	29,235,263

POSSIBLE ORE

Section No.	Short Tons
13,600 E	1,068,878
14,000 E	477,773
14,400 E	1,999,980
	3,546,631

TOTAL PROBABLE PLUS POSSIBLE ORE

32,800,000 short tons @ 4.07% lead
6.30% zinc
1.16 oz. silver
0.12% copper

Future drilling will be for two purposes:

(1) To fill in the grid and confirm the tonnage and grade indicated above, and

(2) To extend the drilling outward and increase the above tonnage.

Drilling to date since January 1, 1966, is now serving these purposes, and the results achieved this year are encouraging.

Check assays were not available for the above estimate, but such are being made and when they come in the estimate will be adjusted accordingly.

Diamond drilling is now being done by wire line to give NQ core with a diameter of $1\frac{23}{32}$ inches. This has improved the core recovery but not to the extent that sludge sampling can be eliminated. A better sludge recovery box is now being employed in order to improve such sampling.

On February 23, 1966, you and I reviewed the exploration program proposed for the coming season by Mr. Robert S. Adamson, Anvil's chief geologist. This will cover much of Anvil's extensive property holdings outside of the Faro No. 1 area. It comprises surveying, reconnaissance geological mapping, detailed geological mapping, line cutting, trenching, ground geophysics by several methods, geochemical testing, and exploration by drilling. Various of these activities are allocated to individual blocks of ground as determined by their apparent merit balanced against the requirement to perform assessment work this year. Mr. Adamson has presented a sound program, and I recommend that it be carried out.

"

E. N. Pennebaker

ENP:mc

cc - Mr. Thurmond, Los Angeles

cc - Mr. Kenneth Lieber

R.M. Parsons
Feb. 1967

FINAL FEASIBILITY STUDY

DEVELOPMENT OF THE FARO No. 1 ORE BODY

THE ANVIL MINING CORPORATION, LIMITED

JOB No. 3746-3

FEBRUARY 1967

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

ENGINEERS-CONSTRUCTORS

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

Engineers • Constructors

617 WEST SEVENTH STREET, LOS ANGELES 17, CALIFORNIA

February 17, 1967

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

ATTENTION of Mr. Kenneth Lieber, President

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

Gentlemen:

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

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

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

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

Very truly yours,

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

By Milton Lewis

Milton Lewis
Executive Vice President

ML:cl
Enclosures (15)
As Above

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SECTION 1

SUMMARY

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

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

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

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

SECTION 4

STRIPPING AND MINING STUDY

ANVIL PROJECT

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SECTION 4

STRIPPING AND MINING STUDY

SECTION 4A

SUMMARY

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

The following pit designs have been prepared:

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

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

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

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

Table 4-1 summarized stripping and mining costs.

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

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

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

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

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

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

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

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

SECTION 4B

INTRODUCTION

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

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

"Mining Study

1. Develop open pit mining plans based on geologic sections supplied by Anvil to include:

   (a) A preproduction stripping plan
   (b) A waste dump plan
   (c) A pit design for first 5 years.
   (d) Final pit design.

   The 5 year pit should be designed to exploit the highest grade ore with minimum preproduction stripping requirements.

2. Develop volume estimates for each mining stage specifying quantities of alluvium, waste rock and ore.

3. Develop a recommended equipment list with specifications.

4. Develop mining plans covering volumes of material to be removed year by year and equipment scheduling year by year.

5. Develop capital cost estimates for preproduction stripping and equipment purchase.

6. Develop a schedule of manpower requirements.

7. Develop operating cost estimates including direct cost consisting of operating labor, operating supplies, maintenance labor maintenance supplies, and indirect cost consisting of supervision and general pit expense."

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

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

SECTION 4C

SCOPE OF THE WORK

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

1. Prepared pit plans and sections for the Faro Number One deposit showing:

   (a) The Preproduction Pit
   (b) The Five-Year Pit
   (c) A Pit which would yield approximately 41,000,000 tons of ore
   (d) An Ultimate Pit which would yield approximately 47,000,000 tons of ore.

2. Prepared a plan showing overburden spoil areas with haulage roads to these areas and to the crusher.

3. Developed quantities of ore and overburden and grade of ore to correspond to the various pit designs.

4. From the drill hole information prepared cross-sections of the ore-body and developed the grade and tonnage of each mining bench.

5. Compiled a list of the major mining equipment required showing the landed and erected cost of the equipment.

6. Prepared operating cost tables and manpower tables for each phase of the mining operation.

7. Developed productivity tables for the major equipment.

8. Prepared basic specifications for all major equipment.

SECTION 4D

DESIGN CRITERIA

The following design criteria were furnished by the client:

1. Staff and hourly paid employees salary and wage rates, hours of work, and the basis for overtime payment. In this regard a 44-hour work week was taken as basic, but the work week is based on 6-8 hour days so that for hourly personnel, payment is for 50 hours per week.

2. The landed cost of diesel oil and gasoline.

3. The specific gravities and densities of alluvium, cap rock and ore.

4. Fifteen percent of alluvium to be considered as cap rock.

5. Minimum temperature -70°F.

6. Pit Criteria - Alluvium slope, to be 1-1/2 to 1 for working and final slopes. Working bench width sufficient for annual stripping and mining requirement, suggested dimensions are: 60' berm, 12' toe and a maximum bench heighth of 35'. Road widths: 60' with 7% maximum grade. The final pit slope to average 40°.

7. Preproduction Stripping - Develop enough ore for one year's milling.

8. After Production Starts - Each expansion should be sufficient for one year's milling.

9. Alluvium Stripping - Assume twin motor scrapers (e.g. Euclid TS-24) will be used. Give consideration to other methods.

10. For Stripping Cap Rock and Mining Ore - That combination of shovels and trucks which will minimize costs taking into consideration the maximum degree of standardization possible. All mining and stripping operations will be on a year-round basis, 24 hours per day for stripping, and 8 hours per day for mining, working 6 days per week on each.

11. Drilling and Blasting - Assume the use of Bucyrus-Erie Model 40R drills. If this class of drill can be obtained with angle drilling capability, use the angle drill. Holes will be 9" diameter with 18' burden and 22' spacing. Ammonia nitrate/fuel oil mixture will be used for blasting. For secondary drilling, use a rubber tired unit carrying compressor and boom mounted drill with full cab controls e.g., "Traveldrill".

Production Factors

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

Production Factors (continued)

1. Alluvium

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

If shovel is used then

Dipper Capacity = Dipper size X .8 (S.F.) X .95 (D.F.) bank c.y. e.g., 5 c.y. dipper

= 5 X .8 X .95
= 3.8 bank c.y.
= 3.8 X 1.68
= 6.38 tons
use 6.40 tons/5 c.y. dipper

2. Cap Rock

Density (S.G. 2.75)	2.3 tons/bank c.y.
Swell Factor (S.F.)	50%
Dipper Factor (D.F.)	85%
One Broken c.y.	= 2.3 X .667
	= 1.53 Tons

Dipper Capacity - Dipper size X 0.667 (S.F.) X .85 (D.F.) bank c.y.
e.g., for a 5 c.y. dipper

= 5 X .667 X .85
= 2.83 bank c.y.
= 2.83 X 2.3 Tons
= 6.5 Tons/5 c.y. dipper

3. Ore

Density (S.G. = 3.76)	3.18 tons/bank c.y.
or 8.5 bank c.f.	= 1 ton
Swell Factor (S.F.)	50%
Dipper Factor (D.F.)	85%
One Broken c.y.	= 3.18 X .667
	= 2.12 tons

Dipper Capacity - Dipper size X 2.12 X .85 (D.F.) tons
e.g., a 5 c.y. dipper

= 5 X 2.12 X .85 tons
= 9.01 tons
use 9.0 tons

Cost Data

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

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

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

SECTION 4E

OVERBURDEN AND OREBODY CHARACTERISTICS

Description of the Orebody

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

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

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

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

The Nature of the Ore

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

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

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

Glacial Debris and Alluvium

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

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

Volume of Materials

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

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

Grade of Mill Feed

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

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

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

Ore Reserve Calculations

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

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

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

SECTION 4F

MINING PLAN

General

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

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

4.8/1

13.8/1

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

6.4/1

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

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

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

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

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

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

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

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

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

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

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

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

Haulage Roads

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

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

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

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

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

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

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

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

Drilling and Blasting

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

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

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

Production

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

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

Support Equipment

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

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

Pumping Equipment

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

General Pit Expense

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

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

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

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

Electrical Power Distribution

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

by portable power and control centers containing the following:

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

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

Service Building

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

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

1. Complete tool sets for servicing and major overhaul of every piece of equipment.

2. Test stands for engine overhaul.

3. Tire vulcanizing and recapping equipment.

4. Overhead traveling bridge crane.

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

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

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

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

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

Table 4-2

VOLUME OF MATERIALS
SUMMARY
ANVIL MINING COMPANY

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

(14,465,408 cu yd)

72,050,000 / 14,465,408 = 4.98 : 1

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

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

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

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

(4) Per Cent Combined Lead and Zinc.

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

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

4-T2

ORE RESERVES
BY BENCH
ANVIL PROJECT

5.0% Combined Pb-Zn = Cut-Off Grade

BENCH	TONS	% Pb	% Zn	COMBINED Pb - Zn
4205	162,500	2.9	6.5	9.4
4170	444,504	4.5	6.5	11.0
4135	372,047	4.6	8.9	13.5
4100	540,871	4.3	7.3	11.6
4065	737,648	4.1	6.3	10.4
4030	683,118	4.9	8.3	13.2
3995	1,377,777	4.3	7.2	11.5
3960	2,073,888	4.3	6.5	10.8
3925	2,778,683	4.3	6.8	11.1
3890	3,165,726	4.2	6.8	11.0
3855	3,556,155 all Z-1	4.7	6.5	11.2
3820	3,448,705	4.1	6.4	10.5
3785	3,534,588	3.0	5.5	8.5
some in dike zone 3750 Z-1	2,717,952	3.6	6.3	9.9
" 3715 Z-1	3,215,059	3.9	6.1	10.0
across dike in N and 3680 Z+1+3	2,989,942	4.0	6.3	10.3
crosses dike all along 3645 Z1+3	3,299,061	3.5	5.7	9.2
half & half 3610 Z1+3	2,456,588	2.2	4.6	6.8
Recom. Pit " 3575 Z1+3	1,361,706	3.2	5.3	8.5 38,916,518
mostly Z3 3540 Z-1+3	2,046,118	2.9	4.6	7.5
41,000,000				
mostly Z3 3505 Z-1+3	1,576,647	3.6	5.7	9.3
barely crosses dike into Z1 3470 Z1+3	1,823,295	3.7	5.4	9.1
" 3435 Z1+3	1,780,471	3.7	5.2	8.9
some in dike 3400 Z1+3	966,706	3.0	4.1	7.1
UH. Pit 3365 Z3	41,176	2.0	5.2	7.2
GRAND TOTAL	47,150,931	3.8	6.1	9.9

TABLE 4-3A

ANVIL PROJECT

TONNAGE AND GRADE DISTRIBUTION BY BENCHES

FARO #1 OREBODY

MILLIONS OF SHORT TONS

1 2 3 4

BENCH	% Pb	% Zn	COMBINED TOTAL	BENCH TONS	CUMULATIVE TONS
4205	2.9	6.5	9.4	162,500	162,500
4170	4.5	6.5	11.0	444,504	607,004
4135	4.6	8.9	13.5	372,047	979,051
4100	4.3	7.3	11.6	540,871	1,519,922
4065	4.1	6.3	10.4	737,648	2,257,570
4030	4.9	8.3	13.0	638.118	2,940,688
3995	4.3	7.2	11.5	1,377,777	4,318,465
3960	4.3	6.5	10.8	2,073,888	6,392,353
3925	4.3	6.8	11.1	2,778,683	9,171,036
3890	4.2	6.8	11.0	3,165,726	12,336,762
3855	4.7	6.5	11.2	3,556,155	15,892,917
3820	4.1	6.4	10.5	3,448,705	19,341.622
3785	3.0	5.5	8.5	3,534,588	22,876,210
3750	3.6	6.3	9.9	2,717,952	25,594,162
3715	3.9	6.1	10.0	3,215,059	28,809,221
3680	4.0	6.3	10.3	2,989,942	31,799,163
3645	3.5	5.7	9.2	3,299,061	35,098,224
3610	2.2	4.6	6.8	2,456,588	37,554,812
3575	3.2	5.3	8.5	1,361,706	38,916,518
3540	2.9	4.6	7.5	2,046,118	40,962,636
3505	3.6	5.7	9.3	1,576,647	42,539,283
3470	3.7	5.4	9.1	1,823,295	44,362,578
435	3.7	5.2	8.9	1,780,471	46,143,049
3400	3.0	4.1	7.1	966,706	47,109,755
3365	2.0	5.2	7.2	41,176	47,150,931
				47,150,931	

TOTAL 47,150,931 t 3.8% Pb 6.1% Zn

+ Pass 539,000T
Tot 39,455,518
2,190,000 Dike
1,034,600
Rem 36,231,518
× 3.3/3.18 = 39,854,670

STRIPPING AND MINI ; EQUIPM
CAPITAL COST DELIVERED TO S
ANVIL PROJECT

	Units	Pound Weight		Equipment Cost U.S.A. Per Unit	Total Equipment Cost U.S.A.	Freight to Dawson Cree Per Unit	Freight to Jobsit
Bucyrus 5 Yard Electric Shovels	(3)	273,500	(1)	$221,600	$664,800	$10,900	$8,200
Bucyrus 5 Yard Diesel Shovel	(1)	274,000	(1)	224,200	224,200	11,000	8,300
Bucyrus 3 Yard Electric Shovel	(1)	116,000	(1)	130,200	130,200	5,000	6,600
Caterpillar Dozers	(4)	76,800	(1)	61,100	244,400	3,100	1,500
ulpak 65 Ton Trucks	(15)	77,500	(2)	101,400	1,521,000	3,100	200
cyrus 40R Rotary Drill	(3)	88,000	(1)	115,900	347,700	3,500	1,800
ey Travel Drill Sniper	(1)	23,800	(1)	41,500	41,500	1,000	500
-24 Rubber Tired Scrapers	(1)	87,200	(1)	94,300	94,300	3,500	500
t 16 Motor Grader	(1)	54,000	(1)	55,900	55,900	2,200	1,100
rder Truck Bulk Chevrolet	(1)	12,000		10,800	10,800	500	200
er Truck 2,500 Gal. Chevrolet	(1)	11,500		6,200	6,200	500	200
ease Truck Lubrivan Chevrolet	(1)	15,000		11,200	11,200	600	200
nch Truck Tandem H.D. Chevrolet	(1)	15,000		9,500	9,500	600	200
un t Bus 44 Passenger Chev.	(1)	15,000		14,600	14,600	600	200
.D. Station Wagon Chevrolet	(1)	4,500		3,600	3,600	200	200
.D. Pickup 1/2 Ton Chevrolet	(3)	4,200		2,900	8,700	200	200
er Center Bkrs. Transformers							
200 HP 15,000/4160V Transformer	(1)	10,000		26,300	26,300	400	100
Type SHD 4160V Power Cable 60% ubber	(3,000')	3,000		9,500	9,500	200	100
Type Cable Connectors 4160V	(3-sets)	500		500	1,500	100	100
.W. Light Plants Diesel	(5)	(500	ea.)	3,700	18,500	100	100
Total Equipment Cost					$3,444,400		

List Price less 5%
List Price less 7%

Drive Unit to Jobsite
Haul Unit to Jobsite (handling)

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

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

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

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

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

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

PREPRODUCTION CAP ROCK STRIPPING SPECIFICATIONS ANVIL PROJECT

TRUCKS

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

SHOVELS

Shovels Required (5 Cubic Yards)	2
Yards per Hour (1 Shovel)	303
Yards per Hour (2 Shovels)	606
Hours per Day (2 Shovels)	39.95
Hours Annually (2 Shovels)	12,467

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

TRUCKS

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

SHOVELS

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

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

TRUCKS

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

SHOVELS

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

ORE PRODUCTION
(1 to 5 YEARS INCLUSIVE)
ANVIL PROJECT

TRUCKS

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

SHOVELS

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

ORE PRODUCTION
(6 to 9 YEARS INCLUSIVE)
ANVIL PROJECT

TRUCKS

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

SHOVELS

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

ORE PRODUCTION
(10 to 17 Years Inclusive)
ANVIL PROJECT

TRUCKS

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

SHOVELS

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

PREPRODUCTION MINING
ORE TO STOCKPILE
ANVIL PROJECT

TRUCKS

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

SHOVELS

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

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

TRUCKS

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

SHOVELS

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

000916

ANVIL MINING CORPORATION LIMITED

Summary of Reserves

January 1, 1973

	Ore (Tons)	% Pb	% Zn	Cu. Yds.	Waste Cu. Yds.	Ratio (Waste):(Ore) (Yds.) (Yds)
* Initial Ultimate Pit Ore Reserve (Ore Zones 1 & 3)	58,314,000	3.4	5.8	18,338,000	118,387,000	6.5 : 1
** Additional Reserve (Ore Zone 2)	5,159,000	3.1	4.8	1,622,000	6,500,000	4.0 : 1
TOTAL RESERVE	63,473,000	3.4	5.7 (9.1)	19,960,000	124,887,000	6.3 : 1
Mined to date December 31, 1972	7,974,530	4.6	6.4	2,507,700	23,815,703	9.5 : 1
*** Remaining as of Jan. 1, 1973	55,498,470	3.2	5.6	17,452,300	101,071,297	5.8 : 1

8.8

Based on Actual All Material movement @ Dec 31 1970 + 1971 to exhaustn from long Term plan, which includes Zone 1, 2, & 3. Zone 2 broke out by best guesstimate at present time.

* Based on an estimate of reserves and a detailed mining plan.

** Based on preliminary ore and waste estimates, but no detailed mining plan.

*** Anticipated remaining life of the ore body is approximately 15 years.

Note April 3/73

This document produced for presentation for Baydad Copper People.

W. Kuntz

PIMA MINING COMPANY
PIMA MINE ROAD
POST OFFICE BOX 7187 - TUCSON, ARIZONA 85713
TELEPHONE 294-1444

4 April 1967

L.A.
629-5771

Mr. R. E. Thurmond
Vice-President & General Manager
ANVIL MINING CORP., LTD.
1200 Pacific Mutual Building
523 West Sixth Street
Los Angeles, California 90014

Dear Mr. Thurmond:

Attached is the report prepared by the undersigned on the mining study for the Faro orebody of Anvil Mining Corporation, Ltd.

The ore reserve estimates prepared in the course of this study are within reasonable agreement with that submitted by E. N. Pennebaker on February 20, 1967.

Capital equipment and stripping costs, as well as the first five year operating costs, are in line with the costs used in the feasibility report prepared by The Ralph M. Parsons Company. Some differences occur between this report and the feasibility report, both in certain equipment selections and in cost calculations, but these differences have compensated in the overall effect. For the period beyond the first five years, the operating costs in this study are somewhat higher than the feasibility costs, due to the higher stripping ratio in this estimate. However, the cost difference is less than 10 percent, and should have no serious effect on the rate of return on investment.

The invaluable assistance of Mr. Murray O. Hampton, Geological Engineer of Anvil Mining Corporation, Ltd., during the major portion of this study, is hereby acknowledged.

Respectfully submitted,

James F. Olk

JFO:cgg

Attachment

Anvil Mining Corp., Ltd.
Faro No. 1
Mining Study

Ore and Waste Estimates

The geological sections prepared by the Geological Staff of Anvil Mining Corporation, and the diamond drill hole logs, were used as a basis of the ore estimate. The midslope ore-waste intercepts for each bench were plotted on horizontal bench plan maps and ore blocks laid out. The ore and waste estimates were made on the bench plans.

The ore cutoff grade used was 5.0% combined lead and zinc, and the density was 3.18 tons per bank cubic yard, or 8.5 cubic feet per ton in place.

The extrapolation of ore beyond a drill hole did not exceed 141 feet, which is one-half the 282 feet spacing of drill holes on the sections. This limit was, with very few exceptions, adhered to. Therefore, this ore estimate can be regarded as a conservative one, which does not include any probable ore according to the parameters established for this estimate. There is a certain tonnage of probable ore that in this estimate is included as waste, but this lends an element of conservatism to the estimate.

The estimated alluvium yardages were reduced by 15 percent, and this was added to the waste rock in all estimates, in order to compensate for a certain amount of material above the apparent alluvium-rock contact that will probably be too hard to rip and take with scrapers, and probably will have to be drilled and blasted.

A considerable amount of thought was given to the subject of probable ore dilution, and the factor to be applied in this estimate. The included waste, that is the waste bands within the ore zones, was estimated for both tonnage and grade. This amounted to about 7 percent of the total ore tonnage. First thoughts were that this could be used as a dilution factor, assuming that a certain amount would be segregated, but that the tonnage segregated out would be balanced by a certain amount of side, top, and bottom waste. Upon further thought, this seemed excessive. At Pima, a dilution factor has been established through considerable study, and the basis of this factor is to regard 10 percent of the included waste as dilution. This requires good shovel segregation at the ore-waste contact points, and since this represents what can be regarded as good mining practice, it was decided to apply the Pima standard dilution factor to this estimate. The included waste estimate is shown in Appendix 35.

The ore in the Faro No. 3 orebody was estimated as of the latest drill hole information received on March 10th. It appears from this information that it is probable that the Faro No. 3 ore is a continuation of the Faro No. 1, and that as further information developes, that the Faro No. 1 and 3 orebodies could be combined and mined as a southeastward expansion of the Faro No. 1 final pit.

Appendix 1 to this report summarizes the ore resources and waste quantities estimated for the various orebodies.

Pit Layout and Design Criteria

For this mining study, three pits were laid out, namely a Preproduction Pit, a Five-Year Pit, and a Final Pit, for the Faro No. 1 orebody.

The Preproduction Pit was designed to make available approximately one year's mill ore requirements, or 2,000,000 tons, on the scheduled mill start-up date of July 1, 1969. The five year pit was laid out to provide a five year ore supply at a grade of at least 12% combined lead-zinc. The final pit was designed to mine all ore in the Faro No. 1 orebody northwest of a major fault that occurs between sections 3 and 4. This structure lowered the ore lying immediately southeast of it approximately 100 feet. No further expansion on to the final pit to recover this ore was estimated, since it appears that this ore at present might be of marginal economic value. This premise can, of course, change with time and further study, but for the purpose of this study it was not considered. The ore below the final pit in Faro No. 1 is estimated in Appendix 36.

A small pit was also designed for the Faro No. 2 orebody in order to recover all ore in this orebody. No pit was designed for the Faro No. 3 orebody since drilling is still incomplete, and only pit design would be shortly superceded by further orebody delineation.

Two final pit slope angles in rock were used in this study. A 40 degree slope in rock was used in most of the pit except in the area north of 10,000 N. coordinate on the west side and west of Section No. 8 on the north side. In this area, a 45 degree slope was used in order to reduce waste rock stripping. The geological sections show that the rock type in this area is mainly a quartz diorite intrusive with no major structure, so that this is apparently competent enough rock to stand at this angle, along with the fact that the final pit rock depth in this area is approximately 500 feet. The Preproduction Pit and Five-Year Pit were designed to finalize as much as feasible in this area. At the top of formation, or base of alluvium, a 50 foot bench was included, and the overall alluvium slope used was 1-1/2:1 for both final and working layouts.

The working slope in rock areas that have not been finalized is approximately 26 degrees. A bench layout with 16 foot crest to toe banks and 56 foot bench levels, or a total of 72 feet from midslope to midslope, was used, which results in a 26 degree slope.

The grade for pit ramps was set at 7%. According to the performance graph for LW65 trucks with Allison transmissions, the performance on this grade should be 8.5 MPH in the start of 2nd range lockup. This is a desirable range and should give good truck performance and economical maintenance. On each bench level, a 100 foot level stretch was left in the ramp system. This is an operating consideration which allows the ramp to the next level down to be started with no interference with haulage from the level, and also allows trucks to enter

the haul road system on level ground, which reduces tire wear and suspension maintenance.

In the Preproduction Pit, throat roads were designed for access to the 4310, 4240, 4205, and 4170 benches. These were put in to eliminate needless lifts or downhill hauls. Since much of the Preproduction Pit and Five-Year Pit alluvium and rock stripping is done in a side hill cut, the placement of these throats can be an operating decision, since they can be put in the alluvium very easily and quickly, as long as the alluvium removals are within the final pit limits.

Appendix 2 of this report summarizes the ore and waste volumes in the various pits. Appendices 6 and 7 show the volume breakdowns for the Preproduction Pit. Appendices 18 and 19 show the same for the 5-year pit, and Appendices 32, 33 and 37, the Final Pit.

Mine Equipment Selection

Appendix 4 to this report summarizes the capital mine equipment requirements and Appendix 5, the requirement calculation details.

The shovel size of five cubic yards selected in the Parsons report appears to be sound for this operation, and the specified number of four shovels will be required. Due to the ore blending problem, in attempting to control the tonnage of high pyrrhotite ore in the mill feed, two shovels will be require for ore mining, and the quantities of waste rock required to be moved demand two shovels in waste rock. If the next largest shovel size is considered a considerable increase in capital expenditure will be required, which is not justified by the scheduled production volumes, and any smaller size of the same number would not give the required production rate, and would also be a mismatch for the proposed 65-ton haulage units. The ore shovels, during the first five years, will have a rather low utilization factor, but it is possible that in actual operations, one of these can handle a certain amount of waste rock, especially included waste, from time to time without an excessive amount of moving. It was assumed that of the first two shovels, one would be a diesel machine and the other an electric, with power provided by a portable generator. The other two shovels would be electric, since a permanent power supply would be available when they are required, that is, when ore delivery to the mill is scheduled to start. The five cubic yard size shovel represents a good balance between the production requirements, the number of shovels needed, and the lower capital expenditure. It should be noted that a Bucyrus-Erie Model 88 shovel is listed in the equipment list. This is because a delivered price was available from the Parsons report on this particular model, and it is assumed that prices and specifications on other makes will be obtained before a purchase is finalized.

The 65 ton truck size is the best selection, since these are the largest size compatible with the shovel size. The LW trucks, operated at Pima with similar specifications, have shown good availability and operating cost during the almost two years of operation, and can be considered a well proven unit. The requirement calculations in Appendix 5 show that during the first five years, one less truck than the total number required would be feasible. However, since this represents only a small fraction of the total cost, it is recommended that all required trucks be purchased initially, mainly to avoid possible price escalation on a later purchase.

The calculation of drill requirements in Appendix 5 show that two drills are adequate for all requirements. This, of course, would mean more moving time, but the utilization factor is such that this time is available. The Bucyrus-Erie 40R drill is a proven, reliable drill unit, and this size is adequate for the drilling requirements.

In addition to the two Caterpillar 619 scrapers available on the job, a Caterpillar model 631 is recommended in addition. This is the largest scraper that can be economically pushed by a D8 tractor, and represents a lower capital cost. This combination of scrapers will handle the preproduction pit stripping requirements, and after that, the utilization factor of the scrapers decreases considerably, as shown in the details in Appendix 5.

Two rubber-tired dozers have been included in the equipment selection, since in the cost estimates, one rubber-tired cleanup dozer is scheduled around the clock. It is felt that more mobile, rubber-tired units would provide more effective and economical shovel cleanup and dump maintenance. One rubber-tired unit should have no trouble servicing two shovels and one dump, and at times, an ore stockpile. Two units are recommended for high availability, The number of D8 tractors have been determined by scraper cut and general dozer requirements, as outlined in Appendix 5.

For stockpile ore loading, a rubber-tired shovel loader is recommended instead of a power shovel. This has proven at Pima to be an economical method of stockpile retrieval, and the stockpile ore requirements are in no year such that they cannot be handled easily by a loader. This also represents a reduction in the capital cost.

Two Caterpillar No. 14 motor graders are also recommended. The reason for this is that it is felt that grader availability must be high as road maintenance, especially during certain times of the year, will be a critical item. Two No. 14 graders represent a cost of only slightly more than a No. 16, and this has proven to be a good unit at Pima. With two graders, two could be scheduled during critical road maintenance periods.

The scraper, tractors, rubber-tired dozers, loader, and graders have all been specified as Caterpillar. One reason is that delivered prices are immediately available or can be closely estimated, and the other is the advantage of standardizing on one make with a source of service and parts readily available in Whitehorse.

The other, smaller service vehicles and miscellaneous equipment was taken verbatim from the Parsons report, since a great deal of work went into the compilation of this, it appears reasonable, and only minor changes might be made with further study. The only change was to recommend more shovel cable (1500 ft. per shovel) and cable connectors.

Equipment Operating Costs

The operating labor rates for various equipment were taken directly from the Parsons report, which in turn were based on the study by Foster Economics Consultant. The hourly rate was established by assuming a 48 hour scheduled work week with 44 hours at straight time and 4 hours at time and one-half. This figure was factored for operating hours by the ratio of 7.5 hours shift scheduled time over 8 hours pay, and an equipment availability figure deemed reasonable for each unit under consideration.

Equipment operating cost experience at Pima was used extensively where applicable, particularly with regard to fuel consumption, tire life, and other operating supplies. The tire cost of $2,120 for 2100 X 35 tires, used on the 65-ton trucks, which was provided by Parsons was also used as a base for estimating other tire prices, by factoring up in the same ratio as Pima's price for 2100 X 35 tires as against the quoted price for Anvil.

Repair costs, labor and parts, were determined in consultation with M. H. Nicholson, Mechanical Superintendent at Pima, who has had considerable experience in equipment maintenance under arctic conditions.

Where no particular cost items were available without extensive research or an "educated guess", the figures used by Parsons were taken, especially for much of the smaller service equipment.

All Pima's costs that were used were converted to equivalent costs in Canadian dollars in the ratio of $0.92 Canadian to $1.00 U.S.

Details of equipment costs are shown in Appendices 48 through 54.

Mining Costs

Mining costs, both totals and unit costs for the three pits, are summarized in Appendix 3. The Preproduction Pit costs are summarized in Appendix 8, and the detail sheets follow as Appendices 9 through 17. It should be noted that the cost for the Preproduction Pit is a total cost, not an annual one, and covers the period when the two 619 scrapers start working, on July 1, 1967, to the end of preproduction stripping on July 1, 1968, so that actually, two years are included.

The annual cost for the five year pit is shown in Appendix 20, and detail sheets in Appendices 21 through 31.

The final, or remaining pit, is shown in Appendix 37, with detail sheets in Appendices 38 through 47.

In each instance, the general pit expense includes the 20% labor overhead as payroll taxes and benefits. Supervision is taken directly from the Parsons report. The general pit expense in each case was prorated to the various mining categories on the basis of proportional yardages to the total.

ANVIL MINING CORP., LTD.
Faro No. 1, 2, and 3
Ore Reserve Summary

Ore	Tons	Grade
Faro No. 1 - Final Pit	33,362,000	10.0
" " " - Dilution	211,000	2.2
" " " - Below Final Pit	8,255,000	8.5
Total Faro No. 1	41,828,000	9.7
Faro No. 2	3,916,000	7.6
Faro No. 3	5,633,000	7.2
Grand Total Ore	51,377,000	9.3

Waste

	Alluvium (cu.yds)	Waste Rock (Cu.yds)
Faro No. 1 - Final Pit	10,308,000	55,830,500
Faro No. 2 - Final Pit	2,338,000	6,284,000

Note: Grade is % combined Pb-Zn

ANVIL MINING CORP., LTD.
Faro No. 1
Ore and Waste Summary

Period	Alluvium (cu. yds)(1)	Waste Rock (cu. yds)	Ore to Stockpile				Ore to Mill	
			High Grade		Low Grade			
			Tons	Grade (2)	Tons	Grade	Tons	Grade
Preproduction	2,576,000	4,695,000	447,000	12.5	270,000	8.2	--	--
1 - 5 Years	4,430,000	14,537,000	--	--	1,848,000	6.7	10,000,000	12.2 (3)
Total 5 Years	7,006,000	19,232,000	447,000	12.5	2,118,000	6.9	10,000,000	12.2
Remaining Pit	3,302,000	36,599,000	--	--	--	--	23,573,000	9.1 (4)
Grand Total	10,308,000	55,831,000	--	--	--	--	33,573,000	10.0 (5)

(1) 15% of original volume subtracted from alluvium and added to waste rock.

(2) Grade is percent combined Pb-Zn.

(3) Includes high grade ore stockpiled during preproduction.

(4) Includes all low grade ore stockpiled during preproduction.

(5) Includes 10% of included waste at a grade of 2.2% as dilution.

Appendix 2

ANVIL MINING CORP., LTD.
Faro No. 1
Capital Cost - Mine Equipment
(in US Funds

	No. Req'd	Unit Cost	Freight Tariff etc.	Total Cost Per Unit	Total Cost
88 B-E 5 yd. Electric Shovels	3	$221,600	$24,800	$246,400	$ 739,200
88 B-E 5 yd. Diesel Shovel	1	224,200	25,000	249,200	249,200
Caterpillar 988 Shovel Loader	1	61,000	10,200(1)	71,200	71,200
Caterpillar D8 Tractors	2	61,100	4,600	65,700	131,400
Cat. 842 Rubber-Tired Dozers	2	60,700	10,800(1)	71,500	143,000
LW 65 Ton Trucks	15	101,400	10,900	112,300	1,684,500
B-E 40R Rotary Drill	2	115,900	14,000	129,900	259,800
Cat. 631 Scraper	1	75,000	12,000(1)	87,000	87,000
Cat. No. 14 Motor Grader	2	31,000	5,000(1)	36,000	72,000
Travel Drill-Athey	1	41,500	4,400	45,900	45,900
Bulk Powder Truck, Chev.	1	20,000	2,000	22,000	22,000
Water Truck 2500 Gal. Chev.	1	6,200	1,800	8,000	8,000
Grease Truck-Lubrivan, Chev.	1	11,200	2,800	14,000	14,000
Winch Truck, Chevrolet	1	9,500	2,300	11,800	11,800
Bus, 44 Pass. Chevrolet	1	14,600	3,400	18,000	18,000
4 W.D. Station Wagon, Chev.	1	3,600	1,000	4,600	4,600
1/2 Ton A W.D. Pickup, Chev.	3	2,900	900	3,800	11,400
Transformer	1	26,300	2,500	28,800	28,800
Shovel Cable - 4500 Ft.		13,700	1,300	15,000	15,000
Cable Connectors	6 Sets	500	300	800	4,800
5 KW light plants	5	3,700	700	4,400	22,000
					$3,643,600

Total Capital Cost - Mine Equipment

(1) Estimated - Freight @ $5.50/cwt and tariff at twice freight.

ANVIL MINING CORP., LTD.
Faro No. 1
Preproduction Pit - Alluvium & Waste Rock Summary

Alluvium

Total estimated alluvium -	3,031,100 cu. yds.
Less -15% to Waste Rock -	454,700 " "
Total Alluvium	2,576,400 cu. yds.

Waste Rock

Bench	Cu. Yds.
4310	237,700
4275	426,900
4240	695,600
4205	852,200
4170	1,121,600
4135	754,000
4100	152,700
TOTAL	4,240,700
Plus 15% of Alluvium	454,700
Total Waste Rock	4,695,400

3,334,000 3,516,000

TO 7/1/69

ANVIL MINING CORP., LTD.
Faro No. 1
Preproduction Pit - Ore to Stockpile

	High Grade		Low Grade	
Bench	Tons	Grade	Tons	Grade
4205	52,400	10.6	50,200	7.0
4170	164,900	12.5	120,700	8.7
4135	229,600	13.0	98,600	8.3
TOTALS	446,900	12.5	269,500	8.2

5500 TRD. x16 =

High grade - 446,900 Tons @ 12.5%

Low grade - 269,500 " @ 8.2%

TOTAL ORE 716,400 " @ 10.9%

Note: Grade is % combined Pb-Zn.

ANVIL MINING CORP., LTD.

Faro #1

Preproduction Stripping Pit
Alluvium Stripping Schedule

Total volume in preproduction pit 2,576,000 Cu. Yds.

Scraper capacity per operating shift:

2 619 Scrapers 957 Cu. Yds. X 2 = 1,914 Cu. Yds.

1 631 Scraper 1,156 Cu. Yds = 1,156 " "

Total . . . . . 3,070 Cu. Yds.

Since 100% of scraper fleet will be scheduled, an average unit availability has to be applied. Pima 1966 average unit availability for scrapers was 89%. Assume 85% to be reasonable for this operation.

Time schedule - Assume two 619 scrapers to start on July 1, 1967, and add the 631 on January 1, 1968.

From July 1, 1967 to January 1, 1968 - 153 working days available, or, at 3 shifts/day, 459 scheduled shifts. Using availability, operating shifts would be:

459 X .85 = 390 operating shifts.

Material moved - 390 X 1,914 = 746,000 Cu. Yds.

Material remaining - January 1, 1969 - 2,576,000 Cu. Yds.
746,000 " "

1,830,000 Cu. Yds.

Starting Jan. 1, 1968, number of operating shifts required, scheduling 3 scrapers per shift:

$$\frac{1,830,000}{3,070} = 596 \text{ shifts (2 scrapers/shift)}$$

or 1,192 - 619 shifts
596 - 631 shifts

No. of scheduled shifts - $\frac{596}{0.85}$ = 701 scheduled shifts.

No. of scheduled days @ 3 shifts per day

$$\frac{701}{3} = 234 \text{ scheduled days}$$

Estimated completion date - $\frac{234}{306}$ = 0.76 year, or 9 months.

Completion on October 1, 1968

Total scraper operating shifts:

	1967	1968	Total
619	780	1,192	1,972
631	--	596	596

Tractor Shifts -

1967 - 1/shift - 390 X 1 = 390 tractor shifts.

1968 - 2/Shift - 596 X 2 = 1,192 tractor shifts

Total ..... 1,582 tractor shifts.

	Loose Yards		Bank	35% swell
	Struck	Heaped		
619	12 c.y.	15.5	10	11
631	16	21	13	16

ANVIL MINING CORP., LTD.
Faro No. 1
Five Year Pit - Total Alluvium & Waste Rock Summary

Alluvium

Total estimated alluvium -	8,242,900	cu. yds.
Less: 15% to Wast Rock -	1,236,400	" "
TOTAL ALLUVIUM	7,006,500	cu. yds.

Waste Rock

Bench	Cu. Yds.
4310	237,700
4275	426,900
4240	698,100
4205	1,013,700
4170	1,896,700
4135	2,195,700
4100	2,290,900
4065	2,267,200
4030	2,142,800
3995	2,151,000
3960	1,354,600
3925	791,100
3890	459,300
3855	70,100
TOTAL	17,995,800
Plus 15% of Alluvium	1,236,400
Total Waste Rock	19,232,200

ANVIL MINING CORP., LTD.
Faro No. 1
Five Year Pit - Ore Summary

369,500 B.2. pre-production

Bench	Ore to Mill Tons	Ore to Mill Grade	Ore to Stockpile Tons	Ore to Stockpile Grade	Total Ore Tons	Total Ore Grade
H.G.Stkpile	446,900	12.5	--	--	446,900	12.5
4100	503,400	13.2	163,100	6.6	666,500	11.6
4065	370,300	11.7	418,300	6.6	788,600	9.0
4030	754,700	12.6	--	--	754,700	12.6
3995	856,400	10.5	11,300	7.4	867,700	10.5
3960	1,151,200	13.6	789,700	6.2	1,940,900	10.6
3925	2,013,700	11.5	301,900	6.9	2,315,600	10.9
3890	1,871,800	11.7	163,200	8.1	2,035,000	11.4
3855	1,890,900	12.7	--	--	1,890,900	12.7
3820	140,700	14.6	--	--	140,700	14.6
TOTALS	10,000,000	12.2	1,847,500	6.7	11,847,500	11.3
				Less: H.G. Stockpile	446,900	
				Total Ore from Pit	11,400,600	

Note: Grade is % combined Pb-Zn

ANVIL MINING CORP., LTD.
Faro No. 1
Five Year Pit
Alluvium Stripping Schedule

Total volume in preproduction pit - 4,430,000 cu. yds.

Average annual - $\frac{4,430,000}{5}$ = 886,000 cu. yds.

Scraper Capacity per operating shift -

2 619 Scrapers - 783 cu. yds. x 2 = 1566 cu. yds.

1 631 Scraper - 918 cu. yds. x 1 = 918 " "

TOTAL 2484 cu. yds.

Operating shifts required/year - $\frac{886,000}{2,484}$ = 357 or 1,071 Scraper shifts

Split - 357 - 631 shifts

714 - 619 shifts

Tractor shifts

one/operating shift = 357 shifts.

ANVIL MINING CORP., LTD.
Faro No. 1
Final Pit - Total Alluvium

Total estimated Alluvium	12,126,900 cu. yds.
Less: 15% to Waste Rock	1,819,000 " "
TOTAL ALLUVIUM	10,307,900 cu. yds.

ANVIL MINING CORP., LTD.
Faro No. 1
Final Pit - Total Waste Rock

Bench	Cu. Yds.
4310	237,700
4275	426,900
4240	959,700
4205	1,645,600
4170	2,879,900
4135	3,424,400
4100	3,812,600
4065	4,491,400
4030	4,768,000
3995	5,087,300
3960	4,761,200
3925	4,289,300
3890	3,750,900
3855	3,125,200
3820	2,752,300
3785	2,525,400
3750	2,141,300
3715	1,518,200
3680	732,400
3645	371,400
3610	252,500
3575	124,100
TOTAL	54,077,700
Plus 15% of Alluvium	1,819,000
TOTAL	55,896,700
Less: dilution	66,200
TOTAL WASTE ROCK	55,830,500

ANVIL MINING CORP., LTD.
Faro No. 1
Final Pit - Total Ore Estimate

Bench	Tons	Grade
4205	102,600	8.8
4170	285,600	10.9
4135	328,200	11.6
4100	666,500	11.6
4065	788,600	9.0
4030	754,700	12.6
3995	902,700	10.6
3960	2,022,800	10.4
3925	2,645,200	11.0
3890	2,592,600	11.5
3855	3,345,900	10.6
3820	3,264,400	10.4
3785	2,520,400	8.1
3750	2,125,200	9.2
3715	2,549,700	9.9
3680	2,762,400	10.5
3645	2,951,400	9.1
3610	1,864,000	7.8
3575	889,300	7.7
TOTAL	33,362,200	10.0
Dilution	210,700	2.2
TOTAL ORE	33,572,900	10.0

3,851,700
733,771

ASSAY TONS
9,568,620
21,037,120
29,097,200
29,814,900
35,466,540
33,949,760
20,415,240
19,551,840
25,242,030
29,005,200
253,148,450

Note: Grade is % combined Pb-Zn.

$$\frac{253,148,450}{24,731,300} = 10.24\%$$

ANVIL MINING CORP., LTD.
Faro No. 1
Final Pit - Included Waste Estimate

Section	Tons	Grade
3	816,100	3.0
4	199,100	1.6
	66,400	1.1
6	159,200	2.6
	119,400	3.2
	82,900	3.4
7	268,700	2.7
9	112,800	0.0
	282,000	0.1
TOTAL	2,106,600	2.2

% of Total Ore - $\frac{2,106,600}{33,572,900}$ = 6.3%

For dilution, use 10% of included waste as dilution factor (Same standard as Pima Mining Company)

Dilution - 2,106,600 x 10% = 210,700 Tons @ 2.2%

Deduct from waste rock $\frac{210,700}{3.18}$ tons = 66,200 cu. yds.

ANVIL MINING CORP., LTD.
Faro No. 1
Ore Estimate - Below Final Pit

Bench	Tons	Grade
3540	1,745,000	7.5
3505	1,727,800	10.1
3470	1,766,100	9.3
3435	1,676,900	8.0
3400	833,700	7.7
3365	505,400	6.6
TOTAL	8,254,900	8.5

Note: Grade is % Combined Pb-Zn.

Pennebaker
Oct. 1967

E. N. PENNEBAKER
CONSULTANT GEOLOGIST.
P.O. BOX 817
SCOTTSDALE, ARIZONA
85252

TELEPHONE 945-8352 CODE 602
RESIDENCE 945-6968

Office: 23 East First Street

October 3, 1967

Mr. R. E. Thurmond
Vice President and General Manager
Anvil Mining Corporation Limited
P.O. Box 2470
Whitehorse, Yukon Territory
CANADA

Dear Bob:

Enclosed are two copies of my ore reserve report. I am also sending an additional copy to your Los Angeles office.

My drilling recommendations will be mailed in a few days.

With best regards,

Yours sincerely

E. N. Pennebaker

ENP:mc
encls.

cc - Mr. Thurmond, Los Angeles

RECEIVED
OCT 10 19[illegible]
WHITEHORSE

ORE RESERVE ESTIMATE
OF THE
FARO MINERALIZED ZONE

THE ORE BODIES

As now known, the Faro mineralized zone consists of three units. Actually the No. 1 and the No. 3 units are parts of the same ore body offset by faulting. On the other hand, the No. 2 ore body, on the southeast, appears to be detached from the others and to form a separate unit, although future drilling might change this conclusion.

In the writer's estimate dated March 2, 1967, the No. 1 ore body was carried southeasterly beyond the Faro fault (to include ore on Section No. 2). At that time No. 3 appeared to be a detached ore body. Later drilling has joined together ore bodies No. 1 and No. 3, with the Faro fault serving as a break in continuity.

It is now proposed to call the ore northwest of the fault (beginning at Section No. 4) the "No. 1" ore body and the ore southeast of the fault the "No. 3" ore body, but keeping in mind that they are actually parts of one ore deposit.

The writer's estimate of March 2, 1967, was concerned with the amount of ore falling within an open pit designed by the Ralph M. Parsons Company and known as the "Tentative Ultimate Pit".

All of this ore was then known as the No. 1 ore body. Under the present proposal, this same pit would extract most of its ore from the No. 1 ore body, as now defined, plus a substantial amount from the No. 3 ore body. Future designs may extend the open pit mine to extract most of the No. 3 ore body.

The No. 1 ore body, as now defined, is about 2,400 feet long and 1,100 feet wide. The No. 3 ore body is about 2,300 feet long and 1,000 feet wide, although it is still open along its southwest flank and future drilling may prove it to be wider. These two bodies vary from 50 to 150 feet in thickness and are gently dipping and gently plunging.

The ore bodies are covered and masked by glacial sands and gravels with an average thickness of about 55 feet, under which the ore crops out on the buried bedrock surface. From this so-called sub-outcrop the Faro No. 1 part of the ore zone plunges down to the southeast under bedrock and attains a rock cover of some 300 feet where it meets the Faro fault. On the southeast this fault drops it down an additional 160 feet (on longitudinal section B). Farther to the southeast, what we are now calling the No. 3 ore body rises toward the surface and almost reaches it on the far southeast. Thus in longitudinal section this bi-partite ore body has the form of a basin.

No. 2 ore body is found about 500 feet south of the southeast end of No. 3. It is about 1,200 feet long and 1,000 feet wide, as now known, but is open on the southeast end. No. 2 ore body is thin and under a relatively shallow cover of overburden.

The relations of these ore deposits are shown on longitudinal section B, accompanying this report. In looking at this section it should be remembered that No. 2 ore body does not occur in the plane of this section; on the contrary it has been projected about 500 feet northerly to meet the section.

METHOD USED

The reserve estimate is based on vertical diamond drill holes. Most of these are located at the corners of grids that are 283 feet square, although a few are located at half intervals.

A cut-off grade of 5% combined lead and zinc was used, along with a tonnage factor of 8.5 cubic feet in place equals one short ton.

The drill holes were plotted on vertical cross sections, the ore intervals were set down, and the ore blocks and their grades were determined. The ore blocks and internal waste derived on the cross-sections were measured by planimeter, and the measurements were reduced by the above tonnage factor to give their sizes. The metal content of the various blocks was then used to obtain a weighted average grade for lead, zinc and silver.

In his estimate of March 2, 1967, the writer found 38.87 million tons of ore to fall within Parson's "Tentative Ultimate Pit". The following estimate does not give the amount of ore falling within any particular open pit design. It does provide an estimate of all the ore discovered to date.

All of the ore presently in the No. 1 ore body is judged to be "assured ore" whose existence and grade can be looked upon with confidence. At the margins of the bodies this ore has been projected out no more than one-half the grid interval (141 feet) and in some places much less. Where ore has been projected farther than this amount (in ore bodies Nos. 2 and 3) it is called "possible" ore, whose existence and grade must be confirmed by additional drilling.

ORE RESERVES

As above mentioned, the Faro No. 1 ore body goes southeasterly to the Faro fault. The No. 3 ore body, as here defined, continues on southeasterly beyond the fault.

TONNAGES OF ASSURED ORE, NOT DILUTED

Ore Body	Tons	% Pb	% ZN	Combined Pb & Zn	Oz. Ag.
1	33,685,529	3.881	6.361	10.242	1.230
3	23,301,661	2.922	5.236	8.158	1.131
2	5,159,335	3.144	4.844	7.988	1.462
TOTAL	62,146,525	3.460	5.814	9.274	1.212
1 + 3 Ore Bodies	56,987,190	3.489	5.800	9.289	1.189

TONNAGES OF POSSIBLE ORE

Ore Body	Tons	% Pb	% Zn	Combined Pb & Zn	Oz. Ag.
3	2,553,914	2.812	5.093	7.905	0.893
2	635,558	2.776	5.794	8.570	1.120
TOTAL	3,169,472	2.805	5.233	8.038	0.939

TONNAGES OF INTERNAL WASTE

Ore Body	Tons	% Pb	% Zn	Oz. Ag.
1	1,738,219	0.690	1.248	0.348
3	2,683,167	0.927	1.476	0.522
2	None			

TONNAGES OF ASSURED ORE DILUTED BY

30% OF INTERNAL WASTE

Ore Body	Tons	% Pb	% Zn	Combined Pb & Zn	Oz. Ag.
1	34,206,995	3.833	6.284	10.117	1.217
3	24,106,611	2.856	5.115	7.971	1.110
* 2	5,159,335	3.144	4.844	7.988	1.462
TOTAL	63,472,941	3.405	5.721	9.126	1.196
1 + 3 Ore Bodies	58,313,606	3.429	5.799	9.228	1.173

* There is no internal waste in No. 2 ore body.

DISCUSSION

DILUTION

During the mining of these ore bodies, dilution will come from two main sources: (1) "internal waste" occurring within the ore bodies and (2) top and bottom waste above and below the ore bodies. Internal waste occurs as layers that average 30 or 40 feet in thickness. It is roughly estimated that about 30% of this will contaminate the ore as it is mined. This relatively high figure is used because of the thinness of the waste layers.

The amount of dilution caused by top and bottom waste has not been estimated for this report.

CHANGES IN TENOR

The grade of Nos. 2 and 3 ore bodies is fairly constant at about 8% combined lead and zinc. On the other hand, No. 1 ore body,

on the northwest, averages about 10%. Going northwesterly through No. 1 ore body there is also a general increase from Section No. 4 (at 9.148% combined lead and zinc) to Section No. 10½ (12.933%). Although Sections Nos. 11 and 11½ show a drop in grade, this may be due to oxidation and leaching near the sub-apex of the ore body.

FUTURE EXPANSION OF RESERVES

The writer expects that the "possible ore" category will be converted to assured ore by continued exploration.

Further drilling along the southwest flank of No. 3 ore body and beyond the southeast end of No. 2 is most promising. Attractive exploration possibilities also exist westerly from the northwest end of No. 1 ore body. Definite drill sites in these areas will be recommended in a forthcoming report.

Additional exploration in the Faro mineralized zone is very attractive.

E. N. Pennebaker

Scottsdale, Arizona
October 3, 1967

Preproduction
Pit Redesign 1968

M.O.H.

Anvil Mining Corporation Limited

To Mr. C. H. Macdonald From J. F. Olk

Subject Mine Engineering Studies Date January 30, 1968

As indicated in my Telex of January 30, Mr. Vernon Gras will arrive in Whitehorse on Friday, February 2. Mr. Gras will be employed on a temporary basis with Anvil Mining Corporation for a period of three months. This may or may not be extended depending upon Mr. Gras' progress in his application for an assignment in India with Parsons Co. During the time that Mr. Gras is with Anvil he will, as we have discussed, assist in the engineering study relative to a re-design of the preproduction pit and of the subsequent five-year pit if time permits. Mr. Gras must work closely with Murray Hampton on all aspects of this mine planning work so that Murray will receive the benefit of Mr. Gras' experience in this field.

With regard to this engineering planning of the preproduction pit the following is a brief outline of my thoughts of what must be accomplished along these lines. It is most important that this be done immediately since we will be in a position to start the rock stripping very shortly.

The basic objective, of course, in this new planning is that any re-design preproduction pit will entail the same dollar cost for stripping as was indicated in the feasibility study. These are the points that will have to be reviewed, recalculated and re-designed as required:

1) The new rock alluvium contact as developed by the recent drilling program will have to be analyzed and this information put on the geological sections and a new rock alluvium line established. As in our past treatment of this alluvium rock contact, a certain percentage of the alluvium will have to be deducted and added to rock to compensate for a certain amount of gradational material at the contact which may be soft enough to drill but too hard to rip. My thinking at this time along these lines is that this should be in the range of five to ten percent although you may want to discuss this among yourselves in light of the recent drilling and the problems in defining the exact contact.

2) Operating cost estimates for mine equipment must be recalculated in the light of new labor rates provided in the labor contract, new tire costs as provided in the tire contract and fuel costs that are now definitely established.

3) Drilling and blasting criteria must also be reviewed, particularly in the light of definite explosive prices obtained from the Canadian Industries Ltd. proposal.

4) Equipment productivity, namely shovel and scraper productivity, will have to be recalculated on the basis of our proposed ten-hour work shift.

A review of the alluvium stripping position in light of the new quantities established by the new rock alluvium contact, particularly in regard to scheduling of scrapers will have to be made. My thoughts are that it is probably feasible to drop one scraper from the schedule. Along with this it may be possible to drop the scrapers from the stripping schedule for two or three months during the extreme cold weather when frost conditions contribute to extreme wear on equipment.

6) A new preproduction pit must be planned and in order to accomplish our objective of keeping the preproduction cost within the feasibility, certain changes must be made. These could include, among others, the following:

(a) The objective of six-months ore availability, including ore in stockpile rather than the year's ore availability used in the initial planning. This will reduce the total size of the pit, hence the volume.

(b) The ramp grades could be steepened from the original design of 7% to 8% and the flat portion left on each bench could be eliminated so that the ramp is a continuous one from top to bottom.

(c) The working slopes could be steepened to a degree.

(d) The bench heights could be increased from 35 feet to 40 feet. However, I would prefer that this last one be utilized only in case of extreme necessity.

(e) I believe that the pit design must incorporate the finalizing of the northwest corner, the same as was done in the initial design. However, further thought and planning may indicate that some modification could be made of this concept.

7) Dump planning will have to be reviewed; new dump areas layed out, and access roads to these dumps planned.

I think that one of the necessities for much of this planning is the preparation of a map covering the pit area showing all existing roads, cuts, diamond drill holes, contours, claim lines, etc. I think that the preparation of this should be initiated. In the planning and estimating of the preproduction pit and the subsequent five-year pit the horizontal bench plans as used in the original mine study can be used.

I hope that this brief outline will suffice to give Mr. Gras and Murray Hampton adequate basis from which to commence work. I plan to return to Whitehorse within a week or ten days and we can review progress at that time and plan for subsequent work.

FO:mjf
cc: RET, MH, VG

March 1, 1968

REDESIGN OF PREPRODUCTION PIT - FARO NO. 1

At the request of Messrs. R.E. Thurmond and J.F. Olk on January 30, 1968, an engineering study was initiated by Murray Hampton and Vernon Gras on the above subject. Costs, production ~~rates~~; alluvium stripping position, ore stockpile, ore availability, waste rock quantities were reviewed as follows:

ALLUVIUM

A new subsurface contour map was made using only recent test hole drilling criteria. The quantity of alluvium amounted to 1,728,000 cubic yards as compared with 3,031,100 cubic yards using diamond drill hole data for the rock-alluvium contact. The smaller redesigned pit further reduced this amount to 1,562,700 cubic yards.

Production rates and quantities of alluvium remaining to be removed as well as costs were estimated from March 1, 1968 to July 1, 1969. The 619 scraper factor was reduced from 15 c.y. to 10 c.y. and the 631 scraper from 18 c.y. to 13 c.y. These factors are bank yards and a swell factor of 50% was used. It should be pointed out that the swell factor may vary plus or minus 10% depending on soil conditions. It was the opinion of the geology staff that a good alluvium-caprock contact was established from the test hole drilling making it unnecessary to subtract any amount from alluvium and allocate this to waste rock. Alluvium quantities were estimated as follows:

	Cubic Yards
Total estimated alluvium	1,728,000
Less amount for reduced pit	165,300
Initial amount to be removed	1,562,700
Estimated removed to March 1, 1968	374,600
Amount remaining March 1, 1968	1,188,100

555,000 reported

This amount of alluvial should be removed in 12 months to keep ahead of the waste shovels. It was estimated that 2 of the 3 scrapers would be scheduled each shift, utilizing the 631 scraper 80% and each of the 619 scrapers 60% of available time to take advantage of the more productive 631 scraper.

/2

ORE AVAILABILITY

The original objective of one year's ore availability, including ore in the stockpile, was reduced to six months in the redesigned pit. The low grade ore will become available as the east pit wall is moved back.

WASTE ROCK

The original preproduction pit including alluvium, waste rock and ore contained a volume of approximately 7,500,000 cubic yards. The redesigned pit contains a volume of approximately 5,350,000 cubic yards. Of this amount, waste rock is estimated at 3,569,300 cubic yards.

Using April 1, 1968 as the startup date for 2 waste shovels working 2 ten hour shifts per day 6 days per week, 1560 shovel shifts are available up to July 1, 1969. Of the 1560 shovel shifts available, 1231 were scheduled for waste rock and 84 shovel shifts for ore to stockpile. This amounts to 84% utilization.

Costs for operating labor and supplies account for the largest increase in shovel and truck costs due to the new wage contract, diesel fuel tax and increased tire prices.

PIT DESIGN

It was necessary to use 8% ramps in the pit and eliminate the flat spot at the 4135 level on the main haul road, in order to reach the ore on the 4100 level. As mining continues after July 1, 1969 and the pit enlarges, it is recommended that 7% ramps and 100 ft. level spots be incorporated into the mining plan.

Bench heights were maintained at 35 feet. It seems advisable to use this bench height with 5 cubic yard shovels.

DRILLING AND BLASTING

Costs and footage rates were evaluated on a 10 hour shift basis and using somewhat higher costs for blasting agents. It was estimated that 30 percent of the holes would be wet requiring a higher cost explosive.

GENERAL PIT EXPENSE

A flat 10 percent increase was applied to the unit cost of material moved for general pit expense. Total cost was previously calculated on a time basis of 15 months.

ANVIL MINING CORPORATION LIMITED

MATERIALS SUMMARY

Preproduction Pit

LEVEL	WASTE ROCK Cubic Yards	ORE			
		High Grade		Low Grade	
		Tons	Grade	Tons	Grade
4345	66,300				
4310	442,000				
4275	721,500				
4240	822,500				
4205	742,700	52,400	10.6	50,200	7.0
4170	540,700	164,900	12.5	120,700	8.7
4135	182,100	267,500	12.7	35,000	8.3
4100	51,500				
Totals	3,569,300	484,800	12.4	205,900	8.1

Note: Grade is % combined Pb-Zn

MEMORANDUM

To: C.H. Macdonald

From: M.O. Hampton

Date: September 25, 1968

Subject: PREPRODUCTION PIT REDESIGN

The preproduction pit has been redesigned to take advantage of leaving ore in place and providing drainage without pumping. As a result of the new design it was possible to leave wider benches where equipment has to resume working at a later date and reduce the grade of the proposed haul road to 6%.

In order to leave the ore in place it is necessary to mine from the side of the body rather than from above if the amount required by the mill is to be maintained. The side approach also lends itself to providing drainage from each bench. Relatively narrow elongate workings are required but their disadvantages are outweighed many times by the advantages provided.

The required amount of work can be done within the preproduction period mining budget and within the capacity of the shovels by planning the start of a third shovel at March 1, 1969.

A cost per cubic yard of $0.50 and a monthly production of 150,000 cubic yards per shovel was used for this study. Both figures appear to be on the conservative side at present but further experience, especially of winter and spring breakup operation, should be gained before any less conservative figures are used. If better production and cost can be maintained a larger amount of waste can be removed during the preproduction period.

This design and calculation was made on the assumption that the scrapers would not be put back into service. However, if conditions permit satisfactory operation at costs better than shovel costs, their use could postpone the start-up date of the third shovel and would require no change in design within the pit.

Bench development should be scheduled in such a way that a shovel is working on 4100 not later than June 30, 1969. Such scheduling is feasible.

The expansion of this pit during the first year of production will require the operation of 6 shovel shifts in waste and 1 shovel shift in ore each day (8 hr. shifts, 6 day week) in order to supply 2,000,000 tons of ore grading 12% combined Zn and Pb. This rather high rate of waste removal requiring more shovel shifts than previously planned appears to be the main disadvantage of this design. However, all of the material is within the 5-year pit and has to be moved in any event.

.....

The pattern of daylighting each bench to provide drainage can be maintained to at least the 3995 bench and if economically desirable even to the 3925 bench (approximately adit level).

The new design changes the preproduction pit boundary somewhat but not so much as to cause any great problem in regard to work already done.

Accompanying please find volume data for preproduction pit and first year production pit, first year ore supply data and 100 scale prints of redesigned preproduction pit and first year production pit.

Presented for your review and comment.

MOH

M.O. Hampton,
Chief Engineer.

MOH/ab

cc: J.F. Olk
D.W. Burns

PREPRODUCTION PIT (September 25, 1968 design)

BENCH	CU. YD.
4345	471,000
4310	889,000
4275	1,340,000
4240	1,662,000
4205	898,000
4170	509,000
4135	414,000
4100	222,000
Total waste rock, alluvium, and minor ore	6,405,000

Total	6,405,000 cu. yd.	
Moved to Aug.31/68	1,906,000 " "	
Remaining	4,499,000 cu. yd.	(To be removed by Sep. 1/69)

Budget to Aug. 31/69	$3,287,000	
Spent to Aug. 31/68	$1,029,000	
Remaining	$2,258,000	≃ 4,500,000 cu. yd. @ 50¢ cu. yd.

2 shovels Sept. 68 thru Feb. 69 = 6 mos. @ 300,000 = 1,800,000 cu. yd.
3 shovels March thru Aug. 69 = 6 mos. @ 450,000 = 2,700,000 cu. yd.
4,500,000 cu. yd.

FIRST YEAR PRODUCTION PIT

BENCH	CUBIC YARDS
4345	471,000
4310	889,000
4275	1,340,000
4240	1,665,000
4205	1,710,000
4170	1,535,000
4135	1,381,000
4100	1,108,000
4065	752,000
4030	334,000
	11,185,000
Removed during preproduction	6,406,000
Remainder	4,779,000
Ore	- 636,000
Alluvium, waste rock, & L.G. Ore	4,143,000

Requires 6 shovel shifts in waste
1 shovel shift in ore.

FIRST YEAR ORE SUPPLY

BENCH	TONS	GRADE (% comb. Zn & Pb)
4205	52,000	10.6
4170	165,000	12.5
4135	230,000	13.0
4100	503,000	13.2
4065	370,000	11.7
4030	331,000	14.4
Various	349,000	8.0
Total	2,000,000	12.07

COMPARISONS

New First Year Production - Alluvium, Waste Rock & L.G. Ore			4,143,000 cu yd
Original Plan (J.F.O.) - Avg. Waste Rock		2,908,000 cu yd	
Avg. Alluvium		886,000	
Avg. L.G. Ore		116,000	3,910,000 cu yd
	DIFFERENCE		233,000 cu yd

Scheduled for removal in original pre-prod period		7,497,000 cu yd
April 68 design pre-prod period		~~5,891,000~~ 6,469,000 cu yd
	DIFFERENCE	~~1,606,000~~ 1,028,000 cu yd

Original pre-prod period		7,497,000 cu yd
September 68 design		6,405,000 cu yd
	DIFFERENCE	1,092,000 cu yd

September 68 design - pre-prod period		6,405,000 cu yd
April 68 design - pre-prod period		~~5,891,000~~ 6,469,000 cu yd
	DIFFERENCE	~~514,000~~ cu yd

First year production stripping schedule was not calculated in April but it appears it would have requirements in excess of September design

MOH
September 30, 1968

PRE-PRODUCTION PIT

ALLUVIUM	Yards	$	Unit Cost
Nov 1/67 - Feb 29/68	554,973	261,666	0.472
March 68	122,300	76,300	0.623
April 68	103,000	46,800	0.455
~~May~~ 68 - Nov 30/68	616,000	234,500	0.380
Mar 1/69 - Aug 31/69	528,000	201,000	0.380
	1,924,~~9~~73	820,266	0.425

WASTE ROCK & ORE			
April 68	50,000	55,000 ✓	1.10
May 68	145,000	72,500 ✓	0.50
June 68 - Nov 30/68	1,740,000	870,000 ✓	0.50
Dec 68 - Feb 28/69	870,000	480,000 ✓	(1)0.55
Mar 69 - Aug 31/69	1,740,000	958,000	(2)0.55
	4,545,000 ✓	2,435,500	0.536

Ore Mines 700,000 tons = 220,000

Waste Mined (yds) 4,325,000

PREPRODUCTION TOTAL

Alluvium	1,924,~~9~~73 yards
Waste Rock	4,325,000 yards
Ore Mined	700,000 tons
Total Cost	$3,255,766

April 23, 1968

ew

M. O. Hampton

Anvil Mining Corporation Limited

To C. H. Macdonald From J. F. Olk

Subject Preproduction Pit Redesign Date November 11, 1968

Referring to M. O. Hampton's memorandum of September 25, 1968, on the above subject, I have a number of comments on this after a review of the plan.

There are advantages to this plan, as well as some disadvantages to consider. The major ones of both are:

1. Advantages

   a. A very minimum of ore, and that practically all low grade, is stockpiled during the preproduction period. In view of the results obtained by Galigher in the aging tests on Anvil ore, it seems almost mandatory that stockpiling of ore, especially high-grade ore, be avoided if at all possible.

   b. Mine drainage, without pumping, is provided through the preproduction period and the first year of ore production.

   c. The ore haul road to the primary crusher can be reduced to a 6% grade.

2. Disadvantages

   a. The change in pit layout reduces the working area for the shovel operation, especially the two lower benches, the 4135 and 4100 benches. Both these are long, narrow cuts, with the width of the 4100 bench cut being 115 feet. Shovel production efficiency would be affected in a long cut of this width.

   b. More total material must be moved in the preproduction period. than that estimated in the April redesign. This proposal calls for a total of 6,405,000 cubic yards of alluvium, waste rock, and minor ore stockpiling as compared with 5,891,000 cubic yards in the April redesign.

6,469,000

This extra volume would have to be moved by starting a third shovel shift during the preproduction period. This material, however, is within the five year pit area, and is not extra volume in the overall mining program.

Before any decision can be made on this proposal, some further work and study should be done. In this layout, the 4275 and 4240 benches, which are presently being worked, have the same limits as the April redesign. Therefore, the final decision on this plan can be deferred until the 4275 bench is completed, which will probably be in February, at the present rate of shovel production. So prior to the first of February the following factors should be studied further:

1. Shovel scheduling will be somewhat complicated by the fact that only one shovel can work the lower benches at a time due to the narrow cuts. This will cut down the shovel productivity available for the preproduction period during the last months, unless material is scheduled to be left on some higher benches for removal during this period. A complete month by month shovel schedule should be worked out to check the feasibility of this.

2. The daylighting of the 4100 bench and possibly lower ones poses some problems in haulage to dumps for the material on the eastern side of higher benches. It practically forces this to be dumped in the Faro Creek valley. This aspect should be studied very carefully, and critical items to consider would be the available capacity of the valley, haulage roads and grades required, and the most economical and practical dump layout.

3. Equipment scheduling and productivity should be reviewed. This proposal calls for no scraper scheduling, and a third shovel starting on March 1, 1969 is to continue until September 1, 1969. This raises the following questions which should be resolved prior to February 1, 1969:

   a. From all indications thus far, the quantity of actual alluvium overburden amenable to scraper removals seems to be constantly diminishing. Alluvium-rock contacts are being found higher than even the revised contact based on further Airtrac drilling. From the appearance of the upper oxidized rock in the banks, it would seem that the fine cuttings from an Airtrac drill could be misinterpreted since they appear the same as alluvium. At the same time, alluvium has been removed by shovels. This raises the question of whether or not it is actually feasible to continue the scraper operation. Can shovels remove it at the same or lesser cost, and is capacity available?

b. If scrapers are discontinued from the stripping program, shovels will have to provide the missing capacity. So far, shovel productivity has exceeded the scheduled yardage, and the question here is can it continue during the extreme cold months, and if so, can it compensate for the missing scraper yardage? By mid-January we should have a reasonable answer to these questions.

c. If more rock is to be moved, both from the fact of increasing the preproduction yardages and the finding of more rock than alluvium, how is the budget for preproduction stripping affected? Will increased shovel productivity for the same dollar cost compensate for the increased yardages? A clearer picture of this should be available by the end of the year, also. This is one area that has to be very carefully studied.

d. A third shovel has been scheduled for March 1, 1969 in this proposal. If present productivity continues, will stripping be sufficiently ahead that this date can be moved back? Can we establish a higher productivity rate for shovels that could move this date further back?

e. Will a third shovel be available on March 1, 1969, to work full time? Will there be enough trucks?

We might have the answers to some of these questions now, but by January, we should have a firmer basis for the answers to all of these. In the meantime, thought should be given to these questions. In summary, the basis for the decision to adopt this plan will rest on whether the stripping capacity is available and it can be done within the preproduction stripping capital budget.

Also, this plan should be expanded into a five year plan to verify that tonnage and grade are definitely available. The first year has been worked out and provides the required grade, but some work should be done for the ensuing years. Also, the quantity of ore developed and available at the start of ore production should be calculated.

JFO:en

cc: R. E. Thurmond
R. L. Cook
D. W. Burns
M. O. Hampton
E. M. Lehbauer

MEMORANDUM

To: C.H. Macdonald

From: M.O. Hampton

Date: December 10, 1968

Subject: PREPRODUCTION PIT REDESIGN

1. Referring to J.F. Olk's memorandum of November 11 on this subject under disadvantages, section (b). The figure of 5,891,000 cubic yards quoted as the total material that was estimated to be moved during the preproduction period in the April redesign report is based on a typographical error. In the summary on page 3 of that report, the figure 3,750,000 should read 4,328,000 (that is 3,750,000 in the pit design plus the 578,000 estimated beyond the pit design during the preproduction period). The actual estimated volume in the April report was 6,469,000 cubic yards as compared with 6,405,000 cubic yards in the September design.

2. The extra shovel capacity required by the September redesign is from anticipating no further use of the scrapers as a major tool in the pit. Production by the shovels during September, October and November exceeded the figure of 150,000 cubic yards per month per shovel used in my calculations. This together with a savings of 200,000 cubic yards on the 4275 bench by steeping the slope to 45° moves the required start up date for the third shovel back to about June 1 rather than March 1. Future production in excess of the plan will probably move the required date for a third shovel even further back. However, such extra production should not be planned on at this time. Moving a third shovel into the pit after NCPC power is available will give the Mine Department greater flexibility even if the operation of extra shovel shifts is not required.

3. The dump area up Faro Creek is capable of taking only approximately 1,000,000 bank yards. The area between the "Final" outline and the adit area is relatively small and is to be crossed with several future roads, therefore should not be considered for dump area at this time.

.....

Some material can probably be dumped in this southern area starting about the 3rd year of production. It should be planned to take the bulk of the waste material to the west. This will entail some relatively long hauls and could conceivably involve some hauling down and then up again. The latter, hopefully, can be kept to a minimum and possibly even avoided with careful planning and scheduling as the pit develops.

M.O. Hampton
Chief Engineer

MOH/ew

cc: J.F. Olk
D.W. Burns

MEMORANDUM

To: C.H. Macdonald

From: M.O. Hampton

Date: December 17, 1968

Subject: PREPRODUCTION PIT DESIGN

The preproduction pit design and a possible shovel schedule has been worked out based on the September 1968 design. It should be noted that the design uses a final pit slope of 45° rather than the 40° used earlier. The scheduling takes into account the production over estimate during September, October and November but does not take into account such probable production in the future. A production of 150,000 cubic yards per shovel per month was used except on 4100 where 111,000 was used.

As noted in my memo of December 10 future production over estimated could move the start up of the third shovel back even further than June 1, as suggested at present.

This plan has four benches in ore at September 1, 1969 (4205, 4170, 4135 and 4100).

The schedule enclosed herewith is only one of many possibilities and is worked out to illustrate that a workable schedule is possible. The particular example is controlled by 4275 remaining static (thus limiting 4240 expansion) until the lower benches are completed to the minimum requirements and by using only one shovel on a bench at a time.

Unless serious blending problems are encountered causing irregular development, the bulk of the waste rock can be moved along the benches and out to the west without having to be hauled down and back up again.

Results of the proposed fill-in drilling may cause some alterations as work progresses.

There is a minimum of some 3,000,000 cubic yards required to be moved by September 1, 1969. At November 30, 1968 the preproduction stripping budget stood at $3,295,000 - $1,548,000 = $1,747,000. The designed volume can be handled within the budgeted funds with a reasonable cushion for expected extra wear and tear during extreme cold and break-up operations.

M.O. Hampton
Chief Engineer

Enclosure

cc: J.F. Olk D.W. Burns

Mr. C. H. Macdonald

J. F. Olk

Preproduction Pit Redesign

January 30, 1969

In reviewing the above subject, particularly Murray Hampton's memos of December 10 and 17, 1968 on this subject, I am inclined to agree that the proposed redesign is the route to take, particularly since the revised redesign, with steepened slopes, reduces the yardage involved in the preproduction stripping. All of the advantages listed in my memo of November 11, 1968 of this redesign still apply, and the disadvantages have been or can be overcome.

According to Murray's memo of December 17th, there was a minimum of 3,000,000 cubic yards of waste and low-grade ore remaining in the redesigned pit as of the end of November, 1968. The total yardage, then, of alluvium, waste rock, and low-grade ore in the redesigned pit would then be:

Remaining - 12/1/68	3,000,000 cubic yards
Removals to 12/1/68	3,288,000 cubic yards
Total scheduled	6,288,000 cubic yards

This compares with 6,469,000 cubic yards presently scheduled for the preproduction pit, or 181,000 cubic yards less.

As of the first of the year, the remaining yardage would be:

Remaining - 12/1/68	3,000,000 cubic yards
December Removals	262,000 cubic yards
Remaining - 1/1/69	2,738,000 cubic yards

Using the present scheduled shovel productivity of 72,500 cubic yards per shovel shift per month (290,000 cubic yards per month with four shovel shifts per day), the present four shovel shifts per day can continue to June 1, 1969, at which time the scheduling must go to six per day. Any overproduction between now and June 1st will move the date for starting the additional shovel shifts back.

Applying the unit cost of $0.55 per cubic yard for four shovel shifts per day, producing 290,000 cubic yards per month, which is the cost being used in the present plan, the total cost of the preproduction stripping would be:

Costs to 1/1/69	$ 1,683,000
2,738,000 cubic yards at $0.55	1,506,000
Total preproduction cost	$ 3,189,000

This compares with the present budget:

Present Preproduction Budget	$ 3,295,000
Revised Preproduction Budget	3,189,000
Saving	106,000

Again, any overproduction during the preproduction period will result in further cost savings. I would not favor projecting the unit cost to date of $0.474 to the remaining yardage, since it is still possible that we will incur some tire expense for the trucks during the preproduction period, and the cost to date reflects very little tire activity. I think that, with continued close attention to prolonging tire life, close control of drilling and blasting costs, and increased production over scheduled, an additional $100,000 can be saved in the preproduction stripping cost. This should be a goal to strive for.

The final details of this redesign will depend, of course, on the outcome of the test drilling program in the upper ore blocks. If the tonnage and grade used in the estimates is verified, there would be no problem. However, if the reverse is true, we will have to take a long, hard review of the entire preproduction stripping program. This is why I am particularly anxious that this testing program be given as much priority as possible. When this is complete, the final decision as to changing the schedule and cost of the preproduction stripping program can be made.

JFO/ps

cc: Messrs. D. W. Burns
R. L. Cook
M. O. Hampton
E. M. Lehbauer
R. E. Thurmond

MEMORANDUM

TO: N. Cornish

FROM: M. O. Hampton

DATE: February 10, 1970

SUBJECT: ORE RESERVE COMPARISONS JANUARY 31/70

The following table illustrates the predicted ore reserves and the amounts produced to date from completed benches.

Bench	Counted by JFO	Produced
4240 & 4275	0 Tons	19,800 Tons
4205	102,600	80,625
4170	285,600	123,905
4135	328,200	157,560*
4100	666,500	157,950*
4065	788,600	121,940*

*Some ore still to be mined from expansion.

M. O. Hampton
Chief Engineer

MOH/kd

cc. J. F. Olk
J. McLachlin

ORE RESERVE COMPARISON

BASED ON J.F.O. MINING STUDY

Actual Losses Bench	(Complete Benches) Tons of Ore Mining Study	Tons of Ore Produced	(Gain) or Loss
4240-4275	0	19,800	(19,800)
4205	102,600	80,600	22,000
4170	285,600	123,900	161,700

Estimated Losses (Incomplete Benches)

Bench	Tons of Ore Mining Study	Tons of Ore Produced	Theoretical Ore Remaining	Estimated Net Loss
4135	328,200	157,600	98,000	68,600
4100	666,500	158,000	163,000	345,500
4065	788,500	122,000	300,000	366,500
			Total Loss - Actual and Estimated	944,500

ORE RESERVE COMPARISON

BASED ON RMPCo FEASIBILITY STUDY

Actual Losses

Bench	Tons of Ore — Feasibility Estimate	Tons of Ore Produced	(Gain) or Loss
4240 & 4275	0	19,800	(19,800)
4205	162,500	80,600	81,900
4170	444,500	123,900	320,600

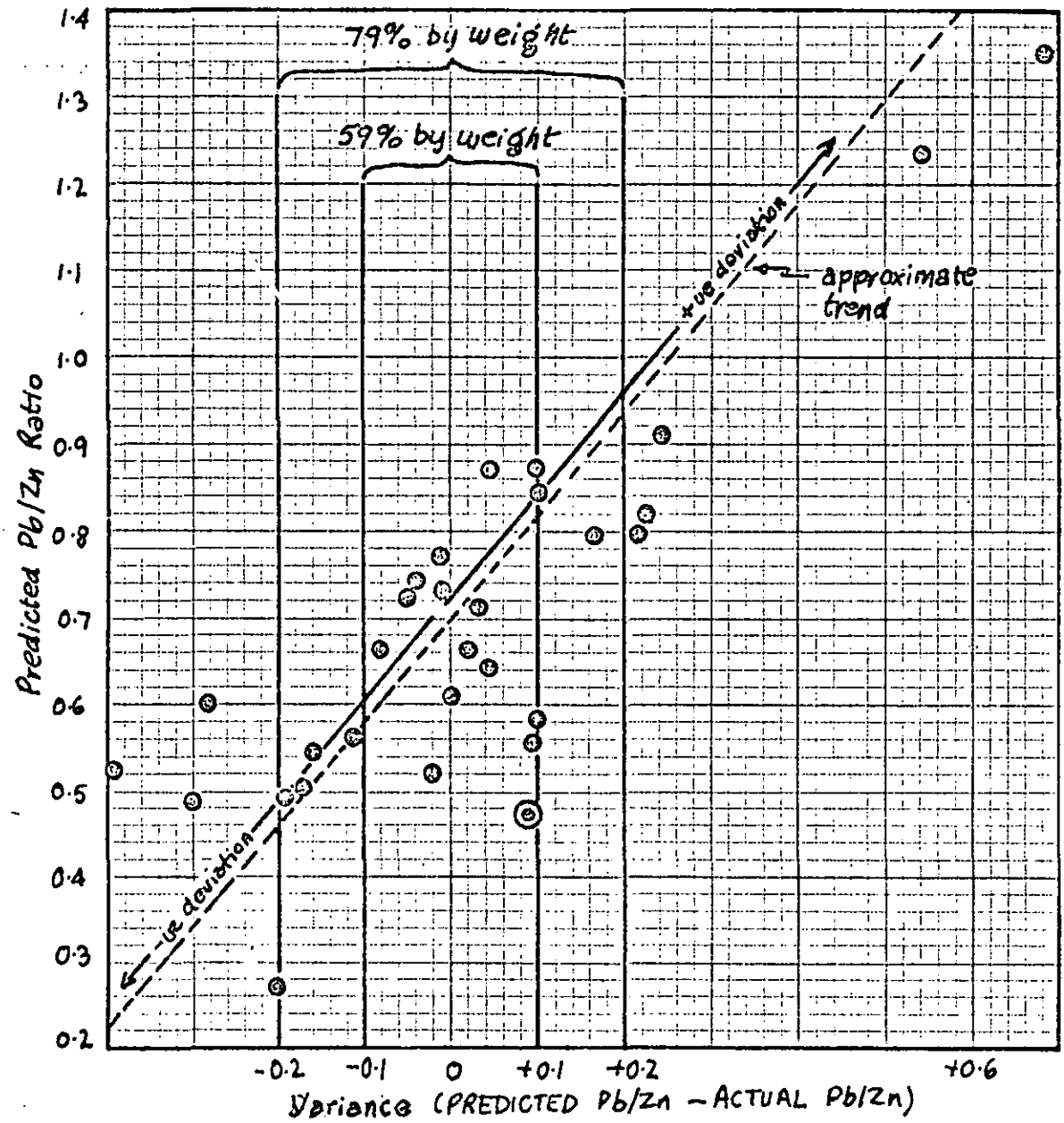
Estimated Losses (Incomplete Benches)

Bench	Tons of Ore Feasibility Estimate	Tons of Ore Produced	Theoretical Ore Remaining	Estimated Net Loss
4135	372,000	157,600	98,000	116,400
4100	540,900	158,000	163,000	219,900
4065	737,600	122,000	300,000	315,600
		Total Loss - Actual and Estimated		1,034,600

JFO - 5 Yr. Plan
1970

higher and lower predicted values seem to be increasingly unreliable.

VARIANCE GRAPH



FIVE YEAR PIT PLAN

The Five Year Pit Plan is designed for the five year period starting on January 1, 1971 and continuing until December 31, 1975. This plan does not include the present 1970 plan, but the five year period beyond.

Attached are Appendices summarizing the various aspects of the plan, and following are comments concerning these aspects:

1. Ore Summary

   A careful reappraisal of the ore benches involved in this plan was made based on the information developed in pit removals to date, review of geological interpretations, and further diamond drilling. It is felt that the ore blocks thus developed for this plan are realistic and the estimated ore tonnages and grades shown are reasonable.

   This plan shows that a grade of mill feed of 12% combined lead and zinc can be maintained for 3.2 years, or until early in 1974. In order to maintain this grade, an estimated 1,369,000 tons of lower grade ore must be stockpiled. After this period, the grade drops to 9.0% for the remaining 1.8 years, and the tonnage requirements were raised 50%, in line with the planned mill expansion.

2. Waste Summary

   All waste in this study was treated as rock. Although there may be some areas of alluvium that could be dug without drilling and blasting, the total volume involved would be minor, so that this treatment lends some conservatism to the estimate.

   The pit layout on which this estimate was based was designed with 45 degree final slopes on the north and west sides, and 26 degree working slopes on the east and south. A portion of the north side that shows as final in this plan will have to be reopened for subsequent operations, but adequate room would be developed in the reopening for efficient operations, so this should present no serious problems. The ramp grades used were 8% throughout.

   If ore to stockpile is considered as part of stripping, the average stripping ratio during this period is 5.7 cubic yards waste per one cubic yard ore to mill.

3. Equipment Scheduling

   The waste and ore removals scheduled for this period are within the capacity of the present shovel fleet, as shown in this Appendix. An average of nine shovel shifts per day, six days per week is capable of the required production. This would mean a shovel fleet utilization of 75%, which is not excessive, and is less than that scheduled for 1970.

   Truck requirements go up with the increasing depth of the pit and longer hauls. An average utilization of 80% of the truck fleet is indicated during this period, which can be achieved if proper and efficient maintenance is carried out.

There may be periods when the utilization may go as high as 87%, when two lower benches are worked simultaneously. Careful scheduling will have to be done so that these periods coincide with the maximum truck availability.

The utilization of present drilling capacity will be 83%, which again is high, but can be obtained with good maintenance. However, travel time may start to become a major factor during the latter part of this period, with the pit expanded and deepened, and the possibility exists that a third drill could be required before the end of this five year period. Drilling capacity is something that should receive constant attention, and if additional capacity is required, the need must be foreseen in adequate time to prepare an AFE justification and obtain delivery.

At some point in time during this period when the financial situation is appropriate, a serious look should be given to the possibility of replacement of some or all of the present mining equipment with larger equipment. There is the definite possibility that substantial cost savings could be achieved, and this should be thoroughly investigated.

4. Annual Mining Cost Estimate
The cost figures developed in this estimate were based on a labor cost using the highest rates shown in the present labor contract. It was felt that this rate would be more appropriate to apply to this period than the present rates. This, along with increased haulage truck scheduling, increases the mine loading and transport cost over the present costs. Drilling and blasting costs were taken from the present year-to-date costs. Since labor is a relatively small percentage of total drilling and blasting costs, the labor rate increase over the present rate should be compensated for in other cost savings.

The total Mine Department costs are about 10% over the 1970 costs due to the factors explained above.

5. Manpower Requirements
This Appendix shows the average manpower level required in the Mine Department based on the equipment scheduling shown herein.

No detailed analysis was made on Mechanical Department costs or manpower projections in this study. An increase in truck scheduling over the present schedule and wage escalation are the only things that might require an increase in this area over present figures. Based on the mine activity, the Mechanical Department increase should not exceed 10% in either costs or manpower.

FIVE YEAR PIT - ORE SUMMARY

First 3.2 Years (From 1/1/71) - 6600 TPD

Bench	To Mill Tons	To Mill Grade	To Stockpile Tons	To Stockpile Grade
3925	1,371,000	12.3%	140,000	6.9%
3890	1,642,000	12.3%	474,000	7.7%
3855	2,422,000	12.0%	755,000	6.6%
3820	2,280,000	11.6%	--	--
Total	7,715,000	12.0%	1,369,000	7.0%

Last 1.8 Years - 9900 TPD

3820	776,000	7.0%	--	--
3785	2,098,000	8.1%	--	--
3750	1,637,000	9.2%	--	--
3715	1,865,000	10.5%	--	--
3680	109,000	9.0%	--	--
Total	6,485,000	9.0%	--	--
TOTAL - 5 Yrs.	14,200,000	10.6%	1,369,000	7.0%

APPENDIX II

FIVE YEAR PIT - WASTE SUMMARY

Bench	Cubic Yards Total Material	Cubic Yards Ore to Mill or Stockpile	Cubic Yards Waste
4240	182,000	--	182,000
4205	384,000	--	384,000
4170	864,000	--	864,000
4135	1,470,000	--	1,470,000
4100	2,223,000	--	2,223,000
4065	2,765,000	--	2,765,000
4030	3,235,000	-	3,235,000
3995	3,298,000	--	3,298,000
3960	3,065,000	--	3,065,000
3925	2,826,000	475,000	2,351,000
3890	2,663,000	665,000	1,998,000
3855	2,276,000	999,000	1,277,000
3820	1,849,000	961,000	888,000
3785	1,461,000	660,000	801,000
3750	932,000	515,000	417,000
3715	586,000	586,000	--
3680	34,000	34,000	--
TOTALS	30,113,000	4,895,000	25,218,000

FIVE YEAR PIT - EQUIPMENT SCHEDULING

1. Shovels

For 5 cubic yard shovels, use productivity as calculated in original mining study, verified by actual productivity experienced:

Waste - 2,250 cubic yards/shovel shift
Ore - 2,000 cubic yards/shovel shift

Total waste capacity required:

$$\frac{25,218,000 \text{ cu. yds.}}{2,250 \text{ cu. yds./s. s.}} = 11,208 \text{ shovel shifts.}$$

Total ore capacity required:

$$\frac{4,895,000 \text{ cu. yds.}}{2,000 \text{ cu. yds./s. s.}} = 2,448 \text{ shovel shifts.}$$

Total shovel shifts required:

Waste	11,208
Ore	2,448
Total	13,656

Average shovel shifts required/day (6 day week)

$$\frac{13,656 \text{ total shovel shifts}}{306 \text{ days/year x 5 years}} = 8.95 \text{ shovel shifts/day.}$$

Required - 9 shovel shifts/day for full 5 year period.

2. Trucks

Waste - Average production/truck shift - 616 cubic yards.

$$\text{Trucks/shovel shift} - \frac{2,250 \text{ cu. yds.}}{616 \text{ cu. yds.}} = 3.7$$

Ore - Average production/truck shift - 1463 tons

$$\text{Trucks/shovel shift} - \frac{6400 \text{ tons}}{1463} = 4.4$$

For 3 shovels/shift - One ore, two waste, average truck requirements:

Waste - 3.7 x 2 = 7.4	
Ore	4.4
Total/shift	11.8 or 12

Fleet utilization - 12/15 = 80%

3. Drills

Cubic yards total material - 30,113,000

Average cubic yards broken/drill hole (last 1/2 - 1969) - 711.3

Average footage drilled/operating hour (last 1/2 1969) - 41.3 ft.

$$\text{No. drill holes required} - \frac{30,113,000}{711.3} = 43,370$$

Drilling footage required - 43,370 holes x 40 ft/hole = 1,734,800 ft.

$$\text{Operating hours required} - \frac{1,734,000 \text{ ft.}}{41.3} = 41,985$$

$$\text{Operating shifts} - \frac{41,985 \text{ Oper. Hrs.}}{7 \text{ hrs. x } 80\% \text{ availability}} = 7,497 \text{ shifts}$$

$$\text{Drill shifts/day} - \frac{7,497 \text{ Total Shifts}}{305 \text{ days x } 5 \text{ years}} = 4.9 \text{ or } \underline{5}$$

APPENDIX IV

FIVE YEAR PIT - ANNUAL MINING COST ESTIMATE

	Annual Cost
Mine Loading and Transport	
Labor	
Shovel Loading	$ 200
Truck Hauling	573
Road Maintenance	121
Supplies	
Shovel Loading	180
Truck Hauling	406
Road Maintenance	77
Tires	135
Total Controllable Costs	$1,692
Cost per cubic yard	0.282
Drilling and Blasting	
Using year to date cost of 0.186/cu. yd.	
Drilling and Blasting - Total Costs	$1,120
Supervision	$ 84

Summary - Average Annual Mine Department Costs - (000's)

	Total Cost	Cost/Cu. Yd.
Drilling and Blasting	$1,120	$ 0.186
Mine Loading and Transport	1,692	0.282
Supervision	84	0.014
	$2,896	$ 0.482

FIVE YEAR PIT - MANPOWER REQUIREMENTS

Mine Department	No. Required
Mine Superintendent	1
Asst. Mine Superintendent	1
General Mine Foreman	1
Mine Foreman	4
Mine Clerk	1
Shovel Operators	9
Rotary Drillers	5
H.D. Truck Drivers	36
Equipment Operators	12
Blasters and Helpers	4
Lubemen	3
Laborers	6
Total	83

Notes:

1. Above requirements based on six day work schedule.
2. Laborers include relief men.

Ultimate Pit
1970

J. McL

MEMORANDUM

TO: J. W. Mossop

FROM: M. O. Hampton

DATE: September 11, 1970

SUBJECT: STRIPPING RATIOS

An ultimate pit plan has been completed by Jim McLachlan which mines virtually all of the known reserves of Faro No. 1 and 3 zones.

TOTAL VOLUME OF WASTE	1970 PIT	119,000,000	cu. yds.
TOTAL ORE	ENP	58,300,000	tons

STRIPPING RATIOS (waste/ore)

$$\frac{119}{58.3} = 2.1/1 \text{ cu. yd. / ton}$$

$$\text{x } 2.15 = 4.5/1 \text{ ton/ton}$$

$$\frac{119}{18.4} = 6.5/1 \text{ cu. yd. / cu. yd.}$$

2.1 | 6.5 (3.1)
6.3
20

M. O. Hampton
Chief Engineer

MOH/mm
cc. N. G. Cornish
R. L. Cook

Engineering 7M 10

MEMORANDUM

TO: W. Krats

FROM: J. McLachlan

DATE: October 23, 1972

SUBJECT: ULTIMATE PIT DESIGN AND REVISION OF GRADES AND TONNAGES - ZONE 1 & 3 SEPTEMBER, 1970

Jaharwalal Gondi compiled a set of revised bench plans for both Zones 1 and 3 during the summer of 1970. This was the first whole scale revision of any bench plans and ore reserves since the initial pre-production bench plans were compiled. The revisions at this time were based upon new information obtained from a "fill-in" diamond drilling program plus past history of what previous mining from September, 1969 had established.

With the help of two summer students, Grant Macdonald and Jim McMillan, ore reserves for each bench, weighted according to the area of influence of each hole, were planimetered. Grade for each hole on the bench appears to have been assigned under the idea that any continuous 15' intersection of material averaging 5% or better could be regarded as ore. But in the published figure on ore reserves it would appear that this 15' intersection could in turn be averaged over the full bench height. This has produced the case where the reserves on some benches may average less than 5%.

These revised bench plans then became the basis for the design of an ultimate pit which was to mine out all the known ore reserves in Zones 1 and 3 without regard to an economic cut-off grade. On this ultimate pit thoughts were at the time to leave every second bench on a wider spacing (54' toe-to-toe on 35' benches) and the pit was drawn up in this way. On all benches, that were at least partially started at that time, quantities measured were from the existing status at September 1, 1970 to the final designed position. Quantities measured were the total yardage to be mined on a bench. Total ore yardage (in both Zones 1 and 3 if applicable) was then subtracted from the total measured yardage to arrive at waste quantities.

These quantities were then tabulated on the accompanying sheet.

J. McLachlan

J. McLachlan
Planning Engineer

JM/mm

Attach.

ULTIMATE PIT DESIGN

FARO NO. 1 & NO. 3 ZONES

COMPLETED REVISION OF SEPTEMBER 11, 1970

Drop Cut in 3990 At Sept 15, 1970

BENCH	TOTAL YARDAGE CONTAINED ON BENCH	ZONE NO. 1 TONS OF ORE	ZONE NO. 1 GRADE	ZONE NO. 1 YD. OF ORE	ZONE NO. 3 TONS OF ORE	ZONE NO. 3 GRADE	ZONE NO. 3 YD. OF ORE	BENCH TOTAL ORE TONNAGE	BENCH TOTAL ORE YARDAGE	WASTE
4240	990,000									
4205	1,829,700									
4170	2,901,600									
4135	3,498,700									
4100	4,983,600									
4065	6,195,500									
4030	6,557,200									
3990	7,294,100									
3950	10,564,000	1,542,112 1,057	10.48 11.49	484,941				1,542,112	484,941	10,079,059
3910	11,239,100	2,613,880 2,319	10.70 10.89	821,975				2,613,880	821,975	10,417,125
3870	10,711,900	3,139,012 2,651	11.65 12.10	987,110				3,139,012	987,110	9,724,790
3830	10,251,400	3,473,477 2,604	11.20 11.46	1,092,288	80,809 81	8.81 8.81	25,411	3,554,286	1,117,699	9,133,701
3790	9,090,800	4,122,308 3,006	10.30 11.35	1,296,323	425,745 304	8.82 10.05	133,882	4,548,053	1,430,205	7,660,595
3750	8,624,400	3,575,901 2,217	7.81 9.16	1,124,497	534,240 306	7.28 10.28	168,000	4,110,141	1,292,497	7,331,903
3710	7,401,200	3,265,451 2,251	9.27 10.91	1,026,871	543,219	6.35 -	170,823	3,808,670	1,197,694	6,203,506
3670	6,799,600	3,854,840 3,178	9.47 10.65	1,212,214	1,007,873	5.40 -	316,941	4,862,713	1,529,155	5,270,445
3630	6,228,400	3,129,411 3,129	10.82 10.81	984,091	715,312	4.16 -	224,941	3,844,723	1,209,032	5,019,368
3590	4,257,700	2,426,080 1,955	9.28 10.01	762,918	791,633 390	6.16 7.98	248,941	3,217,713	1,011,859	4,140,641
3550	4,028,700	1,555,955 631	7.67 9.41	489,294	657,698 349	8.35 0.46	206,823	2,213,653	696,117	3,561,583
3510	3,243,100	870,567 283	7.02 9.77	273,763	690,622 -	7.33 -	217,177	1,561,189	490,940	3,537,760
3470	2,514,000				4,378,672 2295	9.07 12.06	1,376,940	4,378,672	1,376,940	1,866,160
3430	1,919,000				4,363,531 3430	8.74 9.83	1,372,179	4,363,531	1,372,179	1,141,821
3390	989,100				4,791,699 3775	9.20 10.32	1,506,823	4,791,699	1,506,823	~~1,607,177~~ 978,529
3350					2,990,697 975	7.23 8.01	940,471	2,990,697	940,471	~~894,747~~
	137,265,300 17,550,000 ore	33,568,994		10,556,285	300,042 300	8.52 8.51	94,353	300,042	94,353	
	119,715,300				22,271,792 ✓	8.16 ✓		55,812,786 T		

NOTES: 1. Bench 3350 not counted in ultimate pit
2. Benches 4275 - 4030 are 35'; 4030 - 3390 are 40'.

BENCH -- 3990

D.D.H. No.	TONNAGE	COMB. GRADE	TOX COMB. GR.
65 - 1	196,038	16.34	3,203,260
66 - 20	146,654	14.37	1,437,000
66 - 44	293,308	12.49	3,663,417
66 - 24	211,376	8.11	1,714,259
70 - 5	160,122	6.74	1,079,222
70 - 16	106,998	11.38	1,217,637
70 - 2	14,216	10.15	144,292
70 - 4	98,019	9.05	789,053
66 - 41	106,998	7.85	839,934
70 - 8	79,313	10.57	838,338
66 - 39	74,823	9.42	704,833
65 - 14	9,353	7.3	68,277
66 - 18	38,160	10.22	389,995
66 - 18	6,734	10.22	68,821
Total	1,542,112		16,158,338

Average grade -- 10.48

BENCH == 3950

D.D.H. No.	TONNAGE	COMB. GRADE	TOX COMB.
66 - 20	233,824	14.99	3,505,022
66 - 16	270,861	14.07	3,811,014
65 - 11	141,334	10.30	1,455,740
66 [illegible]32	156,381	9.93	1,552,863
66 - 24	82,306	8.61	708,655
66 - 25	88,292	8.37	739,004
66 - 15	172,842	11.09	1,916,817
66 - 44	94,222	8.89	837,634
70 - 5	94,222	11.03	1,039,269
70 - 1	139,172	9.69	1,348,577
70 - 16	94,222	9.86	929,029
65 - 4	109,991	7.66	842,531
70 - 2	117,847	9.94	1,171,399
70 - 3	116,724	10.25	1,196,421
70 - 4	129,445	10.26	1,328,106
66 - 14	94,222	10.17	958,238
66 - 39	99,889	12.91	1,289,567
70 - 9	82,680	9.02	745,774
70 - 11	62,478	9.15	571,674
66 - 9	74,075	8.71	645,193
66 - 18	129,445	9.35	1,210,311
65 - 7	28,058	5.81	163,017
66 - 12	673	11.98	8,063
Total	2,613,880		27,973,918

Average Grade -- 10.7

BENCH -- 3910

D.D.H. No.	TONNAGE	COMB. GRADE	TOGE. X COMB. GRADE
66 - 44	150,021	15.89	2,383,834
66 - 12	218,485	16.56	3,618,112
70 - 7	84,925	14.45	1,227,166
70 - 14	86,795	13,53	1,174,336
70 - 15	86,421	13,53	1,169,276
66 - 16	188,445	15.85	2,986,853
70 - 6	94,222	12,49	1,176,833
65 - 11	141,333	11.55	1,632,396
66 - 15	188,445	11.16	2,103,046
65 - 5A	210,628	9.94	2,093,642
66 - 32	112,609	11.80	1,328,786
70 - 5	94,222	10.83	1,020,424
70 - 16	94,222	7.87	741,527
70 - 2	94,222	5.96	561,563
70 - 4	94,222	9.68	912,069
70 - 9	99,515	10.19	1,014,058
66 - 24	56,118	7.59	425,936
70 - 1	123,085	8.76	1,078,225
65 - 4	107,746	14.14	1,523,528
70 - 3	101,386	10.52	1,066,581
66 - 14	94,652	10.80	1,022,242
70 - 11	101,386	n 4.58	464,348
65 - 7	240,183	9.77	2,346,588
66 - 35	134,682	12.47	1,679,485
66 - 9	88,666	12.31	1,091,478
67 - 34	25,440	15.55	395,592
66 - 25	26,936	12.66	341,010
Total	3,139,012		36,578,934

Av. Grade -- 11.65

BENCH -- 3870

D.D.H. No.	TONNAGE	COMB. GRADE	TOGE. X COMB. GRADE
66 - 25	98,019	18.74	1,836,876
66 - 44	210,254	18.74	3,940,160
65 - 11	235,694	14.08	3,318,572
66 - 32	145,532	7.47	1,087,124
66 - 12	188,555	8.60	1,621,573
70 - 6	137,440	12.73	1,749,611
66 - 15	188,445	11.95	2,251,918
70 - 5	94,222	9.82	925,260
70 - 1	124,955	9.47	1,183,324
70 - 16	94,222	9.24	870,611
65 - 4	105,501	9.92	1,046,570
66 - 23	88,932	14,65	1,302,854
70 - 7	59,929	13.78	825,822
70 - 2	94,222	5.11	481,474
70 - 3	126,452	13.76	1,739,980
70 - 14	106,541	13.78	1,468,135
65 - 5A	235,556	6.98	1,644,181
70 - 4	94,222	6.35	598,310
66 - 14	120,840	6.34	766.126
70 - 15	120,466	13.78	1,660,021
70 - 9	94,222	7.61	717,029
70 - 11	(110,739) (29,929)	8.23	1,157,698
66 - 8	138,424	10.31	1,427,151
66 - 4	(234,946) (55,369)	6.34	1,840,597
66 - 22	215,866	5.45	1,176.470
65 - 7	264,501	8.62	2,279,999
Total	3,473,477		38,917,446

Av. Grade -- 11.20

BENCH -- 3830

D.D.H. No.	TONNAGE	COMB. GR.	TONNAGE X COMB.
66 - 25	77,068	13.96	1,075,869
66 - 44	163,115	13.96	2,277,085
66 - 16	123,085	10.20	1,255,467
70 - 6	107,746	10.20	1,099,009
70 - 7	106,249	12.76	1,355.737
66 - 23	166,856	13.22	2,205,836
65 - 11	141,334	16.21	2,291,024
66 - 15	188,445	9.42	1,775,152
70 - 14	106,541	12.76	1,359,463
70 - 15	118,527	12.76	1,512,405
65 - 5A	235,556	6.38	1,502,847
66 - 8	412,278	12.55	5,174,089
66 - 32	84,551	7.37	623,141
70 - 5	94,222	10.25	965,776
70 - 16	94,222	19.21	1,810,005
70 - 2	94,222	5.32	501,261
70 - 4	94,222	4.03	379,714
70 - 9	94,222	5.70	537,065
70 - 1	61,729	9.90	611,117
65 - 4	86,421	10.08	871,124
70 - 3	90,612	11.07	1,003,075
66 - 14	128,322	17.23	2,210,988
70 - 11	94,222	6.88	648,247
66 - 9	83,802	8.44	707,289
66 - 4	212,000	6,93	1,469,160
66 - 35	131,714	8.41	1,107,715
66 - 22	456,049	8.90	4,058,836
65 - 7	274,976	7.53	2,070,569
Total	4,122,308		42,459,065

Av. Grade -- 10.30

BENCH -- 3790

D.D.H. No.	TONNAGE	COMB. GR.	TONNAGE X COMB. GR.
66.- 23	237,191	10.25	2,431,208
70.- 6	60,981	8.19	499,434
70 - 14	79,313	6.70	531,397
70 - 15	118,527	6.70	794,131
66 - 8	453,431	8.72	3,953,918
65 - 11	83,428	11.67	973,605
66 - 15	175,461	9.42	1,652,843
65 - 5A	227,838	7.46	1,699,671
70 - 5	144,409	11.82	1,706,914
70 - 16	94,222	7.79	733,989
70 - 2	129,819	7.05	915,224
70 - 4	94,222	7.22	680,283
70 - 9	94,222	5.74	540,834
65 - 4	63,600	8.43	536,148
66 - 14	92,033	10.05	924,931
70 - 11	94,222	14.46	1,362,450
66 - 9	50,880	8.44	429,427
66 - 2	270,861	7.47	2,023,332
66 - 22	517,031	5.84	3,019,461
65 - 6	202,024	6.51	1,315,176
66 - 30	170,972	8.30	1,419,067
66 - 57	121,214	5.01	607,282
Total	3,575,901		27,956,594

Av. Grade -- 7.81

BENCH -- 3750

D.D.H. No.	TONNAGE	COMB. GR.	TONNAGE X COMB. GR.
66 - 15	56,866	9.02	512,931
70 - 16	104,005	9.02	938,121
65 - 8	521,894	16.83	8,783,476
70 - 15	90,162	12.18	1,098,173
66 - 8	282,667	7.53	2,128,483
65 - 5A	178,080	9.78	1,741,622
70 - 4	136,553	6.99	954,505
70 - 9	94,222	12.58	1,185,313
70 - 11	126,826	9.67	1,226,407
66 - 4	211,376	7.72	1,631,823
66 - 2	498,699	6.13	3,057,025
66 - 22	395,068	8.79	3,472,648
65 - 6	379,355	5.07	1,923,330
66 - 30	189,678	8.59	1,629,334
Total	3,265,451		30,283,191

Av. Grade. -- 9.27

BENCH -- 3710

D.D.H. No.	TONNAGE	COMB. GRADE	TONGE. X COMB GRADE
66 - 29	130,941	12.93	1,693,067
66 - 26	270,113	11.97	3,233,253
65 - 8	814,038	11.50	9,361,437
66 - 4	248,359	15.44	3,834,663
66 - 8	377,000	8.86	3,340,220
70 - 9	109,187	7.16	781,779
70 - 11	95,026	17.30	1,643,950
66 - 2	376,889	8.02	3,022,650
66 - 22	362,035	3.49	1,263,502
66 - 33	294,001	11.37	3,342,791
65 - 6	462,992	7.81	3,615,968
66 - 6	314,259	4.42	1,389,025
Total	3,854,840		36,522,305

Av. Grade -- 9.47

BENCH -- 3670

D.D.H. No.	TONNAGE	COMB. GRADE	TOGE. X COMB. GRADE
70 - 9	54,621	7.70	420,582
66 - 4	231,149	9.15	2,115,013
66 - 8	275,988	11.96	3,300,817
66 - 22	274,492	11.00	3,019,412
66 - 2	376,889	77.02	2,645,761
66 - 31	439,048	14.79	6,493,520
65 - 12	144,409	12.45	1,797,892
65 - 9	288,015	10.80	3,110,562
66 - 33	403,936	9.07	3,663,699
65 - 6	640,864	11.37	7,286,624
Total	3,129,411		33,853,882

Av. Grade -- 10.82

BENCH -- 3630

D.D.H. No.	TONNAGE	COMB. GRADE.	TONGE. X COMB. GRADE
66 - 2	604,874	9.71	5,873,326
66 - 31	193,419	10.24	1,980,611
65 - 9	406,292	12.20	4,956,762
66 - 33	469,892	6.19	2,908,631
65 - 6	447,071	9.58	4,282,940
66 - 46	304,532	8.21	2,500,208
Total	2,426,080		22,502,478

Av. Grade. -- 9.28

BENCH -- 3590

D.D.H. No.	TONNAGE	COMB. GRADE	TONGE. X COMB. GRADE
65 - 9	376,362	7.31	2,751,206
66 - 33	567,911	6.71	3,810,683
66 - 46	356,534	6.11	2,178,423
66 - 6	255,148	12.53	3,197,004
Total	1,555,955		11,937,316

Av. Grade -- 7.67

BENCH -- 3550

D.D.H. No.	TONNAGE	COMB. GRADE	TONGE. X COMB. GRADE
66 - 46	587,365	5.69	3,342,107
66 - 6	283,202	9.78	2,769,716
Total	870,567		6,111,823

Av. Grade -- 7.02

BENCH -- 3870

ZONE # 3

D.D.H. No.	TONNAGE	COMB. GRADE
66-E-4	80,809	8.81

Average grade for the bench -- 8.81

BENCH -- 3830

D.D.H.	TONNAGE	GRADE
66-E-4	303,783	10.05
66-E-1	121,962	5.77
	425,745	

Average grade for the bench -- 8.82

BENCH -- 3790

D.D.H.	TONNAGE	GRADE
66-E-1	306,028	10.28
66-E-4	228,212	3.26
	534,240	

Average grade for the bench -- 7.28

BENCH -- 3750

D.D.H.	TONNAGE	GRADE
66-E-6	543,219	6.35

Average grade for the bench -- 6.35

BENCH -- 3710

D.D.H.	TONNAGE	GRADE
66-E-9	213,247	3.78
66-E-6	346,433	6.97
66-E-5	448,193	4.96
TOTAL	1,007,873	

Average grade -- 5.40

BENCH -- 3670

D.D.H.	TONNAGE	GRADE
66-E-5	103,256	5.54
66-E-6	106,249	5.77
66-E-9	505,807	3.54
TOTAL	715,312	

Average grade -- 4.16

BENCH -- 3630

D.D.H.	TONNAGE	GRADE
66-E-9	401,054	4.39
67-5	390,579	7.98
TOTAL	791,633	

Average grade -- 6.16

BENCH -- 3590

D.D.H.	TONNAGE	GRADE
67-5	347,929	9.46
66-E-9	309,769	7.11
TOTAL	657,698	

Average grade -- 8.35

BENCH -- 3550

D.D.H.	TONNAGE	GRADE
67-5	656,951	7.33
67-8	33,671	7.33
TOTAL	690,622	

Average Grade -- 7.33

BENCH -- 3510

D.D.H.	TONNAGE	GRADE
70-13	48,635	4.23
67-4	490,842	8.19
67-30	451,934	6.63
66-46	520,772	13.67
66-10	537,233	5.11
67-12	127,948	6.16
66-7	632,259	13.67
66-3	376,362	12.60
67-3	258,889	5.61
67-6	475,878	6.13
67-10	182,569	4.92
67-11	275,351	11.54
TOTAL	4,378,672	

Average grade -- 9.07

BENCH == 3470

D.D.H.	TONNAGE	GRADE
66-5	303,784	5.80
66-52	520,772	13.16
67-9	190,800	4.71
66-3	135,431	5.78
66-46	161,619	9.46
67-8	392,824	9.25
67-4	318,748	6.44
67-10	237,939	10.33
67-3	228,960	8.58
67-30	1,011,614	8.40
67-12	404,047	7.20
66-10	82,306	5.03
70-17	570,155	8.90
66-49	304,532	12.12
TOTAL	4,363,531	

Average grade -- 8.74

BENCH -- 3430

D.D.H.	TONNAGE	GRADE
67-3	506,~~555~~	8.55
67-6	655,454	7.75
67-11	451,934	9.01
67-12	870,198	9.46
70-12	308,273	13.76
66-52	208,758	12.45
66-10	549,205	4.48
66-3	249,911	5.59
67-4	532,743	15.38
66-46	241,680	9.27
66-49	216,988	5.64
TOTAL	4,791,699	

Average Grade -- 9.20

BENCH -- 3390

D.D.H.	TONNAGE	GRADE
67-12	974,951	8.41
67-6	426,494	4.84
66-49	412,278	7.50
66-10	552,198	7.54
66-3	624,776	6.58
TOTAL	2,990,697	

Average grade -- 7.23

BENCH -- 3350

D.D.H.	TONNAGE	GRADE
67-6	140,668	8.69
66-3	159,374	8.36
TOTAL	300,042	

Average grade -- 8.52

Stripping Ratios
1970-71

MEMORANDUM

TO: J. W. Mossop

FROM: M. O. Hampton

DATE: October 1, ~~1970~~

SUBJECT: ORE PROJECTIONS

The following estimates of tonnages and grades to be mined in the next 18 years are based on all data available at the present. All known ore is taken into account, and a mill expansion from 6600 TPD to 9900 TPD considered for 1974.

Faro No. 1 and No. 3 calculations were made by planimetering the areas of the geological projections of the orebody on each bench level plan. A factor of 8.5 cubic feet per ton was used, this is the same factor that has been used on all previous tonnage calculations. A pit has been designed to mine all of the ore from No. 1 and No. 3. The tonnage and grade of Faro No. 2 was taken from E. N. Pennebaker's calculations as no further pit designing or drilling has been done.

The tonnage and grade of ore mined to date has been taken into consideration.

1971, 72, 73 inc.	2,400,000 TPY @	12.0% comb Zn and Pb
1974 - Oct. 1986 inc.	3,600,000 TPY @	9.0% comb. Zn and Pb
Oct. 1986 - 1988 inc.	3,600,000 TPY @	7.9% comb. Zn and Pb

Respectfully submitted,

M. O. Hampton, B.A.Sc., P. Eng.
M.C.I.M., M.A.I.M.E., M.M.A.C.

MOH/mm

cc. N. G. Cornish
R. L. Cook
J. F. Olk

MEMORANDUM

TO: J. W. Mossop

FROM: M. O. Hampton

DATE: October 1, 1970

SUBJECT: PROJECTED MINING

The following are estimated annual volumes to be extracted over life of the mine based on the known ore reserves, planned mill production, and present ultimate pit plan.

	L. G. Ore & Waste (cu. yds.)	Ore (cu. yds.)	Total
1971	6,620,000	751,000	7,371,000
1972, 73	5,500,000	751,000	6,251,000

	Waste (cu. yds.)	Ore (cu. yds.)	Total
1974 - 1987	6,700,000	1,035,000 (pit)	7,735,000
		100,000 (stk. pile)	100,000
1988	- 0 -	1,135,000	1,135,000

M. O. Hampton
Chief Engineer

cc. N. G. Cornish
R. L. Cook
J. F. Olk

Anvil Mining Corporation Limited

February 2, 1971

RECEIVED
FEB 5 1971
FARO

Mr. L. Lilley,
Coopers & Lybrand,
900 West Hastings Street, 6th Floor,
Vancouver 1, B. C.

Dear Mr. Lilley,

Attached please find details of computations arriving at overall stripping ratio of 5.9/1 which is the basis for our deferred development.

We trust you will find this to be in order.

Yours very truly,

R. L. Cook,
Controller.

RLC/sn
Encls.

c.c. E. M. Lehbauer

1333 West Georgia Street, Vancouver 5, B.C., Telephone (604) 687-2586, Telex 04508594

	Waste	Ore	Total	
1971	6,620,000	751,000	7,371,000	11 ea/day 10 ea/day
1972	5,500,000	751,000	6,251,000	8 ea/day
1973	5,500,000	751,000	6,251,000	8 ea/day
1974	6,700,000	1,135,000	7,835,000	11 ea/day
1975	6,700,000	1,135,000	7,835,000	11 ea/day
Totals	31,020,000	1,523,000	36,543,000	
yr. study	25,216,000	1,805,000	30,113,000	
Final Pit	119,000,000 C.Y. Waste 58,300,000 Tons Ore		119,000	
	6.5/1		Stripping Ratio to 1/1/70	7.770
	6.3/1			
976-87	6,700,000	1,135,000	7,835,000	
1988	"	1,135,000	1,135,000	
				Stripping R.
Totals 1971-1988	111,420,000	(61,304,040 Tons 19,278,000 C.Y.)	130,698,000	5.8/1
1970	6,570,000	591,000	7,091,000	11.0/1
	117,720,000	19,869,000	137,787,000	5.9/1
To 1/1/70	7,770,000	134,000	7,904,000	
	125,490,000	20,003,000	145,693,000	6.3/1
		X 3.15		
		63,040,000 Tons		

MEMORANDUM

TO: J. W. Mossop

FROM: M. O. Hampton

DATE: October 1, 1970

SUBJECT: ORE PROJECTIONS

The following estimates of tonnages and grades to be mined in the next 18 years are based on all data available at the present. All known ore is taken into account, and a mill expansion from 6600 TPD to 9900 TPD considered for 1974.

Faro No. 1 and No. 3 calculations were made by planimetering the areas of the geological projections of the orebody on each bench level plan. A factor of 8.5 cubic feet per ton was used, this is the same factor that has been used on all previous tonnage calculations. A pit has been designed to mine all of the ore from No. 1 and No. 3. The tonnage and grade of Faro No. 2 was taken from W. N. Pennebaker's calculations as no further pit designing or drilling has been done.

The tonnage and grade of ore mined to date has been taken into consideration.

1971, 72, 73 inc.	2,400,000 TPY @ 12.0% comb Zn and Pb	
1974 - Oct. 1986 inc.	3,600,000 TPY @ 9.0% comb. Zn and Pb	
Oct. 1986 - 1988 inc.	3,600,000 TPY @ 7.9% comb. Zn and Pb	

61,200,000 T

Respectfully submitted,

M. O. Hampton, B.A.Sc., P. Eng.
M.C.I.M., M.A.I.M.E., M.M.A.C.

Professional Engineers of Ontario
M. O. HAMPTON
Member

MOH/mm

cc. N. G. Cornish
R. L. Cook
J. F. Olk

MEMORANDUM

TO: J. W. Kossop

FROM: M. O. Hampton

DATE: October 1, 1970

SUBJECT: PROJECTED MINING

The following are estimated annual volumes to be extracted over life of the mine based on the known ore reserves, planned mill production, and present ultimate pit plan.

	L. G. Ore & Waste (cu. yds.)	Ore (cu. yds.)	Total
1971	6,620,000	751,000	7,271,000
1972, 73	5,500,000	751,000	6,251,000

	Waste (cu. yds.)	Ore (cu. yds.)	Total
1974 - 1987 (14yr)	6,700,000	1,035,000 (pit)	7,735,000
		100,000 (stk. pile)	100,000
1988	- 0 -	1,135,000	1,135,000
	111,429,000	19,278,000	130,698,000

M. O. Hampton
Chief Engineer

cc. N. G. Cornish

Reserve Memos
1972

October 19, 1972

MEMORANDUM TO THE FILE

Meeting: J. F. Olk, J. Keily, W. Krats, R. L. Haffner, N. G. Cornish

Subject: Reserves & 5 Year Mining Plans

Mr. Olk had just completed a review of the 5 year mining plans.

The conclusions of the discussions were:

1. 1973 and 1974 appeared to be reasonable and to be within acceptable tolerances relative to the original 1970 long term plan.

2. That 1975-76 and 77 were very high in all material movement and the grade deviated from the long term plan which called for 9% combined Pb & Zn feed grades. In both these areas, these estimates deviated from the 1970 long term plan.

3. It was recommended that since the change was not consistent with the long term plan and could not be substantiated without reasonable doubt without significant back-up work on the reserves, that we would revert to the 1970 long term plan for 1975-76-77 figures, adjusting them to 10,000 tons/day of mill feed.

4. The 1974 plan would be slightly modified to reduce all material movement to 7,462,000 cu. yd. and including in it 1,066,000 cu. yd. of mill feed at 3.75% Pb and 5.86% Zn.

5. (1) The Engineering Department is to continue on to evaluating the ore reserves, applying the most recent D.D. data and a logical geological interpretation to bring the cross-sections up to date. A report indicating the parameters used would be made available along with these sections.

   (2) New bench plans would be prepared from these x-sections. The parameters used on these would also be stated as clearly as possible.

   (3) A reserve would be calculated from these bench plans. The parameters would be explained. Perhaps more than one method should be used to verify the reserves.

   (4) New plans for 1975, 76, 77 would then be developed from this new information.

6. As per the original 1970 long term plan and plant A.F.E., the following ground rules would apply.

   (1) As of 1/1/75 plant feed would average 10,000 tons per day.

   (2) Grade to be achieved if possible to be 9% combined in the mill feed.

   (3) Within reason we would attempt to recover as many metal units as possible and still economically maintain the 9% minimum feed grade.

   (4) The lean or low grade stockpiles would be sampled and evaluated for ability to process thru the plant. The stockpiles should be introduced as mill feed as soon as possible after 1/1/75, and still maintain grade and product.

7. Equipment for the 5 year cost study would be the present fleet.

General:

Note J.F.O. comments on the attached sheets.

A dominant factor in the new ore reserve must be general definition of the process, the setting up of it so it can be followed, checked and worked on by many people at erratic times and still maintain the rational continuity of the whole process.

A recapitulation of the reserves must be such that we can see what was there and what has been removed, leaving what is remaining. It must be made simple to update the reserve picture at least once or twice per year.

Note for 5 Year Plan:

	1975, 76, 77
Mill Feed	1,147,000 cu. yds.
	6,767,000 cu. yds.
Total	7,914,000 cu. yds. all mat'l.

I. ACTUAL AND ESTIMATED STRIPPING RATIOS:

		Cubic Yards (000's omitted)	
		Long Term Forecast	Actual or Estimated
1971	Ore	751	840
	Waste	6,620	6,167
	Total	7,371	7,007
	Ratios	8.8/1	8.25/1
1972	Ore	751	926
	Waste	5,500	4,202
	Total	6,251	5,128
	Ratios	7.3/1	4.55/1
1973	Ore	751	918
	Waste	5,500	4,607
	Total	6,251	5,525
	Ratios	7.3/1	5.0/1
1974	Ore	1,135	1,178
	Waste	6,700	7,036
	Total	7,835	8,214
	Ratios	5.9/1	6.0/1
1975 thru 1977	Ore	1,135	1,205
	Waste	6,700	9,545
	Total	7,835	10,750
	Ratios	5.9/1	7.9/1

II. ULTIMATE PIT STRIPPING RATIO
FROM LATE 1970 LONG TERM FORECAST - AS AT JANUARY 1, 1971:

$$\frac{114{,}420{,}000 \text{ c.y. waste}}{19{,}278{,}000 \text{ c.y. ore}} = 5.9/1$$

(61,304,000 tons)

III. PROJECTED MILL FEED GRADES - 1975-1977:

AFE - Mill Expansion	10,000 tons @ 9.0%
Add - Overtonnage	600 tons @ 6.0%
Total	10,600 tons @ 8.6%

	1975	1976	1977
5 year plan grade	8.95	10.10	9.85
AFE forecast	8.60	8.60	8.60
Over	0.35	1.50	1.25

IV. RECOMMENDATIONS:

1. A review of the 5 year plan indicates the following:

   (a) The plans for 1973 and 1974 appear reasonable. I would recommend increasing 1973 stripping to even out both years.

   (b) Years 1975-77 must be reviewed again, due to the large waste yardages shown, which are beyond the capacity of the mine equipment and over the projected grade.

2. In order to properly re-assess the 1975-77 plans, the ore block plans and bench ore estimates, which have been receiving revisions from time to time, must be finalized based on latest available drilling information. The bench plans used for 1973-74 appear to be in good shape, but the ones for 1975-77 must be reviewed.

3. Mining plans for 1975-77 should then be redone, based on the following ground rules.

   (a) The grade for these years should conform to the long term forecast.

   (b) The stripping ratios should not exceed the long term forecast.

   (c) Equipment capacities used in this plan should be that of the existing equipment.

   (d) Consideration should be given to the possible utilization of the low grade ore stockpile as mill feed during these years. Whether or not the degree of oxidation while in stockpile would seriously affect metallurgy at various ratios of feed should be determined.

4. After re-assessment of the ore block bench plans, a new ultimate pit plan should be prepared.

5. It is essential that Engineering and Accounting figures on yearly production agree.

ENGINEERING DEPARTMENT

ESTIMATES

(I) Reserve -

(1) Calculate reserve based on latest D.D. information and interpretation.

- Sections complete week of October 16, 1972
- Plans of individual benches next
- Calculate tons and grade after areas of influence have been determined
- Catalogue all information in a systematic logical fashion to allow maximum efficiency in future usage

(2) Design a preliminary ultimate pit which recovers all the mineralized zone.

- Set up guidelines which will determine uneconomic conditions
- Evaluate application of uneconomic conditions
  - manual?
  - computer?
  - timing?
- Adjust ultimate pit as conditions dictate

(3) Update reserve annually with new data as acquired from future drilling and new geological information and deplete as mining progress. (set up a master log)

(4) Breakdown into increments to exhaustion.

- Increment to be small enough to define all material flow for equipment and cash flow analysis.

(II) Estimates (Annually) -

(1) July 1 - Estimate July 1 - December 31 to determine January 1 banks.

(2) After (#1) do a one year plan for the complete year January 1 - December 31, and breakdown to quarters for budget purposes.

(3) Five Year Forecast

- 1st. year is Item 2
- 2nd. year by quarters
- 3rd., 4th., and 5th. years by years

(III) Estimates (Operating) -

Four month estimates will be done during the year, each overlapping the previous by one month.

Start dates will be December 1
March 1
June 1
September 1

These will detail shorter term ore and development requirements.

Smaller increments than the 4 month overlapping may well be justified for grade control and these will be produced as required.

(IV) Check Estimates -

Volume - Surveys and calculation of volume will be done monthly and accumulated to maintain a running average of the last six months, and also to get any increment we require.

Grade - Diamond drill projections should be compared to blast hole data on some frequency.

- Lead/Zinc ratios - D.D.H. vs. mill feed should be compared

- Adjustment factors should be developed and applied as justified against future estimates

(V) Waste Dumps -

An ultimate waste disposal plan must be evolved.

(1) Step one is to develop the waste disposal plan for the present 5 year plan. This will give us sufficient data to evaluate our immediate haulage truck requirements.

(2) Waste plans are part of mining plans, particularly during the 5 year plans, so we are able to project haulage requirements.

7.11 Engineering 10

MEMORANDUM

TO: J. Keily

FROM: W. Krats

DATE: November 1, 1972

SUBJECT: ULTIMATE PIT PLANS

The plan and progress with respect to an ultimate pit plan and reserve is progressing as follows:

It is the feeling of this department that for a reserve to be meaningful at this time, that we must begin from basic information and interpret this based on reasonable stated ground rules, and check every step of the process so that we are satisfied that mechanical errors and judgement errors are minimized.

Consequently the following program and general guidelines are being followed:

1) D.D. hole data is being transferred to cross-sections and being checked by a second person for accuracy.

2) All increments five feet or greater at 5% (Pb + Zn) or greater are being plotted as geological ore reserves.

3) Material less than 5% (Pb + Zn) within the geological ore zone is being indicated in two categories, high in massive sulphides and low in sulphides content.

4) Continuity of geological data is then projected from hole to hole on the cross-sections and being checked for interpretation.

5) Upon completion of the cross-sections, bench plans will be prepared for each bench. These plans will indicate the extent of the geological ore zones and the areas of influence of the D.D. holes. These will also be checked for accuracy and consistency.

6) Ground Rules will then be established for calculating of reserves from these bench plans. The first reserve will be calculated on a basis quite similar to the methods developed over the past several years and which were used in reserves stated as of Sept. 1970. This will give us a common basis for comparison of these two reserve calculations.

7) Upon completion of Item 6, we will attempt to calculate the reserves by one or two other methods and compare these to the present method. An evaluation of these methods will determine which approach is most suitable for our operation.

8) Upon completion of the reserve calculations, a 5 year mining plan through 1977 will be developed.

9) Emphasis at all times will be placed on defining the ground rules of the process and all steps of the process will be checked by someone other than the person who originally did the process. All this will be suitably documented for future reference.

The general schedule is to do Zone 1 first to completion to allow the completion of an acceptable 5 year plan. Then emphasis will be placed on Zones 3 and 2 to accomplish a total reserve and ultimate mining plan to exhaustion.

Every attempt will be made to set up data in a simple workable fashion to allow for efficiency in future use, for revision based on new information, for comparison to actually achieved results and previous reserves.

W. Krats
Chief Engineer

WK/mm

7.16 - Engineering 10

...AVIL MINING CORPORATION LIMITED

MEMORANDUM

TO: J. Keily FROM: W. Krats

SUBJECT: ORE RESERVE & ULTIMATE PIT PLAN DATE: November 13, 1972

TIME TABLE

In October of 1972 we began work on an ore reserve and ultimate pit plan. The progress to date and planned time table is as follows.

It is planned to initially do Zone I to completion so that a most up to date mining plan to December 31, 1977 can be produced by years as quickly as possible. Following completion of Zone I, we would immediately proceed to Zone 2 and 3 and complete them to give us the necessary information to develop a year by year mining plan to exhaustion.

At present the geological cross-sections for Zone I are complete utilizing the latest information and our best interpretation of it. We are in the process of laying out the bench plans from these cross-sections. It is estimated that Zone I will be complete with a design pit to mine all the material which is estimated in the reserve for that section along with plans for 1975, 1976 and 1977 mining by late December 1972 or early January 1973.

We will then proceed to Zones 2 and 3 which should be completed as far as total ore and waste quantities are concerned and an ultimate pit design by March or April 1973.

Mining plans by years to indicate grade and all material movement to exhaustion for long term planning purposes would then require another two or three months.

During this process a close look at edge ratios and the economics of mining some of the blocks on the fringes of the ore zone has not taken place. Further work in this area will be necessary to help us decide what circumstances are necessary to make mining and processing of certain blocks desirable. This exercise will undoubtedly in some areas change the shape of the first run at an ultimate pit.

An effort will also be made to relate diamond drill hole projections to blast hole analysis and what we are actually getting out of the mined areas. When a factor of some reliability can be established, this will be incorporated into the projections of ore reserves.

We are endeavoring to set up the complete process in as simple as possible a sequence to maximize on the utilization of all the people in the Engineering Department. The process will also be conducive to step by step checking of the complete routine as it progresses.

A memo to yourself on November 1, 1972 regarding Ultimate Pit Plans has more details regarding the complete process.

W. Krats
Chief Engineer

WK/mm

Reserves
1/1/73

ANVIL MINING CORPORATION LIMITED

Summary of Reserves

January 1, 1973

	Ore (Tons)	% Pb	% Zn	Cu. Yds.	Waste Cu. Yds.	Ratio (Waste) : (Ore) (Yds.) : (Yds)
* Initial Ultimate Pit Ore Reserve (Ore Zones 1 & 3)	58,314,000	3.4	5.8	18,338,000	118,387,000	6.5 : 1
** Additional Reserve (Ore Zone 2)	5,159,000	3.1	4.8	1,622,000	6,500,000	4.0 : 1
TOTAL RESERVE	63,473,000	3.4	5.7 (9.1)	19,960,000	124,887,000	6.3 : 1
Mined to date December 31, 1972	7,974,530	4.6	6.4	2,507,700	23,815,703	9.5 : 1
*** Remaining as of Jan. 1, 1973	55,498,470	3.2	5.6	17,452,300	101,071,297	5.8 : 1

8.8

* Based on an estimate of reserves and a detailed mining plan.

** Based on preliminary ore and waste estimates, but no detailed mining plan.

*** Anticipated remaining life of the ore body is approximately 15 years.

Based on Actual Acc Material movement @ Dec 31 1970 + 1971 & estimate from long Term plan, which includes Zone 1, 2, & 3. Zone 2 broke out by best guess estimate at present time.

Note April 3/73

This document produced for presentation for Baydad Copper People.

Kents

STATEMENT of RESERVE

	Mill Feed				Waste	Mining Ratio Waste (Cu Yds) / Mill Feed (Cu Yds)
	Tons	% Pb	% Zn	(Cu Yds)	Cu Yds	
Initial Ultimate P.I Ore Reserve (Ore Zones 1 & 3)	58,314,000	3.4	5.8	19,338,000	118,387,000	6.5 : 1
Pennebaker (Zone 2)	5,159,000	3.1	4.8	1,622,000	6,500,000	4.0 : 1
TOTAL RESERVE	63,473,000	3.4	5.7	19,960,000	124,887,000	6.3/1
MINED To date Dec 31/72	7.974,530	4.6	6.4	2,507,700	23,815,703	9.5/1 ←
REMAINING 1/1/73	55,498,470	3.2	5.6	17,452,300	101,071,297	5.8/1
MINING 1973 Estimate	2,920,000	4.1	6.9	918,000	4,509,000	4.9/1
REMAINING 1/1/74	52,578,470	3.15	5.52	16,534,300	96,562,297	5.8/1
MINING 1974 (Estimate)	3,562,000	3.35	5.33	1,120,126	8,322,848	7.4/1
REMAINING 1/1/75	49,016,470	3.09	5.53	15,414,174	88,239,449	5.7/1
MINING 1975 (Estimate)	3,815,000	3.59	5.17	1,199,684	5,904,677	4.9/1
REMAINING 1/1/76	45,201,470	3.05	5.56	14,214,490	82,334,772	5.8/1
MINING 1976 (Estimate)	3,722,000	3.4	5.09	1,170,481	6,704,677	5.7/1
REMAINING 1/1/77	41,479,470	3.02	5.60	13,044,009	75,630,095	5.8/1
MINING 1977 (Estimate)	3,650,000	3.47	5.28	1,147,798	6,709,677	5.8/1
REMAINING 1/1/78	37,829,470	2.96	5.63	11,896,211	68,925,418	5.8/1
MINING 1978 (Estimate)	3,650,000	3.47	5.24	1,147,798	6,704,677	5.8/1
REMAINING 1/1/79	34,179,470	2.91	5.67	10,748,413	62,220,741	5.8/1 →
MINING 1979	3,650,000	2.91	5.67	1,148,000	8,474,391	7.4/1
REMAINING 1/1/80	30,529,470	2.91	5.67	9,600,413	53,746,350	5.6/1
MINING 1980	3,650,000	2.91	5.67	1,148,000	8,474,391	7.4/1
REMAINING 1/1/81	24,879,470	2.91	5.67	8,452,413	45,271,959	5.4/1
MINING 1981	3,650,000	2.91	5.67	1,148,000	8,474,391	7.4/1
REMAINING 1/1/82	23,229,470	2.91	5.67	7,304,413	36,797,568	5.1/1
MINING 1982	3,650,000	2.91	5.67	1,148,000	8,474,391	7.4/1
REMAINING 1/1/83	19,579,470	2.91	5.67	6,156,413	28,323,177	4.6/1
MINING 1983	3,650,000	2.91	5.67	1,148,000	8,474,391	7.4/1
REMAINING 1/1/84	15,929,470	2.91	5.67	5,008413	19,848,786	4.0/1
MINING 1984	3,650,000	2.91	5.67	1,148,000	8,474,391	7.4/1
REMAINING 1/1/85	12,279,470	2.91	5.67	3,860,413	11,374,395	2.9/1
MINING 1985	3,650,000	2.91	5.67	1,148,000	8,474,391	7.4/1
REMAINING 1/1/86	8,629,470	2.91	5.67	2,712,413	2,900,004	1.1/1
MINING 1986	3,650,000	2.91	5.67	1,148,000	1,200,000	1/1
REMAINING 1/1/87	4,979,470	2.91	5.67	1,564,413	1,700,004	1.1/1
MINING 1987	3,650,000	2.91	5.67	1,148,000	1,200,000	1/1
REMAINING 1/1/88	1,329,470	2.91	5.67	416,413	500,004	1.2/1
MINING 1988	1,329,470	2.91	5.67	416,413	500,004	1.2/1

ANVIL MINING CORPORATION LIMITED

MEMORANDUM

TO: R. L. Haffner FROM: W. Krats

SUBJECT: ORE RESERVES DATE: January 24, 1973

Reserve at start of operations - 63,473,000 tons at 3.41 Pb & 5.72 Zn

(Pennebaker Estimate)

Processed to December 31, 1971 -	5,069,000 Tons 4.6% Pb, 6.54% Zn
Processed in 1972 to December 31 -	2,905,530 Tons 4.63% Pb, 6.22% Zn
Total processed to December 31, 1972 -	7,974,530 Tons 4.61% Pb, 6.42% Zn
Remaining Reserve as of January 1, 1973 -	55,498,470 Tons 3.24% Pb, 5.62% Zn

W. Krats
Chief Engineer

WK/mm

cc. J. Keily

ANVIL MINING CORPORATION LIMITED

MEMORANDUM

TO: W. Krats FROM: P. M. Pettigrew

SUBJECT: TONNAGE AND GRADE ESTIMATES DATE: May 8, 1973

FARO 1-3 AS OF JAN. 1, 1973

TONNAGE AND GRADE ESTIMATE AS OF JAN. 1, 1973

	Tonnage	Pb	Zn	Pb + Zn
Ore remaining in Zones 1 and 3 after Jan. 1, 1973	42,062,619	3.3	5.7	9.0 (a)
Stockpiles at Jan. 1, 1973	1,919,359	3.0	4.9	7.9
Total	43,981,978	3.3	5.7	9.0 (b)
Pennebaker's estimate for Zone 2	5,159,335 (2,928,188)	3.1 (2.8)	4.8 (4.7)	7.9 (7.5)
Total	49,141,313 (46,910,166)	3.3 (3.3)	5.6 (5.6)	8.9 (c) (8.9)

Note: (c) is not yet satisfactorily proven due to lack of sufficient drilling.

P.M. Pettigrew

P. M. Pettigrew
Ore Control Geologist

PMP/nm

Geol. Reserve
Est., June/73

PROGRESS REPORT

TONNAGE AND GRADE ESTIMATES

FARO DEPOSITS

June, 1973

Anvil Mining Corporation Limited

Whitehorse M.D., Yukon Territory

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I Estimated Tonnage and Grade Tables, January and May, 1973

II Estimated Tonnage and Grade Bench Plans, January and May, 1973

I INTRODUCTION

In order to have an up to date accurate estimate of tonnage and grade in the Faro deposits for mine planning, a review of estimation methods and techniques was conducted. During November and December 1972 and January 1973 all the diamond drill hole and rotary drill hole logs were reviewed and updated and new sections and plans were plotted. During the period January to May 1973 a tonnage and grade estimate was conducted in both the Large and Small Deposits.*

II SUMMARY

1) This estimate is a compilation of tonnages and grades of ⩾5% (Pb + Zn) material for ⩾5 foot intervals and also the stockpiled material.

2) In the Large Faro Deposit (North) on the 3910 to 3590 benches inclusive from 8,500 N to 11,400 N and 12,000 E to 16,000 E the total of mined and geologic reserve material is estimated to be 27,749,824 tons at 3.7% Pb and 6.1% Zn.

3) In the Large Faro Deposit (South) on benches 3550 and below from 8,500 N to 11,400 N and 12,000 E to 16,000 E and also all benches 7,000 N to 8,500 N and 12,000 E to 16,000 E, the total geologic reserve material is estimated to be 17,967,396 tons at 3.0% Pb and 5.3% Zn.

4) A trial estimate of tonnage and grade in the Small Faro Deposit totalled 2,928,188 tons of 2.8% Pb and 4.7% Zn.

III BRIEF GEOLOGY

The Faro deposits consist of two near horizontal strataform, stratabound massive pyrite, sphalerite, galena, and pyrrhotite deposits. The zinc and lead values in the massive sulfides range from below the economic cutoff of 5% (Pb + Zn) now used to well above this value. Some major distribution patterns within the massive sulfides have been recognized, although they are not necessarily continuous through the two deposits.

The Large Faro Deposit has a major trend length of up to 4,000 feet at approximately 315° Azimuth, is up to 1,500 feet wide, 300 feet thick and is covered by up to 500 feet of waste. The zinc and lead values are more continuous along the major trend direction and are more consistently above 9% (Pb + Zn) on the north west portion than the north east or south portions of the deposit.

* What is now being called the Large Faro Deposit has in the past been commonly referred to as Zones 1 and 3, and what is now being called the Small Faro Deposit has in the past been commonly referred to as Zone 2.

The Small Faro Deposit is located about 1,500 feet due south of the south east tip of the Large Faro Deposit. It has a major trend length of up to 1,000 feet at approximately 275° Azimuth, is up to 600 feet wide, 80 feet thick and is covered by up to 130 feet of waste.

IV ESTIMATION METHOD

A review of previous tonnage and grade estimates, available literature and attempts with typical plans and sections were conducted to find the best estimation method to give locations, tonnages and grades to the zinc and lead values in the deposits. To make sure that the estimate was complete in areas required for short term mine planning, an order of priority was outlined as follows: 1) Area required for 5 year mining plan from present mining banks in Large Faro Deposit, 2) Small Faro deposit, and 3) Remainder of Large Faro Deposit.

The diamond drill logs were reviewed and in some cases recopied and assay data entered. New longitudinal and transverse sections were prepared on which the drill holes, surface topography and intervals of $\geq$5 feet of $\geq$5% (Pb + Zn) were plotted. New bench plans were prepared for the 40 foot benches required on each of which individual drill hole intersections were plotted.

Using geologic experience gained during the present mining, an almost straight line interpretation of the Large Faro Deposit was outlined on the sections. Points indicating

1) The limit of a full 40 feet of $\geq$5% (Pb + Zn) for each bench, and

2) The limit of $\geq$20 feet of $\geq$5% (Pb + Zn) for each bench

were located on the sections and then transferred to the plans. These points were connected by smooth curves to form the outlines of

1) Full benches of $\geq$5% (Pb + Zn), and

2) Half benches or greater of $\geq$5% (Pb + Zn).

Areas of influence for individual sample locations were then outlined in a stepwise method. A grid was prepared based on a unit block of $100(\sqrt{2}/2)$ feet squared to coincide with the north-south, east-west grid at 315° Azimuth. Intersections of this grid coincide with the planned locations of the diamond drill holes. The planned locations are used to represent the real location of the drill hole for the areas of influence.

The areas are first extended to a maximum 4 unit blocks north west and south east and 2 unit blocks north east and south west centered about the grid intersection. Secondly, the areas are extended to a maximum of 8 unit blocks north west and south east centered about the grid intersection. The third step is to extend the areas to a maximum of 4 unit blocks north east and south west centered about the grid intersection. These extensions are done until all the area within the half bench outline has been allocated or a block is extended 1/2 the distance to the next grid intersection representing a diamond drill hole location.

A tonnage factor of 3.18 tons per cubic yard is used.

Grades are applied to the tonnage blocks in the following three cases (less than 1% of the tonnage has required geologic judgement to apply assumed reasonable grades):

1) If a diamond drill hole intersection is within the full bench outline then the weighted average of the full 40 feet is applied.

2) If the diamond drill hole intersection is between the full bench outline and the half bench outline, then a weighted average of the full 40 feet is applied if the 1/2 bench is determined by internal waste and of a 40 foot intersection into the deposit if the lower or upper contacts of the deposit have determined the 1/2 bench.

3) If the diamond drill intersection is outside the half bench outline then a weighted average of a 40 foot intersection into the deposit is applied from the contact provided that grades are not being projected more than one bench thickness from the upper or lower extremities of the bench concerned.

Tables are prepared to add up the total tonnage and grade by bench and a summary table is prepared to add up the total estimated tonnage and grade.

V LARGE FARO DEPOSIT

a) North:

3910 to 3590 benches inclusive from 8,500 N to 11,400 N and 12,000 E to 16,100 E.

A new tonnage and grade estimate was prepared for this portion of the Large Faro Deposit to have adequate estimates for the five year mining plan.

Estimated Tonnage and Grade Tables, January 1973 (see Appendix I) and Estimated Tonnage and Grade Bench Plans (see Appendix II) show the distribution and the totals. Final copies of the ink drafted bench plans are available on request.

b) South:

3550 bench and below from 8,500 N to 11,400 N and 12,000 E to 16,000 E also all benches from 7,000 N to 8,500 and 12,000 E to 16,000 E.

A new tonnage and grade estimate was prepared for this portion in May and completes the current picture of the Large Deposit prior to 1973 diamond drilling.

Estimated Tonnage and Grade Tables, May 1973 (see Appendix I) and Estimated Tonnage and Grade Bench Plans (see Appendix II) show the distribution and the totals. Final copies of the ink drafted bench plans will be available on request when they have been completed.

VI SMALL FARO DEPOSIT

A total tonnage and grade estimate has shown that the major trend direction is not similar to the Large Faro Deposit.

Formal drawings and tables have not been prepared as this estimate has

1) not been checked in the detail required,

2) the estimate has been prepared for 40 foot benches which might not be the optimum,

3) 1973 diamond drilling in this area is completed but not fully assessed at present, and

4) a revised cross-sectional grid is to be implemented before work can begin on bench plans. The new diamond drill holes have been tied to this grid.

VII RELIABILITY OF DATA

While the ore model constructed for the Large Faro Deposit may be quite correct overall, experience has shown that some discrete assay data are incorrect and/or overweighted. For at least two years the short term use of some of this data has resulted in overestimations of available Zn metal tons and underestimations of available Pb metal tons.

Present work on blasthole data indicates that this is due to:

1) a Pb zonation being missed in the DDH assay data due partly to the interference of Ba with data from one of the zones,

2) the intersection of erratic high Zn values in the original DDH's, and

3) possible errors in the original assays.

These problems are currently under close review and some steps have already been implemented on the short term. Systematic reassaying is about to be carried out in order to determine whether or not a reassessment of the DDH data should be carried out.

APPENDIX I

ESTIMATED TONNAGE AND GRADE TABLES

LARGE FARO DEPOSIT

3910 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

January 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-4	94,222	5.3	8.8	14.1
65-5A	188,444	4.7	5.8	10.5
65-7	92,338	2.6	7.2	9.8
65-11	141,333	6.4	5.2	11.6
65-14	7,538	2.4	4.7	7.1
66-4	15,075	1.8	3.9	5.7
66-8	48,053	4.3	6.2	10.5
66-9	66,898	3.6	8.7	12.3
66-12	178,080	6.9	9.7	16.6
66-14	94,222	3.4	7.4	10.8
66-15	188,444	4.4	6.8	11.2
66-16	211,058	6.8	8.5	15.3
66-18	40,044	2.4	4.7	7.1
66-20	62,658	5.3	7.6	12.9
66-23	12,249	5.8	8.3	14.1
66-24	37,689	2.6	6.3	8.9
66-25	80,089	4.8	8.2	13.0
66-32	101,760	4.0	7.8	11.8
66-35	77,262	4.1	8.4	12.5
66-39	13,191	3.6	8.7	12.3
66-41	10,364	5.3	8.8	14.1
66-44	107,413	7.3	8.6	15.9
67-34	15,075	4.8	8.2	13.0
70-1	92,338	3.1	5.7	8.8
70-2	94,222	2.0	4.0	6.0
70-3	94,222	3.7	6.8	10.5
70-4	94,222	3.4	6.3	9.7
70-5	94,222	3.0	7.8	10.8
70-6	94,222	5.2	7.3	12.5
70-8	9,422	1.8	4.1	5.9
70-9	141,333	4.2	6.0	10.2
70-10	6,596	3.1	5.7	8.8
70-11	94,222	1.4	5.5	6.9
70-14	135,680	6.3	7.2	13.5
70-16	94,222	2.3	5.5	7.8
72-1	59,596	6.9	9.7	16.6
72-3	87,627	5.3	6.9	12.2
72-6	81,973	3.3	6.6	9.9
72-7	86,213	4.3	5.7	10.0
Total	3,243,831	4.6	7.0	11.6

LARGE FARO DEPOSIT

3870 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

January 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
65-4	91,396	3.7	6.2	9.9
65-5A	188,444	6.0	6.7	12.7
65-7	159,236	2.5	6.1	8.6
65-11	165,831	6.7	7.4	14.1
66-2	11,778	3.1	4.0	7.1
66-4	118,720	1.8	3.9	5.7
66-8	151,698	4.3	6.2	10.5
66-9	31,564	2.8	6.0	8.8
66-12	146,987	3.1	5.5	8.6
66-14	66,898	2.6	3.7	6.3
66-15	188,444	5.5	6.4	11.9
66-18	3,769	2.5	6.1	8.6
66-22	98,698	2.8	3 3	6.1
66-23	93,280	5.8	8.3	14.1
66-30	3,769	2.5	6.1	8.6
66-32	135,680	3.6	3.9	7.5
66-35	63,129	1.8	5.5	7.3
66-39	45,698	2.8	3.9	6.2
66-44	130,969	8.0	10.7	18.7
66-53	11,307	2.5	6.1	8.6
70-1	95,164	2.9	6.6	9.5
70-2	94,222	2.0	3.1	5.1
70-3	73,964	4.6	9.2	13.8
70-4	94,222	2.2	4.1	6.3
70-5	94,222	3.3	6.5	9.8
70-6	221,422	4.8	7.9	12.7
70-8	16,018	2.3	3.9	6.2
70-9	141,333	2.5	5.1	7.6
70-11	65,404	2.0	6.2	8.2
70-14	141,333	5.2	8.6	13.8
70-16	94,222	2.7	6.5	9.2
72-1	65,019	2.7	5.4	8.1
72-3	152,640	5.2	6.9	12.1
72-6	219,774	2.7	6.7	9.4
72-7	163,947	6.2	6.3	12.5
Total	3,640,275	4.0	6.3	10.3

LARGE FARO DEPOSIT

3830 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

January 1973

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-4	86,684	4.0	6.1	10.1
65-5A	188,444	4.9	5.7	10.6
65-6	27,795	2.1	5.2	7.3
65-7	105,058	1.8	5.7	7.5
65-8	15,076	6.3	9.2	15.5
65-11	142,040	7.3	8.9	16.2
66-2	98,462	2.5	3.1	5.6
66-4	94,222	2.3	4.6	6.9
66-8	188,444	5.6	7.0	12.6
66-9	64,307	2.7	5.7	8.4
66-12	44,520	3.1	5.6	8.7
66-14	86,920	6.4	10.8	17.2
66-15	188,444	4.0	5.4	9.4
66-18	5,182	1.8	5.7	7.5
66-22	180,907	3.5	5.4	8.9
66-23	164,418	6.4	6.8	13.2
66-30	98,934	2.4	5.9	8.3
66-32	62,187	2.4	5.0	7.4
66-35	48,524	1.9	6.5	8.4
66-39	23,084	2.3	3.9	6.2
66-44	56,769	5.3	8.7	14.0
66-57	2,120	1.8	5.5	7.3
70-1	60,067	3.0	6.9	9.9
70-2	101,996	1.8	3.6	5.4
70-3	82,444	3.2	7.9	11.1
70-5	94,222	4.3	5.9	10.2
70-6	117,778	4.0	6.2	10.2
70-9	131,676	1.5	4.2	5.7
70-11	45,227	2.5	4.4	6.9
70-14	141,333	4.9	7.9	12.8
70-16	94,222	6.8	12.4	19.2
72-1	29,209	2.7	5.4	8.1
72-3	225,426	5.1	5.9	11.0
72-6	215,769	2.4	6.5	8.9
72-7	199,751	3.3	3.7	7.0
Total	3,511,661	3.9	6.1	10.0

LARGE FARO DEPOSIT

3790 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

January 1973

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-4	57,711	2.8	5.6	8.4
65-6	181,849	2.2	4.4	6.6
65-7	122,017	2.7	7.4	10.1
65-8	84,329	6.3	9.2	15.5
65-11	65,013	4.2	8.0	12.2
66-2	195,040	3.2	4.5	7.7
66-4	40,515	2.6	6.8	9.4
66-8	193,155	3.7	5.0	8.7
66-9	81,031	3.2	7.1	10.3
66-14	77,262	3.8	6.2	10.0
66-15	265,706	6.9	9.4	16.3
66-18	1,884	2.7	7.4	10.1
66-22	188,444	2.1	3.7	5.8
66-23	127,200	3.0	4.5	7.5
66-30	144,160	2.4	5.9	8.3
66-32	36,747	1.8	6.8	8.6
66-33	27,795	4.2	7.2	11.4
66-35	79,618	1.6	3.9	5.5
66-39	1,884	2.3	3.9	6.2
66-57	33,920	1.3	3.7	5.0
70-1	9,422	2.9	6.8	9.7
70-2	94,222	2.2	4.8	7.0
70-3	51,822	2.6	5.0	7.6
70-4	94,222	2.7	4.5	7.2
70-5	88,569	4.3	7.5	11.8
70-6	52,764	2.6	5.4	8.0
70-9	43,342	2.2	3.5	5.7
70-11	88,569	4.9	9.6	14.5
70-14	76,555	4.5	7.3	11.8
70-16	94,222	2.6	5.2	7.8
72-1	58,418	2.2	5.0	7.2
72-2	12,720	2.2	5.0	7.2
72-3	235,555	5.6	6.0	11.6
72-7	123,431	6.0	6.4	12.4
72-8	59,360	6.9	8.6	15.5
Total	3,188,473	3.7	6.1	9.8

LARGE FARO DEPOSIT

3750 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

January 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-4	2,827	2.6	5.3	7.9
65-6	317,548	2.1	3.0	5.1
65-7	63,128	2.4	6.2	8.6
65-8	141,333	6.8	10.1	16.9
65-12	1,884	4.5	6.7	11.2
66-2	254,871	2.3	3.9	6.2
66-4	73,493	2.3	5.4	7.7
66-6	2,827	2.4	6.2	8.6
66-8	275,129	3.1	4.4	7.5
66-9	23,556	3.4	7.7	11.1
66-14	40,515	3.6	6.5	10.1
66-22	114,951	2.9	5.9	8.8
66-23	55,591	4.9	6.3	11.2
66-30	214,827	2.4	6.2	8.6
66-31	23,555	5.9	8.4	14.3
66-33	120,604	4.2	7.2	11.4
66-35	148,871	1.2	3.8	5.0
66-57	27,324	3.1	8.0	11.1
70-2	47,111	2.5	6.5	9.0
70-4	116,836	3.0	4.0	7.0
70-9	131,911	4.8	7.8	12.6
70-11	92,338	3.5	6.2	9.7
70-14	27,325	5.4	7.3	12.7
70-16	44,284	2.4	5.2	7.6
72-1	27,324	2.2	5.0	7.2
72-2	163,004	3.9	6.3	10.2
72-3	262,880	5.4	7.3	12.7
72-7	114,951	2.2	5.2	7.4
72-8	126,257	6.7	8.5	15.2
Total	3,057,035	3.5	5.8	9.3

LARGE FARO DEPOSIT

January 1973

3710 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-6	292,324	2.8	5.0	7.8
65-7	30,151	1.2	4.2	5.4
65-8	116,364	5.1	6.4	11.5
65-9	26,382	4.3	6.5	10.8
65-12	94,222	4.3	7.7	12.0
66-2	245,449	3.1	4.9	8.0
66-4	140,391	5.4	10.0	15.4
66-6	33,449	1.5	4.0	5.5
66-8	244,035	3.5	5.4	8.9
66-14	2,355	6.5	10.8	17.3
66-22A	184,204	2.5	4.8	7.3
66-23	2,591	5.1	6.4	11.5
66-26	185,382	4.3	7.7	12.0
66-29	96,107	4.7	6.6	11.3
66-30	182,084	1.2	4.2	5.4
66-31	173,369	5.8	8,4	14.2
66-33	233,671	4.2	7.2	11.4
66-35	36,747	1.2	3.8	5.0
66-46	37,217	3.1	4.2	7.3
66-54	22,142	2.6	5.3	7.9
66-57	14,133	2.7	7.2	9.9
70-4	28,502	3.2	4.6	7.8
70-9	141,333	2.8	4.4	7.2
70-11	74,907	6.5	10.8	17.3
72-2	261,466	3.9	6.3	10.2
72-3	204,226	5.9	7.5	13.4
72-7	197,631	1.8	3.7	5.5
72-8	141,333	5.0	6.0	11.0
Total	3,442,167	3.7	6.1	9.8

LARGE FARO DEPOSIT

3670 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

January 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-6	345,559	3.6	7.8	11.4
65-9	191,977	4.3	6.5	10.8
65-12	166,538	2.6	4.8	7.4
66-2	265,706	2.7	4.3	7.0
66-4	115,187	3.2	5.9	9.1
66-46	140,155	3.1	4.2	7.3
66-55	942	2.6	4.8	7.4
66-6	77,969	1.8	3.6	5.4
66-8	157,587	4.9	7.1	12.0
66-22A	195,747	2.5	4.8	7.3
66-30	162,533	2.5	5.8	8.3
66-31	282,666	6.4	8.4	14.8
66-33	232,729	4.0	5.1	9.1
70-9	99,875	3.3	4.9	8.2
70-11	28,738	4.5	7.3	11.8
72-3	82,208	5.9	7.5	13.4
72-7	282,666	1.6	4.2	5.8
72-8	60,067	4.4	5.9	10.3
Total	2,888,849	3.5	5.8	9.3

LARGE FARO DEPOSIT

January 1973

3630 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-6	392,200	3.1	6.5	9.6
65-9	298,684	5.5	6.7	12.2
66-2	266,177	2.9	6.8	9.7
66-3	24,498	2.2	4.0	6.2
66-4	70,196	3.3	6.0	9.3
66-6	301,511	1.8	3.6	5.4
66-7	7,538	3.5	4.8	8.3
66-8	49,702	4.4	6.8	11.2
66-11	29,444	1.8	3.6	5.4
66-30	149,342	2.0	5.3	7.3
66-31	183,968	5.5	6.7	12.2
66-33	240,266	2.2	4.0	6.2
66-46	270,418	3.5	4.8	8.3
66-55	80,089	4.3	7.1	11.4
72-7	283,138	2.4	5.1	7.5
Total	2,647,171	3.2	5.6	8.8

LARGE FARO DEPOSIT

3590 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

January 1973

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-6	215,769	2.8	5.9	8.7
65-9	234,142	3.3	4.0	7.3
66-2	66,897	2.6	6.8	9.4
66-3	91,867	2.1	3.7	5.8
66-6	458,862	3.9	8.6	12.5
66-7	84,800	3.4	6.5	9.9
66-11	274,186	2.6	5.1	7.7
66-30	35,569	1.7	4.1	5.8
66-31	22,142	5.5	6.7	12.2
66-33	220,951	2.1	3.7	5.8
66-55	37,924	4.3	6.5	10.8
66-46	302,924	2.6	3.5	6.1
66-52	21,200	3.3	4.0	7.3
72-16	13,191	2.6	5.1	7.7
66-47	25,440	3.4	6.5	9.9
72-7	24,498	3.5	6.5	10.0
Total	2,130,362	3.0	5.6	8.6

LARGE FARO DEPOSIT

3870 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
FRH-16	35,568	2.8	4.3	7.1
66-E1	1,884	3.5	6.8	10.3
66-E4	27,795	4.5	6.4	10.9
66-E7	15,075	2.8	4.3	7.1
Total	80,322	3.4	5.1	8.5

LARGE FARO DEPOSIT

May, 1973

3830 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
FRH-16	2,355	2.8	4.3	7.1
66-E1	5,182	3.5	6.8	10.3
66-E4	112,831	3.2	6.8	10.0
66-E7	45,226	2.8	4.3	7.1
Total	165,594	3.1	6.1	9.2

LARGE FARO DEPOSIT

3790 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-E1	102,466	3.5	6.8	10.3
66-E4	165,124	1.6	3.5	5.1
66-E5	25,204	1.7	3.5	5.2
66-E6	40,515	2.1	4.1	6 2
Total	333,309	2.2	4.6	6.8

LARGE FARO DEPOSIT

May, 1973

3750 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-E1	84,328	3.5	6.8	10.3
66-E4	77,026	1.6	3.5	5.1
66-E5	67,839	1.7	3.6	5.3
66-E6	146,044	2.3	4.1	6.4
Total	375,237	2.3	4.5	6.8

LARGE FARO DEPOSIT

3710 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-E5	65,484	2.6	4.6	7.2
66-E6	154,995	1.7	5.2	6.9
66-E9	18,373	2.0	3.0	5.0
Total	238,852	2.0	4.9	6.9

LARGE FARO DEPOSIT

3670 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
66-E6	180,435	1.7	5.2	6.9
66-E9	52,057	1.8	3.2	5.0
67-5	7,537	2.4	5.9	8.3
Total	240,029	1.7	4.8	6.5

LARGE FARO DEPOSIT

3630 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-E6	133,324	1.7	5.2	6.9
66-E9	73,257	1.8	3.2	5.0
67-4	1,884	2.3	5.7	8.0
67-5	119,191	2.3	5.7	8.0
67-7	28,502	2.4	5.0	7.4
Total	356,158	2.0	4.9	6.9

LARGE FARO DEPOSIT

3590 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-E9	25,440	1.9	5.2	7.1
67-5	146,044	3.0	6.5	9.5
67-7	80,324	2.9	5.6	8.5
Total	251,808	2.9	6.1	9.0

LARGE FARO DEPOSIT

3550 Bench

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	% (Pb + Zn)
65-6	95,871	1.8	3.9	5.7
65-9	143,453	3.8	5.2	9.0
66-E9	57,004	1.9	5.2	7.1
66-2	125,315	2.6	6.8	9.4
66-3	150,519	4.3	7.6	11.9
66-5	7,066	2.1	3.7	5.8
66-6	252,515	3.1	7.7	10.8
66-7	231,079	3.4	6.5	9.9
66-11	89,275	3.1	7.7	10.8
66-46	274,893	1.8	3.9	5.7
66-52	72,315	5.4	7.1	12.5
66-55	65,484	3.0	4.4	7.4
67-3	89,039	2.2	4.0	6.2
67-5	172,897	2.1	5.2	7.3
67-7	158,528	3.0	4.8	7.8
67-30	7,537	3.0	4.8	7.8
72-16	270,653	2.4	4.8	7.2
Total	2,263,443	2.8	5.6	8.4

LARGE FARO DEPOSIT

3510 Bench

May, 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
65-6	24,497	2.3	4.8	7.1
66-E9	40,515	3.6	7.6	11.2
66-3	468,048	4.2	7.6	11.8
66-5	77,733	2.3	3.4	5.7
66-7	221,186	4.2	9.5	13.7
66-10	156,644	2.5	2.6	5.1
66-46	256,048	2.3	4.8	7.1
66-49	7,537	5.4	6.7	12.1
66-52	207,995	5.4	7.1	12.5
67-3	115,186	2.1	3.5	5.6
67-4	67,604	3.1	5.1	8.2
67-5	154,524	1.9	4.9	6.8
67-6	229,195	1.9	4.2	6.1
67-7	81,031	3.0	4.8	7.8
67-8	52,528	3.0	6.3	9.3
67-10	180,906	2.1	3.3	5.4
67-11	311,168	3.0	7.0	10.0
67-12	80,559	2.4	3.7	6.1
67-30	208,466	3.0	4.8	7.8
72-15	60,537	1.7	4.6	6.3
72-16	112,359	2.1	5.4	7.5
Total	3,114,266	3.0	5.7	8.7

LARGE FARO DEPOSIT

3470 Bench

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-3	138,271	2.1	3.7	5.8
66-5	136,857	2.3	3.4	5.7
66-7	70,902	1.5	3.6	5.1
66-10	7,537	1.5	3.6	5.1
66-46	151,933	2.9	6.6	9.5
66-49	357,102	5.4	6.7	12.1
66-52	299,626	6.1	7.1	13.2
67-3	231,079	3.2	5.4	8.6
67-4	220,244	1.8	4.6	6.4
67-5	97,284	1.9	4.9	6.8
67-7	179,257	2.7	5.2	7.9
67-8	89,039	2.8	6.4	9.2
67-10	167,479	3.1	7.2	10.3
67-11	60,537	3.0	7.0	10.0
67-12	284,551	2.0	4.3	6.3
67-30	728,573	3.7	4.7	8.4
70-13	52,528	3.1	5.2	8.3
70-17	288,319	3.6	5.2	8.8
71-4	123,195	2.5	4.1	6.6
71-5	241,679	3.1	5.2	8.3
Total	3,925,992	3.4	5.3	8.7

LARGE FARO DEPOSIT

3430 Bench

May, 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
66-3	404,684	2.5	3.1	5.6
66-5	13,426	2.1	3.7	5.8
66-7	85,977	1.5	3.6	5.1
66-10A	146,515	1.9	3.3	5.2
66-46	175,253	3.4	5.9	9.3
66-49	376,888	2.3	3.3	5.6
66-52	231,786	5.4	7.1	12.5
67-3	246,626	2.7	5.9	8.6
67-4	54,884	5.3	10.1	15.4
67-6	207,759	3.2	4.6	7.8
67-7	5,182	2.7	5.9	8.6
67-10	94,928	3.2	7.5	10.7
67-11	376,888	2.7	6.3	9.0
67-12	335,902	3.5	5.9	9.4
67-30	393,613	3.6	5.1	8.7
70-12	180,435	5.7	8.1	13.8
70-17	359,457	3.6	5.2	8.8
71-2	16,959	2.2	4.2	6.4
71-3	16,959	5.7	8.1	13.8
71-4	298,213	2.5	4.1	6.6
71-5	161,591	3.1	5.2	8.3
72-13	33,448	6.0	7.6	13.6
Total	4,217,373	3.2	5.2	8.4

LARGE FARO DEPOSIT

3390 Bench

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	% (Pb + Zn)
66-3	356,631	2.3	4.3	6.6
66-7	5,653	3.6	6.6	10.2
66-10	508,564	2.6	4.9	7.5
66-46	46,168	3.6	6.6	10.2
66-49	231,786	2.9	4.6	7.5
66-52	8,008	4.1	5.8	9.9
67-3	3,297	2.9	6.4	9.3
67-6	249,453	1.9	5.0	6.9
67-10	35,804	1.9	5.0	6.9
67-11	220,479	2.8	5.7	8.5
67-12	292,324	3.3	5.0	8.3
70-12	55,355	5.7	8.1	13.8
71-1	7,302	5.7	8.1	13.8
71-3	9,893	5.7	8.1	13.8
72-13	102,702	6.0	7.6	13.6
Total	2,133,419	2.9	5.2	8.1

LARGE FARO DEPOSIT

3350 Bench

May, 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
66-3	14,133	2.1	5.3	7.4
67-6	42,635	2.6	6.0	8.6
72-13	214,826	4.5	4.5	9.0
Total	271,594	4.1	4.8	8.9

SUMMARY OF ESTIMATED TONNAGE AND GRADE

LARGE FARO DEPOSIT

NORTH END: BENCHES 3910 to 3590 inc.

January, 1973

Bench Summary 8,500 N to 11,400 N
12,000 E to 16,000 E

Bench Number	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
3910	3,243,831	4.6	7.0	11.6
3870	3,640,275	4.0	6.3	10.3
3830	3,511,661	3.9	6.1	10.0
3790	3,188,473	3.7	6.1	9.8
3750	3,057,035	3.5	5.8	9.3
3710	3,442,167	3.7	6.1	9.8
3670	2,888,849	3.5	5.8	9.3
3630	2,647,171	3.2	5.6	8.8
3590	2,130,362	3.0	5.6	8.6
Total	27,749,824	3.7	6.1	9.8

SOUTH END: BENCHES 3870 to 3350 inc.

May, 1973

Bench Summary 7,000 N to 8,500 N
12,000 E to 16,000 E

Bench Number	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
3870	80,322	3.4	5.1	8.5
3830	165,594	3.1	6.1	9.2
3790	333,309	2.2	4.6	6.8
3750	375,237	2.3	4.5	6.8
3710	238,852	2.0	4.9	6.9
3670	240,029	1.7	4.8	6.5
3630	356,158	2.0	4.9	6.9
3590	251,808	2.9	6.1	9.0
3550	2,263,443	2.8	5.6	8.4
3510	3,114,266	3.0	5.7	8.7
3470	3,925,992	3.4	5.3	8.7
3430	4,217,373	3.2	5.2	8.4
3390	2,133,419	2.9	5.2	8.1
3350	271,594	4.1	4.8	8.9
Total	17,967,396	3.0	5.3	8.3

TENTATIVE ESTIMATED TONNAGE AND GRADE TOTALS TO DATE

Deposit	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
Lge. Faro North	27,749,824	3.7	6.1	9.8
Lge. Faro South	17,967,396	3.0	5.3	8.3
Small Faro	2,928,188	2.8	4.7	7.5
Total	48,645,408	3.4	5.7	9.1

Ultimate Pit
1973

ANVIL MINING CORPORATION LIMITED

MEMORANDUM

TO: W. Krats FROM: J. McLachlan

SUBJECT: STRIPPING RATIO - WEST SIDE OF ULTIMATE PIT DATE: February 22, 1973

In order to determine if the west wall of the pit can be stripped economically an investigation into the order of magnitude of various stripping ratios has recently been completed. Basis of the comparison was to analyze Case A against Cases B, C, D and E, all described below.

CASE A: This situation simply set up a hypothetical case which showed how much ore would be left remaining in the west wall if a 1:1 slope (toe-toe spacing of 40') were carried down from the presently existing slope of the wall. No measurement of waste or ore yardage to complete to this design was made since it was simply the difference between this case and the others that was of significance. (Drawing FX-7301).

It became readily apparent that the 3710 bench was controlling the shape of the west wall. A large rather sharp block of ore on this bench protrudes farther to the west than on any other bench. Most of the changes engineered in Cases B to E concerned the movement of the toe line in or out on this bench.

CASE B: All the ore on the extremities of the west side of the 3710 bench is 11% plus. It was thus essential that the placement of the final toe line on this bench mined a maximum of this material yet did not impose too high a stripping burden by "going after" sharp awkward corners. Quantities of ore mined in this case compared to that which would have been left in Case A plus the waste required to achieve this position are summarized in Table 1. Drawing reference is FX-7302 for this case.

CASE C: The second trial at reducing the stripping ratio centered around moving the 3710 toe line "in" a full 40' in one portion of the bench and gradually tapering the change into the previously established toe line in Case B. The object was to pull in all benches above this in the hope that the quantity of waste stripping would be significantly reduced. Results are summarized in Table 2. Drawing reference is FX-7303.

CASE D: Since the result in Case C did not decrease the stripping ratio, but only increased it (lost ore "faster" than waste) the next situation was to move the 3710 toe line out further. This, it was hoped, would gain ore "faster" than the increased load of waste stripping accompanying it. Results are summarized in Table 3. Drawing FX-7304 applies to this case.

CASE E: A further trial run was made with the presently designed ultimate pit to determine what its stripping ratio would be with the revised set of bench plans. Results are in Table 4.

In all cases ore on 3910 was not measured even though there appeared to be some remaining on the bench plans. This was not included because a visual inspection of the present west wall of 3910 bench does not reveal any ore remaining there.

Since the grade of ore and tonnage which any resultant stripping ratio is capable of supporting are closely tied together, Table 5 has been detailed to show the grade and tonnage on each bench in Case B and then averaged overall for the entire west side.

CONCLUSIONS: Results of this exercise have indicated the best possible stripping ratio on the west side to be of the order of 10:1. In order to more fully evaluate if this is economically practical, further work and cost investigations should be carried out on a break even stripping ratio related to grade of ore and market prices.

J.R. McLachlan

J. R. McLachlan
Planning Engineer

JRM/mm

Attach.

TABLE I

TRIAL ONE STRIPPING -- CASE B

Bench	Ht.	Total Yardage (O. & W.)	Waste (Yd.)	Ore (Yd.)	Ore (Tons)
4205	27'	48,750	48,750		
4170	44'	183,580	183,580		
4135	26'	188,360	188,360		
4100	39'	427,000	427,000		
4065	31'	338,930	338,930		
4030	35'	416,890	416,890		
3990	40'	469,560	469,560		
3950	40'	454,670	454,670		
3910	40'	447,780	447,780		
3870	40'	446,220	436,660	9,560	30,400
3830	40'	438,000	426,670	11,330	36,030
3790	40'	416,440	376,220	40,220	127,900
3750	40'	359,560	288,230	71,330	226,830
3710	40'	324,000	125,560	198,440	631,040
3670	40'	157,560	100,000	57,560	183,040
3630	40'	56,000	Ø	56,000	178,080
3590	40'	22,890	Ø	22,890	72,790
			4,728,860	467,330	1,486,110

Stripping Ratio $\frac{4,728,860}{467,330} = \frac{10.1}{1}$

TABLE 2

TRIAL TWO STRIPPING -- CASE C

3710 ORE "PULLED IN". MODIFICATION CARRIED UP TO SURFACE.

(INCREMENTAL CHANGE ONLY MEASURED)

DECREASE IN WASTE & ORE

BENCH	HT.	WASTE (Yd.)	ORE (Yd.)	ORE (Tons)
4205	27	10,500		
4170	44	28,600		
4135	26	20,370		
4100	39	33,800		
4065	31	26,520		
4030	35	30,140		
3990	40	38,890		
3950	40	35,560		
3910	40	36,220		
3870	40	29,330	19,080	60,670
3830	40	29,330	7,110	22,610
3790	40	23,560	22,440	71,360
3750	40	16,000	5,560	17,680
3710	40		14,890	47,350
		358,820	69,080	219,670

Stripping Ratio on Revision Itself $\frac{358,820}{69,080} = \frac{5.2}{1}$

New Ratio $\frac{4,728,860 - 358,820}{467,330 - 69,080} = \frac{4,370,040}{398,250} = \frac{11}{1}$

TABLE 3

TRIAL THREE STRIPPING -- CASE D

3710 ORE "PUSHED OUT". INCREMENT ONLY MEASURED.

INCREASE IN WASTE & ORE QUANTITIES

BENCH	HT.	WASTE (Yd.)	ORE (Yd.)	ORE (Tons)
4205	27	6,450		
4170	44	16,620		
4135	26	12,420		
4100	39	16,900		
4065	31	14,120		
4030	35	16,140		
3990	40	20,220		
3950	40	22,670		
3910	40	20,000		
3870	40	15,110	7,110	22,610
3830	40	24,440	-	-
3790	40	17,110	-	-
3750	40	11,780	7,780	24,740
3710	40	15,550	13,110	41,690
3670			2,670	8,490
		229,530	30,670	97,530

Stripping Ratio on Revision Itself $\frac{229,530}{30,670} = \frac{7.5}{1}$

New Ratio $\frac{4,728,860 + 229,530}{467,330 + 30,670} \quad \frac{4,958,390}{498,000} = \frac{10}{1}$

TABLE 4

CASE E -- STRIPPING RATIO WEST SIDE ULTIMATE

PIT USING REVISED BENCH PLANS

ORE AVAILABLE

BENCH	HT.	YARDS	TONS
3870	40'	20,670	65,730
3830	40'	10,890	34,630
3790	40'	44,890	142,750
3750	40'	80,890	257,230
3710	40'	216,220	687,580
3670	40'	52,880	168,160
3630	40'	64,440	204,920
3590	40'	23,110	73,490
		513,990	1,634,490

NOTE: Did not attempt a measure of the volume of waste stripping involved since the incremental increase in ore in Table 3 compared to Table 4 was very minimal. It appeared obvious that there would be a tremendous increase in stripping for only a small increase in ore. Hence, a final ratio was not calculated.

TABLE 5

CASE B

INDIVIDUAL GRADES & TONNAGES

BENCH	YD.	TONS	PB	ZN	COMB.
3870	8,717	27,720	2.70	5.40	8.10
3830	12,141	38,610	2.78	5.44	8.22
3790	44,928	142,870	2.58	4.76	7.34
3750	72,047	229,110	4.51	6.86	11.37
3710	200,553	637,760	4.21	6.91	11.12
3670	61,289	194,900	3.10	5.02	8.12
3630	44,896	142,770	4.79	6.93	11.72
3590	22,742	72,320	3.85	5.38	9.23
	467,313	1,486,060	3.92	6.31	10.23

7.16 Eng 2.1

A. /IL MINING CORPORATION LIMITED

MEMORANDUM

TO: J. Keily FROM: W. Krats

SUBJECT: PROJECT STATUS DATE: April 23, 1973

Reserve Estimate:

Zone I - Ore tonnage and grade, calculated, accumulated and checked at present.

Zone 3 - Completion will be realized by early May to the same degree as Zone I above.

Zone 2 - A preliminary run was taken at Zone 2 during the winter. It has been concluded that it is desirable to complete the 1973 diamond drilling and analyze the results before completing a publishable reserve figure in this area.

Figures on Zone 1 and 3 will be available internally by mid-year, but the intent is to boil in the 1973 diamond drilling results this fall and publish a formal reserve statement in January of 1974.

Ultimate Pit:

This design is currently in progress and should be complete by early May. This pit will include virtually all 5% or greater lead plus zinc material estimated in the reserve estimate for Zones 1 and 3.

Five Year Plans:

Work is presently underway on plans through to and including 1978. Work at present is progressing along the avenue of supplying all mill feed from Zone I. Stockpiles are not presently being included because no metallurgical data is available regarding the feasibility of processing this material. The end of May should see completion of the first run on a year by year basis through 1978. These will be refined by August 1, 1973 for presentation along with the 1974 mining plan for budget purposes.

W.K.

W. Krats
Chief Engineer

WK/mm

ANVIL MINING CORPORATION LIMIT.

MEMORANDUM

TO: W. Krats FROM: J. McLachlan

SUBJECT: ULTIMATE PIT DESIGN - ZONES 1 and 3 DATE: July 16, 1973

(REVISION OF APRIL, 1973)

A revised ultimate pit plan to mine virtually all the economic ore in Zones 1 and 3 was completed in April, 1973. Completion of the up-dated model of Zones 1 and 3 orebodies in January, 1973 allowed the new estimates of the ultimate pit to be prepared.

Toe lines were drawn for the study but in line with changing all earthwork measurement to bench midslopes, any future work on the ultimate pit will be with midslopes. Quantities measured were those required to complete the pit to its ultimate designed limits from the pit status of Jan. 1, 1973. Final wall slope has all been designed at 45°. But in some areas of high overburden removal it is entirely possible this may have to be flattened out to a 2:1 slope, dependent largely upon field conditions encountered during the mining of the overburden.

Ore Not Included:

Not all ore as stated in the January, 1973 revision of Zones 1 and 3 is included in the revised ultimate pit design. The ore not included is stated below:

	Tons	Pb	Zn
Zone 1	145,810	4.1	6.7
Zone 3	1,487,060	3.3	5.6
	1,632,870	3.4	5.7

Some ore had to be eliminated and the reasons for these eliminations fell into two broad categories: economics and the requirements of a smoother overall pit configuration.

1. Economics:

   (a) On some benches the ore is discontinuous and so widely separated from the ore on the remainder of the bench that to go after the small, discontinuous portion simply produced too great a quantity of waste for the yield of ore derived.

(b) Sharp isolated irregular projections of ore on the eastern side of the Zone 3 orebody produced an awkward shape required to mine it and an extremely high (and probably costly) stripping ratio. (In one particular case examined in detail, 6,448,400 yards of waste had to be removed to mine 608,400 tons at 2.6, 6.5 on benches 3510-3430. Resulting stripping ratio was 33:1.)

2. Pit Configuration:

Very irregular contacts which have resulted in bumps or depressions in the orebody do not produce smooth configurations of the ultimate pit wall. Sharp points in the walls tend to produce areas of stress concentration and further trouble in later years of pit life. Elimination of any possible sources of problems early in the pit development by trimming irregular corners of ore can only be of help later on.

Conclusion:

The ultimate pit design of April, 1973 represents only one of a number of possible alternate designs based on economics, operating efficiency and safety. It is not intended to be a panacea for all ultimate pits, but rather a starting point from which to alter and modify designs at any time based mainly upon prevailing prices of lead and zinc.

The attached table outlines all relevant data on the April, 1973 ultimate pit revision.

J. R. McLachlan
Planning Engineer

JRM/mm

Attach.

ULTIMATE PIT - April 1973 Design

Zones 1 & 3

(Quantities Measured Are Those Remaining At January 1, 1973)

PRE 1973 DRILLING

Bench	Total Yardage	Zone No. 1 Tons of Ore	Zone No. 1 Grade	Zone No. 1 Yd. of Ore	Zone No. 3 Tons of Ore	Zone No. 3 Grade	Zone No. 3 Yd. of Ore	Total Ore Tonnage	Total Ore Yardage	Waste Yardage
										54 ,30
4275	54,430									30 ,70
4210	303,070									979,780
4205	979,780									1,781,480
4170	1,781,480									2,343,680
4135	2,343,680									2,889,560
4100	2,889,560									3,747,840
4065	3,747,840									4,036,510
4030	4,038,070	4,950	1.7, 4.6	1,560				4,950	1,560	6,315,240
3990	6,345,590	96,510	2.4, 5.3	30,350				96,510	30,350	8,332,110
3950	8,457,380	398,360	2.7, 6.4	125,270				398,360	125,270	8,869,750
3910	9,142,640	867,780	3.3, 6.7	272,890				867,780	272,890	8,100,450
3870	8,734,010	2,003,410	3.3, 5.6	630,000	11,310	4.0, 6.3	3,560	2,014,720	633,560	7,359,010
3830	8,438,800	3,340,950	3.9, 6.2	1,050,610	92,810	3.2, 6.8	29,180	3,433,760	1,079,790	6,764,410
3790	7,866,550	3,171,510	3.7, 6.1	997,330	333,310	2.2, 4.6	104,810	3,504,820	1,102,140	6,020,820
3750	7,091,730	3,030,240	3.5, 5.8	952,910	375,240	2.3, 4.5	118,000	3,405,480	1,070,910	5,066,790
3710	6,205,820	3,394,350	3.7, 6.1	1,067,400	227,780	2.0, 5.0	71,630	3,622,130	1,139,030	4,482.490
3670	5,465,230	2,886,490	3.5, 5.8	907,700	238,620	1.7, 4.8	75,040	3,125,110	982,740	3,711 ,0
3630	4,638,200	2,625,500	3.2, 5.6	825,630	321,300	2.0, 5.1	101,040	2,946,800	926,670	3,177,680
3590	3,918,270	2,130,360	3.0, 5.6	669,920	224,720	3.0, 6.2	70,670	2,355,080	740,590	2,567,160
3550 3550	3,255,230				2,188,070	2.8, 5.6	688,070	2,188,070	688,070	1,809,000
3510 3510	2,693,510				2,812,750	3.0, 5.8	884,510	2,812,750	884,510	1,098,080
3470	2,221,110				3,571,240	3.5, 5.3	1,123,030	3,571,240	1,123,030	437,980
3430	1,735,160				4,125,040	3.2, 5.1	1,297,180	4,125,040	1,297,180	3,710
3390	619,490				1,958,170	2.6, 5.0	615,780	1,958,170	615,780	
	102,966,630	23,950,410	3.5, 5.2	7,531,570	16,480,360	3.0, 5.2	5,182,500	40,430,770	12,714,070	90,252,560

7,974,530 moved to 1-1-73

31,924,940

Notes:

(1) Bench 3350 not counted in ultimate pit

(2) Benches 4275 - 4030 are 35'; 4030 - 3390 are 40'.

J. McL.
May 18, 1973

Summary Of Ore Outside Ultimate Pit Design In Each Zone

As Of April-73 Design

Zone 1	145,810	4.1	6.7
Zone 3	1,487,056	3.3	5.6
	1,632.866	3.4	5.7

May 10, 1973 Revision

Zone 1	145,810	4.1	6.7
Zone 3	878,615	3.8	5.5
	1,024,425	3.8	5.7

Stripping Ratio To Mine 608,441 tons of ore in zone 3 was 33/1.

V LARGE FARO DEPOSIT

a) 3910 to 3590 benches inclusive from 8,500 N to 11,400 N and 12,000 E to 16,100 E.

A new tonnage and grade estimate was prepared for this portion of the Large Faro Deposit to allow comparison of the new complete estimate with the blasthole data on some of the benches and to have adequate estimates for five year mining plan.

Estimated Tonnage and Grade Tables, January 1973 (see Appendix I) and Estimated Tonnage and Grade Bench Plans (see Appendix II) show the distribution and the totals completed and checked to date. Final copies of the ink drafted bench plans will be available on request when they have been completed.

b) 3550 bench and below from 8,500 N to 11,400 N and 12,000 E to 16,000 E also all benches from 6,600 N to 8,500 and 12,900 E to 17,000 E.

The sections have been completed and the bench plans have been started. Work on this portion of the deposit is progressing as man power becomes available.

VI SMALL FARO DEPOSIT

A total tonnage and grade estimate has shown that the major trend direction is not similar to the Large Faro Deposit. This trial estimate has outlined approximately 2.9 million tons of 2.78% Pb and 4.76% Zn.

Summary of Ore Outside 1973 Ultimate Pit Design

Zone 3

3870 —	69,016	3.3	4.9	Longitudinal Shape of Ore Is Such That Could Only Economically
3830	72,785	2.9	5.2	Take Portions of It.
3790	—	—	—	
3750	—	—	—	
3710	11,071	1.9	3.4	Small Portions Cut Off Corners Of Ore To Improve
3670	1,413	1.8	3.2	Configuration of Ult. Pit Wall
3630	34,862	1.8	3.3	Discontinuous & Widely Separate
3590	27,089	2.0	5.3	Parts of Ore Benches Are Not Included Because Stripping Req'
3550	75,377	2.1	5.1	To Expose Too Great
3510	301,512	2.6	5.2	Not Economic. Ore Projects Long Way Into Eastern Side
3470	354,747	2.3	5.3	of Pit And Irregular Shape Produced That Is Required To
3430	92,337	4.3	8.1	Mine It Results In High Costly Stripping Ratio.
3390	175,253	5.9	7.8	
3350	271,594	4.1	4.8	Not Incl Ult. Pit. Too Far
	1,487,056	33	5.6	Offset From Main Orebody.

COMPARISON OF 1970 AND 1973 ORE TONNAGE AND GRADE ESTIMATES (zone I)

Below is a table comparing (by bench) the 1970 and 1973 reserve tonnage and grade estimates. Note that the 1970 estimate did not break the grade down into Pb and Zn. The comparison has thus been made on the same terms.

TABLE 1: Comparing 1970 & 1973 Ore Tonnage and Grade Estimates

	1970 Estimate		1973 Estimate	
Bench	Tonnage	Grade	Tonnage	Grade
3910	3,139,012	11.65	3,243,831	11.6
3870	3,473,477	11.20	3,640,275	10.3
3830	4,122,308	10.30	3,511,661	10.0
3790	3,575,901	7.81	3,188,473	9.8
3750	3,265,451	9.27	3,057,035	9.3
3710	3,854,840	9.47	3,442,167	9.8
3670	3,129,411	10.82	2,888,849	9.3
3630	2,426,080	9.28	2,647,171	8.8
3590	1,555,955	7.67	2,130,362	8.6
Total	28,542,435	9.46	27,749,824	9.8

Benches 3830-3670 inclusive have been reduced in tonnage in the new estimate, the remaining three have been increased. The grade has remained essentially unchanged for benches 3910 and 3750. The remaining benches have been changed in varying ways. Overall, the total tonnage has been reduced by 769,055 tons, while the 1970 estimate originally contained 871,902 tons averaging 4.01% Pb + Zn. The original aim of this reappraisal was both to ensure an objective approach and attempt to eliminate this < 5% material.

These tonnage and grade changes can be attributed to the following causes:

Tonnage Changes:

(1) New data acquired through 1972 drilling

(2) Editing cross-sections with respect to 5' intersections <5% which were incorporated into the 1970 sections without any apparent systematic approach.

(3) Reinterpretation of cross-sections based on (1) and (2)

Grade Changes:

(1) New data acquired through 1972 drilling.

(2) Adoption of 40' averages rather than the optimistic approach of averaging only the ore interval intersected within any bench interval. In only one case was it considered necessary to do this in the present estimate. The 1970 estimate frequently employed 10' to 35' values somewhat arbitrarily.

(3) Reweighting of areas of influence using holes surrounding the ore as well as those within. The 1970 estimate employed only those within, frequently resulting in projection of grade to areas very remote from the source of the data yet quite close or almost adjacent to holes intersecting ore in the next bench which were used to project the ½ bench line.

(4) Discarding ore which was projected as being < 5% for a 40' intersection in the 1970 estimate and is still < 5% by the new estimate. This occurred in only a few cases.

The table below categorizes these tonnage and grade changes per bench. It must be understood, however, that only cases where the category has made a significant change are shown. All categories apply in many cases, but the table only outlines the highlights. Also in the table is a comparison of the number of areas of influence employed for the two estimates. This attempts to demonstrate the amount of significant data added to the earlier approach. The number of 1972 holes (new data) used is shown also.

TABLE 2: Showing Sources of Change in Estimates Keyed to Seven Causes Listed Above

Bench	Areas of Influence 1970	Areas of Influence 1973	1972 DDH's Used	Cause of Change: Tonnage (1)	(2)	(3)	Grade (1)	(2)	(3)	(4)
3910	27	39	4	x		x				
3870	26	35	4	x		x	x		x	
3830	28	36	4	x	x	x			x	
3790	22	35	5	x	x	x	x		x	x
3750	14	29	5		x	x				
3710	12	28	4	x	x	x				x
3670	10	17	3	x	x	x		x		
3630	6	15	1	x	x	x			x	
3590	4	16	2		x	x			x	

..NVIL MINING CORPORATION LIMI..

MEMORANDUM

TO: D. W. Philip FROM: J. R. McLachlan

SUBJECT: TOTAL VOLUMES - ULTIMATE PIT DATE: March 7, 1974

In September 1970 an ultimate pit was completed for Zones 1 and 3 which moved virtually all known ore at the time. This was the first time an ultimate pit had been completed for both Zones 1 and 3. Pit configuration at the time was to mine every second bench wide, "stacking up" two benches at a time. The figures measured were for the total amount of material from the beginning of pre-production stripping in October 1967 to the then proposed ultimate pit.

	Total Movement	Ore	Waste	
Ultimate Pit (Zones 1 and 3)	137,265,300 Yd.	18,337,735 Yd. (58,314,000 T.)*	118,927,565 Yd.	6.5/1
Zone 2	6,327,065 Yd.	1,622,325 Yd. (5,159,000 T.)*	4,704,740 Yd.x	2.9/1
Totals	143,592,365 Yd.	19,960,060 Yd. (63,473,000 T.)	123,632,305 Yd.	6.2/1

Total yardage moved (1967-1973), accumulated from production statistics, including all mill feed and stockpiled ore - 32,036,700 yd.

In April 1973 another ultimate pit was completed. It was done after a careful re-analysis of the ore reserves had been completed by P. M. Pettigrew. The pit was designed at that time keeping every bench open with a 40' toe-toe spacing.

* E. N. Pennebaker's ore reserve calculations - October, 1967

x Based on a 2.9/1 stripping ratio which is the most recent figure available on a pit designed for Zone 2.

Measured at January 1, 1973:

Total Movement (Ultimate Pit)	102,966,630 Yd.	Original Reserves	58,314,000 T. (18,337,735 Yd.)
Total Movement (Year-1973	5,569,590 Yd.	Mill Feed (Sept. 69-Dec. 73)	10,873,675 T. (3,419,395 Yd.)
Jan. 1, 1974 (Zones 1 & 3)	97,397,040 Yd.	Remaining Jan. 1-74 (Zones 1 & 3)	47,440,325 T. (14,918,340 Yd.)
Add: Zone 2	6,327,065 Yd.		5,159,000 T. (1,622,325 Yd.)
	103,724,110 Yd.		52,599,325 T. (16,540,670 Yd.)

J. R. McLachlan
Planning Engineer

JRM/mm

Geol. Reserve
Est., Dec./73

PROGRESS REPORT

TONNAGE AND GRADE ESTIMATES

FARO DEPOSITS

December, 1973

Anvil Mining Corporation Limited

Whitehorse M.D., Yukon Territory

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b) 3550 bench and below from 8,500 N to 11,400 N and 12,000 E to 16,000 E, also all benches from 7,000 N to 8,500 N and 12,000 E to 16,000 E

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APPENDIX

I Estimated Tonnage and Grade Tables, Large Faro Deposit, December 1973

II Estimated Tonnage and Grade Tables, Small Faro Deposit, December 1973

III Estimated Tonnage and Grade Bench Plans, Large Faro Deposit, December 1973

IV Estimated Tonnage and Grade Bench Plans, Small Faro Deposit, December 1973

I INTRODUCTION

During November and December 1972 and January 1973 all the diamond drill hole and rotary hole logs were reviewed and updated and new sections and plans were plotted. During the period January to May 1973 a tonnage and grade estimate was conducted for the Large Faro Deposit.* In December 1973 this estimate was revised, based upon the 1973 development diamond drilling.

During October to December 1973 a tonnage and grade estimate was conducted for the Small Faro Deposit. New sections were drawn to coincide with a rotation of the original cross-sectional grid and to incorporate the results of twenty-three 1973 diamond drill holes.

II SUMMARY

1) This estimate is a compilation of tonnages and grades of $\geq 5\%$ (Pb + Zn) material for ≥ 5 foot assay intervals.

2) In the Large Faro Deposit (North) on the 3910 to 3590 benches inclusive from 8,500 N to 11,400 N and 12,000 E to 16,000 E the total of mined and geologic reserve material is estimated to be 26,379,864 tons of 3.8% Pb and 6.1% Zn.

3) In the Large Faro Deposit (South) on benches 3550 and below from 8,500 N to 11,400 N and 12,000 E to 16,000 E and also all benches 7,000 N to 8,500 N and 12,000 E to 16,000 E, the total geologic reserve material is estimated to be 17,176,406 tons of 3.0% Pb and 5.3% Zn.

4) In the Small Faro Deposit, the total geologic reserve is estimated to be 3,424,003 tons of 3.4% Pb and 5.1% Zn.

III BRIEF GEOLOGY

The Faro deposits consist of two near horizontal strataform, stratabound massive pyrite, sphalerite, galena and pyrrhotite deposits. The zinc and lead values in the massive sulfides range from below the economic cut off of 5% (Pb + Zn) now used, to well above this value. Some major distribution patterns within the massive sulfides have been recognized, although they are not necessarily continuous throughout.

* What is now being called the Large Faro Deposit has been referred to in the past as Zones 1 and 3, and what is now being called the Small Faro Deposit has been referred to as Zone 2.

a) Large Faro Deposit

The Large Faro Deposit has a major trend length of up to 4,000 feet at approximately 315° Azimuth, is up to 1,500 feet wide, 300 feet thick, and is covered by up to 500 feet of waste. The zinc and lead values are more continuous along the major trend direction and are more consistently above 9% (Pb + Zn) on the northwest portion than on the northeast or south portions of the deposit.

b) Small Faro Deposit

The Small Faro Deposit is located about 1,500 feet south of the southeast tip of the Large Faro Deposit. It is up to 1,000 feet long, 600 feet wide, 80 feet thick and is covered by up to 130 feet of waste. The deposit strikes at 270° Azimuth and dips approximately 20° to the south.

The sulfide mineralization occurs in a quartzitic horizon enclosed by a quartz muscovite schist. Unlike the Large Faro Deposit, this deposit contains little internal massive pyrite waste.

IV ESTIMATION METHOD

A review of previous tonnage and grade estimates, available literature and attempts with typical plans and sections were conducted to find the best estimation method to give locations, tonnages and grades to the lead and zinc values in the deposit models.

a) Large Faro Deposit

The diamond drill logs were reviewed and in some cases recopied and assay data entered. New longitudinal and traverse sections were prepared on which the drill holes, surface topography and intervals of ≥ 5 feet of $\geq 5\%$ (Pb + Zn) were plotted. New bench plans were prepared for the 40 foot benches. Drill hole intersections were plotted on each bench plan.

Using geologic experience gained during the present mining, an almost straight line interpretation was outlined on the sections. Points indicating -

1) the limit of a full 40 feet of $\geq 5\%$ (Pb + Zn) for each bench, and

2) the limit of ≥ 20 feet of $\geq 5\%$ (Pb + Zn) for each bench

were located on the sections and then transferred to the plans. These points were connected by smooth curves to form the outlines of -

1) full benches of $\geq 5\%$ (Pb + Zn), and

2) half benches or greater of $\geq 5\%$ (Pb + Zn).

A grid was prepared based on a unit block of 100 ($\sqrt{2}/2$) feet squared at 315^{o} Azimuth. Intersections of this grid coincide with the planned locations of the diamond drill holes. The planned locations were used to represent the real locations of the drill holes when outlining the areas of influence.

Areas of influence for individual sample locations were outlined in three steps as follows:

1) Areas were extended to a maximum 4 unit blocks north-west and south-east and 2 unit blocks north-east and south-west centered about the grid intersection.

2) The areas were then extended to a maximum of 8 unit blocks north-west and south east centered on the grid intersection.

3) Finally, areas were extended a maximum of 4 unit blocks north-east and south-west centered on the grid intersection.

These extensions were continued until all the area within the half bench outline had been allocated or a block had been extended half the distance to the next grid intersection representing a diamond drill hole location.

A tonnage factor of 3.18 tons per cubic yard was used.

Grades were applied to the tonnage blocks in the following three cases (less than 1% of the tonnage required geologic judgement to apply assumed reasonable grades):

1) If the diamond drill hole intersection is within the full bench outline, then the weighted average of the full 40 feet is applied.

2) If the diamond drill hole intersection is between the full bench outline and the half bench outline, then -

i) a weighted average of the full 40 feet is applied if the 1/2 bench is determined by internal waste; and

ii) a weighted average of a 40 foot intersection into the deposit is applied if the 1/2 bench is determined by external waste.

3) If the diamond drill intersection is outside the 1/2 bench outline, then a weighted average of a 40 foot intersection into the deposit from the contact is applied, provided that grades are not being projected more than 40 feet from the upper or lower extremities of the bench concerned.

Tables were prepared to add up the total tonnage and grade by bench and a summary table was prepared to add up the total estimated tonnage and grade.

b) Small Faro Deposit

Longitudinal and traverse sections and bench plans were plotted following the full bench, half bench method employed for the Large Faro Deposit. The following modifications were made to meet special requirements.

1) To reduce the effects of dilution, benches are 20 feet high.

2) Due to the significant deviation of actual drill hole locations from their planned positions, areas of influence run half way to the next hole from the actual locations. At the margins, areas of influence are extended half the drill hole spacing (100 feet).

Grades were assigned to tonnage blocks in the same three cases as for the Large Faro Deposit but using 20 foot weighted averages. When forming the averages assays >8.5% Zn and >6.5% Pb were disregarded as erratic highs at the 95% confidence level. Eleven percent of the tonnage required geologic judgement to apply reasonable grades.

Tables were prepared to add up the tonnages and grades by bench.

V LARGE FARO DEPOSIT

a) North

3910 to 3590 benches inclusive from 8,500 N to 11,400 N and 12,000 E to 16,000 E.

A new tonnage and grade estimate was prepared for this portion of the Large Faro Deposit to have adequate estimates for the five year mining plan. "Estimated Tonnage and Grade Tables, Large Faro Deposit, December 1973" (see Appendix I) and "Estimated Tonnage and Grade Bench Plans, Large Faro Deposit, December 1973" (see Appendix III) show the distribution and the totals. Final copies of the ink drafted bench plans will be available on request when completed.

b) South

3550 bench and below from 8,500 N to 11,400 N and 12,000 E to 16,000 E, also all benches from 7,000 N to 8,500 N and 12,000 E to 16,000 E.

A new tonnage and grade estimate was prepared for this portion in May 1973 and was revised in December 1973. "Estimated Tonnage and Grade Tables, Large Faro Deposit, December 1973" (see Appendix I) and "Estimated Tonnage and Grade Bench Plans, Large Faro Deposit, December 1973" (see Appendix III) show the distribution and the totals. Final copies of the ink drafted bench plans will be available on request when completed.

VI SMALL FARO DEPOSIT

A detailed tonnage and grade estimate was prepared for the total Small Faro Deposit in December 1973. "Estimated Tonnage and Grade Tables, Small Faro Deposit, December 1973" (see Appendix II) and "Estimated Tonnage and Grade Bench Plans, Small Faro Deposit, December 1973" (see Appendix IV) show the distribution and the totals. Final copies of ink drafted bench plans will be available on request when completed.

VII RELIABILITY OF DATA

a) Large Faro Deposit

While the ore model prepared for the Large Faro Deposit may be quite correct overall, experience has shown that some discrete assay data are incorrect and/or overweighted. For at least two years the short term use of this data has resulted in over-estimations of available Zn metal tons and under-estimations of available Pb metal tons. Present work on blasthole data indicates that this is due to:

1) a Pb zonation being missed in the DDH assay data due partly to Ba interference with data from one of the zones,

2) overweighting of erratic high Zn values; and

3) possible systematic errors in the original assays.

These problems are currently under review and some steps have already been implemented in the short term.

b) Small Faro Deposit

In general, pre-1973 assay data from the Small Faro Deposit were affected by poor drilling recoveries. The assays for 1967 and 1973 diamond drilling were compared. The two zinc distributions are unimodal and log normal with arithmetic means of 4.9% Zn and 4.8% Zn respectively. The two lead distributions are bimodal with identical arithmetic means of 3.1% Pb.

The lead to zinc ratio in the Small Faro Deposit is estimated at 0.67. This figure agrees with a ratio of 0.7 that has been experienced during the mining of the Large Faro Deposit.

APPENDIX I

ESTIMATED TONNAGE AND GRADE TABLES

LARGE FARO DEPOSIT

3910 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-4	94,222	5.3	8.8	14.1
65-5A	188,444	4.7	5.8	10.5
65-7	61,244	2.6	7.2	9.8
65-11	141,333	6.4	5.2	11.6
65-14	7,538	2.4	4.7	7.1
66-4	15,075	1.8	3.9	5.7
66-8	48,053	4.3	6.2	10.5
66-9	66,898	3.6	8.7	12.3
66-12	178,080	6.9	9.7	16.6
66-14	94,222	3.4	7.4	10.8
66-15	188,444	4.4	6.8	11.2
66-16	211,058	6.8	8.5	15.3
66-18	14,133	2.4	4.7	7.1
66-20	62,658	5.3	7.6	12.9
66-23	12,249	5.8	8.3	14.1
66-24	37,689	2.6	6.3	8.9
66-25	80,089	4.8	8.2	13.0
66-32	101,760	4.0	7.8	11.8
66-35	77,262	4.1	8.4	12.5
66-39	13,191	3.6	8.7	12.3
66-41	10,364	5.3	8.8	14.1
66-44	107,413	7.3	8.6	15.9
67-34	15,075	4.8	8.2	13.0
70-1	92,338	3.1	5.7	8.8
70-2	94,222	2.0	4.0	6.0
70-3	94,222	3.7	6.8	10.5
70-4	94,222	3.4	6.3	9.7
70-5	94,222	3.0	7.8	10.8
70-6	94,222	5.2	7.3	12.5
70-8	9,422	1.8	4.1	5.9
70-9	141,333	4.2	6.0	10.2
70-10	6,596	3.1	5.7	8.8
70-11	94,222	1.4	5.5	6.9
70-14	135,680	6.3	7.2	13.5
70-16	94,222	2.3	5.5	7.8
72-1	59,596	6.9	9.7	16.6
72-3	87,627	5.3	6.9	12.2
72-6	81,973	3.3	6.6	9.9
72-7	86,213	4.3	5.7	10.0
73-27	30,151	2.7	5.6	8.3
Total	3,216,977	4.6	7.0	11.6

LARGE FARO DEPOSIT

3870 Bench . 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-4	91,396	3.7	6.2	9.9
65-5A	188,444	6.0	6.7	12.7
65-7	136,622	2.5	6.1	8.6
65-11	165,831	6.7	7.4	14.1
66-2	11,778	3.1	4.0	7.1
66-4	118,720	1.8	3.9	5.7
66-8	151,698	4.3	6.2	10.5
66-9	31,564	2.8	6.0	8.8
66-12	146,987	3.1	5.5	8.6
66-14	66,898	2.6	3.7	6.3
66-15	188,444	5.5	6.4	11.9
66-22	103,644	2.8	3.3	6.1
66-23	93,280	5.8	8.3	14.1
66-30	5,653	2.5	6.1	8.6
66-32	135,680	3.6	3.9	7.5
66-35	63,129	1.8	5.5	7.3
66-39	45,698	2.8	3.9	6.7
66-44	130,969	8.0	10.7	18.7
66-53	8,951	2.5	6.1	8.6
70-1	95,164	2.9	6.6	9.5
70-2	94,222	2.0	3.1	5.1
70-3	73,964	4.6	9.2	13.8
70-4	94,222	2.2	4.1	6.3
70-5	94,222	3.3	6.5	9.8
70-6	221,422	4.8	7.9	12.7
70-8	16,018	2.3	3.9	6.2
70-9	141,333	2.5	5.1	7.6
70-11	65,404	2.0	6.2	8.2
70-14	141,333	5.2	8.6	13.8
70-16	94,222	2.7	6.5	9.2
72-1	65,019	2.7	5.4	8.1
72-3	152,640	5.2	6.9	12.1
72-6	221,422	2.7	6.7	9.4
72-7	163,947	6.2	6.3	12.5
73-27	29,679	2.2	5.7	7.9
73-29	2,827	2.0	3.7	5.7
Total	3,652,446	4.0	6.3	10.3

LARGE FARO DEPOSIT

3830 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-4	86,684	4.0	6.1	10.1
65-5A	188,444	4.9	5.7	10.6
65-6	27,795	2.1	5.2	7.3
65-7	102,702	1.8	5.7	7.5
65-8	15,076	6.3	9.2	15.5
65-11	142,040	7.3	8.9	16.2
66-2	98,462	2.5	3.1	5.6
66-4	100,818	2.3	4.6	6.9
66-8	188,444	5.6	7.0	12.6
66-9	64,307	2.7	5.7	8.4
66-12	44,520	3.1	5.6	8.7
66-14	86,920	6.4	10.8	17.2
66-15	188,444	4.0	5.4	9.4
66-22	180,907	3.5	5.4	8.9
66-23	164,418	6.4	6.8	13.2
66-30	79,146	2.4	5.9	8.3
66-32	62,187	2.4	5.0	7.4
66-35	40,515	1.9	6.5	8.4
66-39	23,084	2.3	3.9	6.2
66-44	56,769	5.3	8.7	14.0
70-1	60,067	3.0	6.9	9.9
70-2	84,800	1.8	3.6	5.4
70-3	82,444	3.2	7.9	11.1
70-5	94,222	4.3	5.9	10.2
70-6	117,778	4.0	6.2	10.2
70-9	120,133	1.5	4.2	5.7
70-11	45,227	2.5	4.4	6.9
70-14	141,333	4.9	7.9	12.8
70-16	94,222	6.8	12.4	19.2
72-1	29,209	2.7	5.4	8.1
72-3	225,426	5.1	5.9	11.0
72-6	217,653	2.4	6.5	8.9
72-7	199,751	3.3	3.7	7.0
73-29	27,324	2.5	5.1	7.6
Total	3,481,271	3.9	6.1	10.0

LARGE FARO DEPOSIT

3790 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-4	57,711	2.8	5.6	8.4
65-6	181,849	2.2	4.4	6.6
65-7	135,680	2.7	7.4	10.1
65-8	84,329	6.3	9.2	15.5
65-11	65,013	4.2	8.0	12.2
66-2	195,040	3.2	4.5	7.7
66-4	40,515	2.6	6.8	9.4
66-8	193,155	3.7	5.0	8.7
66-9	81,031	3.2	7.1	10.3
66-14	77,262	3.8	6.2	10.0
66-15	265,706	6.9	9.4	16.3
66-18	3,298	2.7	7.4	10.1
66-22	188,444	2.1	3.7	5.8
66-23	127,200	3.0	4.5	7.5
66-30	171,484	2.4	5.9	8.3
66-32	36,747	1.8	6.8	8.6
66-33	27,795	4.2	7.2	11.4
66-35	79,618	1.6	3.9	5.5
66-39	1,884	2.3	3.9	6.2
66-57	8,480	1.3	3.7	5.0
70-1	9,422	2.9	6.8	9.7
70-2	94,222	2.2	4.8	7.0
70-3	51,822	2.6	5.0	7.6
70-4	94,222	2.7	4.5	7.2
70-5	88,569	4.3	7.5	11.8
70-6	52,764	2.6	5.4	8.0
70-9	43,342	2.2	3.5	5.7
70-11	88,569	4.9	9.6	14.5
70-14	76,555	4.5	7.3	11.8
70-16	94,222	2.6	5.2	7.8
72-1	58,418	2.2	5.0	7.2
72-2	12,720	2.2	5.0	7.2
72-3	282,667	5.6	6.0	11.6
72-7	123,431	6.0	6.4	12.4
72-8	59,360	6.9	8.6	15.5
73-29	9,893	2.6	5.8	8.4
Total	3,262,439	3.7	6.1	9.8

LARGE FARO DEPOSIT

3750 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-4	2,827	2.6	5.3	7.9
65-6	291,147	2.1	3.0	5.1
65-7	62,187	2.4	6.2	8.6
65-8	138,507	6.8	10.1	16.9
66-2	257,698	2.3	3.9	6.2
66-4	73,493	2.3	5.4	7.7
66-8	275,129	3.1	4.4	7.5
66-9	23,556	3.4	7.7	11.1
66-14	40,515	3.6	6.5	10.1
66-22	114,591	2.9	5.9	8.8
66-23	55,591	4.9	6.3	11.2
66-30	175,724	2.4	6.2	8.6
66-31	23,555	5.9	8.4	14.3
66-33	131,911	4.2	7.2	11.4
66-35	148,871	1.2	3.8	5.0
66-57	21,671	3.1	8.0	11.1
70-2	47,111	2.5	6.5	9.0
70-4	116,836	3.0	4.0	7.0
70-9	131,911	4.8	7.8	12.6
70-11	92,338	3.5	6.2	9.7
70-14	27,325	5.4	7.3	12.7
70-16	44,284	2.4	5.2	7.6
72-1	27,324	2.2	5.0	7.2
72-2	44,284	3.9	6.3	10.2
72-3	262,880	5.4	7.3	12.7
72-7	114,951	2.2	5.2	7.4
72-8	109,298	6.7	8.5	15.2
73-1	13,191	2.4	6.2	8.6
73-3	2,827	4.2	6.1	10.3
73-29	4,711	2.7	5.9	8.6
Total	2,876,244	3.4	5.7	9.1

LARGE FARO DEPOSIT

3710 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-6	256,284	2.8	5.0	7.8
65-7	31,093	1.2	4.2	5.4
65-8	116,364	5.1	6.4	11.5
65-9	42,871	4.3	6.5	10.8
65-12	47,111	4.3	7.7	12.0
66-2	245,449	3.1	4.9	8.0
66-4	140,391	5.4	10.0	15.4
66-8	244,035	3.5	5.4	8.9
66-14	2,355	6.5	10.8	17.3
66-22A	203,991	2.5	4.8	7.3
66-23	2,591	5.1	6.4	11.5
66-26	185,382	4.3	7.7	12.0
66-29	96,107	4.7	6.6	11.3
66-30	149,813	1.2	4.2	5.4
66-31	149,342	5.8	8.4	14.2
66-33	239,324	4.2	7.2	11.4
66-35	36,747	1.2	3.8	5.0
66-54	22,142	2.6	5.3	7.9
66-57	14,133	2.7	7.2	9.9
70-4	28,502	3.2	4.6	7.8
70-9	141,333	2.8	4.4	7.2
70-11	74,907	6.5	10.8	17.3
72-2	167,715	3.9	6.3	10.2
72-3	204,226	5.9	7.5	13.4
72-7	219,302	1.8	3.7	5.5
72-8	117,777	5.0	6.0	11.0
73-1	10,364	1.2	4.2	5.4
73-2	1,884	4.2	7.2	11.4
73-3	76,320	6.4	8.4	14.8
Total	3,267,855	3.8	6.1	9.9

LARGE FARO DEPOSIT

3670 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-6	305,751	3.6	7.8	11.4
65-8	9,893	4.9	6.3	11.2
65-9	154,053	4.3	6.5	10.8
65-12	118,720	2.6	4.8	7.4
66-2	265,706	2.7	4.3	7.0
66-4	115,187	3.2	5.9	9.1
66-8	157,587	4.9	7.1	12.0
66-22A	195,747	2.5	4.8	7.3
66-26	27,324	2.4	4.1	6.5
66-30	125,315	2.5	5.8	8.3
66-31	212,000	6.4	8.4	14.8
66-33	186,089	4.0	5.1	9.1
70-9	99,875	3.3	4.9	8.2
70-11	28,738	4.5	7.3	11.8
72-2	87,627	4.3	6.2	10.5
72-3	82,208	5.9	7.5	13.4
72-7	282,666	1.6	4.2	5.8
72-8	70,195	4.4	5.9	10.3
73-1	19,787	2.5	5.8	8.3
73-2	15,076	3.9	5.1	9.0
73-3	188,444	6.8	9.1	15.9
73-28	108,355	3.9	6.5	10.4
Total	2,856,343	3.8	6.1	9.9

LARGE FARO DEPOSIT

3630 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-6	371,235	3.1	6.5	9.6
65-9	169,128	5.5	6.7	12.2
66-2	266,177	2.9	6.8	9.7
66-4	70,196	3.3	6.0	9.3
66-6	209,173	1.8	3.6	5.4
66-7	7,538	3.5	4.8	8.3
66-8	49,702	4.4	6.8	11.2
66-11	29,444	1.8	3.6	5.4
66-30	77,262	2.0	5.3	7.3
66-31	154,053	5.5	6.7	12.2
66-33	159,235	2.2	4.0	6.2
66-46	142,275	3.5	4.8	8.3
66-55	37,218	4.3	7.1	11.4
72-7	283,138	2.4	5.1	7.5
73-1	55,591	2.0	5.3	7.3
73-2	20,257	3.9	5.7	9.6
73-3	17,902	5.5	7.2	12.7
73-28	79,146	3.4	6.1	9.5
Total	2,198,670	3.2	5.7	8.9

LARGE FARO DEPOSIT

3590 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-6	212,000	2.8	5.9	8.7
65-9	42,871	3.3	4.0	7.3
66-2	66,897	2.6	6.8	9.4
66-3	26,382	2.1	3.7	5.8
66-6	311,875	3.9	8.6	12.5
66-7	84,800	3.4	6.5	9.9
66-11	274,186	2.6	5.1	7.7
66-30	35,569	1.7	4.1	5.8
66-31	5,653	5.5	6.7	12.2
66-33	178,080	2.1	3.7	5.8
66-46	210,115	2.6	3.5	6.1
66-52	17,902	3.3	4.0	7.3
72-16	13,191	2.6	5.1	7.7
66-47	25,440	3.4	6.5	9.9
72-7	24,498	3.5	6.5	10.0
73-2	28,267	4.9	6.2	11.1
73-28	9,893	3.4	6.1	9.5
Total	1,567,619	3.0	5.6	8.6

LARGE FARO DEPOSIT

3870 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	% (Pb + Zn)
FRH-16	35,568	2.8	4.3	7.1
66-E1	1,884	3.5	6.8	10.3
66-E4	27,795	4.5	6.4	10.9
66-E7	15,075	2.8	4.3	7.1
Total	80,322	3.4	5.1	8.5

LARGE FARO DEPOSIT

May, 1973

3830 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
FRH-16	2,355	2.8	4.3	7.1
66-E1	5,182	3.5	6.8	10.3
66-E4	112,831	3.2	6.8	10.0
66-E7	45,226	2.8	4.3	7.1
Total	165,594	3.1	6.1	9.2

LARGE FARO DEPOSIT

3790 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	% (Pb + Zn)
66-E1	102,466	3.5	6.8	10.3
66-E4	165,124	1.6	3.5	5.1
66-E5	25,204	1.7	3.5	5.2
66-E6	40,515	2.1	4.1	6 2
Total	333,309	2.2	4.6	6.8

LARGE FARO DEPOSIT

May, 1973

3750 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
66-E1	84,328	3.5	6.8	10.3
66-E4	77,026	1.6	3.5	5.1
66-E5	67,839	1.7	3.6	5.3
66-E6	146,044	2.3	4.1	6.4
Total	375,237	2.3	4.5	6.8

LARGE FARO DEPOSIT

3710 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
66-E5	65,484	2.6	4.6	7.2
66-E6	154,995	1.7	5.2	6.9
66-E9	18,373	2.0	3.0	5.0
Total	238,852	2.0	4.9	6.9

LARGE FARO DEPOSIT

3670 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-E6	180,435	1.7	5.2	6.9
66-E9	52,057	1.8	3.2	5.0
67-5	7,537	2.4	5.9	8.3
Total	240,029	1.7	4.8	6.5

LARGE FARO DEPOSIT

3630 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	% (Pb + Zn)
66-E6	133,324	1.7	5.2	6.9
66-E9	73,257	1.8	3.2	5.0
67-4	1,884	2.3	5.7	8.0
67-5	119,191	2.3	5.7	8.0
67-7	28,502	2.4	5.0	7.4
Total	356,158	2.0	4.9	6.9

LARGE FARO DEPOSIT

3590 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

May, 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-E9	25,440	1.9	5.2	7.1
67-5	146,044	3.0	6.5	9.5
67-7	80,324	2.9	5.6	8.5
Total	251,808	2.9	6.1	9.0

LARGE FARO DEPOSIT

3550 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-6	91,395	1.8	3.9	5.7
65-9	56,533	3.8	5.2	9.0
66-E9	57,004	1.9	5.2	7.1
66-2	34,862	2.6	6.8	9.4
66-3	55,591	4.3	7.6	11.9
66-5	7,066	2.1	3.7	5.8
66-6	216,711	3.1	7.7	10.8
66-7	231,079	3.4	6.5	9.9
66-11	93,280	3.1	7.7	10.8
66-46	146,044	1.8	3.9	5.7
66-52	59,360	5.4	7.1	12.5
66-55	44,284	3.0	4.4	7.4
67-3	89,039	2.2	4.0	6.2
67-5	172,897	2.1	5.2	7.3
67-7	158,528	3.0	4.8	7.8
67-30	7,537	3.0	4.8	7.8
72-16	270,653	2.4	4.8	7.2
73-2	148,871	2.6	6.8	9.4
Total	1,940,734	2.8	5.7	8.5

LARGE FARO DEPOSIT

3510 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-6	15,075	2.3	4.8	7.1
66-E9	40,515	3.6	7.6	11.2
66-3	307,635	4.2	7.6	11.8
66-5	77,733	2.3	3.4	5.7
66-7	221,186	4.2	9.5	13.7
66-10	156,644	2.5	2.6	5.1
66-46	138,035	2.3	4.8	7.1
66-49	7,537	5.4	6.7	12.1
66-52	161,591	5.4	7.1	12.5
67-3	115,186	2.1	3.5	5.6
67-4	67,604	3.1	5.1	8.2
67-5	154,524	1.9	4.9	6.8
67-6	229,195	1.9	4.2	6.1
67-7	81,031	3.0	4.8	7.8
67-8	52,528	3.0	6.3	9.3
67-10	180,906	2.1	3.3	5.4
67-11	311,168	3.0	7.0	10.0
67-12	80,559	2.4	3.7	6.1
67-30	208,466	3.0	4.8	7.8
72-15	60,537	1.7	4.6	6.3
72-16	112,359	2.1	5.4	7.5
73-2	129,556	4.2	7.6	11.8
Total	2,909,570	3.0	5.7	8.7

LARGE FARO DEPOSIT

3470 Bench — December 1973

Drill Hole	Tons	% Pb	% Zn	%(Pb + Zn)
66-3	77,969	2.1	3.7	5.8
66-5	136,857	2.3	3.4	5.7
66-7	70,902	1.5	3.6	5.1
66-10	7,537	1.5	3.6	5.1
66-46	98,933	2.9	6.6	9.5
66-49	357,102	5.4	6.7	12.1
66-52	299,626	6.1	7.1	13.2
67-3	231,079	3.2	5.4	8.6
67-4	220,244	1.8	4.6	6.4
67-5	97,284	1.9	4.9	6.8
67-7	179,257	2.7	5.2	7.9
67-8	89,039	2.8	6.4	9.2
67-10	167,479	3.1	7.2	10.3
67-11	60,537	3.0	7.0	10.0
67-12	284,551	2.0	4.3	6.3
67-30	728,573	3.7	4.7	8.4
70-13	52,528	3.1	5.2	8.3
70-17	288,319	3.6	5.2	8.8
71-4	123,195	2.5	4.1	6.6
71-5	241,679	3.1	5.2	8.3
Total	3,812,690	3.4	5.3	8.7

LARGE FARO DEPOSIT

3430 Bench

December 1973

Drill Hole	Tons	% Pb	% Zn	%(Pb + Zn)
66-3	318,000	2.5	3.1	5.6
66-5	13,426	2.1	3.7	5.8
66-7	85,977	1.5	3.6	5.1
66-10A	146,515	1.9	3.3	5.2
66-46	127,200	3.4	5.9	9.3
66-49	376,888	2.3	3.3	5.6
66-52	229,902	5.4	7.1	12.5
67-3	246,626	2.7	5.9	8.6
67-4	54,884	5.3	10.1	15.4
67-6	207,759	3.2	4.6	7.8
67-7	5,182	2.7	5.9	8.6
67-10	94,928	3.2	7.5	10.7
67-11	376,888	2.7	6.3	9.0
67-12	335,902	3.5	5.9	9.4
67-30	393,613	3.6	5.1	8.7
70-12	180,435	5.7	8.1	13.8
70-17	359,457	3.6	5.2	8.8
71-2	16,959	2.2	4.2	6.4
71-3	16,959	5.7	8.1	13.8
71-4	298,213	2.5	4.1	6.6
71-5	161,591	3.1	5.2	8.3
72-13	33,448	6.0	7.6	13.6
Total	4,080,752	3.2	5.2	8.4

LARGE FARO DEPOSIT

3390 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-3	349,564	2.3	4.3	6.6
66-7	5,653	3.6	6.6	10.2
66-10	508,564	2.6	4.9	7.5
66-46	39,573	3.6	6.6	10.2
66-49	231,786	2.9	4.6	7.5
66-52	8,008	4.1	5.8	9.9
67-3	3,297	2.9	6.4	9.3
67-6	249,453	1.9	5.0	6.9
67-10	35,804	1.9	5.0	6.9
67-11	220,479	2.8	5.7	8.5
67-12	292,324	3.3	5.0	8.3
70-12	55,355	5.7	8.1	13.8
71-1	7,302	5.7	8.1	13.8
71-3	9,893	5.7	8.1	13.8
72-13	102,702	6.0	7.6	13.6
Total	2,119,757	2.9	5.2	8.1

LARGE FARO DEPOSIT

3350 Bench

May, 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
66-3	14,133	2.1	5.3	7.4
67-6	42,635	2.6	6.0	8.6
72-13	214,826	4.5	4.5	9.0
Total	271,594	4.1	4.8	8.9

SUMMARY OF ESTIMATED [illegible]

LARGE FARO DEPOSIT

NORTH END: BENCHES 3910 to 3590 inc.

Bench Summary — 8,500 N to 11,400 N; 12,000 E to 16,000 E — December 1973

Bench Number	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
3910	3,216,977	4.6	7.0	11.6
3870	3,652,446	4.0	6.3	10.3
3830	3,481,271	3.9	6.1	10.0
3790	3,262,439	3.7	6.1	9.8
3750	2,876,244	3.4	5.7	9.1
3710	3,267,855	3.8	6.1	9.9
3670	2,856,343	3.8	6.1	9.9
3630	2,198,670	3.2	5.7	8.9
3590	1,567,619	3.0	5.6	8.6
Total	26,379,864	3.8	6.1	9.9

SOUTH END: BENCHES 3870 to 3350 inc.

Bench Summary — 7,000 N to 8,500 N; 12,000 E to 16,000 E — December 1973

Bench Number	Tons	≥ 1	Tot.	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
		3,216,977	3,216,977			
3870	80,322	3,652,446	3,732,768	3.4	5.1	8.5
3830	165,594	3,481,271	3,646,865	3.1	6.1	9.2
3790	333,309	3,262,439	3,595,748	2.2	4.6	6.8
3750	375,237	2,876,244	3,251,481	2.3	4.5	6.8
3710	238,852	3,267,855	3,506,707	2.0	4.9	6.9
3670	240,029	2,856,343	3,096,372	1.7	4.8	6.5
3630	356,158	2,198,670	2,554,828	2.0	4.9	6.9
3590	251,808	1,567,619	1,819,427	2.9	6.1	9.0
3550	1,940,734		1,940,734	2.8	5.7	8.5
3510	2,909,570		2,909,570	3.0	5.7	8.7
3470	3,812,690		3,812,690	3.4	5.3	8.7
3430	4,080,752		4,080,752	3.2	5.2	8.4
3390	2,119,757		2,119,757	2.9	5.2	8.1
3350	271,594		271,594	4.1	4.8	8.9
Total	17,176,406	26,379,864	43,556,270	3.0	5.3	8.3

APPENDIX II

ESTIMATED TONNAGE AND GRADE TABLES

SMALL FARO DEPOSIT

3930 Bench

December 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
67-33	24,969	2.2	5.7	7.9
73-20	5,653	4.3	6.7	11.0
Total	30,622	2.6	5.9	8.5

SMALL FARO DEPOSIT

3910 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-22	10,364	2.0	4.0	6.0
67-26	471	5.1	5.9	11.0
67-33	33,566	2.0	4.5	6.5
73-4	942	4.4	6.5	10.9
73-20	32,035	4.3	6.1	10.4
Total	77,378	3.0	5.1	8.1

SMALL FARO DEPOSIT

3890 Bench

December 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
67-22	35,804	1.7	3.8	5.5
67-26	51,351	5.1	5.9	11.0
67-33	11,306	3.7	5.3	9.0
73-4	28,267	4.4	6.5	10.9
73-18	12,955	2.9	3.0	5.9
73-19	44,755	3.8	4.3	8.1
73-20	9,422	4.3	6.7	11.0
73-21	1,884	4.4	6.5	10.9
Total	195,744	3.8	5.0	8.8

SMALL FARO DEPOSIT

3870 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-13	26,853	4.1	6.7	10.8
67-18	15,546	5.5	1.1	6.6
67-22	18,844	1.7	3.8	5.5
67-26	55,591	4.5	5.7	10.2
73-4	66,427	3.9	6.0	9.9
73-6	1,413	4.5	5.7	10.2
73-18	22,142	2.4	3.7	6.1
73-19	17.902	2.7	5.1	7.8
73-21	4,240	3.9	6.0	9.9
73-25	11,306	5.5	1.1	6.6
Total	240,264	3.8	5.0	8.8

SMALL FARO DEPOSIT

3850 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-13	72,904	4.1	6.3	10.4
67-16	19,315	3.2	5.9	9.1
67-18	28,267	3.8	1.2	5.0
67-20	10,835	2.4	6.3	8.7
67-22	8,479	2.2	3.2	5.4
67-26	36,275	4.4	5.0	9.4
67-28	25,911	3.8	6.7	10.5
73-4	61,715	2.0	3.7	5.7
73-5	1,884	2.0	3.7	5.7
73-6	22,142	4.0	4.0	8.0
73-7	52,764	3.1	5.7	8.8
73-18	38,631	2.2	3.2	5.4
73-19	471	3.1	5.2	8.3
73-21	4,711	2.0	3.7	5.7
73-25	16,959	3.8	1.2	5.0
Total	401,263	3.3	4.6	7.9

SMALL FARO DEPOSIT

3830 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-13	10,364	5.2	7.5	12.7
67-16	62,656	3.2	6.0	9.2
67-17	3,298	2.7	5.0	7.7
67-21	31,564	4.1	6.2	10.3
67-26	6,595	4.4	5.0	9.4
67-28	48,524	1.6	4.1	5.7
73-4	36,747	2.1	4.1	6.2
73-5	471	1.6	4.1	5.7
73-6	56,062	4.8	5.0	9.8
73-7	54,648	3.1	5.7	8.8
73-8	471	4.1	6.2	10.3
73-9	4,711	2.5	5.7	8.2
73-18	50,409	2.3	3.7	6.0
73-21	2,356	2.1	4.1	6.2
Total	368,876	3.1	5.1	8.2

SMALL FARO DEPOSIT

December 1973

3810 Bench

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-13	1,884	3.2	4.2	7.4
67-16	49,938	3.3	5.8	9.1
67-17	25,911	2.7	5.0	7.7
67-18	54,649	3.4	3.6	7.0
67-21	51,822	3.9	5.1	9.0
67-27	57,004	5.4	5.6	11.0
67-28	36,275	2.1	3.8	5.9
67-31	14,604	2.4	6.5	8.9
73-6	58,418	3.2	4.2	7.4
73-7	19,787	3.9	5.1	9.0
73-8	1,884	3.9	5.1	9.0
73-9	38,160	5.5	7.2	12.7
73-10	35,804	3.1	5.6	8.7
73-11	2,355	3.7	3.8	7.5
73-12	8,951	5.4	5.6	11.0
73-18	10,364	2.6	4.1	6.7
73-24	52,764	4.7	7.2	11.9
73-26	32,507	2.4	4.8	7.2
Total	553,081	3.7	5.2	8.9

SMALL FARO DEPOSIT

3790 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-16	5,653	3.3	5.8	9.1
67-17	38,159	3.2	5.5	8.7
67-21	64,542	2.4	5.6	8.0
67-25	48,995	3.5	6.3	9.8
67-27	64,071	3.3	4.1	7.4
67-28	16,489	2.6	4.3	6.9
67-29	52,293	2.3	4.0	6.3
67-31	43,342	3.2	7.0	10.2
73-6	11,307	4.0	4.4	8.4
73-8	706	2.4	5.6	8.0
73-9	88,097	4.2	4.7	8.9
73-10	37,218	4.4	6.4	10.8
73-11	11,307	2.3	3.8	6.1
73-12	1,413	3.0	4.0	7.0
73-16	14,133	3.9	6.8	10.7
73-17	10,835	4.3	6.7	11.0
73-23	9,422	3.2	7.0	10.2
73-24	13,191	4.4	7.4	11.8
Total	531,173	3.3	5.3	8.6

SMALL FARO DEPOSIT

3770 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-15	20,258	4.5	6.2	10.7
67-17	39,573	3.0	5.0	8.0
67-21	48,053	1.6	3.4	5.0
67-25	62,656	3.5	6.3	9.8
67-27	44,284	3.7	4.6	8.3
67-28	2,356	2.1	3.8	5.9
67-29	2,356	2.3	4.0	6.3
67-31	65,955	2.2	5.3	7.5
73-8	20,729	3.0	5.0	8.0
73-9	65,013	4.4	4.8	9.2
73-11	53,235	2.7	4.8	7.5
73-12	34,862	2.0	3.9	5.9
73-13	6,595	2.2	2.9	5.1
73-16	41,458	3.9	6.8	10.7
73-17	35,804	4.3	6.7	11.0
73-22	1,413	2.2	5.3	7.5
73-23	2,356	2.2	5.3	7.5
73-24	38,160	2.8	5.0	7.8
Total	585,115	3.1	5.1	8.2

SMALL FARO DEPOSIT

3750 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-15	7,067	4.5	6.2	10.7
67-17	14,604	2.0	4.8	6.8
67-21	1,413	2.2	5.1	7.3
67-25	23,084	3.5	6.3	9.8
67-27	707	3.7	4.6	8.3
67-31	32,978	2.6	5.9	8.5
73-8	15,547	2.7	5.6	8.3
73-11	4,711	2.7	4.8	7.5
73-12	77,262	3.7	4.1	7.8
73-13	48,995	2.4	3.2	5.6
73-15	1,884	3.7	4.1	7.8
73-16	66,898	4.5	6.1	10.6
73-17	36,747	4.0	6.4	10.4
73-22	12,248	2.6	5.9	8.5
Total	344,145	3.4	5.1	8.5

SMALL FARO DEPOSIT

3730 Bench — December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-21	707	4.0	4.9	8.9
67-23	21,671	3.5	6.3	9.8
73-8	16,018	3.8	6.8	10.6
73-12	13,191	4.3	4.9	9.2
73-13	1,413	4.3	4.9	9.2
73-16	36,275	4.0	4.9	8.9
Total	89,275	3.9	5.6	9.5

SMALL FARO DEPOSIT

3710 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-23	7,067	3.5	3.6	7.1
Total	7,067	3.5	3.6	7.1

SUMMARY OF ESTIMATED TONNAGE AND GRADE

SMALL FARO DEPOSIT

BENCHES 3930 TO 3710 INC.

December 1973

Bench Number	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
3930	30,622	2.6	5.9	8.5
3910	77,378	3.0	5.1	8.1
3890	195,744	3.8	5.0	8.8
3870	240,264	3.8	5.0	8.8
3850	401,263	3.3	4.6	7.9
3830	368,876	3.1	5.1	8.2
3810	553,081	3.7	5.2	8.9
3790	531,173	3.3	5.3	8.6
3770	585,115	3.1	5.1	8.2
3750	344,145	3.4	5.1	8.5
3730	89,275	3.9	5.6	9.5
3710	7,067	3.5	3.6	7.1
Total	3,424,003	3.4	5.1	8.5

ESTIMATED TONNAGE

AND GRADE TOTALS

December 1973

Deposit	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
Large Faro North	26,379,864	3.8	6.1	9.9
Large Faro South	17,176,406	3.0	5.3	8.3
Small Faro	3,424,003	3.4	5.1	8.5
Total	46,980,273	3.5	5.7	9.2

Geol. Estimates
Comparison

ANVIL MINING CORPORATION LIMITED

MEMORANDUM

TO: D. W. Philip FROM: D. Hanson

SUBJECT: COMPARISON OF JUNE 1973 AND DECEMBER 1973 TONNAGE & GRADE ESTIMATES DATE: February 12, 1974

1) Large Faro Deposit:

In 1973 six diamond drill holes were drilled through the sulfide intersection(s) to define the east and west pit walls and to confirm the presence of a dyke system in the vicinity of section 117. Table I is a summary of the net changes per bench.

TABLE I: Comparison of June and December 1973 Tonnage and Grade Estimates - Large Faro Deposit

	Bench	June 1973 Tonnage	June 1973 Pb + Zn	December 1973 Tonnage	December 1973 Pb + Zn	Net Change Tonnage
North	3910	3,243,831	11.6	3,216,977	11.6	- 26,854
	3870	3,640,275	10.3	3,652,446	10.3	12,171
	3830	3,511,661	10.0	3,481,271	10.0	- 30,390
	3790	3,188,473	9.8	3,262,439	9.8	73,966
	3750	3,057,035	9.3	2,876,244	9.1	- 180,791
	3710	3,442,167	9.8	3,267,855	9.9	- 174,312
	3670	2,888,849	9.3	2,856,343	9.9	- 32,506
	3630	2,647,171	8.8	2,198,670	8.9	- 448,501
	3590	2,130,362	8.6	1,567,619	8.6	- 562,743
Total		27,749,824	9.8	26,379,864	9.9	-1,369,960
South*	3550	2,263,443	8.4	1,940,734	8.5	- 322,709
	3510	3,114,266	8.7	2,909,570	8.7	- 204,696
	3470	3,925,992	8.7	3,812,690	8.7	- 113,302
	3430	4,217,373	8.4	4,080,752	8.4	- 136,621
	3390	2,133,419	8.1	2,119,757	8.1	- 13,662
Total		15,654,493	8.4	14,863,503	8.5	- 790,990
Total						-2,159,950

2,190,000 lost from 3750 - 3790 Dike

* Benches 3870-3590 inc. and bench 3350 are unchanged.

The interpretation of the diorite dyke(s?) in plan and cross-section was based upon three diamond drill holes with 600 foot spacing and on the continuity of the dyke in pit outcrop. Most of the net loss in estimated tonnage is attributed to the presence of this dyke system.

2) Small Faro Deposit:

In 1973 twenty-three diamond drill holes were drilled through the sulfide intersection(s) to complete the 200 foot center holes on a N-S, E-W grid. After evaluation of this additional sampling, a revised tonnage and grade estimate was conducted for 20 foot benches. Table II compares this estimate with the June 1973 estimate for 40 foot benches on a 315° Azimuth grid.

TABLE II: Comparison of June and December 1973 Tonnage and Grade Estimates - Small Faro Deposit

	June 1973 (40' Benches)			December 1973 (20' Benches)			Net Change
	Tonnage	Pb	Zn	Tonnage	Pb	Zn	Tonnage
Total	2,928,188	2.8	4.7	3,424,003	3.4	5.1	+495,815

D. Hanson

D. Hanson
Mine Geologist

DH/mm

J. C. Devitt

D. J. Hanson

Massive Pyrite Occurrence in Zone III

September 30, 1974

A comparison of the 1970 and June 1973 Geologic Tonnage and Grade Estimates for Zone III shows a net loss of 5,174,963 tons of ore with a concurrent upgrading from 8.16% Pb + Zn to 8.30% Pb + Zn.

These tonnage and grade changes can be attributed to the following:

1. New data acquired through 1972 diamond drilling.
2. Editing cross-sections with respect to massive sulfide intersections of < 5% Pb + Zn.
3. Re-interpretation of cross-sections based on (1) and (2).

In a memo to R. L. Haffner (1972 Development Diamond Drilling, a Preliminary Summary; August 18, 1972), P. M. Pettigrew estimated that the 1972 drilling added 225,000 tons to the geologic reserve of Zone III. It is estimated, therefore, that the 1970 Tonnage and Grade Estimate for Zone III included 5 - 6 million tons of massive sulfide with <5% Pb + Zn (i.e., massive pyrite).

It is recommended that the occurrence of massive pyrite be delineated on bench plans so that a detailed tonnage and grade estimate can be conducted for this material.

DJH:mb

D. J. Hanson
Mine Geologist

Reserves
1/1/74

ANVIL MINING CORPORATION LIMITED

MEMORANDUM

TO: D.W. Philip FROM: D.J. Hanson

SUBJECT: TONNAGE AND GRADE ESTIMATES DATE: January 24, 1974

FARO DEPOSITS AS OF JAN. 1, 1974

This is a compilation of remaining geologic reserve and stockpiled material for the Faro Deposits as of January 1, 1974.

	Tonnage	Pb	Zn	Pb+Zn
1) Large Faro (North)	19,533,908	3.5	5.9	9.4
2) Large Faro (South)	17,176,406	3.0	5.3	8.3
3) Stockpiles -				
L.G. Yellow	1,994,713	2.7	4.5	7.2
L.G. Red	515,713	3.4	5.8	9.2
H.G.S.P.	86,408	4.7	6.3	11.0
4) Small Faro	3,424,003	3.4	5.1	8.5
5) TOTAL	42,731,151	3.3	5.5	8.8

D.J. Hanson

D.J. Hanson

Reserves
1/1/75

Cyprus Anvil Mining Corporation — Interoffice Correspondence

To: R. H. Marshall

Date: January 16, 1975

From: J.C. Devitt

Subject: ORE RESERVES

Ore reserves remaining at January 1, 1975, are calculated below by subtracting mill reported feed by years from the initial accepted ore reserves submitted by E.N. Pennebaker on October 3, 1967 for zones # 1, 2, and 3.

		Tons	Pb%	Zn%	Pb% + Zn%
Initial Reserves		63,473,000	3.4	5.7	9.1
Milled	1969	435,000	3.5	5.6	9.1
Milled	1970	1,961,000	4.4	6.4	10.8
Milled	1971	2,673,000	4.9	6.8	11.7
Milled	1972	2,905,530	4.6	6.2	10.8
Milled	1973	2,899,145	4.9	6.4	11.3
Milled	1974	2,925,359	4.5	5.6	10.1
Total Milled		13,799,034	4.6	6.2	10.8
Reserve Remaining 1-1-75		49,673,966	3.1	5.6	8.7

CYPRUS

Stockpile inventories as of 1-1-75 obtained from pit production truck count figures in and out of stockpile are as follows:

Stockpile	Tons	Pb%	Zn%	Pb% + Zn%
Low Grade Yellow Pb + Zn = 5% to 8.5%	1,776,492	2.7	4.5	7.2
Low Grade Red Pb + Zn = +8.5% Zn = 5.9%	490,518	3.4	5.8	9.2
High Grade Oxidized Pb + Zn = +8.5% Zn = +5.9%	60,000	4.7	6.3	11.0
Crusher Feed Stockpile	30,000	3.5	4.3	7.8
Wet Stockpile	70,000	4.9	6.4	11.3
TOTAL	2,427,010	3.0	4.9	7.9

Grades for the Low Grade Yellow and Red were derived from rotary test hole cuttings. Other grades are from blast hole averages.

J. C. Devitt
Chief Engineer

JCD/ld

cc. J. F. Olk
B. Cook
N. Cornish
D. G. Bohach
D. Hanson

Geol. Reserve
Est., Apr./75

Cyprus Anvil Mining Corporation — Interoffice Correspondence

To: J. C. Devitt

Date: April 17, 1975

From: D. J. Hanson

Subject: PROGRESS REPORT - TONNAGE AND GRADE ESTIMATES, FARO DEPOSITS - APRIL 1975

The 1974 development diamond drilling is now fully assessed with all information incorporated into working sections and plans. Below is a summary of remaining geologic reserve and stockpiled material as of January 1, 1975.

	Tonnage	Pb	Zn	Pb+Zn
1) Large Faro (North)	16,144,376	3.6	6.1	9.7
2) Large Faro (South)	19,324,575	3.1	5.3	8.4
3) Small Faro	3,424,003	3.4	5.1	8.5
4) Stockpiles				
L.G. Yellow	1,776,492	2.7	4.5	7.2
L.G. Red	490,518	3.4	5.8	9.2
C.F.S.P.	100,000	4.5	5.8	10.3
Total	41,259,964	3.3	5.6	8.9

Typewritten tables are being prepared at present. Inked bench plans will be prepared as drafting time permits.

D. J. Hanson
Mine Geologist

DJH/mm

CYPRUS

PROGRESS REPORT

TONNAGE AND GRADE ESTIMATES

FARO DEPOSITS

APRIL, 1975

CYPRUS ANVIL MINING CORPORATION

Whitehorse M.D., Yukon Territory

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VI Reliability of Data

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I Estimated Tonnage and Grade Tables, Large Faro Deposit, April 1975

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III Estimated Tonnage and Grade Bench Plans, Large Faro Deposit, April 1975
(on file in Engineering Department)

IV Estimated Tonnage and Grade Bench Plans, Small Faro Deposit, December 1973
(on file in Engineering Department)

I INTRODUCTION

In 1973, extensive revisions were made to the tonnage and grade models for the Faro Deposits. New assay cross-sections were established, a new interpretive model was drawn, and new bench plans were prepared.

In April 1975, the Large Faro model was updated to include the results of fourteen 1974 development diamond drill holes.

II SUMMARY

1) This estimate is a compilation of tonnage and grades of $\geq$ 5% (Pb + Zn) material for $\geq$ 5 foot assay intervals.

2) In the Large Faro Deposit (North) on the 3910-3590 benches inclusive from 8,500 N to 11,400 N and 12,000 E to 16,000 E, the total of mined and geologic reserve material is estimated to be 25,029,311 tons of 3.9% Pb and 6.2% Zn.

3) In the Large Faro Deposit (South) on benches 3550 and below from 8,500 N to 11,400 N and 12,000 E to 16,000 E and also all benches 7,000 N to 8,500 N and 12,000 E to 16,000 E, the total geologic reserve material is estimated to be 19,324,575 tons of 3.1% Pb and 5.3% Zn.

4) The tonnage and grade estimate for the Small Faro Deposit remained unchanged from December 1973 at 3,424,003 tons of 3.4% Pb and 5.1% Zn.

III ESTIMATION METHOD

The estimation method remained unchanged from 1973.

IV LARGE FARO DEPOSIT

a) North

3910-3590 benches inclusive from 8,500 N to 11,400 N and 12,000 E to 16,000 E.

An updated tonnage and grade estimate was prepared for this portion of the Large Faro Deposit to have adequate estimates for the five year mining plan. "Estimated Tonnage and Grade Tables, Large Faro Deposit, April 1975" (see Appendix I) and "Estimated Tonnage and Grade Bench Plans, Large Faro Deposit, April 1975" (see Appendix III) show the distribution and the totals. Final copies of the ink drafted bench plans are available in the Engineering Office.

b) South

3550 bench and below from 8,500 N to 11,400 N and 12,000 E to 16,000 E, also all benches from 7,000 N to 8,500 N and 12,000 E to 16,000 E.

An updated tonnage and grade estimate was prepared for this portion in April 1975. "Estimated Tonnage and Grade Tables, Large Faro Deposit, April 1975" (see Appendix I) and "Estimated Tonnage and Grade Bench Plans, Large Faro Deposit, April 1975" (see Appendix III) show the distribution and the totals. Final copies of the ink drafted bench plans are available in the Engineering Office.

V SMALL FARO DEPOSIT

The 1973 tonnage and grade estimate for the Small Faro Deposit remained unchanged in 1975. "Estimated Tonnage and Grade Tables, Small Faro Deposit, December 1973" (see Appendix II) and "Estimated Tonnage and Grade Bench Plans, Small Faro Deposit, December 1973" (see Appendix IV) show the distribution and the totals. Final copies of the ink drafted bench plans are available in the Engineering Office.

VI RELIABILITY OF DATA

For the past three years, the use of this model for the Large Faro Deposit has resulted in the over-estimation of available Zn metal tons and the under-estimation of available Pb metal tons. Present work on blasthole data and on new diamond drill hole data indicates that this is due to:

1) a vertical Pb zonation being missed in the DDH assay data due partly to Ba interference in one of the mineralogical sub-zones.

2) possible systematic errors in original assays.

3) overweighting of erratic high Zn values.

Short term corrections are currently being applied to the model for grade prediction purposes.

APPENDIX I

ESTIMATED TONNAGE AND GRADE TABLES

LARGE FARO DEPOSIT

APRIL 1975

SUMMARY OF ESTIMATED TONNAGE AND GRADE

LARGE FARO DEPOSIT

NORTH END: BENCHES 3910 to 3590 inc.

Bench Summary 8,500 N to 11,400 N — April 1975
12,000 E to 16,000 E

Bench Number	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
3910	3,233,466	4.6	7.0	11.6
3870	3,728,295	4.0	6.2	10.2
3830	3,496,113	4.0	6.0	10.0
3790	2,991,552	3.9	6.1	10.0
3750	2,544,281	3.6	5.8	9.4
3710	3,185,651	3.7	6.0	9.7
3670	2,786,759	4.3	6.9	11.2
3630	2,031,760	3.4	5.9	9.3
3590	1,031,434	3.2	6.1	9.3
Total	25,029,311	3.9	6.2	10.1

SOUTH END: BENCHES 3870 to 3350 inc.

Bench Summary 7,000 N to 8,500 N — April 1975
12,000 E to 16,000 E

Bench Number	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
3870	68,782	3.5	4.6	8.1
3830	127,671	3.0	5.0	8.0
3790	256,284	3.7	6.8	10.5
3750	297,084	3.0	4.9	7.9
3710	404,213	2.4	4.5	6.9
3670	217,653	2.0	4.5	6.5
3630	371,472	2.1	4.3	6.4
3590	384,708	2.9	5.7	8.6
3550	1,929,670	2.9	5.5	8.4
3510	3,987,009	3.2	5.9	9.1
3470	3,836,487	3.4	5.2	8.6
3430	4,644,439	3.2	5.1	8.3
3390	2,103,980	2.9	5.0	7.9
3350	695,123	2.9	4.5	7.4
Total	19,324,575	3.1	5.3	8.4

LARGE FARO DEPOSIT

3910 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-04	94,222	5.3	8.8	14.1
65-5A	188,444	4.7	5.8	10.5
65-07	45,226	2.6	7.2	9.8
65-11	141,333	6.4	5.2	11.6
65-14	7,538	2.4	4.7	7.1
66-04	15,075	1.8	3.9	5.7
66-08	48,053	4.3	6.2	10.5
66-09	66,898	3.6	8.7	12.3
66-12	178,080	6.9	9.7	16.6
66-14	94,222	3.4	7.4	10.8
66-15	188,444	4.4	6.8	11.2
66-16	211,058	6.8	8.5	15.3
66-18	14,133	2.4	4.7	7.1
66-20	62,658	5.3	7.6	12.9
66-23	12,249	5.8	8.3	14.1
66-24	37,689	2.6	6.3	8.9
66-25	80,089	4.8	8.2	13.0
66-32	101,760	4.0	7.8	11.8
66-35	77,262	4.1	8.4	12.5
66-39	13,191	3.6	8.7	12.3
66-41	10,364	5.3	8.8	14.1
66-44	107,413	7.3	8.6	15.9
67-34	15,075	4.8	8.2	13.0
70-01	92,338	3.1	5.7	8.8
70-02	94,222	2.0	4.0	6.0
70-03	94,222	3.7	6.8	10.5
70-04	94,222	3.4	6.3	9.7
70-05	94,222	3.0	7.8	10.8
70-06	94,222	5.2	7.3	12.5
70-08	9,422	1.8	4.1	5.9
70-09	141,333	4.2	6.0	10.2
70-10	6,596	3.1	5.7	8.8
70-11	94,222	1.4	5.5	6.9
70-14	135,680	6.3	7.2	13.5
70-16	94,222	2.3	5.5	7.8
72-01	59,596	6.9	9.7	16.6
72-03	87,627	5.3	6.9	12.2
72-06	65,013	3.3	6.6	9.9
72-07	87,627	4.3	5.7	10.0
73-27	30,151	2.7	5.6	8.3
74-06	48,053	4.2	4.8	9.0
Total	3,233,466	4.6	7.0	11.6

001379

November 28, 1991

TO: N. ROSE, I. BILQUIST

FROM: M. HOFFMAN

COPY: L. HWOZDYK, K. CORNISH

RE: PROPOSAL FOR MINING THE SOUTH MINING BLOCK

SEQUENCE

1. Mine rooms from current accesses in South H, SH800, SH300, SN800.

2. Bench South H from SH-SG100 intersection to SH300 - South H intersection.

3. Bench SH300 - SN800 to FW contact and SH800 to FW contact.

4. Drive 305 ft. waste ramp from SN Acc ramp, +18% to SH800 (El 3153 to El 3208).

5. Haulage from SN mining area (includes SN1000, SN700, SN1200, SN900 and access drifts down dip) through SN Acc ramp to SH800 to SH to SG100 to SG to SD.

6. Benching of the bottom end of South D from SN200 - SG300 access.

ADVANTAGES OF SN ACCESS RAMP - SH800 CONNECTION

1. Require independent access to recover thicker ore in South N area.

2. It is hoped that this access will free up more potential headings which will allow the daily production rate to be as high as possible. The higher the production rate the lower the unit cost/ton mined for Curragh.

LARGE FARO DEPOSIT

3830 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-04	86,684	4.0	6.1	10.1
65-5A	188,444	4.9	5.7	10.6
65-06	6,124	2.1	5.2	7.3
65-07	102,702	1.8	5.7	7.5
65-08	15,076	6.3	9.2	15.5
65-11	142,040	7.3	8.9	16.2
66-02	97,803	2.5	3.1	5.6
66-04	100,818	2.3	4.6	6.9
66-08	188,444	5.6	7.0	12.6
66-09	64,307	2.7	5.7	8.4
66-12	44,520	3.1	5.6	8.7
66-14	86,920	6.4	10.8	17.2
66-15	188,444	4.0	5.4	9.4
66-22	187,973	3.5	5.4	8.9
66-23	148,588	6.4	6.8	13.2
66-30	68,971	2.4	5.9	8.3
66-32	59,360	2.4	5.0	7.4
66-35	40,515	1.9	6.5	8.4
66-39	23,084	2.3	3.9	6.2
66-44	32,507	5.3	8.7	14.0
66-57	2,356	2.5	5.1	7.6
70-01	60,067	3.0	6.9	9.9
70-02	84,800	1.8	3.6	5.4
70-03	82,444	3.2	7.9	11.1
70-05	94,222	4.3	5.9	10.2
70-06	117,778	4.0	6.2	10.2
70-09	120,133	1.5	4.2	5.7
70-11	45,227	2.5	4.4	6.9
70-14	141,333	4.9	7.9	12.8
70-16	94,222	6.8	12.4	19.2
72-01	38,631	2.7	5.4	8.1
72-03	225,426	5.1	5.9	11.0
72-06	123,431	2.4	6.5	8.9
72-07	139,449	3.3	3.7	7.0
73-29	24,027	2.5	5.1	7.6
74-05	96,578	4.2	3.7	7.9
74-06	127,483	2.6	3.3	5.9
74-09	5,182	5.1	5.9	11.0
Total	3,496,113	4.0	6.0	10.0

LARGE FARO DEPOSIT

3790 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-04	57,711	2.8	5.6	8.4
65-06	90,453	2.2	4.4	6.6
65-07	59,831	2.7	7.4	10.1
65-08	84,329	6.3	9.2	15.5
65-11	65,013	4.2	8.0	12.2
66-02	159,236	3.2	4.5	7.7
66-04	40,515	2.6	6.8	9.4
66-08	193,155	3.7	5.0	8.7
66-09	57,004	3.2	7.1	10.3
66-14	77,262	3.8	6.2	10.0
66-15	265,706	6.9	9.4	16.3
66-18	3,298	2.7	7.4	10.1
66-22	119,662	2.1	3.7	5.8
66-23	117,778	3.0	4.5	7.5
66-30	91,396	2.4	5.9	8.3
66-32	36,747	1.8	6.8	8.6
66-33	11,307	4.2	7.2	11.4
66-35	53,707	1.6	3.9	5.5
66-39	1,884	2.3	3.9	6.2
70-01	9,422	2.9	6.8	9.7
70-02	94,222	2.2	4.8	7.0
70-03	51,822	2.6	5.0	7.6
70-04	94,222	2.7	4.5	7.2
70-05	88,569	4.3	7.5	11.8
70-06	52,764	2.6	5.4	8.0
70-09	43,342	2.2	3.5	5.7
70-11	88,569	4.9	9.6	14.5
70-14	76,555	4.5	7.3	11.8
70-16	94,222	2.6	5.2	7.8
72-01	20,729	2.2	5.0	7.2
72-03	282,667	5.6	6.0	11.6
72-07	12,720	6.0	6.4	12.4
72-08	39,573	6.9	8.6	15.5
73-29	3,769	2.6	5.8	8.4
74-05	149,813	4.6	3.8	8.4
74-06	144,631	3.5	7.0	10.5
74-09	48,996	4.1	5.5	9.6
74-13	8,951	3.6	5.3	8.9
Total	2,991,552	3.9	6.1	10.0

LARGE FARO DEPOSIT

April 1975

3750 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-04	2,827	2.6	5.3	7.9
65-06	103,644	2.1	3.0	5.1
65-07	23,084	2.4	6.2	8.6
65-08	138,507	6.8	10.1	16.9
66-02	135,680	2.3	3.9	6.2
66-04	73,493	2.3	5.4	7.7
66-08	275,129	3.1	4.4	7.5
66-09	23,556	3.4	7.7	11.1
66-14	40,515	3.6	6.5	10.1
66-22	90,924	2.9	5.9	8.8
66-23	61,527	4.9	6.3	11.2
66-30	83,858	2.4	6.2	8.6
66-31	6,124	5.9	8.4	14.3
66-33	74,907	4.2	7.2	11.4
66-35	123,431	1.2	3.8	5.0
66-57	21,671	3.1	8.0	11.1
70-02	38,160	2.5	6.5	9.0
70-04	116,836	3.0	4.0	7.0
70-09	131,911	4.8	7.8	12.6
70-11	92,338	3.5	6.2	9.7
70-14	27,325	5.4	7.3	12.7
70-16	44,284	2.4	5.2	7.6
72-02	7,538	3.9	6.3	10.2
72-03	262,880	5.4	7.3	12.7
72-07	30,151	2.2	5.2	7.4
72-08	86,684	6.7	8.5	15.2
73-03	2,827	4.2	6.1	10.3
73-29	4,711	2.7	5.9	8.6
74-05	97,991	4.6	3.8	8.4
74-06	158,764	4.2	9.9	14.1
74-09	141,804	4.1	5.5	9.6
74-13	21,200	3.6	5.3	8.9
Total	2,544,281	3.6	5.8	9.4

LARGE FARO DEPOSIT

3710 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-6	126,258	2.8	5.0	7.8
65-7	27,324	1.2	4.2	5.4
65-8	116,364	5.1	6.4	11.5
65-9	4,240	4.3	6.5	10.8
65-12	18,844	4.3	7.7	12.0
66-2	159,707	3.1	4.9	8.0
66-4	140,391	5.4	10.0	15.4
66-8	244,035	3.5	5.4	8.9
66-14	2,355	6.5	10.8	17.3
66-22A	102,702	2.5	4.8	7.3
66-23	2,827	5.1	6.4	11.5
66-26	140,391	4.3	7.7	12.0
66-29	96,107	4.7	6.6	11.3
66-30	74,436	1.2	4.2	5.4
66-31	55,591	5.8	8.4	14.2
66-33	124,844	4.2	7.2	11.4
66-35	59,831	1.2	3.8	5.0
66-54	20,729	2.6	5.3	7.9
66-57	14,133	2.7	7.2	9.9
70-4	28,502	3.2	4.6	7.8
70-9	141,333	2.8	4.4	7.2
70-11	74,907	6.5	10.8	17.3
72-2	136,151	3.9	6.3	10.2
72-3	204,226	5.9	7.5	13.4
72-7	125,316	1.8	3.7	5.5
72-8	94,222	5.0	6.0	11.0
73-3	72,551	6.4	8.4	14.8
74-5	159,236	2.4	5.0	7.4
74-6	242,151	1.3	4.7	6.0
74-9	282,667	4.1	5.3	9.4
74-12	50,880	4.3	7.7	12.0
74-13	41,458	3.2	4.9	8.1
74-14	942	3.5	6.6	10.1
Total	3,185,651	3.7	6.0	9.7

LARGE FARO DEPOSIT

3670 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-06	136,151	3.6	7.8	11.4
65-07	14,322	2.5	4.8	7.3
65-08	9,893	4.9	6.3	11.2
65-09	94,693	4.3	6.5	10.8
65-12	94,222	2.6	4.8	7.4
66-02	144,631	2.7	4.3	7.0
66-04	115,187	3.2	5.9	9.1
66-08	157,587	4.9	7.1	12.0
66-22A	188,444	2.5	4.8	7.3
66-26	13,662	2.4	4.1	6.5
66-30	74,907	2.5	4.8	7.3
66-31	141,333	6.4	8.4	14.8
66-33	78,204	4.0	5.1	9.1
66-35	8,480	2.4	5.7	8.1
66-46	22,613	3.1	4.2	7.3
66-54	6,407	2.6	4.8	7.4
70-09	99,875	3.3	4.9	8.2
70-11	28,738	4.5	7.3	11.8
72-02	103,173	4.3	6.2	10.5
72-03	82,208	5.9	7.5	13.4
72-07	119,191	1.6	4.2	5.8
72-08	48,053	4.4	5.9	10.3
73-03	141,333	6.8	9.1	15.9
73-28	87,061	3.9	6.5	10.4
74-01	29,209	2.7	5.3	8.0
74-05	64,071	2.3	5.0	7.3
74-06	267,591	6.7	14.0	20.7
74-09	308,578	6.1	6.5	12.6
74-12	12,720	2.6	4.8	7.4
74-14	94,222	2.7	5.9	8.6
Total	2,786,759	4.3	6.9	11.2

LARGE FARO DEPOSIT

3630 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-06	230,844	3.1	6.5	9.6
65-09	104,587	5.5	6.7	12.2
66-02	187,973	2.9	6.8	9.7
66-04	76,320	3.3	6.0	9.3
66-06	100,818	1.8	3.6	5.4
66-08	49,702	4.4	6.8	11.2
66-30	41,929	2.0	5.3	7.3
66-31	92,337	5.5	6.7	12.2
66-33	61,056	2.2	4.0	6.2
66-46	136,151	3.5	4.8	8.3
66-55	6,124	4.3	7.1	11.3
72-07	189,858	2.4	5.1	7.5
73-28	63,129	3.4	6.1	9.5
74-01	13,945	1.8	3.6	5.4
74-05	178,551	3.2	5.9	9.1
74-06	228,489	3.0	7.0	10.0
74-09	269,947	4.3	5.4	9.7
Total	2,031,760	3.4	5.9	9.3

LARGE FARO DEPOSIT

3590 Bench 8,500 N to 11,400 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
65-06	51,351	2.8	5.9	8.7
65-09	19,787	3.3	4.0	7.3
66-02	37,689	2.6	6.8	9.4
66-03	26,382	2.1	3.7	5.8
66-06	167,716	3.9	8.6	12.5
66-07	39,573	3.4	6.5	9.9
66-33	72,551	2.1	3.7	5.8
66-46	226,133	2.6	3.5	6.1
66-52	15,076	3.3	4.0	7.3
73-02	23,556	4.9	6.2	11.1
73-28	5,653	3.4	6.1	9.5
74-01	20,729	3.9	8.6	12.5
74-05	184,376	3.8	8.1	11.9
74-06	86,684	2.4	5.7	8.1
74-07	16,960	3.9	8.6	12.5
74-09	37,218	4.3	5.4	9.7
Total	1,031,434	3.2	6.1	9.3

LARGE FARO DEPOSIT

3550 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-06	63,600	1.8	3.9	5.7
65-09	69,724	3.8	5.2	9.0
66-E9	46,169	1.9	5.2	7.1
66-02	34,862	2.6	6.8	9.4
66-03	55,591	4.3	7.6	11.9
66-05	7,066	2.1	3.7	5.8
66-06	154,996	3.1	7.7	10.8
66-07	207,760	3.4	6.5	9.9
66-11	6,596	3.1	7.7	10.8
66-46	138,507	1.8	3.9	5.7
66-52	72,551	5.4	7.1	12.5
66-55	57,004	3.0	4.4	7.4
67-03	97,991	2.2	4.0	6.2
67-05	159,236	2.1	5.2	7.3
67-07	99,404	3.0	4.8	7.8
72-16	242,622	2.4	4.8	7.2
73-02	146,515	2.6	6.8	9.6
74-07	90,924	2.9	7.1	10.0
74-08	151,698	3.5	3.3	6.8
74-10	19,787	3.2	7.5	10.7
74-20	7,067	2.1	5.2	7.3
Total	1,929,670	2.9	5.5	8.4

LARGE FARO DEPOSIT

3510 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
65-06	15,547	2.3	4.8	7.1
66-E9	25,440	3.6	7.6	11.2
66-03	178,551	4.2	7.6	11.8
66-05	77,733	2.3	3.4	5.7
66-06	86,684	3.1	7.7	10.8
66-07	314,702	4.2	9.5	13.7
66-10A	138,978	2.5	6.6	9.1
66-11	942	2.9	7.1	10.0
66-46	160,178	2.3	4.8	7.1
66-52	165,831	5.4	7.1	12.5
67-03	154,524	2.1	3.5	5.6
67-04	47,111	3.1	5.1	8.2
67-05	121,076	1.9	4.9	6.8
67-06	229,195	1.9	4.2	6.1
67-08	52,528	3.0	6.3	9.3
67-10	180,906	2.1	3.3	5.4
67-11	311,168	3.0	7.0	10.0
67-12	98,933	2.4	3.7	6.1
67-30	258,169	3.0	4.8	7.8
71-05	66,898	3.1	5.2	8.3
72-15	60,537	1.7	4.6	6.3
72-16	152,169	2.1	5.4	7.5
73-02	120,133	4.2	7.6	11.8
74-07	183,733	2.9	7.1	10.0
74-08	228,018	2.7	4.1	6.8
74-10	72,080	2.9	5.9	8.8
74-15	338,258	5.1	5.9	11.0
74-17	146,987	4.3	6.5	10.8
Total	3,987,009	3.2	5.9	9.1

LARGE FARO DEPOSIT

3470 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
66-03	59,831	2.1	3.7	5.8
66-05	145,573	2.3	3.4	5.7
66-06	35,804	2.9	5.6	8.5
66-07	18,844	1.5	3.6	5.1
66-46	44,284	2.9	6.6	9.5
66-49	376,888	5.4	6.7	12.1
66-52	299,626	6.1	7.1	13.2
67-03	189,387	3.2	5.4	8.6
67-04	220,244	1.8	4.6	6.4
67-05	97,284	1.9	4.9	6.8
67-08	89,039	2.8	6.4	9.2
67-10	167,479	3.1	7.2	10.3
67-11	57,476	3.0	7.0	10.0
67-12	301,982	2.0	4.3	6.3
67-30	376,888	3.7	4.7	8.4
70-12	30,622	5.7	8.1	13.8
70-13	31,093	3.1	5.2	8.3
70-17	204,462	3.6	5.2	8.8
71-04	139,449	2.5	4.1	6.6
71-05	189,387	3.1	5.2	8.3
74-07	72,551	2.9	5.6	8.5
74-08	283,138	2.2	3.8	6.0
74-15	40,516	4.3	4.5	8.8
74-17	364,640	3.4	4.3	7.7
Total	3,836,487	3.4	5.2	8.6

LARGE FARO DEPOSIT

3430 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-03	229,431	2.5	3.1	5.6
66-05	13,426	2.1	3.7	5.8
66-07	111,182	1.5	3.6	5.1
66-10A	208,231	1.9	3.3	5.2
66-46	127,200	3.4	5.9	9.3
66-49	376,888	2.3	3.3	5.6
66-52	229,902	5.4	7.1	12.5
67-03	253,929	2.7	5.9	8.6
67-04	54,884	5.3	10.1	15.4
67-06	207,759	3.2	4.6	7.8
67-10	94,928	3.2	7.5	10.7
67-11	376,888	2.7	6.3	9.0
67-12	376,888	3.5	5.9	9.4
67-30	193,627	3.6	5.1	8.7
70-12	180,435	5.7	8.1	13.8
70-17	200,222	3.6	5.2	8.8
71-02	16,959	2.2	4.2	6.4
71-03	16,959	5.7	8.1	13.8
71-04	298,213	2.5	4.1	6.6
71-05	161,591	3.1	5.2	8.3
72-13	33,448	6.0	7.6	13.6
74-07	154,996	3.2	5.7	8.9
74-08	119,662	1.9	3.1	5.0
74-15	365,582	2.7	4.4	7.1
74-17	241,209	3.8	3.4	7.2
Total	4,644,439	3.2	5.1	8.3

LARGE FARO DEPOSIT

3390 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
66-03	168,186	2.3	4.3	6.6
66-07	15,547	3.6	6.6	10.2
66-10A	338,729	2.6	4.9	7.5
66-46	41,458	3.6	6.6	10.2
66-49	232,258	2.9	4.6	7.5
66-52	8,008	4.1	5.8	9.9
67-03	3,297	2.9	6.4	9.3
67-06	217,653	1.9	5.0	6.9
67-10	36,276	1.9	5.0	6.9
67-11	219,538	2.8	5.7	8.5
67-12	275,600	3.3	5.0	8.3
70-12	55,355	5.7	8.1	13.8
71-01	7,302	5.7	8.1	13.8
71-03	9,893	5.7	8.1	13.8
72-13	102,702	6.0	7.6	13.6
74-15	372,178	2.6	3.8	6.4
Total	2,103,980	2.9	5.0	7.9

LARGE FARO DEPOSIT

3350 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-03	97,991	2.1	5.3	7.4
66-10A	41,458	1.2	3.9	5.1
66-49	3,298	2.5	4.2	6.7
67-06	42,635	2.6	6.0	8.6
72-13	214,826	4.5	4.5	9.0
74-15	294,915	2.3	4.2	6.5
Total	695,123	2.9	4.5	7.4

LARGE FARO DEPOSIT

3590 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
66-E6	754	3.0	6.5	9.5
66-E9	25,440	1.9	5.2	7.1
67-03	35,333	4.0	7.1	11.1
67-05	111,182	3.0	6.5	9.5
67-07	119,662	2.9	5.6	8.5
74-08	52,764	2.3	3.6	5.9
74-20	39,573	3.0	6.0	9.0
Total	384,708	2.9	5.7	8.6

LARGE FARO DEPOSIT

3630 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
66-E6	79,147	1.7	5.2	6.9
66-E9	21,200	1.8	3.2	5.0
67-04	2,120	2.3	5.7	8.0
67-05	63,129	2.3	5.7	8.0
67-07	2,356	2.4	5.0	7.4
74-10	121,076	2.3	3.3	5.6
74-20	82,444	2.3	3.9	6.2
Total	371,472	2.1	4.3	6.4

LARGE FARO DEPOSIT

3670 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
66-E6	102,702	1.7	5.2	6.9
66-E9	13,191	1.8	3.2	5.0
67-05	4,711	2.4	5.9	8.3
74-10	12,720	2.4	3.8	6.2
74-20	84,329	2.3	3.9	6.2
Total	217,653	2.0	4.5	6.5

LARGE FARO DEPOSIT

3710 Bench — 7,000 N to 8,500 N; 12,000 E to 16,000 E — April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
66-E5	35,333	2.6	4.6	7.2
66-E6	101,289	1.7	5.2	6.9
66-E9	16,018	2.0	3.0	5.0
74-10	99,876	3.1	4.6	7.7
74-11	33,920	2.6	4.7	7.3
74-19	18,373	1.7	3.3	5.0
74-20	99,404	2.5	4.2	6.7
Total	404,213	2.4	4.5	6.9

LARGE FARO DEPOSIT

3750 Bench — 7,000 N to 8,500 N, 12,000 E to 16,000 E — April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
66-E1	15,076	3.5	6.8	10.3
66-E4	6,124	1.6	3.5	5.1
66-E6	29,680	2.3	4.1	6.4
74-10	32,036	3.1	4.6	7.7
74-11	94,034	3.4	6.2	9.6
74-18	22,142	2.3	3.9	6.2
74-19	19,316	1.8	3.2	5.0
74-20	78,676	3.4	4.2	7.6
Total	297,084	3.0	4.9	7.9

LARGE FARO DEPOSIT

3790 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
66-E1	56,533	3.5	6.8	10.3
66-E4	24,498	1.6	3.5	5.1
74-10	21,671	4.5	8.2	12.7
74-11	72,080	4.5	8.2	12.7
74-18	62,658	3.5	6.8	10.3
74-20	18,844	3.4	4.2	7.6
Total	256,284	3.7	6.8	10.5

LARGE FARO DEPOSIT

3830 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
66-E1	7,538	3.5	6.8	10.3
66-E4	33,920	3.2	6.8	10.0
66-E7	23,084	2.8	4.3	7.1
74-18	35,333	2.5	4.2	6.7
74-21	27,796	3.5	4.0	7.5
Total	127,671	3.0	5.0	8.0

LARGE FARO DEPOSIT

3870 Bench 7,000 N to 8,500 N
12,000 E to 16,000 E

April 1975

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
FRH-16	9,893	2.8	4.3	7.1
66-E4	13,191	4.5	6.4	10.9
66-E7	11,307	2.8	4.3	7.1
74-11	1,884	3.3	6.1	9.4
74-21	32,507	3.5	4.0	7.5
Total	68,782	3.5	4.6	8.1

APPENDIX II

ESTIMATED TONNAGE AND GRADE TABLES

SMALL FARO DEPOSIT

DECEMBER 1973

SUMMARY OF ESTIMATED TONNAGE AND GRADE

SMALL FARO DEPOSIT

BENCHES 3930 TO 3710 INC.

December 1973

Bench Number	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
3930	30,622	2.6	5.9	8.5
3910	77,378	3.0	5.1	8.1
3890	195,744	3.8	5.0	8.8
3870	240,264	3.8	5.0	8.8
3850	401,263	3.3	4.6	7.9
3830	368,876	3.1	5.1	8.2
3810	553,081	3.7	5.2	8.9
3790	531,173	3.3	5.3	8.6
3770	585,115	3.1	5.1	8.2
3750	344,145	3.4	5.1	8.5
3730	89,275	3.9	5.6	9.5
3710	7,067	3.5	3.6	7.1
Total	3,424,003	3.4	5.1	8.5

APPENDIX II

ESTIMATED TONNAGE AND GRADE TABLES

SMALL FARO DEPOSIT

3930 Bench

December 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
67-33	24,969	2.2	5.7	7.9
73-20	5,653	4.3	6.7	11.0
Total	30,622	2.6	5.9	8.5

SMALL FARO DEPOSIT

3910 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-22	10,364	2.0	4.0	6.0
67-26	471	5.1	5.9	11.0
67-33	33,566	2.0	4.5	6.5
73-4	942	4.4	6.5	10.9
73-20	32,035	4.3	6.1	10.4
Total	77,378	3.0	5.1	8.1

SMALL FARO DEPOSIT

3890 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-22	35,804	1.7	3.8	5.5
67-26	51,351	5.1	5.9	11.0
67-33	11,306	3.7	5.3	9.0
73-4	28,267	4.4	6.5	10.9
73-18	12,955	2.9	3.0	5.9
73-19	44,755	3.8	4.3	8.1
73-20	9,422	4.3	6.7	11.0
73-21	1,884	4.4	6.5	10.9
Total	195,744	3.8	5.0	8.8

SMALL FARO DEPOSIT

3870 Bench

December 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
67-13	26,853	4.1	6.7	10.8
67-18	15,546	5.5	1.1	6.6
67-22	18,844	1.7	3.8	5.5
67-26	55,591	4.5	5.7	10.2
73-4	66,427	3.9	6.0	9.9
73-6	1,413	4.5	5.7	10.2
73-18	22,142	2.4	3.7	6.1
73-19	17.902	2.7	5.1	7.8
73-21	4,240	3.9	6.0	9.9
73-25	11,306	5.5	1.1	6.6
Total	240,264	3.8	5.0	8.8

SMALL FARO DEPOSIT

3850 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-13	72,904	4.1	6.3	10.4
67-16	19,315	3.2	5.9	9.1
67-18	28,267	3.8	1.2	5.0
67-20	10,835	2.4	6.3	8.7
67-22	8,479	2.2	3.2	5.4
67-26	36,275	4.4	5.0	9.4
67-28	25,911	3.8	6.7	10.5
73-4	61,715	2.0	3.7	5.7
73-5	1,884	2.0	3.7	5.7
73-6	22,142	4.0	4.0	8.0
73-7	52,764	3.1	5.7	8.8
73-18	38,631	2.2	3.2	5.4
73-19	471	3.1	5.2	8.3
73-21	4,711	2.0	3.7	5.7
73-25	16,959	3.8	1.2	5.0
Total	401,263	3.3	4.6	7.9

SMALL FARO DEPOSIT

December 1973

3830 Bench

Drill Hole	Tons	Grade % Pb	Grade % Zn	Grade %(Pb + Zn)
67-13	10,364	5.2	7.5	12.7
67-16	62,656	3.2	6.0	9.2
67-17	3,298	2.7	5.0	7.7
67-21	31,564	4.1	6.2	10.3
67-26	6,595	4.4	5.0	9.4
67-28	48,524	1.6	4.1	5.7
73-4	36,747	2.1	4.1	6.2
73-5	471	1.6	4.1	5.7
73-6	56,062	4.8	5.0	9.8
73-7	54,648	3.1	5.7	8.8
73-8	471	4.1	6.2	10.3
73-9	4,711	2.5	5.7	8.2
73-18	50,409	2.3	3.7	6.0
73-21	2,356	2.1	4.1	6.2
Total	368,876	3.1	5.1	8.2

SMALL FARO DEPOSIT

3810 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-13	1,884	3.2	4.2	7.4
67-16	49,938	3.3	5.8	9.1
67-17	25,911	2.7	5.0	7.7
67-18	54,649	3.4	3.6	7.0
67-21	51,822	3.9	5.1	9.0
67-27	57,004	5.4	5.6	11.0
67-28	36,275	2.1	3.8	5.9
67-31	14,604	2.4	6.5	8.9
73-6	58,418	3.2	4.2	7.4
73-7	19,787	3.9	5.1	9.0
73-8	1,884	3.9	5.1	9.0
73-9	38,160	5.5	7.2	12.7
73-10	35,804	3.1	5.6	8.7
73-11	2,355	3.7	3.8	7.5
73-12	8,951	5.4	5.6	11.0
73-18	10,364	2.6	4.1	6.7
73-24	52,764	4.7	7.2	11.9
73-26	32,507	2.4	4.8	7.2
Total	553,081	3.7	5.2	8.9

SMALL FARO DEPOSIT

3790 Bench

December 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
67-16	5,653	3.3	5.8	9.1
67-17	38,159	3.2	5.5	8.7
67-21	64,542	2.4	5.6	8.0
67-25	48,995	3.5	6.3	9.8
67-27	64,071	3.3	4.1	7.4
67-28	16,489	2.6	4.3	6.9
67-29	52,293	2.3	4.0	6.3
67-31	43,342	3.2	7.0	10.2
73-6	11,307	4.0	4.4	8.4
73-8	706	2.4	5.6	8.0
73-9	88,097	4.2	4.7	8.9
73-10	37,218	4.4	6.4	10.8
73-11	11,307	2.3	3.8	6.1
73-12	1,413	3.0	4.0	7.0
73-16	14,133	3.9	6.8	10.7
73-17	10,835	4.3	6.7	11.0
73-23	9,422	3.2	7.0	10.2
73-24	13,191	4.4	7.4	11.8
Total	531,173	3.3	5.3	8.6

SMALL FARO DEPOSIT

3770 Bench

December 1973

Drill Hole	Tons	Grade % Pb	% Zn	%(Pb + Zn)
67-15	20,258	4.5	6.2	10.7
67-17	39,573	3.0	5.0	8.0
67-21	48,053	1.6	3.4	5.0
67-25	62,656	3.5	6.3	9.8
67-27	44,284	3.7	4.6	8.3
67-28	2,356	2.1	3.8	5.9
67-29	2,356	2.3	4.0	6.3
67-31	65,955	2.2	5.3	7.5
73-8	20,729	3.0	5.0	8.0
73-9	65,013	4.4	4.8	9.2
73-11	53,235	2.7	4.8	7.5
73-12	34,862	2.0	3.9	5.9
73-13	6,595	2.2	2.9	5.1
73-16	41,458	3.9	6.8	10.7
73-17	35,804	4.3	6.7	11.0
73-22	1,413	2.2	5.3	7.5
73-23	2,356	2.2	5.3	7.5
73-24	38,160	2.8	5.0	7.8
Total	585,115	3.1	5.1	8.2

SMALL FARO DEPOSIT

3750 Bench

December 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
67-15	7,067	4.5	6.2	10.7
67-17	14,604	2.0	4.8	6.8
67-21	1,413	2.2	5.1	7.3
67-25	23,084	3.5	6.3	9.8
67-27	707	3.7	4.6	8.3
67-31	32,978	2.6	5.9	8.5
73-8	15,547	2.7	5.6	8.3
73-11	4,711	2.7	4.8	7.5
73-12	77,262	3.7	4.1	7.8
73-13	48,995	2.4	3.2	5.6
73-15	1,884	3.7	4.1	7.8
73-16	66,898	4.5	6.1	10.6
73-17	36,747	4.0	6.4	10.4
73-22	12,248	2.6	5.9	8.5
Total	344,145	3.4	5.1	8.5

SMALL FARO DEPOSIT

3730 Bench

December 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
67-21	707	4.0	4.9	8.9
67-23	21,671	3.5	6.3	9.8
73-8	16,018	3.8	6.8	10.6
73-12	13,191	4.3	4.9	9.2
73-13	1,413	4.3	4.9	9.2
73-16	36,275	4.0	4.9	8.9
Total	89,275	3.9	5.6	9.5

SMALL FARO DEPOSIT

3710 Bench

December 1973

Drill Hole	Tons	Grade		
		% Pb	% Zn	%(Pb + Zn)
67-23	7,067	3.5	3.6	7.1
Total	7,067	3.5	3.6	7.1

Pit Ore
Reserves 1/1/75

Cyprus Anvil Mining Corporation — Interoffice Correspondence

To: J. C. Devitt

Date: April 18, 1975

From: D. J. Hanson

Subject: MINING RESERVES - LARGE FARO DEPOSIT - JANUARY 1, 1975

Table I is a bench compilation of the geologic reserve (April 1975) material that falls within the Laurich Kennedy Ultimate Pit (1974) from the January 1, 1975 surveyed banks.

Summary:

Geologic Reserve as of January 1, 1975			Geologic Reserve Outside L. K. Ultimate Pit			Mining Reserve as of January 1, 1975		
Tons	% Pb	% Zn	Tons	% Pb	% Zn	Tons	% Pb	% Zn
35,468,951	3.3	5.7	3,160,158	3.0	5.9	32,308,793	3.4	5.7

Some of the "outside" geologic reserve material can be mined without additional waste stripping.

D. J. Hanson
Mine Geologist

DJH/mm

Attach.

CYPRUS

TABLE I

LARGE FARO DEPOSIT

MINING RESERVE AS OF JAN. 01, 1975

Bench	Geologic Reserve as of January 01, 1975			Geologic Reserve Outside L. K. Ultimate Pit			Mining Reserve as of January 01, 1975		
	Tons	% Pb	% Zn	Tons	% Pb	% Zn	Tons	% Pb	% Zn
3910	212,000	4.3	7.8	64,306	4.6	7.9	147,694	4.1	7.8
3870	760,848	2.9	5.5	28,267	6.6	9.5	732,581	2.8	5.3
3830	1,497,425	3.1	5.6	6,124	3.4	7.1	1,491,301	3.1	5.6
3790	2,131,538	3.3	5.8	4,240	2.7	6.0	2,127,298	3.3	5.8
3750	2,506,961	3.8	6.1	46,640	2.4	5.2	2,460,321	3.8	6.1
3710	3,185,651	3.7	6.0	34,391	4.7	6.6	3,151,260	3.7	6.0
3670	2,786,759	4.3	6.9	99,592	5.6	11.6	2,687,167	4.3	6.7
3630	2,031,760	3.4	5.9	140,391	3.0	6.9	1,891,369	3.4	5.8
3590	1,031,434	3.2	6.1	181,849	3.0	6.8	849,585	3.2	5.9
3350	695,123	2.9	4.5	695,123	2.9	4.5	-	-	-
3390	2,103,980	2.9	5.0	27,796	2.4	5.5	2,076,184	2.9	5.0
3430	4,644,439	3.2	5.1	334,017	3.2	5.9	4,310,422	3.2	5.0
3470	3,836,487	3.4	5.2	500,082	2.4	5.3	3,336,405	3.6	5.2
3510	3,987,009	3.2	5.9	631,286	2.8	6.3	3,355,723	3.3	5.8
3550	1,929,670	2.9	5.5	186,089	2.8	6.7	1,743,581	2.9	5.4
3590(S)	384,708	2.9	5.7	-	-	-	384,708	2.9	5.7
3630(S)	371,472	2.1	4.3	25,912	2.2	3.5	345,560	2.1	6.1
3670(S)	217,653	2.0	4.5	2,827	1.8	3.2	214,826	2.0	4.5
3710(S)	404,213	2.4	4.5	36,746	2.2	3.6	367,467	2.4	4.6
3750(S)	297,084	3.0	4.9	5,182	3.2	6.2	291,902	3.0	4.9
3790(S)	256,284	3.7	6.8	3,298	1.6	3.5	252,986	3.7	6.8
3830(S)	127,671	3.0	5.0	49,466	3.2	4.2	78,205	2.9	5.5
3870(S)	68,782	3.5	4.6	56,534	3.3	4.3	12,248	4.4	6.0
Total	35,468,951	3.3	5.7	3,160,158	3.0	5.9	32,308,793	3.4	5.7

Cyprus Anvil Mining Corporation — Interoffice Correspondence

To: R. H. Marshall

Date: July 25, 1975

From: J. C. Devitt

Subject: ORE RESERVES

Enclosed is a copy of the memorandum "Mining Reserves, Large Faro Deposit, January 1, 1975" from D. Hanson dated April 18, 1975. This was compiled using:

1) L-K Ultimate Pit
2) 1974 Diamond Drilling Results
3) Unadjusted Grades
4) Standard Ore-Block Projections (no rotary test hole results)
5) January 1, 1975 Status

This was calculated to correlate exactly with the recent reserve analysis by E. N. Pennebaker. A comparison follows:

	Cyprus Anvil			E.N.P.		
	Tons	Pb	Zn	Tons	Pb	Zn
Inside Pit	32,308,793	3.4	5.7	34,277,283	3.2	5.4
Outside Pit	3,160,158	3.0	5.9	7,656,438	2.9	5.7
Total	35,468,951	3.3	5.7	41,933,721	3.2	5.4

The only discrepancy noted in the comparison is the tonnage factor. We have used 3.18 tons/yd.^{3}, whereas Mr. Pennebaker used 3.15. However, this factor is to be revised soon based on an extensive analysis of core specific gravity as related to assay values.

Metal contents of both estimates inside the design pit are very close (within 0.5%). The considerable discrepancies outside the pit are explained in pg. 15 of Mr. Pennebaker's report.

A redesign of the ultimate pit is in order to consider the economics of recovering at least a portion of the ore outside the present pit design. This study is held up pending results of slope design parameters, which should be established by the end of August unless additional information (i.e. diamond drilling) is required to obtain final results.

Reserves in the unmined Small Faro Deposit, also calculated on a comparative basis are:

CYPRUS ANVIL

	Cyprus Anvil*			E.N.P.		
	Tons	Pb	Zn	Tons	Pb	Zn
Inside Pit	3,424,003	3.4	5.1	3,463,942	3.4	5.0
Outside Pit	0	-	-	582,296	2.8	4.6
Total	3,424,003	3.4	5.1	4,046,238	3.3	4.9

These figures are all calculated on the basis of 20 ft. bench heights in ore. The ore shown in Mr. Pennebaker's report for outside the pit is thought to be isolated blocks away from the main orebody, which were not counted in Cyprus Anvil's calculations. These variances can be identified when Mr. Pennebaker brings the cross-sections up in August.

J. C. Devitt
Chief Engineer

JCD/mm

cc. J. F. Olk (w/attach.)
D. J. Hanson (w/o attach.)

* From the December 1973 Tonnage and Grade Estimate

Pennebaker
July, 1975

RESERVE ESTIMATE

OF THE

FARO ORE BODIES

YUKON TERRITORY, CANADA

Submitted to:

Cyprus Anvil Mining Corporation, Ltd.

July 8, 1975

By E. N. Pennebaker

E. N. Pennebaker



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SUMMARY

The results of the estimate described in this report, as of January 1, 1975, are as follows:

THE LARGE FARO ORE BODY

WITHIN THE "L.K. ULTIMATE PIT"

TONNAGES OF PROVED PLUS PROBABLE ORE

(Ore Bodies 1 + 3)

Estimate by Longitudinal Sections

Category	Cubic Yards	Short Tons	% Pb	% Zn	Combined Pb + Zn
Proved	4,839,712	15,245,076	3.5	5.8	9.3
Probable	6,041,978	19,032,207	3.0	5.0	8.0
	10,881,690	34,277,283	3.2	5.4	8.6

OUTSIDE OF THE "L.K. ULTIMATE PIT"

TONNAGES OF PROVED PLUS PROBABLE ORE

(Ore Bodies 1 + 3)

Estimate by Longitudinal Sections

Category	Cubic Yards	Short Tons	% Pb	% Zn	Combined Pb + Zn
Proved	527,961	1,663,075	3.3	6.4	9.7
Probable	1,902,655	5,993,363	2.7	5.5	7.7
	2,430,616	7,656,438	2.9	5.7	8.6

TOTAL PROVED PLUS PROBABLE ORE

(Inside and Outside of the "L.K. Ultimate Pit")

Cubic Yards	Short Tons	% Pb	% Zn	Combined Pb + Zn
13,312,306	41,933,721	3.2	5.4	8.6

THE SMALL FARO ORE BODY

WITHIN THE GIVEN PIT OUTLINE

Average of Results by Longitudinal and Cross Sections

TONNAGE OF PROBABLE ORE

(Ore Body 2)

Cubic Yards	Short Tons	% Pb	% Zn	Combined Pb + Zn
1,099,664	3,463,942	3.4	5.0	8.4

OUTSIDE THE GIVEN PIT OUTLINE

Average of Results by Longitudinal and Cross Sections

TONNAGE OF PROBABLE ORE

(Ore Body 2)

Cubic Yards	Short Tons	% Pb	% Zn	Combined Pb + Zn
184,855	582,296	2.8	4.6	7.4

TOTAL PROBABLE ORE

(Ore Body 2)

Cubic Yards	Short Tons	% Pb	% Zn	Combined Pb + Zn
1,284,519	4,046,238	3.3	4.9	8.2

REGISTERED GEOLOGIST
CERTIFICATE EXPIRES
E. N. PENNEBAKER
D. C. 31, 19[illegible]
ARIZONA, U.S.A.

E. N. Pennebaker

E. N. Pennebaker
Registered Geologist
in the States of
Arizona
California
Idaho
Nevada

ORE RESERVE
OF THE
FARO ORE BODIES

INTRODUCTION

The Faro mineralized zone contains two ore deposits. One is the large Faro ore body, made up of what was formerly known as the No. 1 and No. 3 ore bodies; the other is the small Faro ore body, once known as No. 2.

The Faro ore bodies are valuable for their lead, zinc and silver content. A minor amount of copper is also present and some day may become a by-product. The present estimate is for the amount of contained lead and zinc only. From past estimates silver is known to amount to a little over one troy ounce per ton.

The large ore body trends about N40°W and is about 4,800 feet long by 1,500 feet wide in plan view. It varies from about 50 to 300 feet in vertical thickness. It dips gently toward the southwest and plunges irregularly. The large Faro ore body is cross-cut by a steep zone of barren quartz diorite dikes at about its mid-point. This dike zone is about 100 to 200 feet wide at the horizon of the ore zone, and it "robs" the deposit of a considerable amount of ore.

The small Faro ore body is found southeast of the large body, distant about 700 feet. It strikes about N45°W and is approximately 1,600 feet long by 1,000 feet wide. It averages from

about 20 to 50 feet in thickness and dips to the southwest.

In 1967 the writer estimated the tonnage and grade of both the Faro ore bodies based on about 93 vertical diamond drill holes. Since that time some 76 additional holes have been drilled in or near the ore bodies.

The 1967 estimate was made prior to mining. Since then 13.8 million tons of better than average grade have been mined and milled.

The ore bodies are covered and masked by glacial sands and gravels with an average thickness of about 55 feet, under which the ore crops out on the buried bedrock surface. From this sub-outcrop the northwesterly part of the large Faro ore body plunges down to the southeast under bedrock and attains a rock cover of some 300 feet where it meets the dike zone. Beyond the dike zone, to the southeast, it is dropped down about 150 feet. Farther to the southeast the ore body rises toward the surface and approaches it on the far southeast.

METHOD USED IN MAKING THE ESTIMATES

The large Faro ore body was estimated in two segments: one is northwest of the dike zone where the ground is more closely drilled; the other is to the southeast.

For this estimate a set of 11 longitudinal and 19 cross-sections were furnished by Cyprus Anvil's engineering department. These sections showed the profiles of the so-called "L.K. Ultimate Pit," including the profile of the mined section as of

January 1, 1975. Also shown were the vertical diamond drill holes, and bench intervals at 40 foot spacing.

Diamond drill hole logs were supplied, including bench interval average assays at 40 feet (20 feet for the small ore body). The writer did not systematically re-calculate all of these assay averages, but many were re-calculated for various reasons, and it is believed that all such averages are accurate.

From the above data horizontal ore blocks 40 feet thick, from bench to bench, were constructed from the drill holes shown on the sections, each block extending laterally from one drill hole halfway to the adjacent holes shown on a section. The ore body was extended out only 50 feet from marginal holes.

Thus for the purpose of the estimate the ore lenses are distorted to form flat blocks bounded by open pit bench levels. Tonnages so derived are reliable, but the ore has been moved somewhat out of its true position, depending upon the amount of dip and/or plunge of the ore layers.

A cut-off grade of 5.0% combined lead plus zinc was used along with a tonnage factor of 1 cubic yard contains 3.15 short tons of ore in place. This approximately equals 8.50 cubic feet per ton, the factor used in 1967. The specific gravity of the ore is now being given close study, but the results of this study are not yet available. When they are, the following estimates can be adjusted accordingly.

Where a block 40 feet thick averaged below cut-off but was made up of one part below ore grade and another part above ore grade, the ore part was carried into the estimate as ore if the two could be separated by visual differences, such as white barren schist in contact with massive sulfides. In these cases it was assumed that the ore could be separated from the waste and sent to the mill. Otherwise the entire bench interval was considered to be waste.

The ore blocks derived on the cross-sections were measured by scaling where of regular shape or by planimeter if irregular. These measurements were reduced to cubic yards and to tons. The average metal content of the various blocks was then used to obtain the weighted average grade for lead and zinc.

THE ORE

Faro ore is mostly a massive sulfide mixture of pyrite, pyrrhotite, galena, and sphalerite with a little chalcopyrite. A small amount of silver is present, and some barite has been determined to be present. Magnetite is found in places.

A subordinate amount of ore appears as banded or disseminated sulfides in quartzite.

Granular pyrite occurs as lenses and zones of internal waste associated with the massive sulfides, and close-spaced drilling is required to outline them.

RESULTS OF THE ESTIMATE

THE LARGE FARO ORE BODY

The tonnage and grade of the segment of the large Faro ore body lying northwest of the dike zone was derived from some 75 vertical diamond drill holes. Most of these were drilled at the corners of a grid pattern that is 141 feet on a side, but there is some irregularity in the pattern and there are some gaps, where the holes are 282 feet apart.

Southeast of the dike zone, drill hole spacing for the most part is on or close to a 282-foot grid with a few holes at closer but irregular intervals. For this segment the estimate is based on some 45 vertical diamond drill holes.

Because the plunge of the ore layers is somewhat flatter than their dip, ore blocks on the longitudinal sections give a truer picture than do the cross-sections. Furthermore, longitudinal sections cross the barren dike zone at about right angles, whereas it falls between two cross-sections and was not properly considered in the writer's earlier estimates. Consequently the longitudinal sections give a more reliable estimate. Also, it has been determined by the Cyprus Anvil staff that grade zones are more regular along the NW-SE longitudinal direction.

For the large Faro ore body three items were estimated:

1. Ore within the "L.K. Ultimate Pit."
2. Ore below and outside of this pit.
3. Ore layers less than 40 feet in thickness that were

"diluted to waste" by being averaged with the waste making up the remainder of the bench interval.

The results of the estimate for the northwest part of the large Faro ore body by longitudinal and by cross-sections, as of January 1, 1975, are as follows:

	Short Tons	% Pb	% Zn	Pb + Zn
By Cross Sections	15,653,285	3.46	5.73	9.2%
By Long. Sections	15,245,076	3.48	5.80	9.3%
Average	15,449,180	3.47	5.76	9.2%

For the southeast part of the large Faro ore body, as of January 1, 1975:

	Short Tons	% Pb	% Zn	Pb + Zn
By Cross Sections	19,351,127	3.02	4.98	8.0%
By Long. Sections	19,032,207	3.02	5.02	8.0%
Average	19,191,667	3.02	5.00	8.0%

Combining the results from the longitudinal sections for the northwest and southeast sectors we have:

	Short Tons	% Pb	% Zn	Pb + Zn
NW Sector	15,245,076	3.48	5.80	9.3%
SE Sector	19,032,207	3.02	5.02	8.0%
Total	34,277,283	3.22	5.37	8.6%

This is the ore now remaining within the "L.K. Ultimate Pit." These are the figures set forth in the SUMMARY.

For that portion of the large Faro ore body that lies below and outside of the "L.K. Ultimate Pit" we have from the cross-sections:

	Short Tons	% Pb	% Zn	Pb + Zn
NW Sector	2,229,241	3.22	6.31	9.5
SE Sector	5,651,088	2.85	5.69	8.5
Total	7,880,329	2.95	5.87	8.8

From the longitudinal sections:

	Short Tons	% Pb	% Zn	Pb + Zn
NW Sector	1,663,075	3.30	6.45	9.7
SE Sector	5,993,363	2.74	5.50	7.7
Total	7,656,438	2.86	5.70	8.6

This is the ore below and outside of the "L.K. Ultimate Pit." These figures derived from longitudinal sections are set forth in the SUMMARY.

For the ore that is "diluted to waste" we have from the cross-sections:

	Short Tons	% Pb	% Zn	Pb + Zn
NW Sector	332,050	2.80	5.04	7.8
SE Sector	1,383,853	2.52	4.36	6.9
Total	1,715,903	2.57	4.49	7.1

From the longitudinal sections:

	Short Tons	% Pb	% Zn	Pb + Zn
NW Sector	208,340	2.35	4.78	7.1
SE Sector	1,445,662	2.28	4.39	6.7
Total	1,654,002	2.29	4.44	6.7

Results from the longitudinal sections are believed to be the more reliable.

THE SMALL FARO ORE BODY

The tonnage and grade of the small Faro ore body, as yet unmined, were derived from some 45 vertical diamond drill holes.

For the estimate a set of seven N-S and seven E-W sections were furnished by the Cyprus Anvil engineering department. On these were plotted the upper and lower profiles of an open pit with bench levels at intervals of 20 feet. The sections are spaced at 200 feet and run at 45 degrees to the NW-SE trend of the ore body. Consequently an average of the results obtained from the two sets of sections are presented.

The estimate was made for ore within and for ore falling outside of the open pit design provided by the Cyprus Anvil engineering department.

The ore within the pit design, as of January 1, 1975, is as follows:

	Short Tons	% Pb	% Zn	Pb + Zn
N Sections	3,770,087	3.38	4.93	8.3%
E Sections	3,157,797	3.46	5.03	8.5%
Average	3,463,942	3.41	4.98	8.4%

The ore below and outside the pit design is as follows:

	Short Tons	% Pb	% Zn	Pb + Zn
N Sections	479,941	2.65	4.61	7.3%
E Sections	217,349	2.83	4.61	7.4%
Average	582,296	2.76	4.61	7.4%

No estimate was made of "ore diluted to waste" for the small Faro ore body because of the 20-foot bench interval and the scarcity of internal waste.

THE COMBINED ORE BODIES

Within the Two Pit Designs

	Short Tons	% Pb	% Zn	Pb + Zn
	32,308,793	3.4	5.7	9.1
Large Ore Body	34,277,283	3.22	5.37	8.6
	3,424,003	3.4	5.1	8.5
Small Ore Body	3,463,942	3.41	4.98	8.4
	35,732,796	3.4	5.6	
Total	37,741,225	3.24	5.33	8.6

Within and Outside the Two Pit Designs

	Short Tons	% Pb	% Zn	Pb + Zn
Large Ore Body	41,933,721	3.16	5.43	8.6
Small Ore Body	4,046,238	3.32	4.92	8.2
Total	45,979,959	3.17	5.38	8.6

ORE "ROBBED" BY DIKE ZONE

That portion of the ore zone invaded by the post-mineral quartz diorite dikes has been estimated by the writer and converted into short tons to give an approximate measure of the ore lost. This amounts to:

3,321,184 short tons

This item can be subtracted from the writer's ore reserve estimate of October 1, 1967.

CATEGORIES OF RELIABILITY

The results of ore reserve estimates are generally grouped in several categories, reflecting varying degrees of reliability. Many terms and definitions have been proposed for these categories, but those set forth many years ago by Herbert Hoover are, in the writer's opinion, the simplest and the best for defining "proved ore" and "probable ore."

Proved ore is defined by Mr. Hoover* as "ore where there is practically no risk of failure of continuity."

Probable ore, according to his definition, "is ore where there is some risk, but warrantable justificiation for assumption of continuity."

Possible ore is a term generally employed where there are few if any samples or measurements, and where assumed continuity is mostly based on geologic evidence.

*Hoover -- "Principles of Mining", 1909, pg. 19.

These various degrees of reliability were really set up for veins that were exposed in underground mines by drifts, cross-cuts and raises. In the case of large deposits explored by vertical diamond drill holes, the drill holes are analogous to raises, but we are lacking horizontal exposures analogous to those in drifts. Consequently we have to rely on the drill hole spacing considered with respect to the structure and variability of the ore body.

What may seem to be proper drill hole spacing before some mining of the deposit has been carried out may turn out to be too wide after the deposit has been exposed.

This is somewhat the case at Faro where mining has shown the internal waste to be more erratic and more abundant than was originally expected. This problem has been solved, in part, by the holes more recently drilled. Nevertheless, the writer's earlier categories of reliability need to be re-examined.

In the large Faro ore body in the sector northwest of the dike zone, we now have 32 additional holes in addition to the original 43. These and their spacing, along with the experience gained by mining, influence the writer to believe that the ore estimated for the Northwest Sector should now be designated "proved."

In the Southeast Sector, on the other hand, there are only 27 old holes plus 19 drilled since 1970. These are widely spaced (mostly at 282 feet). In view of this, ore estimated for the

Southeast Sector should, in the writer's opinion, be designated as "probable," along with the recommendation that many more holes be drilled at half-intervals to aid in ore control during mining and to provide a more secure ore reserve estimate.

In the small ore body, there are 23 old holes plus 22 new ones. These together give a much better definition of the small ore body than was possible in 1967. Nevertheless, because of the thinness of the ore body, the number of bench intervals that are only partially filed by ore, and with hole intervals greater than 200 feet in a number of places, the estimated tonnage is designated as "probable."

DISCUSSION

The writer's estimate of reserves of the large Faro ore body in the "L.K. Ultimate Pit" (34.3 million tons @ 3.2% Pb and 5.4% Zn) is larger than that estimated by Cyprus Anvil engineering department (32.3 million tons @ 3.4% Pb plus 5.7% Zn). These are about equal as regards the gross poundage of lead plus zinc (294.8 million vs. 293.93 million grade tons).

The larger ore tonnage of the writer is probably due to his including in the reserve many ore blocks whose vertical thickness is less than the bench interval of 40 feet. To realize this tonnage sharp ore control will need to be exercised during mining.

The writer's estimate of ore below and outside of the "L.K. Ultimate Pit" (7.7 million tons by longitudinal section or 7.9 million tons by cross-sections) is much greater than that estimated by the Cyprus Anvil engineering department (3.1 million tons). This is possibly because the writer's estimate includes a great many small blocks below all parts of the ore body. It is not confined to places where the outside ore is abundant and offers the best opportunities for being included in a revised pit design. Obviously, all of the writer's tonnage cannot be recovered in a modified open pit layout.

The writer's estimate for the small ore body (4.0 million tons) is less than that obtained in 1967 (5.2 million tons). This is because the more recent drilling has invalidated some of the longer projections made in 1967 and because for the present estimate ore was projected only 50 feet beyond marginal holes.

The writer's estimate of ore "lost" in the dike zone (3.3 million tons) is greater than that obtained by Cyprus Anvil's engineering department (2.2 million tons). Probably the Cyprus Anvil tonnage was derived from more geologic information than that used by the writer, and the figure of 2.2 million tons is accepted.

In the writer's 1967 estimate the large Faro deposit was estimated as a geologic body (not confined to a given pit layout) and without dilution. Thirty percent of the estimated internal waste was then added as a diluent.

In the present estimate, the averaging of assays for bench intervals gives a certain amount of "built in" dilution. Additional dilution for interfaces where the ore block is less than 40 feet (20 feet for the small ore body) and for dilution at the lateral ends of blocks have not been taken because Cyprus Anvil's mining is very close to expected grade. Consequently, as far as dilution is concerned, the writer's figures should be between the 1967 undiluted and diluted ore.

If ore block data from the writer's sections of the large ore body are projected onto bench level plans, the longitudinal sections should be favored.

RECONCILIATION

The present minable reserves do not include all of the ore embraced by the writer's 1967 estimate. Excluded items are ore outside of the open pits, the tonnage mined out, ore stored in stockpiles, and ore "diluted to waste." The ore "robbed" by the dike zone was also not apparent in 1967. These various items are summarized below.

	Million of Tons	E.N.P. 1967 Estimate
Reserves, 1/1/75		
Large Ore Body	34.3	
Small Ore Body	3.5	
Ore Outside of Pits		
Large Ore Body	7.7	
Small Ore Body	0.6	
Ore mined and milled	13.8	
Ore in stock piles	2.4	
Ore lost in dike zone	2.2	
Ore "diluted to waste"	1.7	
	66.2	62.1 Not Diluted 63.5 Diluted
If we accept the Cyprus Anvil tonnage of ore outside the "L.K. Ultimate Pit" as mostly recoverable (3.1 million tons), the total is	61.6	

RECOMMENDATIONS

1. Review the open pit design to determine how much of the ore below and outside of the "L.K. Ultimate Pit" can be recovered.

2. Also determine if bench intervals less than 40 feet are economic. It may be possible to recover more of the reserve if this be feasible.

3. The small ore body is still open on the southeast. Drill the holes proposed in this area by the writer in 1967.

4. Segregate the low-grade sulfide waste in a separate stockpile. In the future it may become valuable. The available tonnage and grade of this material can be readily estimated from the sections submitted.

5. Only carry ore reserves for a few years "by subtraction of annual production." Re-estimate every two or three years.

6. Drill the Southeast Sector of the large Faro ore body at intermediate locations to facilitate ore control during mining and to improve the reserve estimate.



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APPENDIX

The copy of this report going to the engineering department of Cyprus Anvil has at its back the summary calculation sheets for the estimate.

Also, a full set of ore body sections will be delivered to the engineering department.

The following legend applies to the calculation sheets and to the sections:

Material	Abbreviation	Color on Sections
Sulfide ore, NW Sector of large Faro ore body	NWS	Yellow
Sulfide ore, SE Sector	SES	Yellow
Sulfide waste	ws	Bright green
Parts of sulfide waste above cut-off	Dotted	Bright green
Sulfide ore outside of "L.K. Ultimate Pit"	WSB&ESB	Red
Sulfide waste outside of "L.K. Ultimate Pit"	ws	Grayish green or uncolored
Sulfide ore, small Faro ore body	S	Yellow
Sulfide ore, small Faro ore body, outside of pit	B	Red
Ore "robbed" by dike zone	Cross-hatched	Red
Rock waste	w	Uncolored