CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

006984

To

From P. Clarke

Date October 2, 1982

Subject FARO COMPUTER MODEL

During September of this year a study was initiated, in preparation for development of Phase NA, on the nature of one of several sulphide rock types occurring in the Faro deposit. This was a graphitic quartzite, known locally as 2A. See Appendix I.

As it progressed the study showed that lead and zinc grades assigned to this sulphide facies in the computer ore reserve estimation process were in excess of what would be normally expected. Further investigation has shown that the ore reserve estimation process hitherto used was not well adapted to this section of the deposit especially.

A widely used approach in the mining industry to computer mine modeling for metal deposits is 3-D block modeling and this has been used at Faro.

The part of the process most in question is that one known as "interpolation" and "extrapolation". This is the algorithm that assigns metal grades to blocks (blocks representing small parts of benches to be mined) from surrounding or nearby diamond drill hole (DDH) composite assay values. A composite assay value is calculated from length and specific gravity weighted individual assays taken from samples of several feet in length located down a DDH between bench crest and toe elevations. See Appendix 2. The algorithm used for the interpolation process did not explicitly prevent the use of a composite made up of predominantly one facies from being used is the interpolation of a block of another facies, except for non-sulphide blocks. Prior to 1982 the relatively widely spaced drill pattern made a facies selective approach difficult to apply. Instead the process relied more upon inverse distance squared (IDS) weighting interpolation to correctly assign block grades. Here the influence of any one composite in the assignment of a block grade decreases with the square of the distance of the composite from the block undergoing interpolation. Thus a composite would influence the grades of blocks near to it quite significantly and far away very little.

Prior to the above mentioned study it was not perceived that the ore reserve estimation process was greatly in error on a mining phase scale. The previous 2 yearly (1980 and 1981) tonnage and grade predictions from the computer model vs. actually mined vs. milled showed quite good agreement for a complex sulphide orebody situation. See Appendix 3.

CYPRUS ANVIL

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

CASL#6 (1-27)

5393

To J. Purkis

From R. Lopaschuk

Date October 19, 1981

Subject PRELIMINARY COMPARISON OF MODEL.L1 vs. MODEL.F3

The existing Mine Plan quantities were determined using Model.L1 mining reserves. The mine model has since been updated to include new drilling and assaying data along with some new geologic interpretation to create version Model.F3. It is important to compare this latest version of the mine model with Model.L1 to determine if the Mine Plan is still valid.

The comparison of Phase quantities between Model.L1 and Model.F3 is represented in Table 1. Overall, Model.F3 has 248,000 SDT's less than Model.L1, which is not too significant except that 176,000 SDT's of this shortfall occur before the ore gap, in Phases NA, OA, PA and 7B. The ore tonnages in Model.F3 do not indicate any major shifts in the location of the orebody in comparison to Model.L1, indicating that a continuation of our present phasing scheme would still be viable.

New approaches to the current phases will be investigated, hopefully using the Mintec "Stripper" system.

The geological reserves for Model.L1, Model.L2 and Model.F3 are tabulated below for comparison. All geological reserves are quoted as under the topography dated June 30, 1980. Obviously, ore has been removed since that time but this does not affect the relative comparisons (all with clipped composites, new tonnage factors, and 4.0% cut-off).

	Ore (SDT's)	% Pb	% Zn	Ag (g/mt)
Model.L1	39,474,656	3.0	4.6	34.7
Model.L2	40,046,124	3.0	4.6	34.9
Model.F3	39,713,476	3.0	4.6	35.2

R. Lopaschuk
Senior Project Engineer

RL/mh

Attach.

Arctic Star Printing

CYPRUS

TABLE I

COMPARISON OF PHASES (BENCHES 3910 DOWNWARD)

MODEL.L1 vs. MODEL.F3

(000's)

	MODEL.L1					MODEL.F3				
Phase	Waste (BCY)	Ore (SDT)	% Pb	% Zn	Ag (g/mt)	Waste (BCY)	Ore (SDT)	% Pb	% Zn	Ag (g/mt)
NA	6,949	2,080	2.8	4.4	38.4	6,935	2,053	2.7	4.2	40.4
OA	3,611	1,502	2.4	3.9	29.1	3,620	1,438	2.6	3.8	33.3
PA	3,417	1,767	2.8	4.6	30.3	3,427	1,682	2.9	4.5	35.1
7B	5,075	760	3.3	4.7	40.8	5,075	760	3.3	4.7	40.8
AU	7,560	94	3.5	4.2	45.9	7,551	127	3.0	3.9	39.0
AV	2,058	1,013	2.4	4.5	22.8	2,058	1,024	2.4	4.4	23.0
AW	7,776	789	3.3	4.9	46.4	7,830	631	3.2	4.7	43.5
AX	3,007	11,280	3.0	4.5	35.3	3,032	11,304	3.0	4.4	36.0
AY	10,798	11,471	3.2	4.7	39.5	10,794	11,489	3.2	4.8	40.1
TOTAL	50,251	30,756	3.0	4.6	36.5	50,322	30,508	3.0	4.5	37.5

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To J. Purkis R. Lopaschuk

From P. Clarke

Date October 23, 1980

Subject MEDS SYSTEM MINE PLANNING COMPUTER MODEL (DIPPER)

(Application of Concentrate Economics and Metallurgical Data in the Development of an Equivalent Pb Grade)

SECTION 1.1:

Concentrate Economic Data:

Kindly supplied to the Engineering Department and documented here for reference.

Concentrate Revenues at Loadout:

Costs removed are: Transportation Faro to Skagway; Port Charges; Shipping Charges; and Smelter Charges.

Pb Concentrate: 60% Pb with 18 oz. Ag per MT Pb Concentrate

Note: In this paper MT refers to Dry Metric Tonne and oz. refers to troy oz.

$ Value/Weight Pb Concentrate, including Ag = $705.00/MT

Adjustments for concentrate Pb grade and Ag content:

± 1% Pb Grade = ± $ 9.50/MT

± 1 oz. Ag = ± $21.00/oz.

Zn Concentrate: 50% Zn

$ Value/Weight Zn Concentrate = $294.00/MT

(Note: $294.00/MT refers to years 2-12 from present. For year 0-1 = $150.00/MT, for year 1-2 = $190.00/MT)

Adjustment for concentrate grade:

± 1% Zn Grade = ± $12.50/MT

Note: Increase in $ value/MT of concentrates with increase in concentrate grades is not linear.

i.e., $294.00/MT ÷ 50 = $5.88/MT/%

c.f., $12.50/MT/%

SECTION 2.1:

Metallurgical Data:

Reproduced here with minor simplifications and additions for reference.

The following data was originally developed and documented (more comprehensively) by Al McIntyre and members of the Metallurgical Lab, especially Stan Chmelyk and Bill Muir.

(Refer to "Interim Report Metallurgical Evaluation of Zone 3 Ores," A. McIntyre to R. C. Smith, October 15, 1980).

ZONE 3

AVERAGE METALLURGICAL PERFORMANCE BY ORE TYPE

Ore Type	Pb		Zn		Ag	
	Recovery %	Grade %	Recovery %	Grade %	Recovery %	Amount oz./MT Pb Con.
2A	85	60	80	50	60	19
2BCD	80	60	75	50	50	20
2CE	80	60	75	50	50	15
2EF	81*	60	80*	50	51*	15
2H	50	50	50	50	40	9
2G	85	65	85	50	65	17

* Simplifications:

Concerning 2EF Pb, Zn and Ag recoveries, the actual data supplied was:

Ore Type	Pb Recovery %	Zn Recovery %	Ag Recovery %
2E	80	80	50
2F	85	80	55

The computer model does not differentiate between 2E and 2F by rock type code. The difference between these rock types is basically part grade, part textural. The model does distinguish grade.

In order to overcome this identification problem, Pb and Ag recoveries of 81% and 51% were assigned respectively on the basis of an approximate ratio of 4:1 of 2E:2F in Zone 3.

SECTION 3.1:

Separate $ Values/Weight of 2A derived Concentrates and Metals at Mill Loadout

($ value/weight here is discounted as described in Section 1.1 but not discounted for mining, milling and other associated costs).

2A Concentrates:

Pb Concentrate: 60% Pb with 19 oz. Ag/MT Pb con.

Zn Concentrate: 50% Zn

$ Value/Weight of 1 MT of Pb con. with Ag = $705.00 + $ [21.00 (19-18)]

= $705.00 + $21.00

= $726.00

$ Value of 19 oz. Ag = 19 x $21.00 = $399.00

∴ $ Value of 1 MT Pb con. minus Ag component = $726.00 - $399.00 = $327.00

$ Value of 1 MT Zn con. = $294.00

2A Metals:

2A Pb con. is 60% Pb

∴ $ Value/Weight of Actual Pb Metal = $327.00/MT x 100/60 = $545.00 MT*

* That is MT Pb as opposed to MT Pb con.

$ Value/Weight of Ag = $21.00/oz. = $21.00 x 31.103/g = $0.675/g

2A Zn con. is 50%

∴ $ Value/Weight of Actual Zn Metal = $294.00/MT x 100/50 = $588.00/MT

SECTION 3.2:

Development of Equivalent % Pb Grade Equation for Rock Type 2A

In-pit $ values/weight of Pb, Zn and Ag occurring in a rock type are obtained by multiplying the actual metal $ values/weight by their recoveries from that rock type.

In-pit $ values/weight of Pb, Zn and Ag in 2A are:

Pb	$545.00/MT x 85/100	=	$463.25/MT
Zn	$588.00/MT x 80/100	=	$470.40/MT
Ag	$0.675/g x 60/100	=	$0.405/g

$$\text{Eq. \% Pb} = \text{\% Pb} + \frac{470.40}{463.25}\ \text{\% Zn} + \frac{0.405}{463.25} \times 100\ \text{g/MT Ag}$$

(Note: Ag x 100 to allow for conversion to %)

$$= \text{\% Pb} + 1.015\text{\% Zn} + 0.087\ \text{g/MT Ag}$$

SECTION 4.1:

Separate $ Values/Weight of 2BCD Derived Concentrates and Metals at Mill Loadout

2BCD Concentrates:

Pb Concentrate: 60% Pb with 20 oz. Ag/MT Pb con.

Zn Concentrate: 50% Zn

$ Value/Weight of 1 MT of Pb con. with Ag = $705.00 + $[21.00 (20-18)]

= $705.00 + $42.00

= $747.00

$ Value of 20 oz. Ag = 20 x $21.00 = $420.00

.˙. $ Value of 1 MT Pb con. minus Ag component = $747.00 - $420.00 = $327.00

$ Value of 1 MT of Zn con. = $294.00

2BCD Metals:

2BCD Pb con. is 60% Pb

.˙. $ Value/Weight of Actual Pb Metal = $327.00/MT x 100/60 = $545.00/MT

$ Value/Weight of Ag = $21.00/oz. = $21.00 x 31.103/g = $0.675/g

2BCD Zn con. is 50% Zn

.˙. $ Value/Weight of Actual Zn Metal = $294.00/MT x 100/50 = $588.00/MT

SECTION 4.2:

Development of Equivalent % Pb Grade Equation for Rock Type 2BCD

In-pit $ values/weight of Pb, Zn and Ag in 2BCD are:

Pb	\$545.00/MT x 80/100	=	\$436.00/MT
Zn	\$588.00/MT x 75/100	=	\$441.00/MT
Ag	\$0.675/g x 50/100	=	\$0.337/g

$$\text{Eq. \% Pb} = \text{\% Pb} + \frac{441.00}{436.00}\text{\% Zn} + \frac{0.337}{436.00} \times 100 \text{ g/MT Ag}$$

$$= \text{\% Pb} + 1.011\text{\% Zn} + 0.077 \text{ g/MT Ag}$$

SECTION 5.1:

Separate $ Values/Weight of 2CE Derived Concentrates and Metals at Mill Loadout

2BCD Concentrates:

Pb Concentrate: 60% Pb with 15 oz. Ag/MT Pb con.

Zn Concentrate: 50% Zn

$ Value/Weight of 1 MT of Pb con. with Ag = $705.00 - $[21.00 (18-15)]
= $705.00 - $63.00
= $642.00

$ Value of 15 oz. Ag = 15 x $21.00 = $315.00

.˙. $ Value of 1 MT of Pb con. minus Ag component = $642.00 - $315.00 = $327.00

$ Value of 1 MT of Zn con. = $294.00

2CE Metals:

2CE Pb con. is 60% Pb

.˙. $ Value/Weight of Actual Pb Metal = $327.00/MT x 100/60 = $545.00/MT

$ Value/Weight of Ag = $21.00/oz. = $21.00 x 31.103/g = $0.675/g

2CE Zn con. is 50% Zn

.˙. $ Value/Weight of Actual Zn Metal = $294.00/MT x 100/50 = $588.00/MT

SECTION 5.2:

Development of Equivalent % Pb Grade Equation for Rock Type 2CE

In-pit $ values/weight of Pb, Zn and Ag in 2CE are:

Pb	$545.00/MT x 80/100	=	$436.00/MT
Zn	$588.00/MT x 75/100	=	$441.00/MT
Ag	$0.675/g x 50/100	=	$0.337/g

$$\text{Eq. \% Pb} = \%\ \text{Pb} + \frac{441.00}{436.00}\ \%\ \text{Zn} + \frac{0.337}{436.00} \times 100\ \text{g/MT Ag}$$

$$= \$\ \text{Pb} + 1.011\%\ \text{Zn} + 0.077\ \text{g/MT Ag}$$

SECTION 6.1:

Separate $ Values/Weight of 2EF Derived Concentrates and Metals at Mill Loadout

2EF Concentrates:

Pb Concentrate: 60% Pb with 15 oz. Ag/MT Pb con.

Zn Concentrate: 50% Zn

$ Value/Weight of 1 MT of Pb con. with Ag = $705.00 - $[21.00 (18-15)]

= $705.00 - $63.00

= $642.00

$ Value 15 oz. Ag = 15 x $21.00 = $315.00

∴ $ Value of 1 MT of Pb minus Ag Component = $642.00 - $315.00 = $327.00

$ Value of 1 MT Zn con. = $294.00

2EF Metals:

2EF Pb con. is 60% Pb

∴ $ Value/Weight of Actual Pb Metal = $327.00/MT x 100/60 = $545.00/MT

$ Value/Weight of Ag = $21.00/oz. = $21.00 x 31.103/g = $0.675/g

2EF Zn con. is 50% Zn

∴ $ Value/Weight of Actual Zn Metal = $294.00/MT x 100/50 = $588.00/MT

SECTION 6.2:

Development of Equivalent % Pb Grade for Rock Type 2EF

In-pit $ values/weight of Pb, Zn and Ag in 2EF are:

Pb	\$545.00/MT x 81/100	=	\$441.45/MT
Zn	\$588.00/MT x 80/100	=	\$470.40/MT
Ag	\$0.675/g x 51/100	=	\$0.344/g

$$\text{Eq. \% Pb} = \text{\% Pb} + \frac{470.40}{441.45}\text{\% Zn} + \frac{0.344}{441.45} \times 100 \text{ g/MT Ag}$$

$$= \text{\% Pb} + 1.065\text{\% Zn} + 0.078 \text{ g/MT Ag}$$

SECTION 7.1:

Separate $ Values/Weight of 2H Derived Concentrates and Metals at Mill Loadout

2H Concentrates:

Pb Concentrate: 50% Pb with 9 oz. Ag/MT Pb Concentrate

Zn Concentrate: 50% Zn

$ Value of 1 MT of Pb con. with Ag = $705.00 - $[9.50 (60-50)] - $[21.00 (18-9)]

= $705.00 - $95.00 - $189.00

= $421.00

∴ $ Value of 9 oz. Ag = 9 x $21.00 = $189.00

∴ $ Value of 1 MT of Pb con. minus Ag Component = $421.00 - $189.00 = $232.00

$ Value of 1 MT of Zn Con. = $294.00

2H Metals:

2H Pb Concentrate is 50% Pb.

∴ $ Value/Weight of Actual Pb Metal = $232.00/MT x 100/50 = $464.00/MT

∴ $ Value/Weight of Ag = $21.00/oz. = $21.00 x 31.103/g = $0.675/g

2H Zn concentrate is 50% Zn

$ Value/Weight of Actual Zn Metal = $294.00/MT x 100/50 = $588.00/MT

(Note: The value of actual Pb metal at mill loadout deriving from 2H is lower than that from 2A, 2BCD, 2CE and 2EF due to its occurrence in a lower grade concentrate and the aforementioned non-linearity of $ value/ concentrate weight with increase or decrease of concentrate grade. c.f. 2H - $464.00/MT Pb with 2A, 2BCD, 2CE, 2EF - $545.00/MT Pb.)

SECTION 7.2:

Development of Equivalent % Pb Grade Equation for Rock Type 2H

In-pit $ values/weight of Pb, Zn and Ag in 2H are:

Pb	$464.00/MT x 50/100	=	$232.00/MT
Zn	$588.00/MT x 50/100	=	$294.00/MT
Ag	$0.675/g x 40/100	=	$0.270/g

$$\text{Eq. \% Pb} = \text{\% Pb} + \frac{294.00}{232.00}\text{\% Zn} + \frac{0.270}{232.00} \times 100 \text{ g/MT Ag}$$

$$= \text{\% Pb} + 1.267\text{\% Zn} + 0.116 \text{ g/MT Ag}$$

SECTION 8.1:

Separate \$ Values/Weight of 2G Derived Concentrates and Metals at Mill Loadout

2G Concentrates:

Pb Concentrate: 65% Pb with 17 oz. Ag/MT Pb Concentrate

Zn Concentrate: 50% Zn

\$ Value of 1MT of Pb Con. with 19.5 oz. Ag = \$705.00 + \$[9.50(65-60)] - \$[21(18-17)]

= \$705.00 + \$47.50 - \$21.00

= \$731.50

∴ \$ Value of 17 oz. Ag = 17 x \$21.00 = \$357.00

∴ \$ Value of 1 MT Pb Con. minus Ag Component = \$731.50 - \$357.00 = \$374.50

\$ Value of 1 MT of Zn Con. = \$294.00

2G Metals:

2G Pb concentrate is 65% Pb

∴ \$ Value/Weight of Actual Pb Metal = \$374.50/MT x 100/65 = \$576.15/MT

∴ \$ Value/Weight of Ag = \$21.00/oz. = \$21.00 x 31.103/g = \$0.675/g

\$ Value/Weight of Actual Zn Metal = \$294.00/MT x 100/50 = \$588.00/MT

SECTION 8.2:

Development of Equivalent % Pb Grade Equation for Rock Type 2G

In-pit $ values/weight of Pb, Zn and Ag in 2G are:

Pb	\$576.15/MT x 85/100	=	\$489.73/MT
Zn	\$588.00/MT x 85/100	=	\$499.80/MT
Ag	\$0.675/g x 65/100	=	\$0.439/g

$$\text{Eq. \% Pb} = \text{\% Pb} + \frac{499.80}{489.73}\text{\% Zn} + \frac{0.439}{489.73} \times 100 \text{ g/MT Ag}$$

$$= \text{\% Pb} + 1.021\text{\% Zn} + 0.090 \text{ g/MT Ag}$$

SECTION 9.1:

Comparison of the Equivalent % Pb Grade Technique with a Conventional Method

The following is a comparison of techniques by which the Mintec system requires the $ value/weight of in-pit ore to be handled as against a more conventional method.

Example: To calculate the $ value of 1 MT of 2CE rock type in place.

(The $ value will not include mining and milling cost reductions).

Say, 1 MT of 2CE with grades of Pb 2.8%, Zn 4.6%, and Ag 36 g/MT.

The in-pit $ values/weight of the actual metals as calculated in Section 5 are:

Pb = $436.00/MT

Zn = $441.00/MT

Ag = $0.337/g

SECTION 9.2:

Method 1 - Conventional Approach:

$ Value of Pb in 1 MT of 2CE Pb Grade 2.8%	=	$436.00 x 2.8/100	=	$12.21
$ Value of Zn in 1 MT of 2CE Zn Grade 4.6%	=	$441.00 x 4.60/100	=	$20.29
$ Value of Ag in 1 MT of 2CE Ag Grade 36 g/MT	=	$0.337 x 36.0	=	$12.13
Total $ Value of 1 MT of described 2CE excluding mining and milling costs	=	$12.21 + $20.29 + 12.13	=	$44.63

SECTION 9.3:

Method 2 - Computer Approach:

From Section 5,

Eq. % Pb for 2CE = % Pb + 1.011% Zn + 0.077 g/MT Ag

= 2.80 + (1.011 x 4.60) + (0.077 x 36.0)

= 2.80 + 4.65 + 2.77

= 10.22

= 10.2 as computer rounds to 1 d.p.

$ Value of 1 MT of Described 2CE Excluding Mining and Milling Costs = $\frac{10.2}{100}$ x \$436.00 = \$44.47

Comparison:

Conventional - \$44.63

Computer - \$44.47

The slight discrepancy between the two methods is due to rounding off errors at certain points in their calculations.

In this example, the % difference is: $\left(\frac{44.63 - 44.47}{44.63}\right)$ = 0.0036

or 0.36 of a percent

SECTION 10:

The Dipper Model:

The Dipper Model is a condensed version of the main model file.

The constants in the equivalent % Pb grade equations for each rock type are part of the input run deck to program M612V1 when linked with customized sub-routine UR612B.FD. This program takes each model block in turn and according to rock type and the block Pb, Zn and Ag grades calculates the equivalent % Pb grade. This value is stored in each block in the main model as Item #6 label 'EQ2', (EQ1 is the Pb + Zn equivalent grade) along side the already existing data there - % Pb, % Zn, Ag g/MT, % Cu, % Pb + Zn, rock type, and % of block below topography.

Having previously run program M717CY to create the Dipper Model project control file, program M718CY can be executed to extract the equivalent % Pb value and rock type code in each block, and write these data to the Dipper Model. Some other data to do with topography is also written over at the same time.

Program M720CY is the customized moving cone program and requires as part of its input deck:

1) A single $ value/weight for Pb, and

2) A factor for each rock type allowing for the different in pit $ values/weight of Pb occurring in the different rock types. Call this latter item the relative Pb value factor.

By multiplying a block equivalent % Pb grade by the $ value/weight of Pb, the relative Pb value factor and block weight, a block can be (theoretically) reduced to a single appropriate $ value. (To shrieks of delight from the bean counters, no doubt.)

The $ value/weight of Pb and the relative Pb value factor work in conjunction with each other and so there is a certain amount of choice as to exactly which figures, and logic, is followed, provided that they are compatible.

It is suggested by the author that the $ value/weight of Pb be the $ value/weight of actual Pb metal as occurring in a normal 60% Pb concentrate - developed in Sections 3.1, 4.1, 5.1, and 6.1 - that is $545.00/MT.

The relative Pb value factor for each rock type would be:

$$\frac{\text{In-Pit \$ Value/Weight of Pb Occurring in the Rock}}{\text{\$ Value/Weight of Pb Specified in 1)}}$$

However, as yet no account has been taken of an adjustment factor for the metal grades to allow for dilution and other systematic discrepancies inherent in considering the straight equivalent % Pb grade as a grade likely to be delivered to the mill. This adjustment factor is derived from experience.

The adjustment factor used, at the present state of the art, is -5% on Pb, Zn and Ag grades and is based on statistics documented elsewhere.

(Note: If different adjustment factors are to be applied to the different elements Pb, Zn and Ag, then this calculation would have to be done when deriving the equivalent % Pb grade, which would become an adjusted equivalent % Pb grade. The above mentioned adjustment factor to the in-pit $ value/weight of Pb would then be redundant.)

The adjusted relative Pb value factor for a rock type is then:

$$\frac{\text{In-Pit \$ Value/Weight of Pb occurring in the Rock Type x 0.95}}{\text{\$ Value/Weight Specified in 1)}}$$

Referring to the example in Section 9.3, the in-pit $ value/weight of Pb for 2CE (a normal 60% Pb concentrate producer) was $436.00/MT. This figure was derived in Section 5.2, as follows:

$545.00/MT x 80/100 = $436.00/MT where 80% was the Pb recovery from rock 2CE.

Therefore, using $545.00/MT as the $ value/weight of Pb and 80% as the relative Pb value factor, the correct* in-pit $ value for Pb occurring in 2CE is arrived at. *(The adjustment factor allowing for grade reduction is ignored in this example).

Basically, this exercise is simply to multiply the equivalent % Pb grade in a particular rock type by an appropriate $ value/weight of Pb. The way it's done above is to choose a single $ value/weight of Pb and factor it for each rock type so as to arrive at an appropriate adjusted in-pit $ value/weight of Pb for each rock type.

Before putting the single $ value/weight of Pb which will be in $/MT into the M720CY run deck, it must be converted to $/lb. as the program has been set to work in these units along with SDT and cu. yds.

Program M720CY also requires a number of mining and milling parameters including: tonnage adjustment factor, mining cost, incremental mining cost with depth, cut-off grade, milling cost, pit slope angles, etc.

In order to convert an operating cut-off grade, such as a Pb + Zn one, to an equivalent % Pb grade, the average Ag grade at the operating cut-off grade needs to be taken into account. The average Ag grade at Pb + Zn cut-off can be derived from some statistics output from program M608V1 that currently exists in the Engineering offices. To derive the average grade at the cut-off, as opposed to average grade above the cut-off, as given in the program output, requires only a few calculations.

SECTION 11:

Derivation of Adjusted Relative Pb Value Factors:

Adjustment factor for Pb, Zn and Ag Grades = -5%

Base \$ Value/Weight of Pb = \$545.00/MT

Adjusted Relative Pb Value Factors:

For 2A = $\frac{\$463.25 \times 0.95}{\$545.00}$ = 0.807, or 80.7%

For 2BCD = $\frac{\$436.00 \times 0.95}{\$545.00}$ = 0.760, or 76.0%

For 2CE = $\frac{\$436.00 \times 0.95}{\$545.00}$ = 0.760, or 76.0%

For 2EF = $\frac{\$441.45 \times 0.95}{\$545.00}$ = 0.769, or 76.9%

For 2H = $\frac{\$232.00 \times 0.95}{\$545.00}$ = 0.404, or 40.4%

For 2G = $\frac{\$489.73 \times 0.95}{\$545.00}$ = 0.854, or 85.4%

SUMMARY

Rock	-5% Adjusted Relative Pb Value Factor %	Equivalent % Pb Equation (Zn factor to 2 d.p.)
2A	80.7	% Pb + 1.01% Zn + 0.087 g/MT Ag
2BCD	76.0	% Pb + 1.01% Zn + 0.077 g/MT Ag
2CE	76.0	% Pb + 1.01% Zn + 0.077 g/MT Ag
2EF	76.9	% Pb + 1.06% Zn + 0.078 g/MT Ag
2H	40.4	% Pb + 1.27% Zn + 0.116 g/MT Ag
2G	85.4	% Pb + 1.02% Zn + 0.090 g/MT Ag

Base \$ Value/Weight of Pb for Input to M720CY =

$$\$545.00/MT = \$545.00 \times \frac{1}{1.1023} \times \frac{1}{2000} / \text{lb.} = \$0.247/\text{lb. (to 3 d.p.)}$$

Equiv
% Pb Grade × 545 × Pb factor

It is hoped that this has elucidated upon the use of an equivalent % Pb grade in the MEDS System mine planning model.

The methodology used here is open to refinement just as actual values are open to revision. The main purpose of this paper is to document the present method of deriving and using an equivalent % Pb grade.

P. Clarke

P. Clarke
Engineering Geologist

cc. R. Lopaschuk
H. Hong
B. Cron
J. Bowers
A. McIntyre
F. Gay
S. Chmelyk

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To J. Purkis — R.L.

From P. Clarke

Date July 3, 1980

Subject ZONE 3 PIT DESIGN - NOTES ON BASIC METHODOLOGY

As an introduction, refer to attached memo, "Zone 3 Pit Design," P. Clarke to J. Purkis, June 25, 1980.

At the outset one might note that the use of mining simulation programs are in effect mining engineering tools. The techniques are not able, or in fact intended, to replace the necessary 'manual' input of a mining engineer in the later stages of a detailed mine design.

This memo traces the basic method of incorporating various factors into the design technique and gives a brief outline of the procedure.

It all goes something like this:

1) Having established some values for the necessary parameters (see "Zone 3 Pit Design"), or set of values for a particular design case, one is then in a position to work out 'in-pit' values per unit weight for Pb, Zn and Ag from the recovery and concentrate information.

   (Eventually every mineralized block in the model needs to have a single value assigned to it representing the grades and $ values of its contents.)

2) From the concentrate details a $ value per unit weight, at the loadout, for Pb, Zn, and Ag is established.

3) Assuming average recoveries of Pb, Zn and Ag can be estimated for each rock type, a table could have been drawn up:

TABLE OF AVERAGE PB, ZN, AG RECOVERIES BY ROCK TYPE

Rock Type/Element	Pb	Zn	Ag
2A	a	b	c
2BCD	d	e	f
2EF	g	h	i
etc.		etc.	

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To work out the \$ values per unit weight of in-pit Pb, Zn and Ag in, say, rock type 2A, the \$ values per unit weight for Pb, Zn and Ag at loadout are multiplied by their respective recoveries from 2A type rock.

This is done for each rock type, thus giving in-pit Pb, Zn and Ag \$ values per unit weight as they occur in the various rock types.

4) The above in-pit \$ values per unit weight for Pb, Zn and Ag now have to be incorporated into a single equivalent grade equation for each rock type.

If in-pit \$ values/unit weight for rock type 2A are:

Pb = \$ x/lb.

Zn = \$ y/lb.

Ag = \$ z/g (Ag is stored in g/mt in the computer model)

It is convenient to choose an equivalent % Pb grade and express \$ values of Zn and Ag in terms of Pb.

Equivalent % Pb grade (equation) =

$$\% \text{ Pb} + \left\{\frac{y}{x}\right\}.\% \text{ Zn} + C.\left\{\frac{z}{x}\right\}.\text{g/mt Ag}$$

(C is an appropriate constant to allow conversion of units)

At this point the grade adjustment factors also need to be included.

Assuming Pb = 6%, Zn = -4%, Ag = -6%

Therefore,

Adjusted equivalent % Pb grade =

$$0.94.\% \text{ Pb} + 0.96 \left\{\frac{y}{x}\right\}.\% \text{ Zn} + 0.94\ (C).\left\{\frac{z}{x}\right\}.\text{g/mt Ag}$$

Such an equation is developed for each rock type reflecting the different recoveries from each of the main rock types.

Then, a copy of the main model is produced, renamed, and program M612V1 (linked with a customized user subroutine performing the above type of calculation) is run on the file assigning an adjusted equivalent % Pb grade to each mineralized block, each according to its rock type and previously interpolated Pb, Zn and Ag grades from surrounding DDH's.

5) Programs M717V1 and M718V1 allow extraction of the data from the renamed model file and creation of a 'Dipper' control file, block file, and surface file. (The 'Dipper model,' as Mintec call it, is basically a condensed version of the main model file.)

The block file has the adjusted equivalent % Pb grade in each block and a rock type code.

(The rock type code is needed to differentiate the variable tons/block associated with the rock types. As most Mintec programs were originally developed for use by porphry copper mines that are not as concerned about changes in rock density, we will require a customized program here.)

The surface file stores the pit topographic data. (This will be current topography, as opposed to original topography.)

6) Program M720V1 is a 'moving cone' program. (It utilizes a user subroutine which will be written to suit the necessary calculations.)

It is this program that requires:

i) $ mining cost/unit volume

ii) $ milling cost/unit tonnage

iii) Mine and mill cut-off grades

iv) In-pit $ value Pb/unit weight (to go with the adjusted equivalent % Pb grades)

v) % Rate of Return
(also, pit slope angles in various directions)

If a normal Pb + Zn cut-off grade is to be used for the mine and mill, then this will need to be translated into adjusted equivalent % Pb grade terms. This can be approximated using statistics from the average Pb, Zn, Ag ratio at the chosen cut-off.

Program M720V1 examines blocks within the deposit individually or in small groups as 'base blocks' for a series of cones.

Basically, the program expands a 'warped' cone upwards from the base (reflecting different slope angles in different directions) and calculates the net value of all the blocks lying within the cone and beneath the surface.

The net value of a cone is a fairly straightforward calculation summing the net value of all the individual blocks within it.

The net value of a block is calculated on the basis of:

N.V. = (adjusted equivalent % Pb grade x 'in-pit' $ value Pb/unit weight x block tonnage)

- ($ block mining cost + $ block milling cost)

A tonnage adjustment factor can easily be incorporated at this point in the calculations. Blocks with a grade below the mill cut-off have only a mining cost associated with them.

Thus a 'pit expansion' or 'phase' would consist of a number of overlapping cones, each having had a net positive value.

The cone can be caused to move within a particular area and on certain benches by the artful use of input variables to M720V1. While not altogether easy, given some conditions, it is possible to approximate phases. (Where phases run parallel to a mine model co-ordinate system it is more straightforward, however at Faro ....)

As base blocks are set deeper, the mining cost can be slightly increased.

If a 'break even' pit is required, then those cones down to where net value equalled cost will be 'mined.'

If a rate of return is required (undiscounted) then only those cones where the net values were greater than cost by a certain pre-specified percentage will be taken.

Subsequent phase expansions can be examined using a new surface file produced by the program in the previous run. The pit can so be 'pushed back' in phases, the program output assisting this approach. One can easily 'go back' and so advance using an iterative procedure.

Often the actual surface files left from the moving cone are somewhat irregular and cannot be said to be smooth or practical designs in themselves.

The actual procedure is for the mining engineer to manually create the final design using the Dipper output files as his basis.

Using a digitizer and additional programs, the engineer can input a design back to the computer for plotting on mylar, detailed reserves calculations and subsequent scheduling.

It is hoped that the above gives sufficient understanding of the basic technique to allow discussion of it, and various attendant economic philosophies.

Part of the approach at this stage might be to examine some design cases, based on sets of economics at upper and lower limits of some ranges, and therefore, gauge the sensitivity of the design to these changes.

Peter

P. Clarke
Engineering Geologist

PC/mm

To	J. Purkis	cc.	J. Carrington	B. Cron
From	P. Clarke		D. Gregoire	T. Biggs
Date	June 25, 1980		R. C. Smith	G. Wight

Subject ZONE 3 PIT DESIGN

In order to carry out mine design studies on Zone 3, we are in need of some predictive estimates for the forthcoming years (basically, at this point in time, 1982-1985).

The following parameters will be required:

a) $ Mining cost per unit volume.

b) $ Milling cost per unit tonnage.

(The above should include all relevant costs, including capital, admin., etc.).

c) % Mine cut-off grade.

(This can be a Pb + Zn + Ag version).

d) % Mill cut-off grade.

(If above mine cut-off, this is equivalent to stockpiling).

e) Adjustment factors for grade and tonnage (as stored in the computer model) to allow for dilution and other systematic departures of the model from mining reality.

f) If we are able to estimate recovery by rock type, these should be included at this point. This would require:

% Recovery of Pb, Zn, Ag for each of the ~~five~~ six main rock type groupings (i.e. 2A, 2BCD, 2EC 2EF, 2G, 2H).

If not, then the following simplification can be made:

Average % recovery of Pb, Zn, Ag from all feed material.

Concentrate details required are:

g) Average % Pb in Pb concentrate.

h) Average % Zn in Zn concentrate.

i) Average oz./SDT (or g/mt) of Ag in Pb concentrate.

(As the study is concerned with the period of post mill modifications, bulk production is not considered).

j) $ Value of Pb concentrate per unit tonnage.

k) $ Value of Zn concentrate per unit tonnage.

l) $ Value of Ag in Pb concentrate per oz. (or g).

($ Values of concentrate will be needed as f.o.b. loadout, i.e. transportation, smelting, etc. discounted).

Also, if the pit is to be designed with some rate of return in mind, then this should also be considered at this point. Probably the best thing to do would be to have a meeting to agree on some values for the above-mentioned parameters and to develop the necessary philosophical approach for mine planning.

Peter

P. Clarke
Engineering Geologist

PC/mm

OPEN PIT MINE DESIGN - DIPPER SUBSYSTEM

7.21 INTRODUCTION

The acronym DIPPER stands for Dynamic Interactive Pit Planner & EvaluatoR. The DIPPER subsystem is used to rapidly generate a series of pit designs based on economic criteria, allowing the computer to automatically perform much of the design calculations with a small amount of engineering assistance. The basic method used is a variation of the multiple moving cone method which has been used by various mining companies for over a decade.

DIPPER works with a special condensed matrix which incorporates sophisticated packing techniques to allow an entire mineral inventory to be stored in core memory during program execution. These techniques remarkably reduce the cost and computer time required for MINTEC's version of the multiple moving cone method.

DIPPER calculations are made on a whole block basis, where a block is either entirely inside or entirely outside of a given pit. Volume calculations for an entire pit will be quite accurate, regardless of the pit slope or block dimensions. However, it should be remembered that volume calculations for a few benches or a limited area are less satisfactory.

The next page contains the generalized procedure to set up and use the DIPPER subsystem for open pit mine design.

7.27 DESIGNING PITS WITH DIPPER

Designing pits with the DIPPER subsystem is an iterative procedure with the primary purpose being to minimize the cost of arriving at design objectives. To achieve minimum cost, the user is asked to provide a small amount of assistance by making various decisions.

The DIPPER subsystem could be used to obtain a 'one-shot' ultimate pit, but the cost of such a run will be much greater than the cost of the iterative procedure. Also, such an ultimate pit could easily contain features which make the outlines unreasonable.

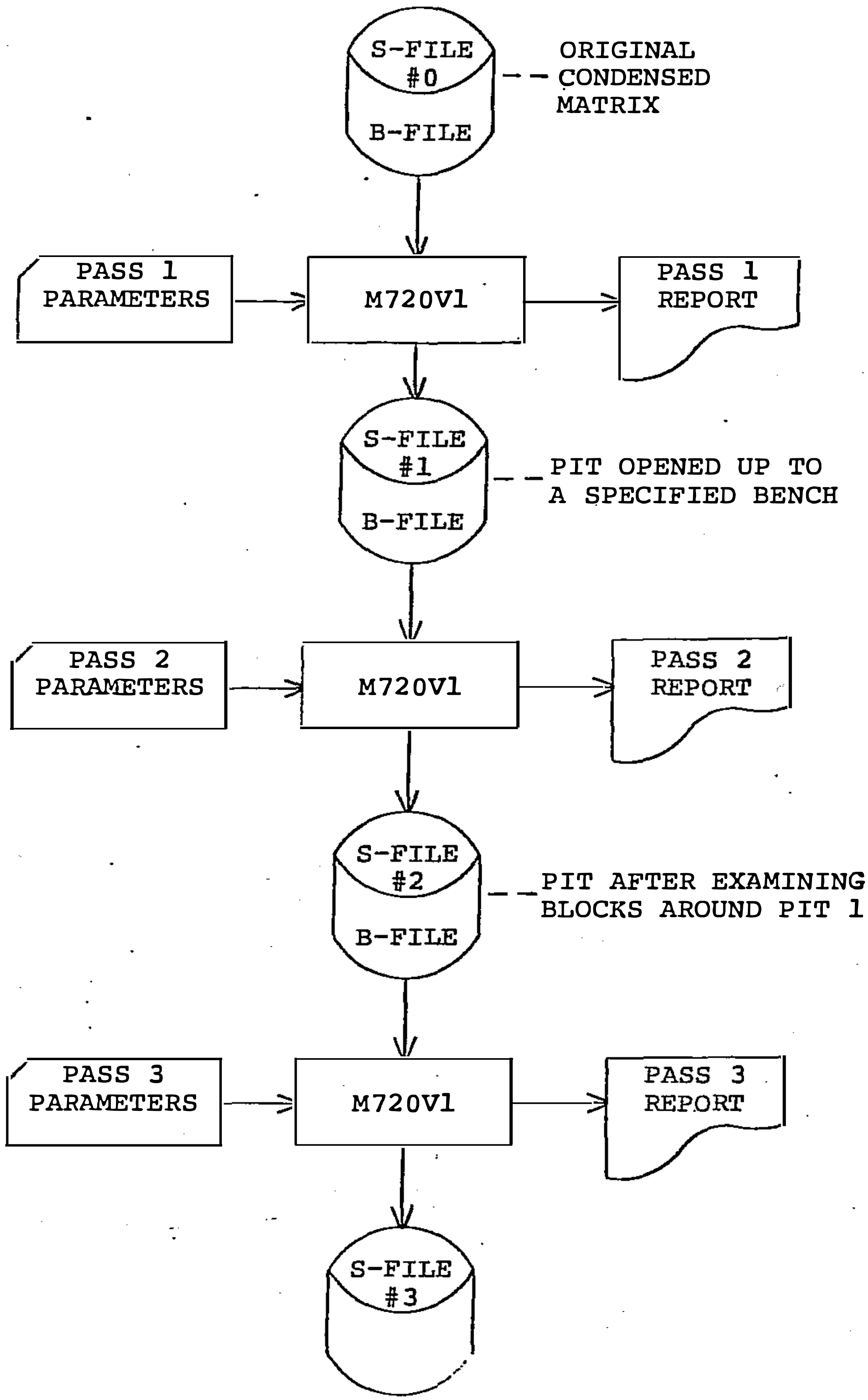
The basic procedure consists of:

(1) Opening up a pit to a specified bench, and

(2) Examining blocks within an envelope surrounding the pit from (1) until no or very little additional tonnage is economic.

The procedure is illustrated in the following figure.

With the use of the iterative procedure, the user may request that various S-FILES be stored, so that a possible sequence of mining is represented. With a series of S-FILES, incremental reserves are easily determined.



ITERATIVE PROCEDURE FOR DESIGNING PITS WITH DIPPER

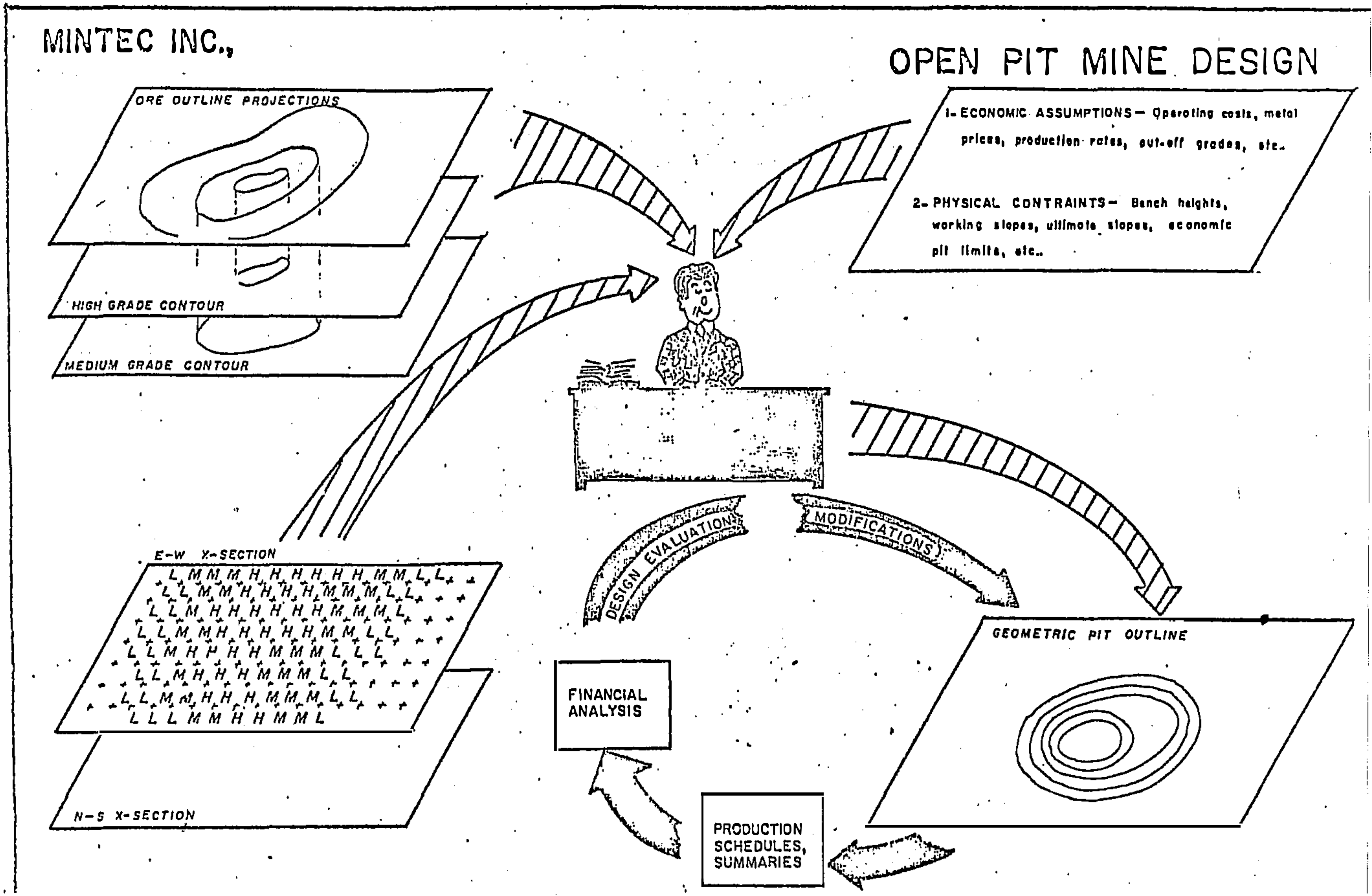

MINTEC INC.,
OPEN PIT MINE DESIGN
ORE OUTLINE PROJECTIONS
HIGH GRADE CONTOUR
MEDIUM GRADE CONTOUR
1- ECONOMIC ASSUMPTIONS— Operating costs, metal prices, production rates, cut-off grades, etc.
2- PHYSICAL CONTRAINTS— Bench heights, working slopes, ultimate slopes, economic pit limits, etc.
E-W X-SECTION
N-S X-SECTION
DESIGN EVALUATION
MODIFICATIONS
GEOMETRIC PIT OUTLINE
FINANCIAL ANALYSIS
PRODUCTION SCHEDULES, SUMMARIES

FIGURE 7.1

7.10 THE DIPPER SUBSYSTEM FOR PIT DESIGN

Programs M718V1 or M719V1 are used to set up a condensed form of the mine model grades which defines the surface topography and contains a single grade for each block. This condensed form reduces the matrix to a size that can be read into core. The size of the condensed form of the matrix depends upon both the size and grade distribution of the deposit. Usually, the grades of 4 or 5 mine model blocks can be packed into a single word of the condensed model. The condensed model is composed of two parts, the S-File describing the surface and the B-File containing the grades.

Program M720V1 is used to generate pit limits for either a specified pit slope or for a set of assumptions regarding pit slope and economics. The pit slope can be varied in different areas of the pit. The accuracy of the calculation for tonnage can be examined with M725V1.

The logic used by M720V1 to evaluate the economics of mining blocks is essentially that of the multiple cone to approximate the removal of pit material at the desired pit slopes. The geometry of the pit is approximated by a series of connecting cones containing whole blocks. The current implementation allows for variable bench heights and multiple pit slopes both horizontally and vertically. The calculations are summarized on the next page.

PROGRAM	DESCRIPTION
M718V1	CONDENSE BLOCK MODEL. The block model developed with MEDS is usually quite large, often greater than 500,000 words. To use the grade matrix in the DIPPER System, it must be condensed to a size which can be contained in the computer's core space. Using M718V1 this condensed matrix is composed of two files, the S-File containing the surface and the B-File containing the grade values.
M719V1	LOAD CONDENSED MATRIX FROM CARDS. This program allows the User to load grades or topography directly into the condensed form. This may be required for using DIPPER with a grade matrix not developed with MEDS.
M720V1	ECONOMIC PIT LIMITS CALCULATION. Given geometric and economic constraints, M720V1 can be used to define economic pit limits, and store them for use as the starting pit limits for further expansion. Used with a series of increasingly favorable economic constraints, this program can be used to determine a mining sequence.
M721V1	PIT/LIMITS TOPOGRAPHY PLOT. M721V1 prints symbol or scale maps of pits generated.
M722V0 M722V1 M722V2 M722V3	PLAN/SECTION PRINTER GRADE MAPS FROM DIPPER MODEL. M722 prints scale maps or symbol maps of grade values below pit limits.

CYPRUS ANVIL MINING CORPORATION INTEROFFICE CORRESPONDENCE

To J. Purkis cc. F. Gay

From P. Clarke R. Lopaschuk

Date May 5, 1980

Subject ZONE 2 TONNAGE AND GRADE MODEL

An examination of Zone 2 DDH recoveries was undertaken in the light of relatively large differences encountered in the mining of the upper benches of Zone 2.

Figures from the 7 DDH's where recoveries were worst are attached. They show that, generally, recoveries from these DDH's, most of which are located in one particular area, are worst on the upper benches - 3910, 3890, 3870 and 3850. There are cases where 3830 and 3810 are poor also.

A comparison of a 500' x 500' area on 3830 bench showed:

	SDT	% Pb	% Zn
Model	310	3.3	4.5
Blastholes	250	3.3	4.4

In any given area within a bench, tonnages may be out to the degree seen above. Taken over a complete bench, tonnages usually come to smaller variances.

This is largely a problem inherited from the structure of the orebody and the DDH spacing thereon. Gaps in the 140' x 140' pattern have usually been able to provide some kind of surprise.

In short, most of the problems have come from poor recovery and not being able to make an accurate geological interpretation. The latter is most affected by DDH spacing.

The 3830 part bench grade comparison and improved recoveries from DDH's in the lower benches encourage the belief that the Zone 2 tonnage and grade model will provide better predictive estimates than has been experienced to date.

Based on the above, examination of various statistics, and discussion with F. Gay, it was agreed that until next reviewed the following adjustment factors will be applied to the Zone 2 tonnage and grade model:

Tonnage	No Factor
Pb	-5%
Zn	-5%
Ag	-5%

It has become readily apparent that Zone 2 and Zones 1/3 are sufficiently different in the nature of their occurrence that inter-application of the above factors is not meaningful.

P. Clarke

P. Clarke
Engineering Geologist

PC/mm

Attach.

ZONE 2

FOOTAGE RECOVERIES FROM 7 DDHS THROUGH 20.0 FOOT BENCHES

BENCH \ DDH	67-18	67-20	67-22	67-29	67-31	78-10	78-20
3910			2.0			1.0	5.0
3890		0.5	7.5		12.0	6.0	4.0
3870	6.0	1.2	12.3		19.0	12.8	16.0
3850	6.1	2.5	12.0		20.0		16.8
3830	10.4	1.3		10.8	20.0		
3810	16.1			8.0			
3790	15.0			10.5			
3770	7.0			15.8			

Cyprus Anvil Minin(ˀorporation

Interoˀ ˀe Correspondence

To D. Gregoire

Date September 7, 1978

From P. I. Clarke

Subject PROPOSED SCHEDULE FOR COMPUTER MINE MODEL

September:

1. Complete outstanding re-assaying of Zone 3 samples and re-assaying of samples from part of Zone 1. Re-assaying in Zone 1 has been limited to Phase VI and later phases.

2. Continue preparation of DDH data into presentable format.

3. Continue to have DDH keypunched and loaded to magnetic tape files.

4. Continue drawing Zone 3/Zone 1 geological sections.

5. Start coding topography into computer format.

October:

1. Continue preparation of DDH data.

2. Continue keypunching and loading of DDH data.

3. Continue drawing Zone 3/Zone 1 geological sections and produce geological bench plans.

4. Start coding geological bench plans into computer format.

5. Finish coding topography.

6. Calculation of Zone 3/Zone 1 bench assay composites by computer.

7. Label composites as to predominant rock type.

November:

1. Compilation of frequency distribution of composites by computer.

2. Develop grade interpolation technique (comparison with a mined out area of Zone 1).

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3. Compilation of frequency distribution of block grades by different interpolation techniques for comparison.

4. Interpolate grade values into Zone 3/Zone 1 blocks.

5. Print bench plans of Zone 3/Zone 1.

6. Start organizing Zone 2 samples for re-assaying.

7. Install computer terminal, diskette and digitiser.

December:

1. Check bench plans.

2. Calculate Zone 3/Zone 1 mine reserve inventory by bench (using current mine design).

3. Prepare Zone 2 DDH data.

4. Prepare Zone 2 geological sections.

5. Start re-assaying Zone 2 samples.

January:

1. Ongoing work on Zone 3 model.

2. Start keypunching and loading Zone 2 DDH data.

3. Draw Zone 2 geological sections.

4. Produce geological bench plans of Zone 2.

5. Code Zone 2 bench plans into computer format.

6. Continue re-assaying Zone 2 samples.

February:

1. Code Zone 2 topography.

2. Continue preparation and loading of Zone 2 DDH and in situ data.

3. Continue re-assaying Zone 2 samples.

4. Prepare First Quarter Review.

March:

1. First Quarter Review.

2. Complete model of Zone 2.

3. Start new ultimate pit designs.

Peter

P. Clarke
Mine Geologist

PC/mm

Cyprus Anvil Minir Corporation

Interc e Correspondence

RL, HM

To D. Gregoire

Date October 26, 1978

From P. I. Clarke

Subject COMPUTER MODEL

Since its inception, the computer mine model has broadened in scope and now includes Zone 1 (Phase 6 onwards) and Zone 2, in addition to the originally envisaged Zone 3 model.

The Zone 1/Zone 3 model encompasses 130 diamond drill holes, all of which are now logged and plotted on the latest lithological coding system. (In addition, a further 20 DDH's were re-logged during July of this year for predicting Phase 5 metallurgical problems.)

The re-assay program of Zone 1 and 3 has involved the handling and organization of over 2,700 samples for 14,000 assay determinations. The program proved the need for a great deal more time and effort to organize than anticipated, due to poor sample storage, duplicated sample numbers, lost samples (and the ensuing searching for, and re-sampling of core where possible) in addition to the splitting and assaying of mineralized core not previously sampled.

The majority of assaying was contracted out to Kamloops Research and Assay Laboratory due to the work load on the Cyprus Anvil Assay Lab. However, the program was completed expediently by: the Cyprus Anvil buckers handling 1,100 reject samples in addition to their normal work; the Assay Lab performing assays for barium on all of these, and, on composite samples, for pyrite, pyrrhotite and manganese; and the Met Lab completing specific gravity determinations on all the 1,100 samples.

Coding of the re-assay data, together with previous results (for some elements) from 1975, 1976 and 1977 assaying, has involved the compilation of 29,000 assay values for input to the C.S.C. computer. The coding is 95% completed now.

Mr. R. Willie (of Willie Associates) is organizing the keypunching of this data (being performed at Tetrad, Vancouver) and the computer file creation (at C.S.C., Vancouver). The basic DDH file is largely complete.

The other main files that need to be created are: topography; drill hole composite geology; and block geology. The two latter files first require all the geological cross sections to be interpreted and bench plans inferred. This represents a considerable amount of data to be prepared and transferred into computer format.

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The Zone 2 model presently encompasses 53 DDH's, nearly all of which have been previously re-logged.

The re-assay program for this zone involves some 400 samples. 300 have already been sent to KRAL, while the remainder still have to be located, or taken from, or split from the original core.

It should be clear that a considerable amount of data has still to be prepared on all zones.

Also, in the light of the forthcoming drill programs on Zones 1 and 2, adding more data for preparation, previous schedules for stages of model completion should be re-appraised.

Daryl Hanson will be managing the project, in my obligatory absence after the end of October, for an interim period, until it can be taken over on a full and longer term basis again.

Cheers,

Peter

P. I. Clarke
Mine Geologist

PIC/mm

cc. P. Taggart
J. Devitt
J. Mustard
D. Hanson
B. Ferguson
B. Muir
G. Chapman

Cyprus Anvil Mining Corporation

Interoffice Correspondence

To D. Gregoire

Date October 5, 1978

From P. Clarke

Subject PROPOSED SCHEDULE FOR COMPUTER MINE MODEL

October:

1. Continue preparation and keypunching of D.D.H. (Zone 3 re-assaying nearly complete, results of Zone 1 due shortly).
2. Continue drawing Zone 1 and Zone 3 geological cross-sections. (There is still a considerable amount of work to be completed on these due to their complexity.)
3. Create the topography file.
4. Prepare Zone 2 samples for re-assaying.
5. Familiarization of Engineering personnel with the whole computerization project in readiness for departure of Peter Clarke.

November:

1. Complete D.D.H. file.
2. Continue drawing geological cross-sections and bench plans.
3. Start coding block geological data.
4. Check and prepare files for subsequent stages.
5. Start Zone 2 re-assaying.

December:

1. Calculation of Zone 1/Zone 3 bench assay composites.
2. Label composites as to predominant rock type.
3. Develop grade interpolation technique.
4. Install computer terminal, diskette and digitizer.
5. Continue Zone 2 re-assaying.

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January:

1. Calculate Zone 1/Zone 3 mine reserve inventory.
2. Ongoing work to check and evaluate the new model.
3. Prepare and start keypunching Zone 2 D.D.H. data.
4. Start coding Zone 2 in-situ geology.
5. Complete re-assaying Zone 2.

February:

1. Complete Zone 2 files and develop model.
2. Prepare First Quarter Review.

March:

1. First Quarter Review.
2. Complete model of Zone 2.
3. Start new ultimate pit designs.

To clarify this program in relation to the original, it should be on record that the overall scope of the model has increased from that of Zone 3 to also include Zone 1 (Phase VI onwards) and Zone 2.

Changes to last month's schedule are that Zone 2 data acquisition and processing (prior to input to computer) have been brought forward at the expense of the eventual completion of the Zone 1/Zone 3 model.

In the last month, the majority of the D.D.H. data has been sent off for keypunching.

Peter

P. Clarke
Mine Geologist

PC/mm

Cyprus Anvil Minir~ ~orporation   Intero~ e Correspondence

To D. Gregoire

Date December 8, 1978

From D. Hanson

Subject LARGE FARO COMPUTER MODEL - PROGRESS REPORT

During the month of November, progress on the computer model for the Large Faro Deposit centered around the construction of detailed geologic sections. This stage of the project is currently 60% complete with 13 longitudinal and 11 cross sections interpreted in rough form. Also during the month the DDH files and assay files were completed with the exception of 73-29 and all files were transferred into MINTEC compatible format.

Two observations were made regarding data quantity which severly affect section interpretation. In the area of the dike separating Zones 1 and 3 there are still sections with no data. A recommendation was made to the Engineering Department for a combination drilling and mapping programme to further define the dike in these sections.

An up-to-date pit geology map would improve the reliability of the section interpretations, especially for short range planning. It is recommended that manpower requirements and/or priorities be evaluated so that this important function is not further neglected.

A tentative schedule for December is as follows:

1) Complete detailed geologic sections in rough form.
2) Check assay file.
3) Installation of terminal, diskette and digitizer.
4) Bench assay composite calculations.
5) Drill hole assay statistics calculations.

M.M.
for D. Hanson
Exploration Geologist

DH/mm

CYPRUS ANVIL

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Cyprus Anvil Mining Corporation

Interoffice Correspondence

To D. Gregoire

Date January 10, 1979

From D. Hanson

Subject COMPUTER MINE MODEL - PROGRESS REPORT

During December the detailed geologic sections were completed in rough form. These sections require thorough checking to remove inter-sectional inconsistencies before the next steps of creating simplified sections and transferring contacts to bench plans can begin.

Manual verification of the assay file was initiated in the latter part of the month and should be completed in January.

No progress was made on computing drill hole assay statistics and bench assay composites due to technical problems between MINTEC and CSC.

During December CSC will sponsor an introductory course to the CSC system for representatives of all interested departments.

Daryl.

D. Hanson
Exploration Geologist

DH/mm

CYPRUS ANVIL

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Cyprus Anvil Mining Corporation Interoffice Correspondence

To D. Gregoire

Date February 9, 1979

From D. Hanson

Subject COMPUTER MINE MODEL - PROGRESS REPORT

Progress for the month of January was as follows:

1) Detailed geologic sections completed and cross-checked.

2) Completed and checked assay file.

3) Bench geologic composite file completed.

4) Simplified geologic sections - 2/3 complete.

Also during the month, CSC gave an introductory course on their computer system. The course was attended by twenty Cyprus Anvil employees from the Accounting, Engineering, Mine, Mechanical, Warehouse, Metallurgy and Exploration Departments. As a direct result of this course, CSC will develop prototype computerized reports for mine production, mine tire usage, townsite, drilling and blasting, and pit volume calculations.

A tentative schedule for February is as follows:

1) Complete simplified geologic sections.

2) Define 1/2 bench geologic contacts on section overlays as construction drawings for bench plan creation.

3) Input bench geologic composite file.

4) Initiate usage of some MINTEC programmes by calculating basic drill hole assay statistics and bench assay composite values.

M. MacInnis

for D. Hanson
Exploration Geologist

DH/mm

CYPRUS ANVIL

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Cyprus Anvil Mining Corporation

Interoffice Correspondence

To D. Gregoire

Date April 5, 1979

From J. Mustard

Subject COMPUTERIZATION OF THE ORE MODELS

During the month of March, all outstanding re-assaying and re-logging was completed for Zone 2. Computer drawn plots of each drill hole were completed. A complete file for Zone 2 is now ready for computer entry. Cross-sectional interpretation was virtually complete at month end.

During April, it is anticipated that all data will be on computer storage and that trial runs can be attempted late in the month.

J. W. Mustard

J. W. Mustard
Mine Geologist

JWM/mm

CYPRUS ANVIL

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Cyprus Anvil Mining Corporation

Interoffice Correspondence

To D. Gregoire

Date April 9, 1979

From D. Hanson

Subject LARGE FARO COMPUTER MODEL - PROGRESS REPORT

During March, 1979 work on the Large Faro computer mine model centered around the production of geologic bench plans and the familiarization of mine personnel with MINTEC programs.

Geologic bench plans were completed for the Large Faro Deposit. Elevations 4270 to 3910 were constructed on 40 foot benches and 3910 to 3290 on 20 foot benches. Our experience using the two different bench heights has led to the conclusion that all further models should use standard 20 foot benches. Too much simplification and the resulting loss of important detail occurs with 40 foot bench heights.

Also during March, a week was spent in Vancouver with Fred Banfield of MINTEC. The main purposes of this session were to familiarize mine personnel with the techniques and operation of the computer programs, and to identify any problems in the next stages of model development.

During the session, a mine model was initiated using real drillhole and geological data for the 3510 bench. The following types of programs were demonstrated:

a) Setting model parameters
b) Input of manually coded geological data
c) Input of assay and survey drillhole data
d) Calculation of bench assay composites using specific gravity and sample length
e) Calculation of statistics for raw assay data and for composite assay data
f) Plotting of block geology and assay composite values on printer
g) Testing a hypothetical grade interpolation technique both with and without geologic control
h) Calculation of statistics for interpolated block grades

We came away with valuable operating experience and a very favourable impression about the versatility of the program package.

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Three problem areas were identified during the session. In checking the bench assay composite calculation, a number of anomalously low specific gravity values were observed. This problem has since been investigated and should be resolved shortly. The entire problem is complex and will be treated in a forthcoming memo.

The second problem dredges up a philosophical question about any modelling technique. The question is whether or not external grade dilution should be included in the model. It was decided that, since it is easier to add a dilution factor than to remove included dilution, external dilution will not be included in this model.

It was further decided to use a standard 20 foot composite length to reduce the effects of erratically high assays near ore/waste contacts. When a drill hole intersects less than 20 feet of the sulfide horizon on a bench, the composite calculation will use the average grade from the bench below or above to make a full 20 foot composite. This technique is similar to the dip projection method of previous models.

The third problem concerns the technique used by the programs to incorporate geologic control into the grade interpolation. The present programs use drill hole geology composites for this and we wish to use block geology. MINTEC will alter their programs to satisfy this condition.

At the end of the month, M. O. Hampton left the project and Vivian Schmidt was hired as a temporary secretary to code and keypunch block geology.

Daryl.

D. Hanson
Exploration Geologist

DH/mm

Cyprus Anvil Minir Corporation — Interc e Correspondence

To D. Gregoire

Date April 30, 1979

From J. W. Mustard

Subject SMALL FARO COMPUTER MODEL - PROGRESS REPORT

During the first half of April, time was mainly devoted to the preparation of geologic bench plans from completed cross sectional interpretation. Assay data and block geology files were created during the latter half of the month.

Drafted copies of the geologic bench plans were completed from 3990 to 3770 on 20 foot elevations. These are now on file in the Engineering office, as are the pencilled cross sectional interpretations.

All assay and block geology data were entered on the local terminal for storage on CSC hardware. Local entry proved invaluable since priority was established under direct control. In addition, each record entry was checked to ensure that no errors were made.

During May, a first generation model will be completed, with a view to complete scheduling by the end of May.

A distance weighting technique within an elliptical search pattern for grade interpolation will be used on the first generation model. The dimensions of the ellipse will be such that not less than 4 boreholes affect the grade in any 1 block, except near the sulfide margins.

While the above technique does <u>not</u> allow for any estimate of the errors of estimation nor does it necessarily minimize the errors of estimation, it has worked in a modified form in the Large Faro Deposit.

The method of interpolation by elliptical smoothing appears to be satisfactory in all cases except in areas of external contact (ore/waste). Here the geological boundary can be such that a portion of the ore bench will actually be waste at any given economic cut-off grade. A method whereby this "waste" is accounted for is a necessary part of any grade interpolation technique.

M. M.

J. W. Mustard
for Mine Geologist

JWM/mm

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CYPRUS ANVIL MINING CORPORATION — INTEROFFICE CORRESPONDENCE

To: D. Gregoire — cc R. Visagie, R. Buckley

From: R. Tolbert — D. Hogan, I. Hall

Date: September 12, 1984

Subject: Comparison of F-4 Model Reserves with Dome Hand Calculated Reserves for Faro

Since the F-4 reserve modelling is only half complete (x-sections 124+22 to 134+47) it has not been possible to compare reserves in the F-3 model with those derived in the F-4 model.

It is possible, however, to compare the F-4 model and the Dome hand calculated reserves for x-sections 124+22 to 135+54.

The Minable Reserves outlined below are directly compariable taking into account their different interpolation methods.

To enable the Geological Reserves to be comparible, the Dome hand calculated reserves for x-sections 134+47 and 135+54 were added to the F-4 reserves covering x-sections 124+22 to 133+00.

Results

Geological Reserves (124+22 to 135+54)

Model	Tonnes	Pb %	Zn %	Ag g/mt
F4 + (D.H.C.)	13,768,695	3.2	5.0	41.9
Dome H.C.	12,746,023	3.18	5.12	42.1
*Variance %	-7.4	0	+2.4	0

Minable Reserves (124+22 to 133+00)

Model	Tonnes	Pb %	Zn %	Ag g/mt
F4	9,980,457	3.1	4.9	40.5
Dome H.C.	9,131,432	3.09	4.91	40.93
*Variance %	-8.5	0	0	+1

$$\text{*Variance \%} = \frac{\text{Dome H.C.} - \text{F4}}{\text{F4}} \times 100$$

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CYPRUS

Re: Comparison of F-4 Model Reserves

The following is a comparison between the F3 model and the Dome hand calculated Minable Reserves for the same volume (118+00 to 133+00):

Minable Reserves (118+00 to 133+00)

Model	Tonnes (000,000)	Pb %	Zn %	Ag g/mt
F-3	26.4	2.9	4.3	36.0
Dome H.C.	22.7	3.17	4.87	40.8
**Variance %	-14	+9.3	+13	+13.4

$$\text{**Variance \%} = \frac{\text{Dome H.C.} - \text{F3}}{\text{F3}} \times 100$$

A comparison between the F3 and the F4 models within the NA, OA, and PA phases (R. Tolbert, October 1983) shows the following variances:

	Tonnes	Pb %	Zn %	Ag g/mt
***Variance %	-10	+ 5	+ 8	+ 1

$$\text{***Variance \%} = \frac{\text{F4} - \text{F3}}{\text{F3}} \times 100$$

A review of the Faro Zone 3 by Simpson et al. in May 1983 compared the reserves from the F3 model with their hand calculated reserves (unfortunately at 3% and 5% Pb + Zn cutoff) in the NA, OA, and PA phase volumes. Their results showed an increase in grade and a decrease in tonnage. The decrease in tonnage ranged from -27% (at 3% cutoff) to -8% (at 5% cutoff). It is estimated that the variance between the F3 model tonnage and the Exploration hand calculation would be in the same order of mangitude as the F3/Dome hand calculation, i.e. in the order of -14%.

Discussion

The Dome hand calculation and the F4 model show the closest correlation. The difference in tonnage can be explained due to the difference in interpolation method. One might expect, since there is more detail in the F-4 model, that it would be correct. If this is assumed to be the case then it is possible to derive an anticipated F-4 tonnage and grades for the whole deposit. This would be a tonnage 8.5% higher than the Dome hand calculation with grades the same as the Dome hand calculation as follows:

	Tonnes	Pb %	Zn %	Ag g/mt
F-4 (anticipated)	24.6	3.18	5.12	42.1

Examining the total metal content the following variance between Dome H.C. and F3 and between F4 (anticipated) and F3 can be made as follows:

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Re: Comparison of F-4 Model Reserves

Model	Tonnes (000,000)	Pb Tonnes	Zn Tonnes	Ag Tonnes
F3	26.4	765,600	1,135,200	950.4
Dome H.C.	22.7	719,600	1,105,500	926.2
F4 (anticipated)	24.6	779.800	1,198,000	1003.7
Variance %: $\frac{\text{Dome H.C. - F3 x 100}}{\text{F3}}$	-14%	- 6.4%	- 2.3%	- 2.5%
Variance %: $\frac{\text{F4 (ant) - F3 x 100}}{\text{F3}}$	- 6.8%	+ 1.9%	+ 5.5%	+ 5.6%

The value of the above total metal content (in the ground) for silver, lead and zinc only is as follows:

Model	Value * (C$ Billion)	Variance to F3	Difference to F3* (C$ 000,000)
F3	2.219	-	-
F4 (anticipated)	2.328	+4.9%	+ 109
Dome H.C.	2.148	-3.2%	- 71

*Silver U.S. $10/oz
Lead U.S. $0.25/lb.
Zinc U.S. $0.40/lb.
C$1.00 = U.S. $0.76

It is pointed out here that the above discussion is simply an indicator of the possible discrepancy that may be anticipated between the F3 model and the F4 model once it is completed.

Since:
1. there may be differences, in the unmodelled volume remaining (x-sections 118+00 to 123+00), between the Dome hand calculated reserve and the to be completed F4 model that are not readily apparent:
2. new 1984 drill assay results have to be included in the new F4 model;

the above discussion should be viewed with due discretion.

Clearly if the above F4 (anticipated) reserve were to be real then the effect, if the same ore tonnage throughput to the mill were maintained as in the F3 based plans, would be to deplete the reserves sooner. This would require sooner start up in the Vangorda Plateau.

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Re: Comparison of F-4 Model Reserves

If the same concentrate tonnage output were to be maintained, as in the F3 based plans, then since the F4 (anticipated) grades are higher the ore tonnage throughput to the mill would have to be reduced.

Summary

1. Comparing the F4 model with the Dome hand calculation for half of the Faro deposit shows excellent correlation between grades and a reasonable correlation between tonnages of the two methods in both the Geological and Minable Reserves.

2. Comparing the F3 model with the Dome hand calculation of the Minable Reserve for all the Faro deposit shows a decrease in tonnage from the F3 to the Dome hand calculation and an increase in grade.

3. Various other comparisons also suggest that the F3 model overestimated the tonnage and underestimated the grades.

4. An anticipated reserve from the F4 model may contain less ore but at a higher grade.

5. If the F4 (anticipated) reserve were valid then this would affect milling and mining rates as well as start up dates on the Vangorda Plateau.

Recommendations

1. Once the 1984 drill logs and assays have been entered to the database an updated revision of the Dome hand calculated reserve should be carried out.

2. The F4 modelling should be completed as soon as possible, on schedule. To this end it is imperative that the computer handling (be it Mintec or a 'standalone' system)be available by the end of December 1984.

Robin S. Tolbert

RST/DBC