Cyprus Anvil Mining Corporation

Interoffice Correspondence

000207

To J. C. Devitt

Date October 31, 1975

From D. J. Hanson

Subject TONNAGE FACTOR STUDY
SMALL FARO DEPOSIT

Introduction:

As a follow-up to the Large Faro Tonnage Factor Study, samples were obtained from the Small Faro Deposit for specific gravity determination.

The lower observable pyrite content in the core and the more disseminated appearance of the economic sulfides indicate that the tonnage factor for this zone should be less than that recommended for the Large Faro Deposit.

Summary and Recommendations:

1) The distribution of disseminated sulfide specific gravities is approximately log normal with an arithmetic mean of 3.03 g/cm^3. Disseminated sulfides accounted for 56% of the samples.

2) The distribution of massive sulfide specific gravities is approximately normal with an arithmetic mean of 4.18 g/cm^3.

3) The weighted average of massive and disseminated sulfide specific gravities is 3.54 g/cm^3.

It is recommended that the value of 3.00 tons/BCY be adopted as the tonnage factor for the Small Faro Deposit to replace the current factor of 3.18 tons/BCY.

Method:

1) Sampling:

Seventy-eight core samples were selected at five foot intervals from the sulfide zone of ten diamond drill holes on one longitudinal and one cross section (Figure I).

2) Specific Gravity Determinations:

Specific gravity determinations were made by the Cyprus Anvil Metallurgical Department. Samples were weighed in air and water and specific gravities were calculated by the formula:

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$$S.G. = \frac{Wa}{Wa - Ww}$$

Wa = Weight in Air
Ww = Weight in Water

The accuracy of this technique is estimated at ± 5%.

Results:

For the purpose of analysis, samples were divided into massive and disseminated sulfides. Disseminated sulfides accounted for 56% of the samples.

1) The frequency distribution of disseminated sulfide specific gravities is approximately log normal with an arithmetic mean of 3.03 g/cm^3 and a range of 2.4-3.9 g/cm^3 (Figure II).

2) The frequency distribution of massive sulfide specific gravities is approximately normal with an arithmetic mean of 4.18 g/cm^3 and a range of 3.5-4.7 g/cm^3 (Figure III).

Discussion of Results:

1) No specific gravity checks or Pb-Zn analyses were done on the Small Faro samples. Results from the Large Faro tonnage factor study showed that:

   1) There is no correlation between specific gravity and Pb-Zn assays.

   2) The external accuracy on the specific gravity check determinations was excellent.

2) The following table compares the specific gravity arithmetic means for the Large and Small Faro Deposits:

	S.G. (g/cm^3)			
	Disseminated Sulfides	Massive Sulfides	Weighted Average	% Massive Sulfides
Large Faro	3.02	4.29	4.15	89
Small Faro	3.03	4.18	3.54	44

D. Hanson
Mine Geologist

DH/mm

Attach. (3)

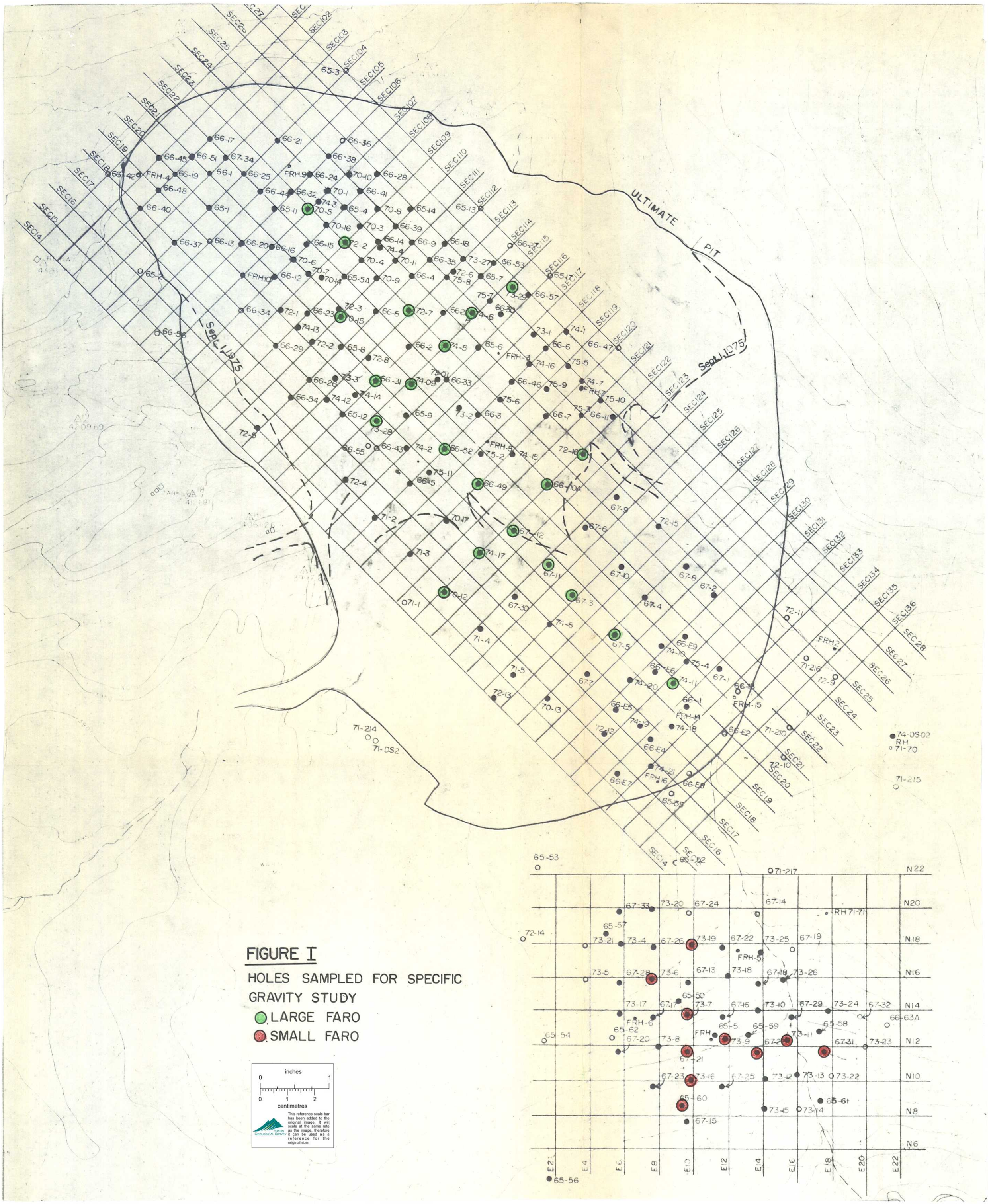

FIGURE I

HOLES SAMPLED FOR SPECIFIC GRAVITY STUDY

- LARGE FARO
- SMALL FARO

inches
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centimetres

This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.

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FIGURE II

DISSEMINATED SULFIDE
SPECIFIC GRAVITY
HISTOGRAM

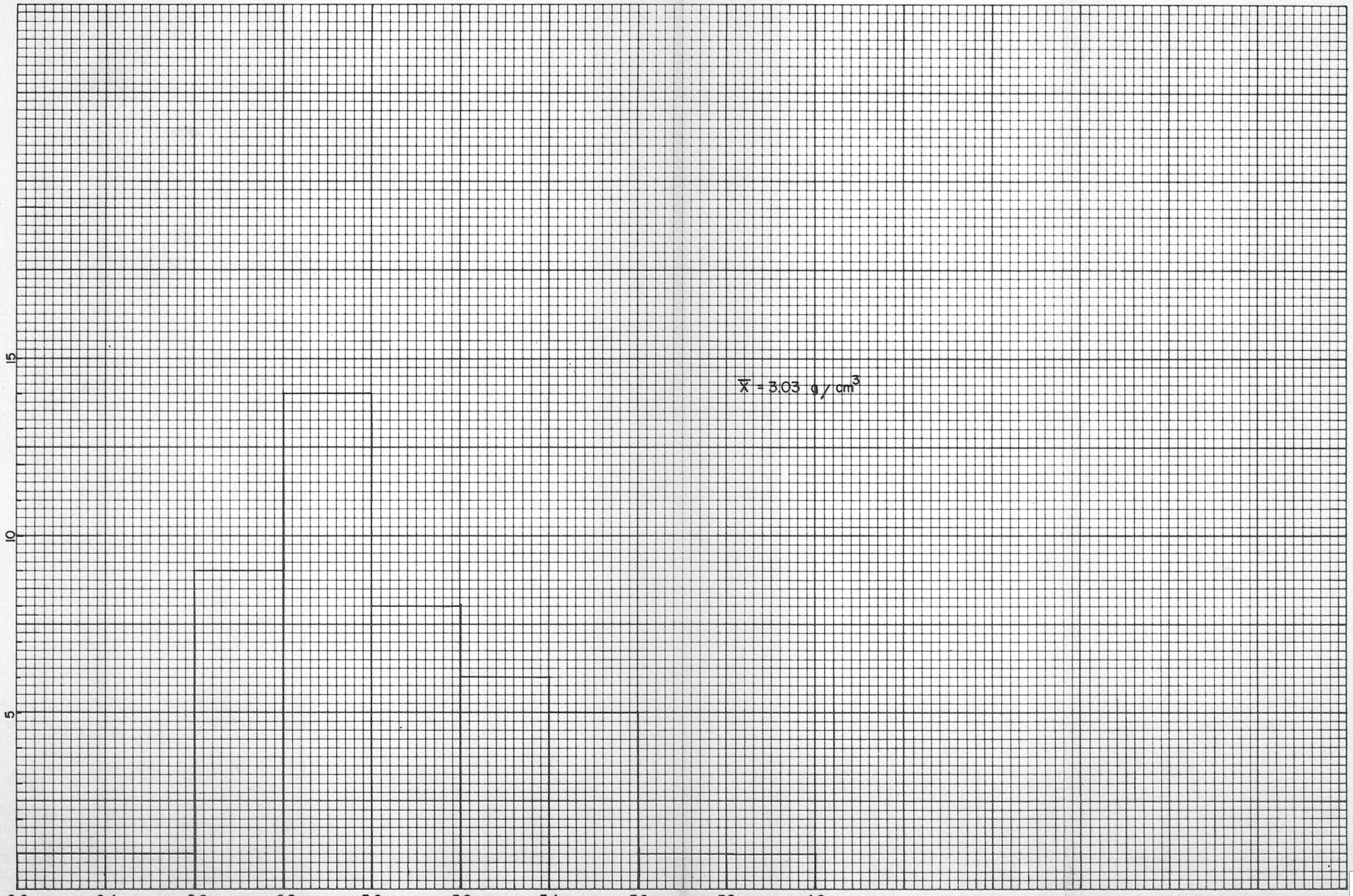
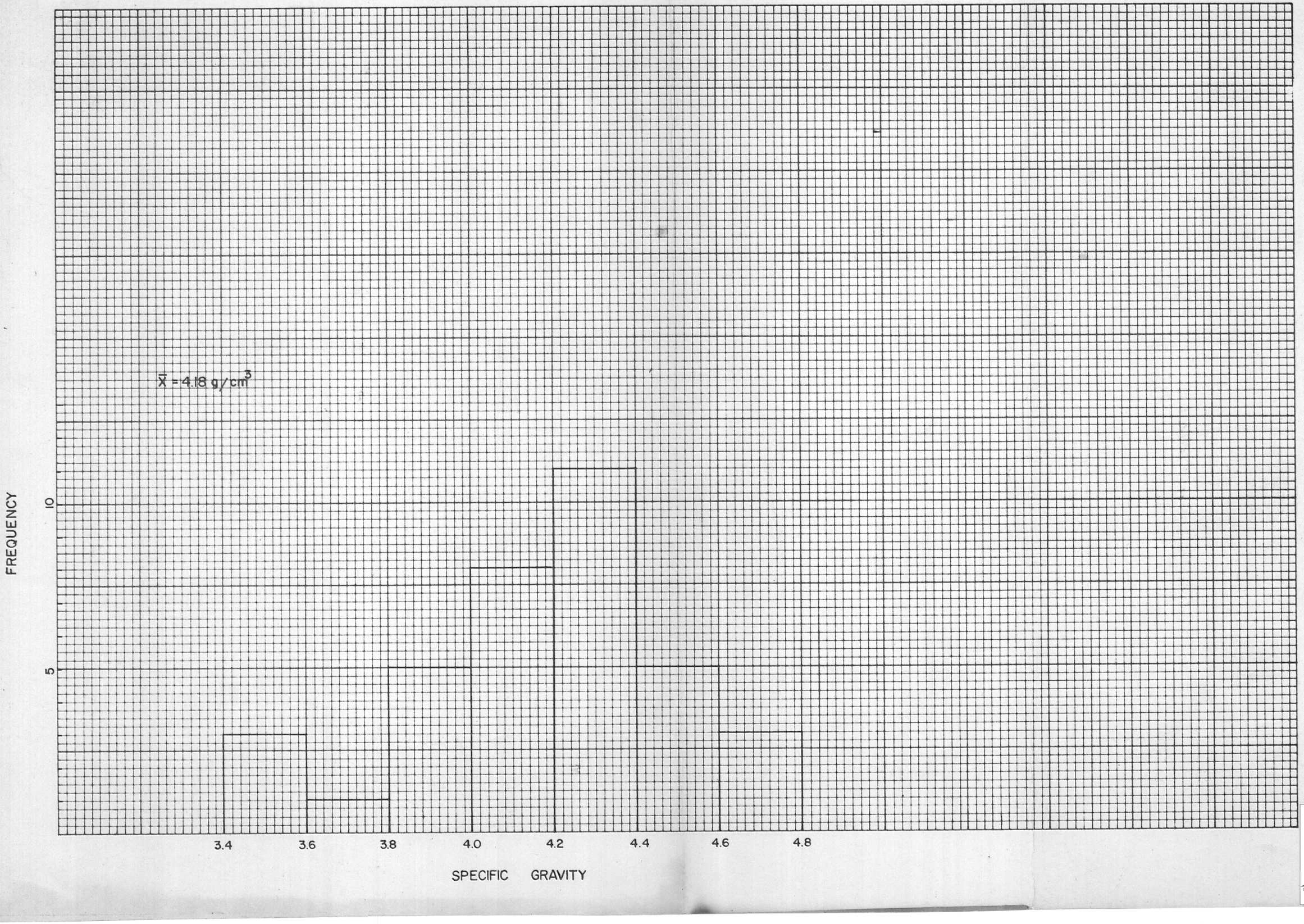


FIGURE III

MASSIVE SULFIDE
SPECIFIC GRAVITY
HISTOGRAM



Cyprus Anvil Mining Corporation — Interoffice Correspondence

To: J. C. Devitt

Date: October 9, 1975

From: D. J. Hanson

Subject: TONNAGE FACTOR STUDY
LARGE FARO DEPOSIT

Introduction:

The accuracy of our tonnage factor for the Large Faro Deposit has been in doubt for some time. Production comparisons, available specific gravity data, and theoretical specific gravity calculations have indicated that 3.18 tons per bank cubic yard is too low.

In order to accurately estimate the tonnage factor, three hundred core samples were selected for specific gravity determinations.

Summary and Recommendations:

1) The distribution of massive sulfide specific gravities is approximately anti-log normal with an arithmetic mean of 4.29 grams/cm^3. Massive sulfides accounted for 89% of the samples.

2) The distribution of disseminated sulfide specific gravities is approximately normal with an arithmetic mean of 3.02 grams/cm^3.

3) The weighted average mean of massive and disseminated sulfides is 4.15 grams/cm^3.

It is recommended that the value of 4.15 grams/cm^3 (3.50 tons/yd^3) be adopted as the tonnage factor for the Large Faro Deposit to replace the current factor of 3.77 grams/cm^3 (3.18 tons/yd^3).

Due to the wide range of specific gravity values, an attempt should be made, at least in the short term, to determine the tonnage factor for each ore block either by direct determination on core samples or by establishing a correlation between Pb, Zn, Fe, BaO assays and specific gravity.

Method:

1) Sampling:

The number of samples required to estimate the true mean within $\pm$ 0.05 tons/$yd.^3$ at the 95% confidence level was determined using normal sampling theory. The standard deviation was estimated from available data.

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$$n = \left(1.96 \frac{s}{d}\right)^2$$

$$n = \left[1.96 \left(\frac{0.55}{0.06}\right)\right]^2 = 322$$

n = number of samples

s = estimated standard deviation

d = maximum allowable difference between true mean and sampled mean

Three hundred core samples were selected at ten foot intervals from the sulfide zone of twenty-one drill holes on one longitudinal section and two cross sections (Figure I). All recognizable mineralogical sub-zones were sampled.

2) Specific Gravity Determinations:

Specific gravity determinations were made by the Cyprus Anvil Metallurgical Department. Samples were weighed in air and water and specific gravities calculated by the formula:

$$S.G. = \frac{Wa}{Wa - Ww}$$

Wa = weight in air

Ww = weight in water

The accuracy of this technique is estimated at $\pm$ 5%.

3) Assay Analysis and Specific Gravity Checks:

Samples were sent to Kamloops Research and Assay Laboratory for Pb-Zn analyses.

Thirty specific gravity checks were also performed by KRAL. Pulverized samples were weighed and displacements were measured. Accuracy of this technique is estimated at $\pm$ 4%.

Results:

For the purpose of analysis, samples were divided into massive and disseminated sulfides. Massive sulfides accounted for 89% of the samples.

1) There is no correlation between specific gravity and combined Pb-Zn grades for massive or disseminated sulfides (Figure II).

2) The frequency distribution of massive sulfide specific gravities is approximately anti-log normal with arithmetic mean of 4.29 g/cm^3 and a range of 3.1-5.1 g/cm^3 (Figure III). This mean is calculated from the geometric mean by the formula:

$M_x = M_G e^{0.5 \sigma^2_L}$

M_x = arithmetic mean

M_G = geometric mean

σ^2_L = variance of the natural logarithms of the S.G. values

3) Disseminated sulfide specific gravities display an approximately normal frequency distribution with a mean of 3.02 g/cm^3, a standard deviation of 0.17, and a range of 2.8-3.6 g/cm^3 (Figure IV).

4) A comparison of the thirty check determinations gives the following results:

	$\bar{X}$	s	n
Anvil	3.975	0.57	30
KRAL	4.086	0.59	30
Diff.	-0.111		

Discussion of Results:

The difference between the Anvil and KRAL specific gravity values can be explained by the porosity of the core samples. A porosity of only 3% would increase the specific gravity of the pulverized sample by 0.12 g/cm^3.

D. J. Hanson
Mine Geologist

DJH/mm

Attach.

FIGURE IV

DISSEMINATED SULFIDE SPECIFIC GRAVITY

FREQUENCY HISTOGRAM.

$\bar{X} = 3.02$ g/cm^3

S = 0.17

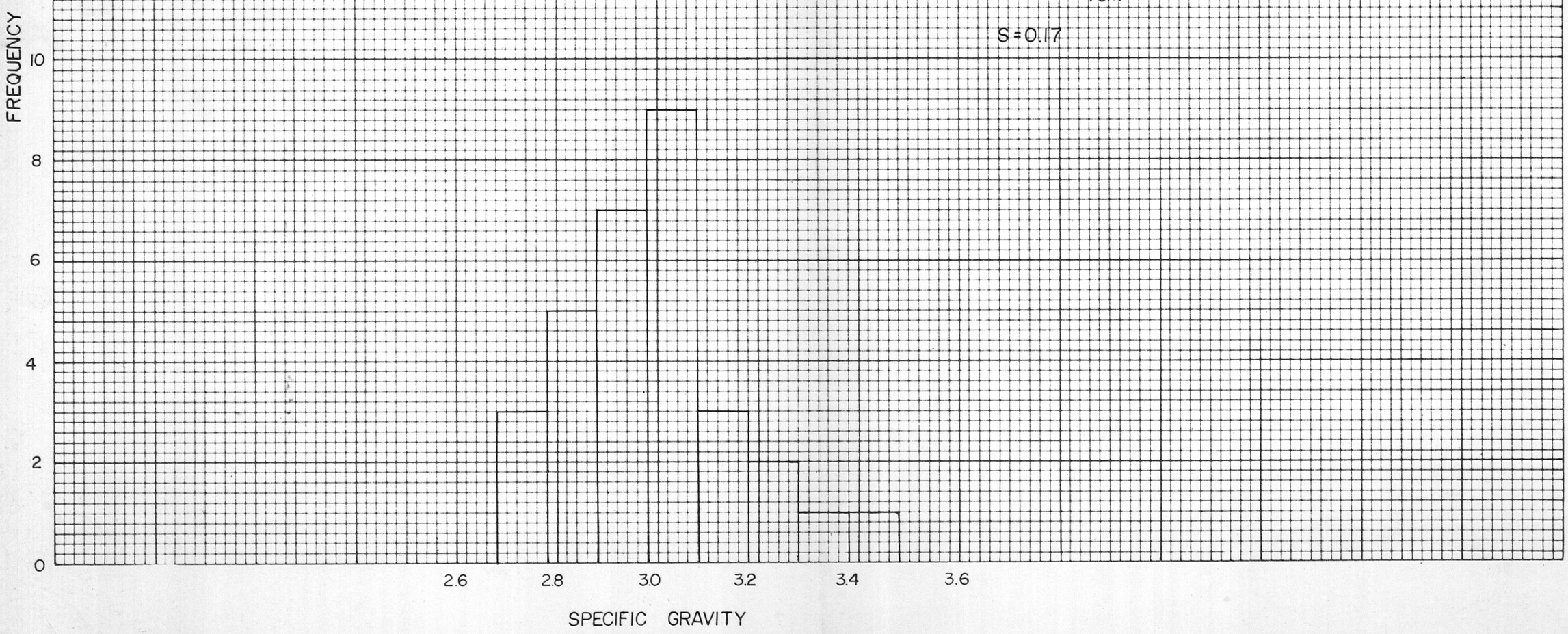

FIGURE III

MASSIVE SULFIDE SPECIFIC GRAVITY

FREQUENCY HISTOGRAM

$M_G = 4.31$ g/cm^3

$\sigma^2_L = 0.01$

$M_X = 4.29$ g/cm^3

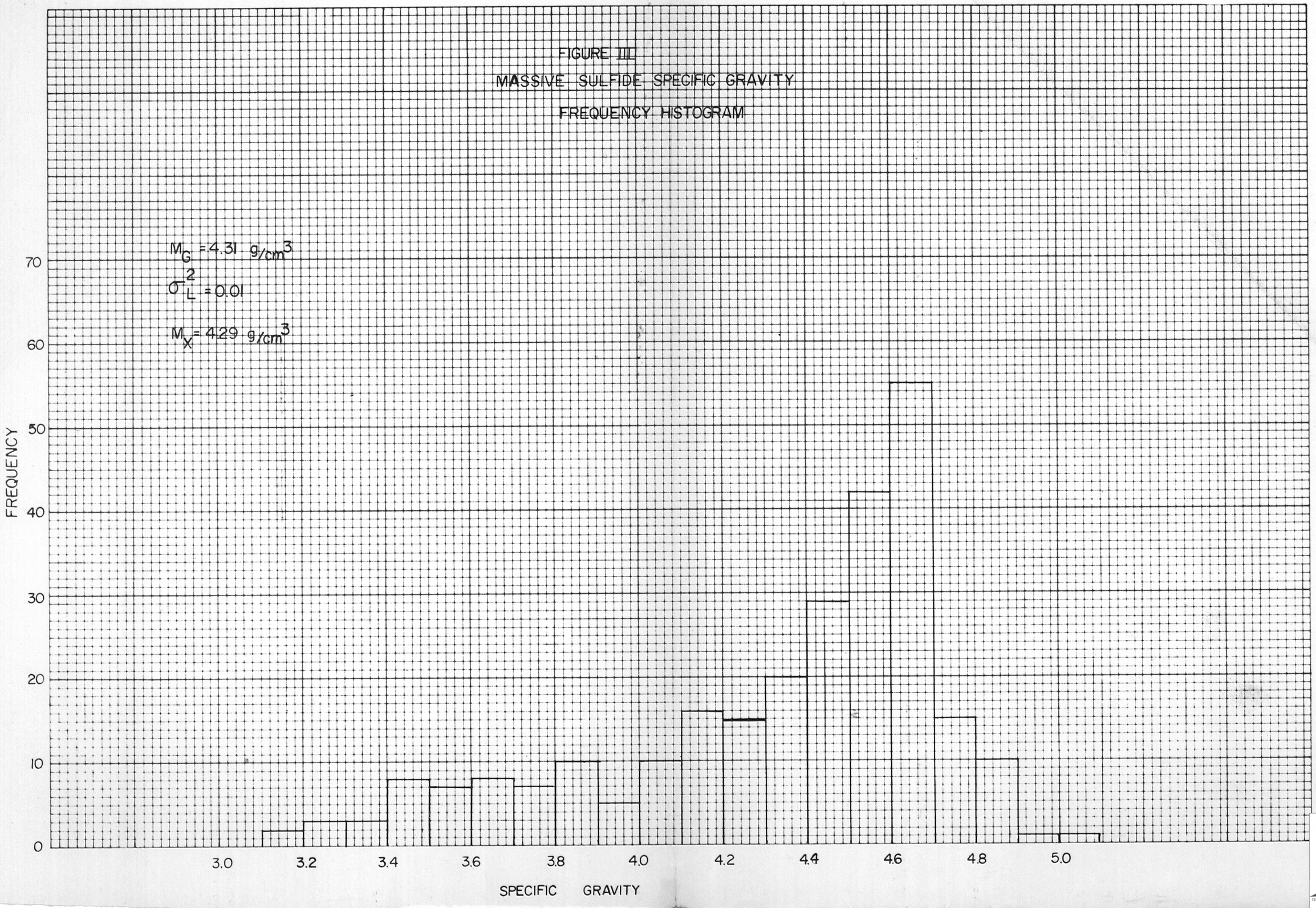

FIGURE III

SPECIFIC GRAVITY vs. % (Pb+Zn)

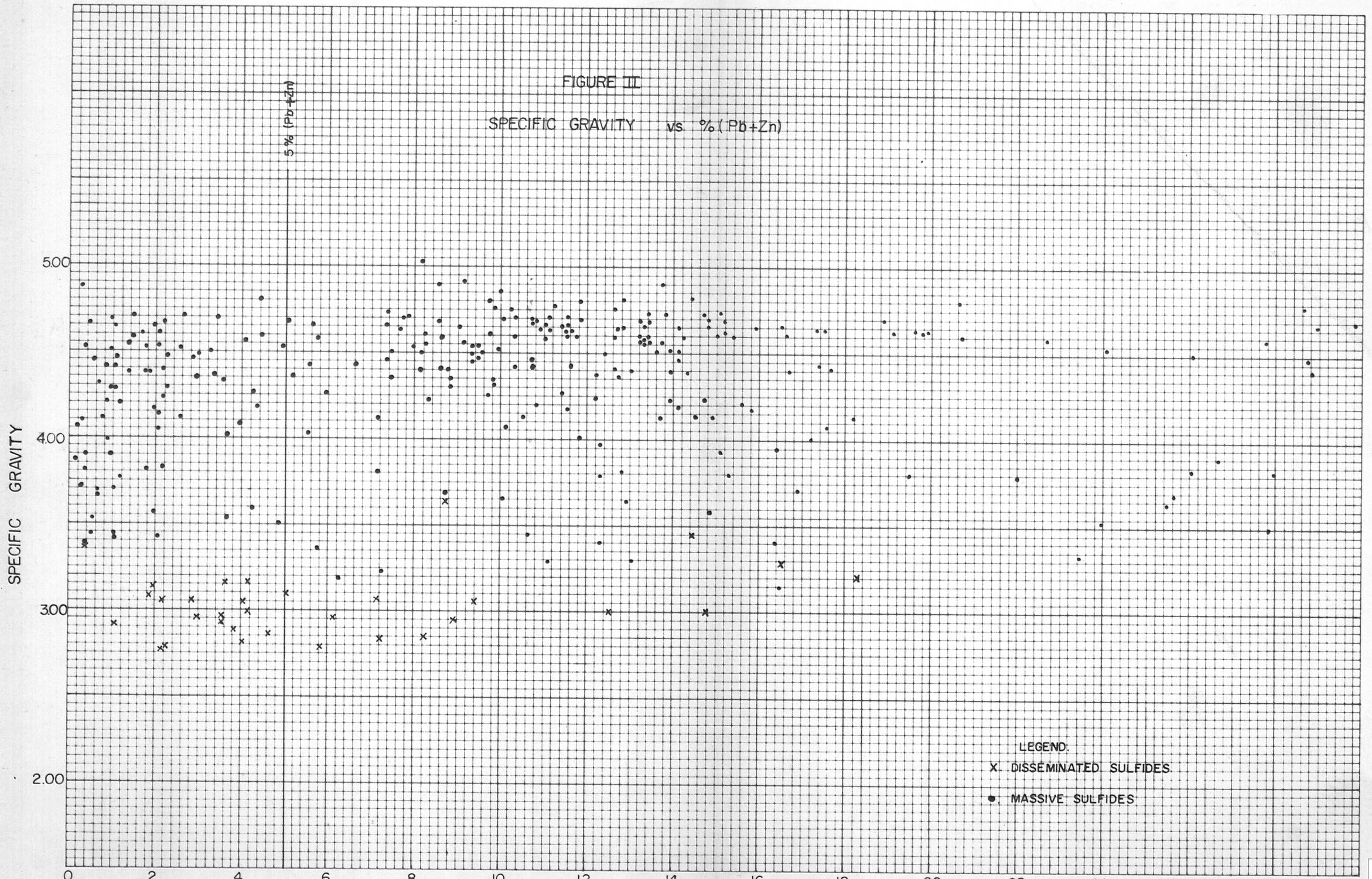

FIGURE I

HOLES SAMPLED FOR SPECIFIC GRAVITY STUDY

FA-7120

FA-7201

FX-72104

inches
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centimetres

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Cyprus Anvil Mining Corporation — Interoffice Correspondence

RAINE

To: A. von Kursell — Date: June 22, 1977

From: P. Taggart — Denis Gregoire

Subject: Re-logging and Re-assaying of Diamond Drill Core

A meeting was convened on Tuesday, June 14, 1977 at 1:00 pm in the Conference Room with the following in attendance:

Dave Jennings, Daryl Hanson - Exploration
Jack Devitt, ~~Denis Gregoire~~ - Engineering
Bob Smith, Bruce Ferguson - Mill
P. Taggart

The following items were discussed and agreed upon:

1. There is considerable benefit to be gained by re-assaying and re-logging core. Such benefits will relate to plant throughput and metallurgical performance. The accuracy of production forecasts will be greatly improved by the knowledge gained through the proposed program.

2. The following information is considered to be of value and accessible:

   a) Chemical Analysis, %:
   Pb, Zn, Ag, Cu, S, Py, Po, BaO, Mn, C(as graphite)

   b) Visual Analysis:
   Grain size, degree of mineral separation

   c) Other
   Hardness, microprobe or x-ray diffraction analysis of selected samples to determine the extent of iron dissolution within sphalerite lattice (marmatite), specific gravity determinations on selected samples, selected samples of Zone 3 core will be required for gradability tests.

A brief study prepared by D. Jennings and D. Hanson indicated that samples would be required from approximately 50 drill holes. The re-logging of each hole would consume three man days - two days for the Geologist and one day for his assistant. The program would take three months to complete in its entirety and will require the re-assaying of approximately 2150 samples. It was recommended and agreed that the Geologist should be a permanent staff employee, to "retain in-house experience." It was also generally agreed that the program should be continuous and not carried out in discreet "phases". Ideally it is recommended that the additional Geologist be hired in the immediate future, if approval for the program is forthcoming. By so doing the Geologist could expedite the completion of the mine model and then assume the responsibility for organizing and implementing the proposed program.

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STAR PRINTING 9-1016-75

Re: Diamond Drill Core

An approximate estimate for the initial program could be:

Analytical costs	$ 80,000
Personnel	11,000
Data processing	4,000
Sub total	$ 95,000

or $100,000 including some contingency

It is quite conceivable that as a result of this program other projects could be generated which may involve extensive grinding and or flotation testwork. This work may be performed by "outside labs" or more ideally by an expanded C.A.M.C. metallurgical staff. The need for additional facilities could arise. It should be noted that no provision for these costs is included or could even be estimated at this time.

In view of the ever increasing complexity of the Faro ore deposit as the mine develops it is the recommendation of all present that the program be implemented and that early consideration be given to recruiting a competent Geologist for this purpose. The anticipated expenditures required to generate the required information will be more than offset by the increased value of production achieved as a result of this program and subsequent related projects.

L.P. Taggart

/dbc
cc all present

Cyprus Anvil Mining Corporation — Interoffice Correspondence

C - JM
RL
HH

To A. H. von Kursell — Date July 28, 1977

From J. F. Olk

Subject Re-logging and Re-Assaying of Diamond Drill Core

As a result of the meeting between Glenn Simpson, yourself and myself on the above subject on July 26, 1977, the re-logging and re-assaying program is approved in principle, subject to a report on the following items:-

- The chemical analysis list is quite extensive and some of these items may or may not be justified at this time. I think some second thoughts should go into the cost vs value of some of these items.

- Also some thought should be given to the requirement at this time for microprobe or x-ray diffraction work. Don't need.

- Because of the substantial cost of this program, the following should be determined:-

  - Can a substantial amount of this assaying be done in our lab with resultant lower costs?

  - Where in the operating budget, as prepared for the Second Quarter Review, will these costs be assigned? Can this be done within the present operating budget? If so, what items will be deleted or deferred? If not, what will be the net increase in the operating costs as a result of this program?

- Within what period of time will these costs be incurred?
  between now and Christmas

J.F. Olk

JFO/sn

c.c. Bill Wallberger (3)
Dave Marr
[redacted]

c.c. R.E.G. Davis
R.L. Cook
J.G. Simpson
L.P. Taggart

2nd/8/77.

RECEIVED
AUG 02 1977
FARO
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To All Concerned:
Reviewed above by telephone Aug. 2nd with WW & J.D
Requested that J.D, in his official capacity as project co-ordinator convene a meeting with all concerned to formulate a response to the questions above.
Generally it is not practical to use our own assay facilities. Also no costs have been included in the budget but can be shown in our 3rd Qtr review of 1978 PLAN.

Cyprus Anvil Mining Corporation — Interoffice Correspondence

To: A. von Kursell — Date: August 10, 1977

From: J. C. Devitt — Randy

Subject: LOGGING AND RE-ASSAYING OF DIAMOND DRILL CORE

In response to J. F. Olk's memo of July 28, a meeting was convened August 8 to finalize details of the project. Present were J. Devitt, H. Hong, J. Mustard, R. Lopaschuk, P. Clarke, B. Ferguson, B. Smith and D. Garries. A memorandum was prepared by B. Ferguson and D. Garries answering many of Mr. Olk's queries, and is attached. In summary, replies to Mr. Olk's queries are:

1) Chemical Analysis List:

Required (see attached) are Pb, Zn Ag, Cu, S, Py, Po, BaO, Mn, C and S.G. Cost per sample done at Kamloops Research and Assay Lab would be $2.86 per element, except BaO @ $8.08, C and S @ $5.00, and S.G. @ $3.50.

Total cost will be reduced through compositing of samples. Each five foot sample will be analyzed for Pb, Zn, Ag and Cu for a total cost of these elements of 2,150 x 4 x $2.86 = $24,596

Other elements will be run on a 20 foot composite. Costs for these will be:

Cost/Sample:		
	S	$ 5.00
	C	5.00
	S.G.	3.50
	BaO	8.08
	Py	2.86
	Po	2.86
	Mn	2.86
		$30.16

Total Cost = 2,150/4 x $30.16 = $16,211

2) No x-ray diffraction required. Microprobe work required on composited samples (see attachment).

Cost: 2,150/4 x $8.50 = $4,569

Total analytical costs would therefore be:

Pb, Zn, Cu, Ag	$ 24,596
Other Analytical	16,211
Microprobe	4,569
Total	$45,376

3) No possibility of running analytical work in our lab (see attached).

4) Analytical work can be accommodated in 2nd. Quarter uncommitted funds in the plant testing budget (31141) - see attached. Personnel costs are covered in the 2nd. Quarter Review Engineering Department operating budget - salaries (35210). Data processing can be covered under Engineering Department Outside Services account (35219) presently uncommitted funds. In short, the entire project can be accommodated in the already existing 2nd. Quarter Review Budget without increasing operating costs.

5) Time Frame - Analytical costs will be incurred in the 3rd. Quarter and 4th. Quarter 1977. Data Processing costs will be incurred in the 4th. Quarter 1977 and first half of 1978. Personnel costs will be evenly distributed through the next year.

J. C. Devitt
Chief Mining Engineer

JCD/mm

cc. D. Marr
B. Ferguson
D. Garries
W. Wallinger
J. Mustard
R. Lopaschuk
P. Clarke
B. Smith
H. Hong

To W. N. Wallinger

Date 9th August 1977

From R. B. Ferguson, L. D. Garries J. DEVITT

Subject Relogging and Reassaying of Diamond Drill Core

In response to queries by J. F. Olk:

In general, it should be noted that if this is not done now, it will never be done, something that could be very much regretted in future. The objective is to increase forecast accuracy and reduce surprise frequency.

1. Chemical Analysis

a.) Lead and Zinc

b.) Silver

Referring to recent experience with declining silver values in lead concentrate causing severe surprise and consternation, it is necessary to have accurate silver reserve models to forecast cash flows.

c.) Copper

With the existing circuit, copper values in feed in excess of 0.15% cause considerable difficulty. Copper will normally deport to zinc concentrate causing considerable grade reduction as well as havoc with the customers electrolytic recovery operation. The alternative is to promote copper recovery in the lead circuit which maintains a relatively copper free zinc product but at the expense of lead recovery and subsequent elevated lead in zinc concentrate.

Preliminary testwork indicates that modifications to the process might extricate us from this dilema, thus allowing production of either a copper rich lead concentrate or a separate copper concentrate with no loss in lead recovery.

However, in order to prepare a cash flow for any capital project to achieve this objective, we must have an accurate copper reserve, which makes this analysis mandatory.

d.) Sulphur

If this analysis is available, a metal sulphide balance can be determined. With that information available, the non sulphide gangue levels can be calculated. Non sulphide gangues

generally contain troublesome talc which is a lead concentrate diluent and a large consumer of reagents. If we determine that there could be large volumes of talcose material, then a program could be launched to alleviate future difficulties with it.

e.) Pyrite, Pyrrhotite

Total iron is the sum of these. Generally, comminution becomes more difficult as total iron increases. In order to predict mill throughput more accurately, or to determine if more grinding capacity may be required, a total iron analysis is required.

Pyrrhotite is a much more reactive form of iron sulphide than pyrite. It oxidizes very readily and as such as quite susceptible to collector attachment. Pyrite/pyrrhotite ratios less than 3:1 have historically been associated with concentrate grade problems from excess iron dilution.

There does not appear to be a solution, but forewarning would allow more accurate concentrate grade prediction.

f.) Barite

If barite levels are to continue at present low levels, then we can relieve ourselves of any concerns about the present non operable status of the barite rejection circuit. However, if barite levels similar to those of 1973 and 1974 experience are forecasted by this program, then we had best start acting.

Besides reactivating the barite rejection circuit at a cost of about $40,000, we could also look into a barite recovery project if we have an accurate barite reserve model.

g.) Manganese

This is a diluent in zinc concentrate. In the interest of more accurate grade forecasting, we should have this. Metallurgically there is nothing that can be done.

h.) Carbon (as graphite)

This is a diluent in lead concentrate. It also is a voracious consumer of collector. Presently some testwork is being done with graphite depressents both by us and by Lakefield. Even if this work is not fruitful, forecasting of graphite levels would enable more accurate lead grade and collector consumption prediction.

2. Visual Analysis

This can be qualitatively performed by the geologist during logging.

3. Other

a.) Microprobe analysis

This is the only means by which to determine the extent of iron dissolution within the sphalerite crystal lattice (marmatite). If levels of marmatitization are shown to continue at or near experience levels, then there is no need for concern. If, however, there turns out to be large volumes of marmatitc sphalerite ahead of us, then there are two alternatives which have to be considered, accept either reduced zinc grade or recovery, or install a zinc leach plant at considerable expense.

Approximate cost for this work is $7 to $10 per sample which is rather pricey. In order to reduce the total cost, the number of samples could be vastly reduced by compositing.

b.) X-ray diffraction

This is probably of the least value as it yields information of academic interest only. It could be dropped.

4. Analytical Work

After doing the following calculation, I feel it would be impossible for the assay laboratory to cope with the extra work load these samples would create.

Analysis per sample (Pb, Zn, Cu, Mn, Ag, Py, Po, BaO):		8
Total samples		2,150
Therefore, total analysis	2,150 x 8 =	17,200
Number of days available for analysis to December 1st, 1977, assuming four (4) days per week to work on these samples is		61
Therefore, number of samples per day is	2,150 ÷ 61 =	35
or number of analysis per day	35 x 8 =	280.

From now until the end of January, my total staff is reduced to five assayers due to holidays. One assayer is used in the water laboratory and one assayer is used to maintain and program the x-ray. Therefore, only three assayers are available to the assay laboratory. These assayers are used to provide the basic routine services to the user departments, and would only be available for a maximum of two hours per day to work on the reassay program if all goes well.

Moving to the new laboratory will probably reduce the laboratory staff by one assayer for the month of October in order to set up the new laboratory. I would, therefore, advise that these samples be sent out for assay.

5. Costs

The plant testing budget (31141) has, as of the second quarter review, $217,000 for 1977 of which $58,000 has been discharged as of June 30. This leaves $159,000.

Referring to the 1977 Budget, the following changes have been made:

1. Re-routing zinc retreat tails to zinc rougher feed

   Reagents : (354,053)
   Mechanical, materials and labour : (30,000)

   After due consideration of the dubious benefits against the demand this project would place on our maintenance group, it was postponed.

2. Test of automatic level control on one bank each of rougher/scavenger and cleaners

   This is still a very high priority project but progress has been much slower than expected. It appears that only one unit will be tested this year.

   Reduced by: (15,000)

3. Test of on-line moisture analyser on dryers (rental)

   We may still be able to complete this by year end.

   No reduction.

4. Testing new types of screen decks, crushing plant

   Open circuiting the tertiary crushers has eliminated the need for this program.

   Reduced by: (5,000)

5. Miscellaneous circuit changes, mechanical

   Have spent $3,000 on lead regrind, and expect to spend at least another $3,000.

   Reduced by: (6,000)

6. On-line samples
   Probably will be unable to make any progress this year.
   Reduced by: (24,000)
7. Grinding media tests
   No change.
8. Testing process control schemes on grinding circuits
   No change.
9. Testing froth control schemes on concentrate thickeners
   No change.

Total reduction from 1977 plan	434,053
Still committed	25,000
Expenditure to date	58,000
Total	83,000
Second quarter review	217,000
Uncommitted	134,000

Since this program will assist in promoting the efficient operation of the Anvil mill, its objectives are in line with the philosophy of a plant testing scheme.

It is, therefore, recommended that the costs related to the program be collected in the plant testing account.

R. B. Ferguson
Chief Metallurgist

cc: R. Smith; W. Muir; M. Holm; D. Garries; J. Devitt; J. Mustard

/peg

Cyprus Anvil Mining Corporation — Interoffice Correspondence

To: L. P. Taggart — Date: July 13, 1978

From: P. J. Brown

Subject: Phase V Ore Test Program

During the next three months the laboratory and plant test program involving Phase V ore will be carried out. Since the program execution will necessitate the cooperation of several departments, a meeting has been scheduled for Wednesday, July 19, at 10:00 a.m. in the Conference Room, to inform you of some of the program details.

At the meeting we intend to review the results of the test drilling and assays on the Phase V ore, with particular reference to possible mining sequence alternatives.

Peter

P. J. Brown
Consulting Metallurgist

/klc

cc: W. N. Wallinger
R. C. Smith
R. B. Ferguson
P. Clarke
G. Chapman
W. Muir
A. Clifford
J. Devitt
D. Marr

Cyprus Anvil Mining Corporation | Interoffice Correspondence

To: R. B. Ferguson | Date: July 26, 1978

From: P. J. Brown

Subject: Testing Phase V Ore

Test commenced Monday, July 24 on Phase V ore samples obtained during test drilling. The test procedure is presently based on the Anvil standard procedure plus some tests utilizing a fairly heavy dose of sodium sulphite to aid zinc-iron separation. To date we have completed preliminary work on the 3730 samples which are a fraction below cutoff at about 4.35% combined; work on 3710 samples is proceeding. I anticipate the first assay results of this work will be available for discussion toward the end of next week.

I expect that the preliminary testing phase will be complete by August 8 or 9 and that provided the laboratory regrind apparatus can be activated we can then start on cleaning tests.

Peter.

P. J. Brown
Consulting Metallurgist

/klc

cc: W. N. Wallinger
R. C. Smith
L. P. Taggart
W. W. Muir
B. M. Crawford
Met. Techs (4)
Flotation Operators (1)
Mill Shifters (1)
P. Clarke
J. Mustard
L. Hyde

CYPRUS

To B. Cron

From J. G. Simpson

Date June 27, 1980

Subject Drill Core for Metallurgical Tests from 1980 Exploration

Copy to R. E. G. Davis
R. C. Smith
J. C. Carrington
J. Purkis ✓
P. Clarke
W. Muir
L. P. Taggart

I have, somewhat belatedly, been handed a copy of your memo dated 7th March on the above subject, addressed to Hanson and Mustard, c.c. et al. Your request and reasoning highlight the alarming gap in understanding between Exploration and F & D ideas and objectives on mine models and predictive data, and those of the Production and Metallurgical groups. I had hoped that numerous previous discussions with all Departments concerned might have lessened the gap, and I understand that quite recently metallurgical testing on an ore type basis, using different grades, was being done on Zone 3 material. These data, promised some two years ago, will hopefully be used as a control alongside the "bench test" metallurgical data to establish the validity of the currently-available ore type (facies) and grade mine models. This is the only way that the models will be convincingly tested, and essential if we wish to use similar models to optimise the development and recovery of the Vangorda Plateau Deposits. This work would be the single most productive input that the Faro metallurgical group could provide.

It is an incontrovertible fact that widely-spaced vertical intercepts of ore would be representative of a large area of a horizontal bench only if the ore in the bench were either:

(a) reasonably homogeneous as was the case with Faro No. 1 orebody, or

(b) if the layering within the ore were itself horizontal.

This is decidedly not the case in Faro No. 3, No. 2, or any orebody in the Vangorda Plateau.

A tremendous effort has been expended by Exploration, together with Daryl Hanson and Peter Clarke, to develop a usable mine model that can be applied to all the Anvil area orebodies. We understand the Faro models are in use for tonnage and grade predictions. The model should also be thoroughly tested for predictive metallurgical purposes,

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particularly as the quality of data input is far superior to the old "bench sample" which has no relevance to the geometry and distribution of ore types in the known orebodies. The only alternative is to drill the deposits to death at a spacing and cost that I am sure is not contemplated.

With regard to your request for Exploration metallurgical samples, we do carry out metallurgical tests using assay reject samples, via commercial research labs under controlled conditions. These provide preliminary data on new deposits and can be made available on a confidential basis on request. The test material is in limited supply and under no circumstances would I allow premature destruction of half-core. It is the only permanent record of any given deposit in existence and often the last resort - witness most of Faro Zone No. 3 geological data and re-assays were taken from retained split core and sample rejects. Samples for F & D metallurgical testing are taken more specifically, but still undergo multiple tests with multi-element analyses for Hg, Au, Mn, Cd, Ba, etc., in addition to the more usual base metal assays. Again, where this information is likely to be of use to the Faro operation, i.e. potential mill feed, there would seem no reason why these data should not be readily available to you. While respecting your interest in new deposits, it would still seem to me that solving the very present problem of Faro metallurgical prediction and control should be the primary objective of the Faro Metallurgical department, and any available test and lab time could well be devoted to this end.

J. G. SIMPSON

JGS/ck

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To	J. Purkis/F. Gay	cc. B. Cron	Assay Lab
From	P. Clarke	B. Voisey	Met Lab
Date	October 21, 1980		
Subject	ZONE 1/3 TONNAGE FACTORS		

Further to my July 21, 1980 memo concerning Zone 1/3 tonnage factors, further statistical analysis of rock type densities with grade has lead to the following revised figures.

Revised Zone 1/3 Tonnage Factors for Pb + Zn > 4.0%:

Rock	Tonnage Factor SDT/Cu. Yd.
2A	2.69
2BCD	2.93
2CE	3.18
2EF	3.65
2H	3.39
2G	3.74

-5% ?

P. Clarke

P. Clarke
Engineering Geologist

PC/mm

CYPRUS

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To J. Purkis/F. Gay cc. B. Cron Assay Lab

From P. Clarke B. Voisey Met Lab

Date August 19, 1980

Subject ZONE 2 TONNAGE FACTORS

Further to the July 21, 1980 memo on tonnage factors, a recent statistical analysis of rock type densities with grade in Zone 2 has called for a modification to earlier estimates.

As yet figures are available for Zone 2 only. The new tonnage factors for > 4.5% Pb + Zn Zone 2 are:

Rock	Tonnage Factor SDT/Cu. Yd.
2A	2.64
2BCD	2.67
2EC	2.92
2EF, 2G, 2H	3.21

These figures agree quite well with an independent study made from production statistics by F. Gay.

P. Clarke

P. Clarke
Engineering Geologist

PC/mm

END OF OUTPUT

```
LIST JCL710.V1
   1      M710V1****  FCFM.V1     MODEL.V1    PITS.V1     RESRVS.V1   1
   2         1          0
   3      **  VANGORDA MODEL, RESERVE DESCRIPTORS SET#1 1981  **
   4           9   -1    0    0    0    9    0   26    1    0    0    0    1    0
   5      MINING       RX            MINING    27/04/81
   6        PB    ZN    AG    AU  EQ1
   7       O/B    1     5   0.363   0.363
   8      5/4L    2     5   0.363   0.363
   9        4G    3     5   0.243   0.243    0.01     3.0      3.5      4.0      4.5
  10      4EFH    4     5   0.243   0.254    0.01     3.0      3.5      4.0      4.5
  11       4GE    5     5   0.241   0.241    0.01     3.0      3.5      4.0      4.5
  12       4CE    6     5   0.268   0.260    0.01     3.0      3.5      4.0      4.5
  13      4BCD    7     5   0.282   0.289    0.01     3.0      3.5      4.0      4.5
  14        4A    8     5   0.331   0.339    0.01     3.0      3.5      4.0      4.5
  15      UND.    9     5   0.363   0.363
```

M³/Mt. ORE

M³/Mt. WASTE

ROCK	ORE S.G.
5/4L	2.75
4G	4.11(5)
4EFH	4.11(5)
4GE	4.15
4CE	3.73
4BCD	3.55
4A	3.02

To See Distribution

Copy to J.G. Simpson
L.P. Taggart
J. Purkis
R.S. Tolbert
J.F. Olk
D.S. Jennings
M.O. Hampton

From B.V. Hall/P.I. Clarke

Date October 26, 1981

Subject

Specific Gravity Study on Faro and Dy Core

In reference to a memo dated June 26, 1981 by B.V. Hall, the results of the study conducted by D.M. Wyslouzil of Lakefield Research are now in hand. As predicted, the whole core specific gravities are significantly less than both the -10 and -100 mesh samples, due to the destruction of intergranular voids upon crushing.

Unfortunately, laboratory imprecision on the behalf of Lakefield Research has rendered this data unsatisfactory. The reason being is the standard deviation (4.01 $\pm$ 0.08) for the whole core based upon 10 replicate determinations on the same piece of core was too high. When three standard deviations are considered each side of the mean (4.01), which is the minimum confidence level (97%) acceptable for engineering work, the range ($\pm$ 0.24) encompasses the difference obtained between the different methods (Appendix 1). In other words, the range of values obtained by water immersion method on the whole core is greater than the difference between the values of the whole core versus the -10 and -100 mesh air pycnometer determinations. The problem results from the methodology used by Lakefield Research, in that a water displacement method was used. In this method the water level can only be read to $\pm$ 2.5 ml with the sample displacing approximately 160 ml of water, consequently a large standard deviation is inherent in the method. This only became apparent the other day when the final report was received from Lakefield Research. Gravimetric methods, using a Jolly balance, are known to be much more accurate than the water displacement method that was employed. Unfortunately, the procedure specified by Cyprus Anvil called for the core to be crushed to -10, then -100 mesh fractions after the specific gravities were determined. This means that we have no sample left and tha data will have to be collected over again on a new set of samples. Because of our past problems, it has been decided to use a facility like U.B.C. or B.C. Research. Preliminary work will soon be undertaken to determine what analytical procedure is best and once this has been determined the study will be repeated.

Appendix 1 shows the results obtained from 38 samples of various ore facies from the Faro Deposit. The difference is evidently greatest in the buckshot facies (approximately 9%) of massive sulphides and generally less in other rock types. This is what one would expect on the basis of the number and size of intergranular voids.

... 2

For the DY Deposit, based upon 52 samples, an average difference of 4.5% was attained, with no large discrepancies evident between the different ore types.

As previously mentioned, the quantity of these differences by deposit and rock type have yet to be determined accurately.

However, based on currently available information, the average differential appears to be approximately 5%.

At the present, the affect on current reserve estimates would be:

Faro - No change. Current estimates include -5% on tons. Difficulty with tonnage estimation was considered probable.

Vangorda - Tonnage reduction of 5%.

Grum - No change. Currently used Kerr-Addison estimates, the best available presently, are probably conservative to the degree that no change is necessary.

DY - Tonnage reduction of approximately 4.5%, which translates to a net loss of approximately 1 million tonnes.

B.V. Hall

P.I. Clarke

APPENDIX 1

Test Results - Lakefield Research

Sample No.	Core No.	A Core Bulk Density g/cm^3	B S.G.(Beckman) ACP -10 Mesh	C S.G.(Beckman) ACP -100 Mesh	Difference (C-A)/C	Rock	Av. % Difference By Rock Type	No. of Samples
1	75-3-643	2.76	2.87	2.86	0.035			
2	75-3-658	2.70	2.88	2.86	0.056	Graphitic Quartz		
3	75-9-560	3.30	3.32	3.34	0.012			
4	75-9-627	2.87	2.99	2.97	0.034	2A	3.3	7
5	75-2-620	2.88	2.99	3.01	0.043			
6	75-11-480	2.78	2.85	2.87	0.031			
7	75-11-487	3.30	3.39	3.40	0.029			
8	75-2-609	4.31	4.49	4.50	0.042	Pyrrhotitic M.S.		
9	75-2-534	4.35	4.47	4.48	0.029			
10	75-11-520	4.32	4.38	4.40	0.018	2H	3.5	4
11	75-2-530	4.26	4.47	4.50	0.053			
12	75-9-359	3.32	3.46	3.47	0.043			
13	75-3-369	3.20	3.55	3.50	0.086	Pyritic Quartz		
14	75-4-215	3.06	3.37	3.41	0.103			
15	75-5-591	3.58	3.69	3.71	0.035	2CD	5.1	8
16	75-3-536	3.25	3.21	3.22	0.009			
17	75-11-495	3.47	3.65	3.65	0.049			
18	75-5-392	3.39	3.49	3.49	0.029			
19	75-5-404	3.45	3.64	3.66	0.057			
20	75-5-330	3.46	3.50	3.48	0.006			
21	75-4-306	4.51	4.70	4.72	0.044			
22	75-4-290	4.68	4.70	4.74	0.013			
23	75-2-566	4.56	4.65	4.65	0.019	Pyritic M.S.		
23	75-9-390	4.72	4.74	4.74	0.004			
24	75-9-352	4.74	4.82	4.83	0.019	2E	3.3	9
25	75-2-600	4.62	4.89	4.88	0.053			
26	75-5-427	4.03	4.29	4.32	0.067			
27	75-11-500	4.35	4.51	4.53	0.040			
28	75-9-536	4.44	4.61	4.62	0.039			
29	75-9-329	4.64	4.70	4.72	0.017	Barite M.S.		
30	75-9-351	4.72	4.76	4.78	0.013			
31	75-2-587	4.59	4.71	4.74	0.032	2G	2.6	4
32	75-9-330	4.55	4.73	4.76	0.044			
33	75-11-550	4.46	4.66	4.69	0.049			
34	75-2-536	4.62	4.76	4.78	0.033	'Buckshot' M.S.		
35	75-2-540	4.59	4.80	4.83	0.050			
36	75-2-571	4.48	4.94	4.98	0.100	2F	9.2	6
37	75-2-550	4.03	4.75	4.76	0.153			
38	75-2-520	4.06	4.87	4.88	0.168			

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

CMC # 6 (D-2)

5631

To R. Tolbert cc. J. Purkis

From J. Keir D. Gregoire

Date February 1, 1982 S. Chmelyk

Subject CALCULATION OF TONNAGE FACTORS FROM Fe EQUIVALENTS

The tonnage determination for an ore blast in the pit was calculated using an average tonnage factor for the bench. However, the inhomogeneity of the orebody resulted in the over-estimation and under-estimation of the tonnage. The solution was that we required tonnage factors that "fingerprinted" the blast in the same manner as the Pb, Zn and Fe assays. A graph was established using daily milled specific gravities vs. the Pb, Zn, Fe assays. The assays were grouped as Fe equivalents using the following formula:

Fe Equivalents = % Fe + (1.5 x % Pb assay) + (0.8 x % Zn assay)

The constants 1.5 and 0.8 were determined when the specific gravities of galena and sphalerite were normalized to the specific gravity of pyrite.

A curved line was the best fit through the 234 sample points, the results of which are summarized in the following chart.

Fe Equiv.	T.F.	S.G.	Fe Equiv.	T.F.	S.G.	Fe Equiv.	T.F.	S.G.	Fe Equiv.	T.F.	S.G.	Fe Equiv.	T.F.	S.G.
1			11	2.46		21	2.65		31	2.99		41	3.59	
2			12	2.47		22	2.68		32	3.03		42	3.68	
3			13	2.49		23	2.72		33	3.08		43	3.77	
4			14	2.50		24	2.75		34	3.12		44	3.87	
5	2.36	2.80	15	2.52	2.99	25	2.78	3.30	35	3.16	3.75	45	3.96	4.70
6	2.38		16	2.54		26	2.81		36	3.23				
7	2.39		17	2.56		27	2.85		37	3.30				
8	2.41		18	2.58		28	2.88		38	3.36				
9	2.42		19	2.60		29	2.92		39	3.43				
10	2.44	2.89	20	2.62	3.11	30	2.95	3.50	40	3.50	4.15			

Note: Tonnage Factor = S.G./1.1865

The range of specific gravities and the error of estimation for the different Fe equivalents are as follows:

Fe Eq.	Max. S.G.	Min. S.G.	Diff. S.G.	Diff. T.F.	Error of Estimation T.F.
15	3.12	2.87	0.25	0.21	±0.10
20	3.23	2.96	0.27	0.23	±0.12
25	3.45	3.14	0.31	0.26	±0.13
30	3.69	3.33	0.36	0.30	±0.15
35	4.00	3.58	0.42	0.35	±0.18
40*	4.40	3.90	0.50	0.42	±0.21
45	4.80	4.27	0.53	0.45	±0.22
AVERAGE			0.37	0.31	±0.15

* For example, a sample having an Fe equivalent of 40 would have a T.F. of 3.50 ± 0.21

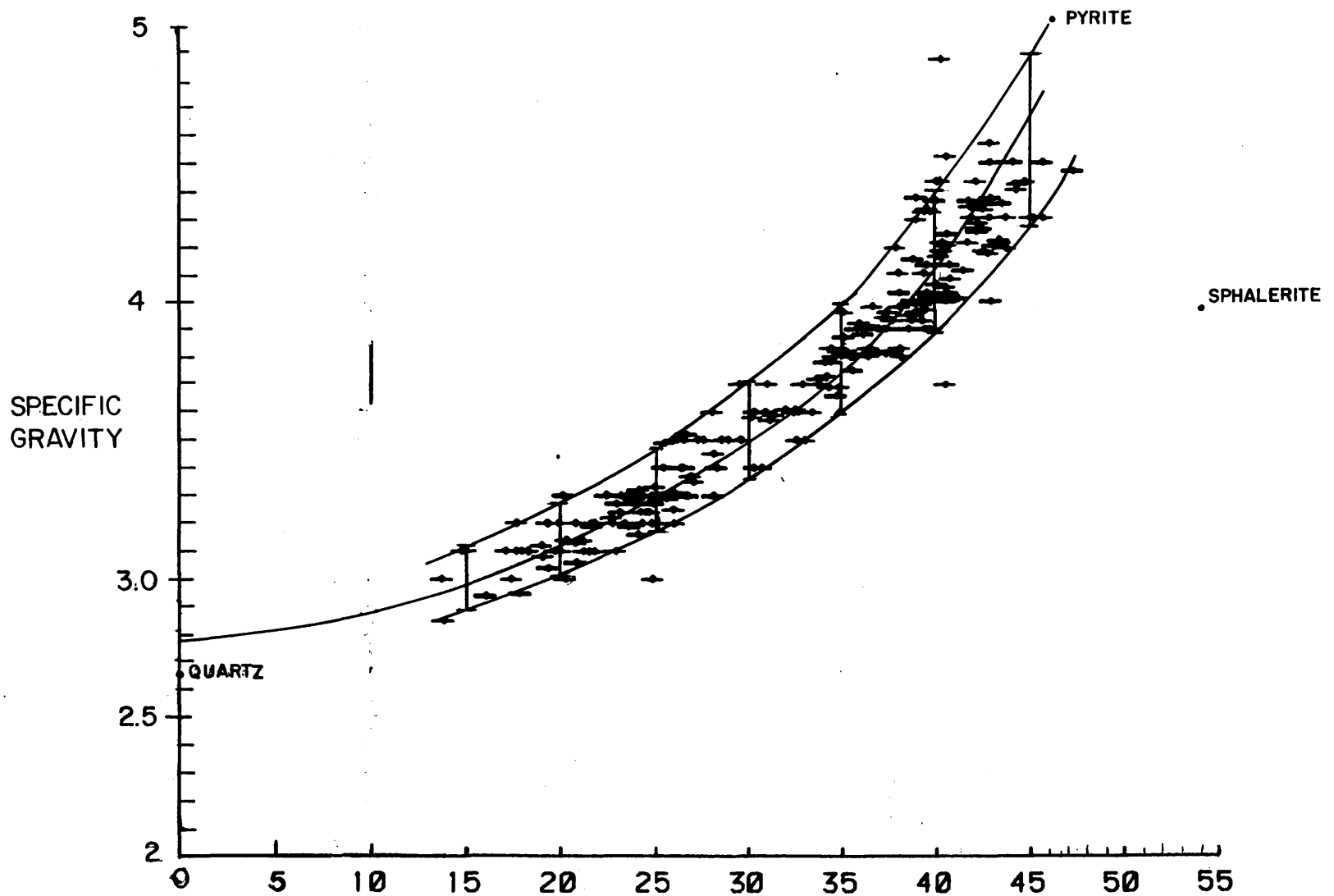
Tonnages for individual ore blasts will now be calculated from tonnage factors derived from the Pb, Zn and Fe assays according to the Fe equivalent formula.

Fe equivalents = % Fe + (1.5 x % Pb assay) + (0.8 x % Zn assay)

Jim Keir

J. Keir
Mine Geologist

CR TO CONTINUE SPECIFIC GRAVITY VS Fe EQUIVALENTS



To

From R.S. Tolbert

Date June 15, 1983

Subject STATUS OF GEOLOGIC WORK IN THE ANVIL DISTRICT

INTRODUCTION

This report will briefly cover the status of, and work required to be completed, in the various deposits as follows:

- Faro - status of work
  - work to be completed in short term
  - work schedule
  - longer term work to be carried out
- Grum, Dy, Vangorda - status and work to be completed (covered by G. Jilson).
- Considerations with respect to decisions on scheduling of work.
- Diamond Drilling

FARO - STATUS OF WORK

The geologic reserves at Faro that have been used for mine modeling purposes up to this time were derived from a gelogic interpretation carried out in 1978 based, to a large degree, on 280 foot spaced drill holes.

Since 1978, due to short falls in waste stripping at the mine, it was decided to change the mine plans in order to avoid an ore gap. It was therefore decided to access ore in the shallower Zone 3 area.

From 1980 to 1982 additional holes have been drilled in Zone 3 from Sections 124 to 135.

In 1981 with the additional drill hole data a preliminary interpretation was completed for the initial phases of Zone 3.

The reliability of the reserve figures from this model was thrown into some doubt during 1982 when it was shown that the predicted tonnage and grade was significantly different from the tonnage and grade mined in the initial ore benches. It was determined that the discrepancy was in large part due to an incorrect interpolation within the computer modeling process at the margin of the deposit.

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Arctic Star Printing Inc

Faro - Status of Work (Cont'd)

However it was also decided after examining the core of several drill holes that the structure and some ore lithologies had to be re-evaluated.

So from October to December 1982 all the core between cross sections 124 to 135 was re-examined and appropriate changes made as follows:

- The ore lithologies were standardized with respect to assays as well as visual logging.
- Discontinuities (faults, etc.) were logged more assiduously.
- Post D-2 features were included in the structural logs.
- The assay logs were updated to the 1981 format to include the ore types being assayed.
- The drill hole surveys were checked and marginal drill holes were included in the interpretation.

All this information was checked, corrected and entered in the Faro diamond drill hole data base. Presently the data is considered to be > 90% correct.

New developments in computer plotting capability have allowed:

- Plotting of plunge and trend corrected drill hole traces.
- Plotting of all drill holes onto a particular section.

New cross and long sections were constructed and geologic interpretation commenced in January 1983.

The interpretation was completed in early May 1983, so that now we have geologic interpretations for cross sections 124 to 131 and long sections 14 to 24.

In the next stage from May to June 1983 the sections were digitized to enable computer plotting of geologic contacts on bench plans. The first of these have just been received and interpretation has commenced.

The next step is complete - the geologic interpretation of the 30 bench plans and then digitize them. This digitizing of bench plans will enable computer block coding of the benches.

As part of the digitizing process towards the mine model, a new code system was developed to enable more detailed definition or ore types. Once the block coding has been completed a manual check will take place to insure that the coding is reasonably correct and that the best block size has been chosen. The coded blocks will be entered into the Mintec Mine Modeling system and will have grade and tonnage interpolated into them. After the interpolated blocks have been checked a geologic reserve will be generated. This will be completed by late August 1983.

Faro - Status of Work (Cont'd)

The other import[illegible]nt part of the geologic interpretation has been, and will continue to be, [illegible]he geologic mapping of the pit.

In 1981-82 geolo[illegible]ic mapping was carried out in Faro Zone 1, 2, and 3. This was the first time g[illegible]ologic mapping has been carried out at Faro since 1974. This is a very import[illegible]nt ongoing process since the mine model is based on a relatively few data points. It is therefore imperative at the mining stage to keep the pit mapping up t[illegible] date in order to effect changes to the mine model where necessary so tha[illegible] correct ore definition is assured.

A standardized m[illegible]pping grid base has been set up for Faro at a scale of 1" = 50 feet. A[illegible]l bench plans, blast hole plans, mine model plans, status maps and geologic map[illegible] will be set to this standard grid system.

SHORT TERM SCHE[illegible]LE OF WORK

Faro

June - August 31 1983	- Completion of bench plan interpretation, sections 124 to 133
	- Completion of geologic reserve calculation, sections 124 to 133
	- Mineralogical study of ore to determine Cu and the mineralogy at Faro
	- Pit mapping
September - Nove[illegible]ber 1983	- Relog core for sections 117 to 123
November - Decem[illegible]er 1983	- Check and correct DDHDB for above relogged holes
January - April [illegible]984	- Grade distribution, geostatistical studies Faro
	- Research and familiarization with Anvil District
May 1 - July 15, 1984	- Diamond drilling sections 117 to 123 (14,000')
June 1 - August [illegible]984	- Logging, assaying, entering above acquired data to DDHDB

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Short Term Schedule of Work - Faro (Cont'd)

September 1984 - January 15, 1985	- Plotting, interpretation of sections, bench plans
January 15 - March 31, 1985	- Final compilation 117 to 123 intergration with sections 124 to 133
April 1 - May 1, 1985	- Mintec modeling. Final reserve figure for Faro. - Continuous geologic mapping at Faro

LONG TERM WORK

Faro
- Drill define reserves in ramp area and northeast wall Zone 3 - 5,000'.
- Drill define underground reserves southwest of Zone 3 - 8,000'.
- Determine exploration potential of area immediately to northwest of deposit.

Grum
- Drill problem areas (G. Jilson).
- Define u/g potential northwest of Grum, sections 92W to 128W, ie) framework deposit - 60,000 feet - 40 ddhs.
- Dumpsite holes - 9,000' - 6 holes.

Vangorda
- Drill define southeast area - 5,000'.
- Dumpsite holes - 6,000' - 4 holes.

Dy
- Minor exploration southwest of deposit.
- Pilot hole for shaft.

Swim Basis and Other Exploration - G. Jilson.

.../5

Long Term Work (Cont'd)

Considerations With Respect to Scheduling of Work

- Confidence limits on reserves of each deposit, ie) state of the geologic knowledge.
- Confidence limits on figures used in financial planning.
- Ten years of open pit reserves (see Table 1).
- Five years to develop u/g mine.
- MAPR84-30 extended; assumes April '84 start-up - Faro mined out by 1990.
- 1989 preproduction stripping Vangorda - 1990 Vangorda in production.
- 1989-1991 preproduction stripping Grum - 1991 Grum in production as open pit.
- 1991 start shaft on Dy - 1996 Dy into production.
- Mill rate - 11,160 m tonnes/day to 1996.
- Manpower required to complete work.

Diamond Drilling

- 1984	- Faro	- 117 to 123, 14,000' at $35/foot	= $	490,000
		*Optional Zone 3 ramp, NE wall 5,000	= $	175,000
	- Grum	- problem areas (G. Jilson)		
		- NW to Firth, 35,000' at $45/foot	= $1,575,000	
- 1985	- Faro	- *If deferred from above		
- 1986-87	- Vangorda	- SW extension, 5,000' at $45/foot	= $	225,000
	- Grum	- NW area if required		
	- Vangorda-Grum	- Dumpsite holes, 15,000' at $45/foot	= $	675,000
- 1988	- Faro	- Drill define u/g reserves, 8,000'		
		- Explore SW of Dy		
	- Dy	- Drill Dy shaft pilot hole		

R.S. Tolbert
RST/gg

CYPRUS

TABLE 1

ANVIL DISTRICT TONNAGE AND GRADE COMPILATION

DEPOSIT	RESERVES (000 Tonnes)	GRADE Lead %	Zinc %	Silver g/mt	Cutoff %Pb+Zn	SOURCE
FARO						
Geological Reserves Before Mining	57,583	3.4	5.7	-	5.0	1
Remaining Geological Reserves	33,000	3.0	4.6	36	4.0	2
Minable Open Pit Reserves	25,200	2.9	4.3	36	4.0	2
Oxide Stockpile	1,363	2.9	4.7	33	n/a	2
GRUM (62W-86W only)						
Geological Reserves *	30,781	3.1	4.9	49	4.0	3
Minable Open Pit Reserves	16,875	3.0	4.9	47	4.0	3
VANGORDA						
Geological Reserves (2W-28E)	7,080	3.4	4.3	48	4.0	5
Minable Open Pit Reserves (2W-18E)	5,189	3.4	4.2	47	4.0	5
DY						
Geological Reserves	20,267	5.7	7.0	82	9.0	6
	11,049	6.7	8.0	100	12.0***	6
SWIM						
Geological Reserves	4,750	3.8	4.7	42	6.0**	7
TOTAL						
Geological Reserves - Before Mining	120,461	3.7	5.6	-	n/a	
- Remaining	95,878	3.7	5.2	51	n/a	
Minable Open Pit Reserves	47,264	3.0	4.5	41	4.0	

* NO REFINED ESTIMATES OF UNDERGROUND GEOLOGICAL RESERVES AVAILABLE.

** INFERRED MINIMUM %Pb+Zn USED IN RESERVE CALCULATIONS.

*** FOR GEOLOGICAL RESERVE TOTAL DY 9% CUTOFF RESERVE USED.

All tonnage figures adjusted for 1981 specific gravity reductions.

All geological and open pit reserves adjusted for dilution.

SOURCES

1) Pennebaker, E.N. (1967) Report to Anvil Mining Corp. dated 3 October, 1967 reported in Cyprus Anvil correspondence to J.F. Olk from J.C. Devitt 4 December, 1975.
2) Purkis, J. (1982) November 19, 1982 Reserve Summary; Cyprus Anvil Mining Corporation in-house report.
3) Clarke, P.I. (1981) October 1, 1981 Reserve Summary; Cyprus Anvil Mining Corporation in-house report.
4) Tolbert, R.S. (1982) November 19, 1982 Reserve Summary: Cyprus Anvil Mining Corporation in-house report.
5) Rollings, R.W. (1982) January 1, 1982 Reserve Summary; Cyprus Anvil Mining Corporation in-house report.
6) Kavanaugh, P.M. (1966) Ore Reserves Calculations, Swim Lakes "A" Group, Yukon dated 27 December, 1966; Kerr-Addison in-house memorandum.

June 13, 1983

Cyprus Anvil Mining Corporation | Interoffice Correspondence

To

Date June 28, 1983

From R. S. Tolbert

Subject FARO GEOLOGICAL MAPPING

A Faro Geological Mapping Guide System has now been set up for the area around and including Zone 1 and Zone 111. It consists of 12 base maps each at a scale of 1"=50'(1:600) covering an area 2000'east (40" wide) by 1500 north (30" high) (see Fig. 1). With margins, the dimensions of the maps are 44" wide by 32" high. This will facilitate fitting into appropriate binders.

For field mapping purposed these 12 base maps have been divided into twelve 500' (10") by 500' (10") blocks (see Figure 1).

The twelve base maps will be used for recording geological mapping, status maps, blasthole plans, drill plan, mintec block model plans, bench plans, structural maps etc. and will be stored in appropriate binders.

The field maps (10' x 10") used for geological mapping will be filed by map number and letter eg. 5A, 5B etc. bench and date.

It is the geologist's responsibility to note on the field map:

- the map number and letter
- the bench
- the date of mapping
- his/her initials
- the appropriate symbols (Table 1)

Where four field maps overlap (see Figure 2) seperate maps will be constructed if necessary. These will be filed as "mixed maps" using the map number at the lower right of intersection.

Information from field maps is to be transferred to the appropriate base map as soon as possible.

Geological notes are to be paged numbered and dated. These should be filed by date in the appropriate file with the field maps.

Adhering to this system will allow geological records for Faro to be more readily accessable in the future.

R. S. Tolbert
District Geologist

RST/vm

CYPRUS

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To M. Nicholson cc D. Gregoire

From R. Tolbert

Date October 11, 1983

Subject Anvil District Geology Department - Progress Report - September 1983

Faro Geologic Ore Reserve Estimation

The completion of ore reserve estimation has been delayed due to a number of factors:

- Progress was delayed to September 21 in order to allow entry of late gold assays into the data base. This will allow estimation of total and local gold grades in Zone III.

- Since September 21 a number of problems in the interpolation program and the reserves calculation program have appeared and been resolved.

The problems occurring at each step have each been resolved and it appears that both the geologic reserves and the mine reserves will be completed during the week of October 10, the latter on schedule.

It should be pointed out that the Mintec Mine Modelling System has been modified at Cyprus Anvil's request to allow better incorporation of the particular problems encountered in this highly deformed strataform deposit.

These modifications were agreed to in November 1982 when the NA phase problem was observed by the Geology Department. The design and implementation of the appropriate modifications have been a step wise progression along some completely new paths towards the final mine model.

The innovations have caused time consuming problems but the result is a 'state of the art' mine modelling program which will serve and ultimately save time and expense on the other complex deposits in the district far better than the previous mine models available.

Gold-Copper-Silver Study

As mentioned above the gold assaying program on 853 samples has been completed. Initial calculations indicate the average gold grade in Zone III to be 0.18g Au/tonne. This is three times higher than that indicated in the 1980 Vangorda Plateau A.F.E. Furthermore, initial analysis confirms that gold values are zoned as predicted within particular ore types and areas.

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Re: Anvil District Geology Department
Progress Report - September 1983

Gold-Copper-Silver Study (contd)

This indicates the possibility of several benches of a particular ore type having higher than average gold values. Being able to predict these areas may allow better metallurgical preparation resulting in increased recovery.

Present geological studies are aimed towards definition of high gold ore types and areas.

Zone III Waste Stripping

As mentioned in the August Progress Report, the Geology Department has been compiling blasthole drilling rates in an attempt to quantify drilling and blasting problems in the calc-silicate breccia.

The penetration rate planned by the Engineering Department was 65 feet/hour. The results of the compilation indicate an average penetration rate of 42 feet/hour with rates as low as 20 feet/hour.

As a result of this information the Engineering Department and Drilling and Blasting Department has taken steps to counteract this problem with the assistance of Dr. Bill Crosby of Mining Resource Engineering Ltd.

Manpower

A number of candidates for the position of Geologist have been obtained and interviews will take place in early October.

1983 Winter Drill Program Faro

Due to potential changes in the mine plan for Faro and a potential advanced mill startup in 1984 it was deemed necessary to advance diamond drilling in areas of Zone III that might impact on these decisions.

A drilling budget was drawn up in late September for approval. The proposed program of 13,000 feet of drilling for a maximum of 33 holes is estimated to cost under $400,000.

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Re: Anvil District Geology Department
Progress Report - September 1983

1983 Winter Drill Program Faro (contd)

It is estimated that the program would take 2 1/2 months to complete with two drill rigs, providing we do not experience unduly cold weather. Thus it becomes imperative that the program commence as soon as possible. Written bids for the program have been requested from Arctic Diamond Drilling Ltd. and Caron Diamond Drilling Ltd. - both of Whitehorse.

R.S. Tolbert,
District Geologist

RST/DBC

CYPRUS ANVIL MINING CORPORATION **INTEROFFICE CORRESPONDENCE**

To Those Listed R Lopaschuk

From D. Gregoire

Date Feb. 27, 1984

Subject Diamond Drilling Program Faro Deposit

The following was decided at the February 24 Geology meeting in regards to diamond drilling for 1984:

1. The southwest underground extension will be drilled as proposed in R. Tolbert's memo of February 21, 1984.

2. The northwest Zone I extension will be drilled as proposed in R. Tolbert's memo of February 21, 1984.

3. In regards to section 117 - 123, Zone III:

   a. Ian Hall, Dome Calgary, will compile reserves of Zone III based on sections. Ian will relocate temporarily to Faro for this compilation, a two month period is estimated.

   b. If the reserves compiled by Ian Hall compare satisfactorily with the F_3 model, then no in-fill drilling of sections 117 - 123 will take place.

      The Faro Geology group would then update the F_3 model (117 - 123) to an F_4 level to match with the remaining F_4 model already compiled for phases NA, OA, and PA in view of obtaining a complete Faro pit model usable for planning, ore control, etc.

      A new time schedule for the Faro group would then be published reflecting the above changes.

   c. If the reserves compiled by Ian Hall differ significantly with the F_3 model, then in-fill drilling would be required between sections 117 - 123 and R. Tolbert's proposal would apply.

4. A maximum of three test holes between section 117 and 123 were approved and could take place regardless of (3) above.

5. The position of the Geological Technician with the Faro Geology group was made permanent.

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Re- Diamond Drilling Program Faro Deposit

6. Depending on the above points, the hiring of a temporary Geologist for the summer 1984 was also approved.

D. Gregoire

DG/DBC

DISTRIBUTION

E. Forgues
M. Nicholson
R. Lopaschuk
J. Keir
L. Pigage
R. Visagie
P. Rouleau
J. McKibbon
R. Tolbert
R. Buckley (Hall)
A. Chevalier
G. Jilson
D. Bazowski

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To Quarterly Geology/Exploration Meeting

From R. Tolbert

Date February 21, 1984.

Subject 1984 Diamond Drill Program

There is budgeted in 1984 a total of $458,000.00 for direct drilling costs. It is estimated that the drilling charges in 1984 will be $23/foot. This is equivalent to approximately 20,000 feet of drilling.

It is proposed here that drilling be carried out in three areas as follows:

S.W. Underground Extension.

It is proposed that four to six drill holes totalling 4350 to 6200 feet be completed in this area as discussed in my memo (dated February 21, 1984).

Faro Zone III 117-123

It is proposed that a total of 14,000 feet in 28 drill holes be completed between cross-sections 117 and 123. This is to complete the sections at a drill spacing of 140 feet, approximately. This is to allow completion of the F4 mine model by May 1985.

Though undesirable, drilling elsewhere could reduce the footage available to drill in this area.

NW Zone I Extension

The area NW of Zone I has been examined for open pitable ore potential.

The area is limited to the northwest by a unit 10D quartz diorite intrusion, and by definitive holes 67-35 and 67-36.

To the southwest potential is limited by depth and definitive hole 66DS-03. To the south and southeast holes 66-56 and 75456-16 limit potential.

Therefore an area of 700 feet by 800 feet peripheral to the pit has been defined as having limited potential for open pit reserves. There are six holes drilled to the northeast of this area with varying results as follows:

1984 Diamond Drill Program - (Cont'd):

DDH	Ore Thickness (Ft)	Cut off Pb+Zn %	Pb %	Zn %	Ag g/mt	Comments
65-02						Too short
66-37	15	4	3.3	6.3	48.0	Faulted
66-40	5	4	4.1	8.2	70.0	Faulted
66-42						Too short
66-45	35	4	3.5	5.5	50.0	
66-48	35	4	2.8	4.1	49.0	

It could therefore be inferred that if within the above area of 700 feet x 800 feet at least 35 feet of ore >4% Pb+Zn exsisted, with an S.G. of 3.6 then there would be a potential of up to 2 million tonnes of ore with at least 8% Pb+Zn grade.

To test this hypothesis it is proposed that from 3 to 10 drill holes be completed in this area. Dependant on results this will total 1300 to 5000 feet of drilling.

R. S. Tolbert
District Geologist

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To D. Gregoire cc: R. Lopaschuk

From R. Tolbert

Date June 27, 1984

Subject SPECIFIC GRAVITY OF WASTE ROCKS
Units 3D, 3DBx and 1D in
Solid Core vs Pulverised Core

A total of 28 samples of each rock type were individually tested for;

i) S.G. using volume displacemnt of whole core in water.

ii) S.G. using Hubbard pynometer, of same samples pulverised to -100 mesh. (See Appendix I)

Results

Rock Type	$\bar{M}$ S.G. Solid Core	$\bar{M}$ S.G. Pulverised Core	
3D	2.68	2.75	28 samples
3DBx	2.69	2.76	28 samples
1D	2.69	2.76	28 samples
Avg.	2.69	2.76	

From the above results there is clear evidence that for waste rocks, which visibly have little porosity, pulverised core S.G.'s ave. 2.6% higher than whole rock S.G.'s.

For porous ore types 2F, 2G, this will be much higher.

Additional study is required to quantify this variance of S.G. in ore types, but this study supports the reduction of S.G.'s in ore types by 5% that was used in the F-3 model and the reduction in tonnage by 5% on the Vangorda Deposit.

$1\ BCY = \frac{2.69 \times 62.4 \times 27}{2000} = 2.27\ tons$

R. Tolbert
District Geologist

RT/cg

SPECIFIC GRAVITY OF DRILL CORE

SAMPLE NUMBER	SAMPLE DESCRIPTION			SPECIFIC GRAVITY	
	ORE TYPE = 1D			OF SOLID SAMPLE	OF PULVERIZED SAMPLE
	D.D.H.#	FOOTAGE	MID BENCH		
57	82F-09	230	3810	2.80	2.71
58		270	3770	2.73	2.75
59		310	3730	2.77	2.72
60		350	3690	2.69	2.76
61	82F-05	65	3850	2.73	2.74
62		105	3810	2.63	2.76
63		145	3770	2.69	2.74
64		185	3730(1D2)	2.70	2.70
65		225	3690	2.80	2.81
66	80-02	290	3770	2.59	2.76
67		330	3730	2.66	2.70
68		370	3690	2.75	2.79
69		410	3650	2.70	2.79
70	81-09	173	3850	2.71	2.76
71		213	3810	2.71	2.77
72		253	3770	2.84	2.82
73		333	3690	2.72	2.79
74	74-08	167	3850	2.39	2.66
75		207	3810	2.81	2.69
76		247	3770	2.53	2.74
77		287	3730	2.69	2.73
78		327	3690	2.67	2.76
79	81-11	93	3930	2.63	2.70
80		133	3890	2.76	2.78
81		173	3850	2.78	2.79
82		213	3810	2.60	2.75
83		253	3770	2.72	2.76
84		293	3730	2.84	2.86
				$\bar{M}$ = 2.69	$\bar{M}$ = 2.76

SPECIFIC GRAVITY OF DRILL CORE

SAMPLE NUMBER	SAMPLE DESCRIPTION			SPECIFIC GRAVITY	
	ORE TYPE = 3D			OF SOLID SAMPLE	OF PULVERIZED SAMPLE
	D.D.H.#	FOOTAGE	MID BENCH		
1	82F-09	30	3970	2.72	2.79
2		70	3930	2.64	2.75
3		110	3890	2.92	2.93
4	77-16	35	3970	2.50	2.77
5		75	3930	2.74	2.75
6		115	3890	2.68	2.76
7	72-13	110	3890	2.48	2.70
8		150	3850	2.58	2.79
9	82F-14	62	3930	2.63	2.67
10		102	3890	2.52	2.69
11		142	3850	2.61	2.78
12		182	3810	2.70	2.73
13	82F-16	75	3930	2.64	2.65
14		115	3890	2.72	2.75
15		155	3850	2.67	2.85
16	82F-10	9	3930	2.87	2.88
17		49	3890	2.65	2.75
18		89	3850	2.89	2.82
19		129	3810	2.69	2.70
20	82F-11	79	3930	2.60	2.66
21		119	3890	2.57	2.71
22	82F-13	56	3890	2.80	2.82
23		96	3850	2.78	2.75
24		136	3810	2.58	2.69
25	82F-15	53	3970	2.70	2.74
26		93	3930	2.66	2.72
27		133	3890	2.71	2.74
28		173	3850	2.69	2.73
				$\bar{M}$ = 2.68	$\bar{M}$ = 2.75

SPECIFIC GRAVITY OF DRILL CORE

SAMPLE NUMBER	SAMPLE DESCRIPTION			SPECIFIC GRAVITY	
	ORE TYPE = 3D 'breccia'			OF SOLID SAMPLE	OF PULVERIZED SAMPLE
	D.D.H.#	FOOTAGE	MID BENCH		
29	67-09	175	4010	2.58	2.70
30		215	3970	2.92	2.88
31		259	3930	2.55	2.69
32		295	3890	2.63	2.75
33		341	3850	2.87	2.87
34	82F-02	61	3970	2.71	2.78
35		101	3930	2.64	2.72
36		141	3890	2.74	2.74
37		181	3850	2.72	2.79
38	77-09	87	4010	2.70	2.80
39		129	3970	2.66	2.69
40		167	3930	2.73	2.80
41		207	3890	2.79	2.76
42		247	3850	2.64	2.87
43	81-14	62	4010	2.57	2.71
44		87	3970	2.71	2.73
45		126	3930	2.63	2.67
46		167	3890	2.63	2.67
47		215	3850	2.64	2.70
48	81-19	96	4010	2.64	2.69
49		132	3970	2.72	2.75
50		174	3930	2.91	2.80
51		212	3890	2.61	2.72
52		251	3850	2.63	2.72
53	77-06	130	4010	2.59	2.82
54		170	3970	2.69	2.75
55		210	3930	2.67	2.79
56		251	3890	2.57	2.76
				$\bar{M}$ = 2.69	$\bar{M}$ = 2.76

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To Denis Gregoire cc: R. Visagie A. Chevalier

From Robin Tolbert R. Hogan

Date September 21, 1984 J. Keir

Subject ORE RESERVES IN FARO OXIDE STOCKPILE

It has come to my attention that the Oxide stockpile reserves for Faro reported on November 19, 1982 by J. Purkis (memo attached) are incorrect, and this error has been carried to present.

The error occured by reporting the oxide tonnage as tonnes where it should have been reported as short dry tons.

The silver grade reported in the November 19, 1982 memo was also incorrect at 37.6 g/t Ag.

The correct reserve figures for the oxide stockpile is as follows:

Tonnes	Pb%	Zn%	Ag g/mt
1,236,000	2.9	4.7	33.0

1,363,000 tons

R. S. Tolbert
District Geologist

RST/cg

CYPRUS

CYPRUS ANVIL MINING CORPORATION **INTEROFFICE CORRESPONDENCE**

To Liaison Committee Members

From R. Tolbert

Date October 18, 1984

Subject Gold Occurrences in Zone III

As follow up to discovery of significantly high gold values in assays from two holes (84F-01 and 84F-06) at the northwest end of Zone III a total of 377 pulp samples from eight surrounding diamond drill holes were sent to Chemex Labs, Vancouver for gold assay.

The purposes of these additional assays were to:

1. determine the grade and extent of the gold mineralization.

2. determine the locus of the mineralization.

To date a total of 212 assays have been received from six diamond drill holes. The significant assays are appended to this memo.

Results

1. From presently received results it appears that there is no potential for significant tonnage of high grade gold ore (i.e. > 1.0 g/mt) in favourable horizons.

2. There is still potential for low grade gold mineralization (0.2 - 1.0 g/mt Au) in favourable horizons.

3. The majority of the gold mineralization averaging > 0.2 g/mt occurred in quartzitic ore types (2A, 2B, 2C, 2D).

4. A significant number of gold values occur in mineralized quartz veins (OP9) and altered metavolcanics (IH4). This latter rock type (see 84F-19) has in the past not generally been assayed and has been considered waste.

   This rock type occurs within the ore in areas of Faro Zone III and its equivalent (5C4*, 3F4*) also occurs interbanded with ore at the Grum deposit. No extensive gold assaying has been carried out on this rock type.

. . ./2

Re: Gold Occurrences in Zone III

Recommendations

1. Gold assaying of ore samples from relogged core samples from Faro should continue.

2. Additional gold assaying of altered metavolcanics, mineralized altered metopelites and mineralized large quartz veins should be carried out.

3. The possibility of gold occurring in altered metabasites interbanded with ore at the Grum deposit should be examined.

R. Tolbert

RST/DBC
attch

L SECTION 22

DDH	FOOTAGE FROM	TO	INT.	AU(g/mt)	ROCK-TYPE
74-15	631	636	5	.64	2E8
	636	641	5	.82	2E8
	641	646	5	.89	2E1
AVG.			15	.78	
66-03	553	558	5	.27	2D0
	558	563	5	1.10	2D0
	563	568	5	.82	2D0
AVG.			15	.73	
	586	590.6	5.6	.85	2A0
	590.6	595	4.6	.89	2A0
	595	600	5	.13	2B0(1D4)
	600	605	5	.59	2B0(1D4)
	605	610	5	.82	2B0(1D4)
AVG.			25.2	.66	
84F-01	338.0	341.1	3.1	.40	2B0[2L14]
	341.1	348.2	7.1	T.B.A.	2B0[2L14]
	348.2	355	6.8	.60 >	1E19
	355	359.6	4.6	5.80 >	2L14[2D0]
	359.6	364.2	4.6	<17.56 >8.92	2L14[2D0]
	364.2	367.5	3.3	17.9<18.36 >	2D09
	367.5	372	4.5	.10	2E46
84F-19	409.2	410.5	1.3	.66	2B0
	410.5	412.7	2.2	.10	2E4
	412.7	414.7	2.0	1.65	1H4
	414.7	421	6.3	1.17	1H4
	421	422.8	1.8	.22	1H4
	422.8	425.8	3	.62	2D63
	425.8	429.3	3.5	.25	2G4
	429.3	433.8	4.5	.25	2G4
AVG			24.6	.65	
	467.4	469.1	1.7	3.33	1H4
	469.1	474	4.9	.54	2D09(0Q0)
AVG			6.6	1.26	
84F-25	404.8	410	5.2	2.19	2C3
	410	415	5	.60	2C3
AVG			10.2	1.39	

SECT119.ROBIN.GEOLOGY THU. OCT 18. 1984. 6:56 PM

X SECTION 119

DDH	FOOTAGE FROM	TO	INT.	AU(g/mt)		ROCK-TYPE
84F-01	338.0	341.1	3.1	.40		2B0[2L14]
	341.1	348.2	7.1	T.B.A.		2B0[2L14]
	348.2	355	6.8	.60 >		1E19
	355	359.6	4.6	5.80 >		2L14[2D0]
	359.6	364.2	4.6	<17.56 >8.92		2L14[2D0]
	364.2	367.5	3.3	17.9<18.36 >		2D09
	367.5	372	4.5	.10		2E46
76-13	481.5	486.5	5	< .52		2AC
	486.5	490.3	3.8	.89<1.65		2D0
	490.3	494.2	3.9	< .62		2D0
	494.2	499.2	5	.25>		2G4
	499.2	504.2	5	.26>		2G4
	504.2	509.2	5	.20>		2E4
	509.2	514	4.8	.25>	.24	2E4
	514	519	5	.14>		2E4
	519	524	5	.30>		2D0
	524	529	5	.27>		2A0
	529	534	5	< .59		2A0
	534	539	5	< .33		2A0
	539	544	5	< .43		2A0
	544	549	5	.40< .19		2A0
	549	554	5	< .49		2H0
	554	559	5	< .45		2G0
	559	564	5	< .34		2H0
	629	634	5	.26		2A0
	634	639	5	.60		2A0
	639	644	5	.25		2A0
	644	649	5	.23		2A0
AVG			20	.34		
76-14	474	479	5	.43		1D4

X SECTION 118

DDH	FOOTAGE FROM	TO	INT.	AU(g/mt)	ROCK-TYPE
66-03	553	558	5	.27	2D0
	558	563	5	1.10	2D0
	563	568	5	.82	2D0
AVG.			15	.73	
	586	590.6	5.6	.85	2A0
	590.6	595	4.6	.89	2A0
	595	600	5	.13	2B0(1D4)
	600	605	5	.59	2B0(1D4)
	605	610	5	.82	2B0(1D4)
AVG.			25.2	.66	
84F-06	255.1	259.5	4.4	.30	2C79(2H0)
	259.5	263.9	4.4	4.30	2C79(2H0)
AVG			8.8	2.30	
	289.8	294.6	4.8	5.00	0Q9(2L0)
66-46	520	525	5	.33	2G0
	525	530	5	.51	2G0
	530	535	5	.40	2G0
	535	540	5	.28	2G0
	540	545	5	.17	2G0
	545	550	5	.21	2E8
	550	555	5	1.44	2E8
AVG			40	.50	
	735	740	5	.34	2C0
	740	745	5	.16	2C0
	745	750	5	.22	2C0
	750	755	5	.15	2C0
	755	760	5	.20	2C0
AVG			25	0.21	

CYPRUS ANVIL MINING CORPORATION **INTEROFFICE CORRESPONDENCE**

To Engineering - R. Hogan/I. Booth cc: D. Gregoire R. Tolbert

From Geology - J. Keir/A. Chevalier M. Nicholson

Date September 27, 1984 N. Cornish

Subject INSITU INVENTORY FOR CALC-SILICATE (3D) AND CALC-SILICATE BRECCA (3Dbx) AS OF SEPTEMBER 5, 1984

1. Calc-silicate (3D)

Section (Geo. X-Sect)	Volume (BCY)	Phase Distribution (%)				
		WA	7D	YA	UB	NA
118 + 00	261,000	98	2			
120 + 00	883,000	82	15	3		
122 + 20	1,009,000	70	20		10	
124 + 22	941,000	100				
126 + 23	1,015,000	90		5		5
128 + 20	654,000	80		5		15
130 + 00	89,000					100
Sub-Total	4,852,000	84	7	2	2	5

2. Calc-silicate Brecca (3Dbx)

Section (Geo X-Sect)	Volume (BCY)	Phase Distribution (%)			
		7D	UB	PA	OA
122 + 20	23,000	100			
124 + 22	920,000	95	5		
126 + 23	967,000		25	70	5
128 + 20	769,000			50	50
130 + 00	74,000				100
Sub-Total	2,753,000	33	10	39	18

	Volume (BCY)	Phase Distribution (%)						
		WA	7D	PA	OA	NA	UB	YA
Total (3D & 3Dbx)	7,605,000	65	8	14	5	3	4	1

Notes:
1. The initial estimation of the calc-silicate for the Abandonment scheme II was 14,585,000 B.C.Y. (P. Nakai Feb. 1, 1982), therefore the calculated figure for what has been removed to date is 6,980,000 B.C.Y.

2. The Phase Distribution is a visual estimation with a ± 5% error per section.

J. Keir

CYPRUS

Brenda please cc D Hogan / J Bowers / J Stephens Thks Mike

Cyprus Anvil Mining Corporation — Interoffice Correspondence

To [redacted] — Copy to T. Brown, R. Tolbert, N. Cornish

From Gregg Jilson

Date February 6, 1985

Subject LEASES/COAL

Tony Brown has requested that I give you some idea of the land situation around the Faro operation and as regards coal. Here is a synopsis of the highlites:

A. Surface Leases

We have records for 4 surface leases near the Faro operation -

Lease #	Date Granted	Term	File #	(Acs) Area	Purpose	Annual Rental
1646	1/12/67	21 yrs	105K060000N0001	3700	Mine Site	$1,480.00
1690	01/4/68	21 yrs	105K060000N0002	413	Water Storage	$ 164.80
1777	1/10/68	21 yrs	105K060000N0003	158	Waste Rock Dump	$ 64.00
4945	1/1/79	21 yrs	105K060000N0004	131	Waste Rock Dump	$ 131.00

These leases require payment of annual rental, in advance, by their anniversary dates. This is currently being handled through the Vancouver office. In addition, taxes are charged on the same parcels, this is handled through Faro.

The approximate limits of these leases is shown on the accompanying diagram. There is no intention to surrender any of these leases!

B. Mineral Leases

There are 12 mineral leases covering the Faro ordebody (see attached sheet). These require no annual payments and renewal on expiry of their term is routine provided the company has complied with all provision of the Quartz Act. (i.e. any differences over royalties not settled by November 1988 could jeopardize renewal).

In addition there are 41 mineral leases on Vangorda Plateau (covering Grum and Vangorda deposits). 17 of these expired on January 28, 1985 and the process of renewal is underway. This involves filing an affadavit of compliance which has already been prepared and will be submitted this month. The leases can be renewed within a 3 month period following the expiry of their 21 year term and a rental payment of $200 each will keep them for another 21 years.

C. Carmacks Coal

The Carmacks coal area is divided into three sub areas as shown on the attached map. Only the Carmacks north area is of interest at the present time; this

is the location of the mine. There are 3 types of land parcels involved:

a) lots owned by Cyprus Anvil
b) leases in the name of Cyprus Anvil
c) leases under option by Cyprus Anvil from Teslin Explorations

There are lots in each of the three areas; as these are owned there is no annual rental but taxes must be paid. It is my understanding this is being done through Faro. Each of the clusters of lots is the site of an old coal mine. The coal resource potential of the lots is not well known to me but it seems that the product is not desireable for use at the Faro operation, thus those lots may have little more than historic value. Since the environmental/reclamation impact is also poorly known to me there will be no initiatives made from this office to dispose of these lots.

There are 4 Cyprus Anvil leases remaining in North Carmacks and 3 in South Carmacks; the remainder have been surrendered. The Northern leases cost an aggregate of $2413 per year to retain; the pit is on one of them (2959) and a stockpile on another (2955?). We plan to retain all of them. The southern leases presently cost nothing, however, as of June 10, 1986 they will cost $1,583 to retain. On the basis of undesireable product, all other Cyprus Anvil leases have been dropped in South Carmacks; presumably these three will be also since they are geologically less attractive (and less well known) than leases that have already been surrendered. These three leases have only a few cut lines on them, no roads or drill holes. Thus, they should pose no reclamation problem, certainly far less than #2949 that was surrendered last year, apparently without any major repercussions yet.

The remaining leases are the Teslin Option leases; there are 6 leases costing $3,459 per year to retain. Apparently there is a work commitment on this ground with is overdue thus Calgary has negotiated an extension of the agreement in exhcange for keeping the ground in good standing.

The target at South Carmacks was apparently a large deposit of good quality coal. If a deposit like the Carmacks North Pit were to be the target, then there is a significant change that such a deposit would remain. The most likely area, in my opinion on structural grounds, would be on 2965; 2967 and parts of 2949, 2979 and 2978 that have already lapsed. The purported low quality of the coal however renders this an academic point thus there seems to be lettle reason other than financial to continue with this agreement. Unless the required work or the penalty for not doing it is in excess of $35,000 this isn't a very good return.

Several new roads and drill holes were located on the Teslin leases, thus there may be some reclamation needed on this ground. The damage here should not be significantly worse than on lease #2949 which, as noted above, has already been surrendered (I believe without reclamation having been done).

To summarize:

a) No further reductions will be proposed for Carmacks North nor for the old coal lots.

b) Leases 2981,2982 and 2983 should be surrendered when their advance assessment runs out in 1986.

c) Teslin option should be terminated unless the $3,500 annual lease is deferring a sufficiently large expenditure and reclamation considerations are favourable.

D. Ross River Coal

There are three coal leases at Ross River. At present these leases cost nothing since there is advance work filed until April 1986. After that date, the leases will cost $1,920 each year in rental. This is apparently good quality coal, thus this ground may warrent further exploration or development.

E. Claims

Advance assessment is being filed on claims around the Faro Deposit using expenditures from the 1984 drilling.

Road work (E & L Maintenance building up haul road) is being filed on some claims needing assessment between Grum and Faro deposits.
Cash in lieu has been requested and approved for claims in Swim basin (Eacho, Capa, Sea) and Urn Barite areas.

North Anvil (Tim, Jet, Zan and Ming) and lower Anvil (Lorna, Roto, Aro) will be dropped. 3 Dana claims will be dropped.

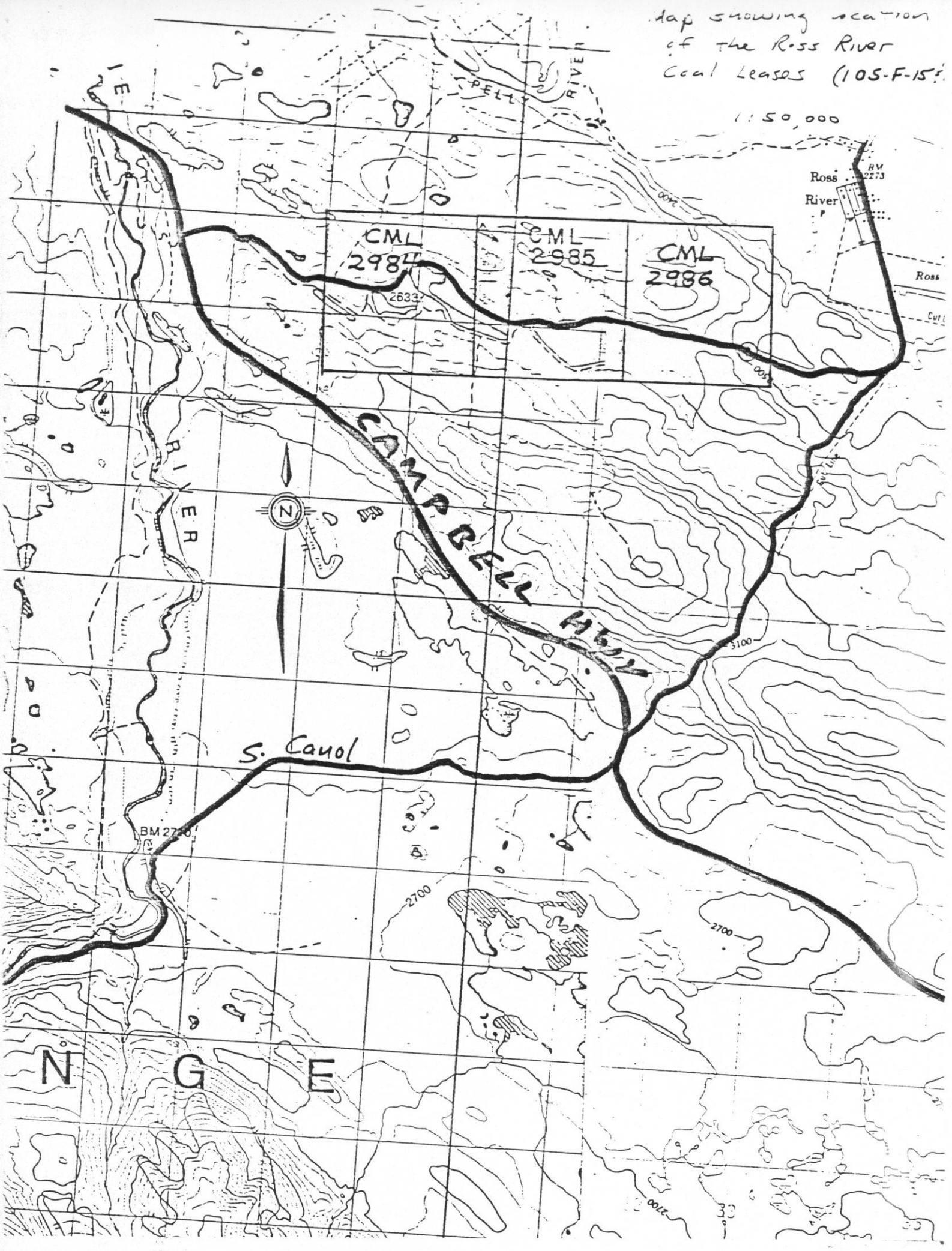
A new map of expiry dates for the Anvil District claim block will be sent to Faro as soon as the dust settles here from the recent flurry of land activity, hopefully in March.

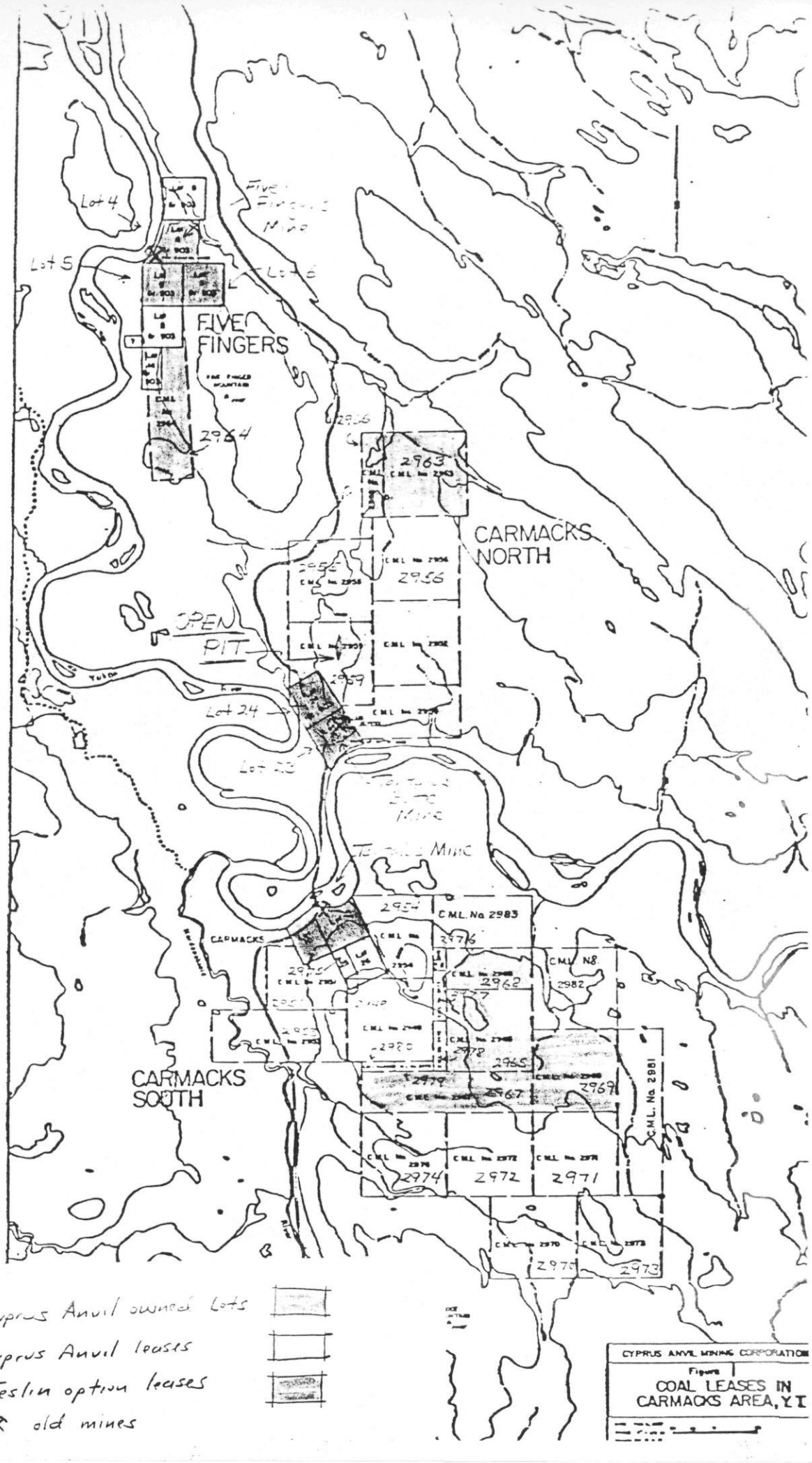
F. Limestone Leases

Cyprus Anvil had several limestone quarry leases near Ross River and Carmacks. These were kept for possible use as flux in a smelter. They were dropped last year, thus, there are no longer any limestone leases.

Gregg Jilson

GJ/lp


Map showing location of the Ross River Coal Leases (105-F-15)
1:50,000
CML 2984
CML 2985
CML 2986
Ross River
Ross
CAMPBELL HIGH
S. Canol
RIVER
PELLY
BM 2700
2700
3100
N
G
E

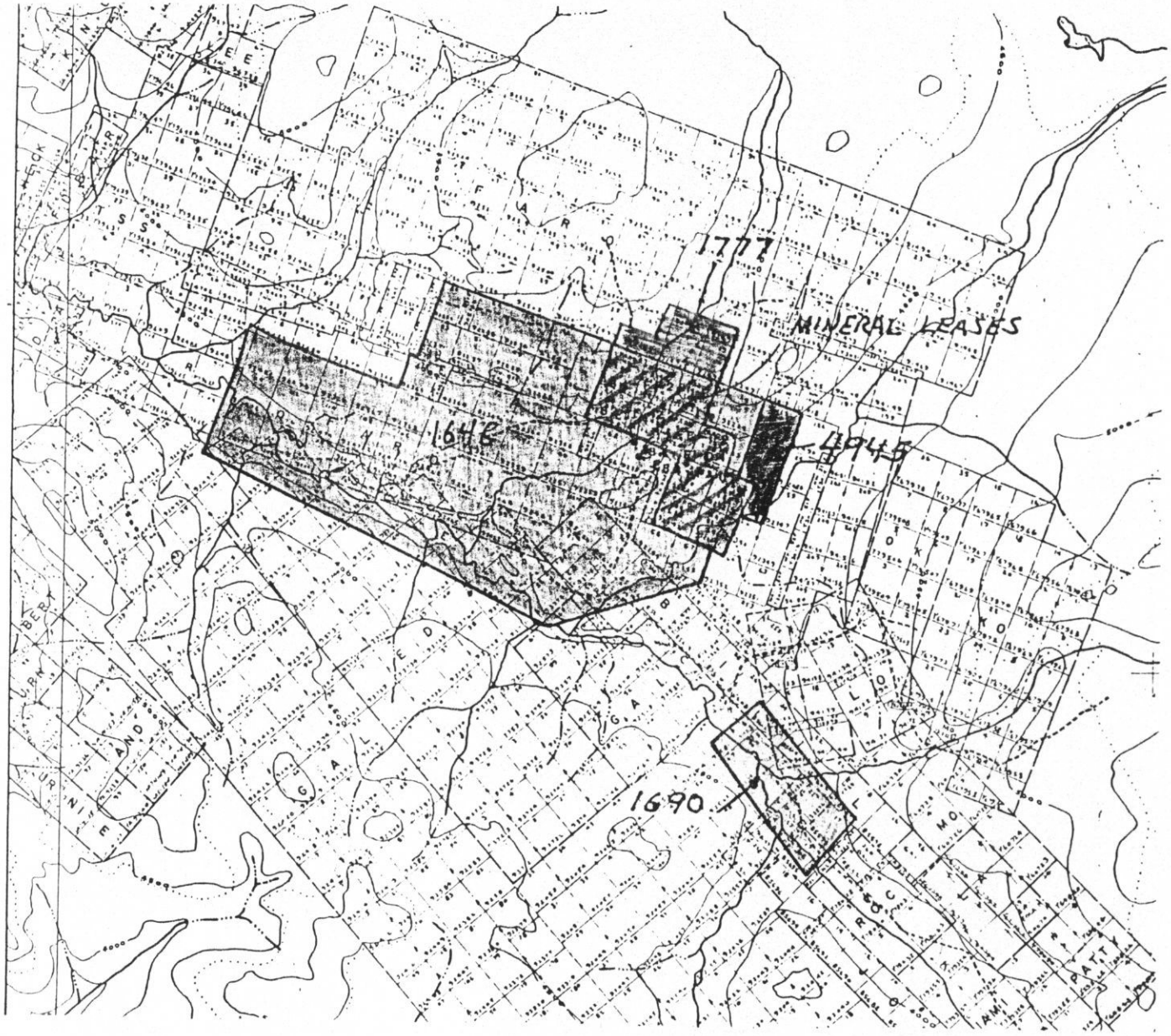

Lot 4
Lot 5
Lot 6
FIVE FINGERS
2964
2963
CARMACKS NORTH
2956
OPEN PIT
2959
Lot 24
Lot 23
CML No 2983
2954
2976
CARMACKS
2962
2982
2965
2969
2967
CARMACKS SOUTH
2974
2972
2971
2975
2973
Cyprus Anvil owned Lots
Cyprus Anvil leases
Teslin option leases
old mines
CYPRUS ANVIL MINING CORPORATION
Figure
COAL LEASES IN CARMACKS AREA, YT

RETAIN

MINERAL LEASES

(CYPRUS ANVIL MINING CORPORATION)

Lease Number			Acreage	Expiry Date	Renewal Fee
#1462	(92225)	Faro 39 - Lot 39 - Group 955 21 yrs. from November 16, 1967	52.0 acres	November 16, 1988	$220.00
#1463	(92227)	Faro 41 - Lot 41 - Group 955 21 yrs. from November 16, 1967	46.6 acres	November 16, 1988	$200.00
#1464	(92228)	Faro 42 - Lot 42 - Group 955 21 yrs. from November 16, 1967	47.0 acres	November 16, 1988	$200.00
#1465	(92229)	Faro 43 - Lot 43 - Group 955 21 yrs. from November 16, 1967	48.1 acres	November 16, 1988	$200.00
#1466	(92230)	Faro 44 - Lot 44 - Group 955 21 yrs. from November 16, 1967	49.9 acres	November 16, 1988	$200.00
#1467	(92231)	Faro 45 - Lot 45 - Group 955 21 yrs. from November 16, 1967	50.8 acres	November 16, 1988	$200.00
#1468	(92232)	Faro 46 - Lot 46 - Group 955 21 yrs. from November 16, 1967	49.6 acres	November 16, 1988	$200.00
#1469	(92239)	Faro 53 - Lot 53 - Group 955 1 yrs. from November 16, 1967	49.8 acres	November 16, 1988	$200.00
#1470	(92240)	Faro 54 - Lot 54 - Group 955 21 yrs. from November 16, 1967	47.6 acres	November 16, 1988	$200.00
#1471	(92241)	Faro 55 - Lot 55 - Group 955 21 yrs. from November 16, 1967	51.1 acres	November 16, 1988	$200.00
#1472	(92242)	Faro 56 - Lot 56 - Group 955 21 yrs. from November 16, 1967	50.9 acres	November 16, 1988	$200.00
#1473	(94573)	WHI 8 Frac. - Lot 90 - Group 955 21 yrs. from November 16, 1967	1.27 acres	November 16, 1988	$200.00



N

Cyprus Anvil Mining Corporation

SURFACE & MINERAL LEASES

FARO DEPOSIT AREA CLAIMS

84-F-62 diamond drillhole

0 1
kilometers

0 1
miles

portion of claim sheet 105-K-6