CYPRUS ANVIL MININ CORPORATION INTEROF. CE CORRESPONDENCE

019585

To	J. Levanaho	cc: P. Taggart, J. Purkis, D. Gregoire, A. Laird
From	J. K. Carrington	
Date	December 22, 1982	
Subject	Oxide Metallurgy	

From Pete Taggart's memo of December 20th, Jaakko, I have the following comments:

(1) Oxide Stockpile

We do not have a good estimate of the percentage of fines the screening scenario would reject. However, Pete's guesstimate of 25-30% would indicate a mill feed tonnage loss of 300,000 - 350,000 tonnes or approximately one month's feed. Any mine plan based on this scenario must be adjusted accordingly, something you should discuss with John Purkis.

Please have John & Denis confirm the calculations in the Appendix re the time required to crush the entire pile. They should calculate the costs of delivery for crushing, removal of the fines, stockpiling and reclaim costs. Such costs must be included in the budget when milling resumes.

I understand you have already started on estimating the capital costs for this project. Please work closely with Allan, John and Denis. We need to know the cost of the modifications, what has to be done and how long it will take. John can then do a proper economic analysis of the benefit.

(2) Fineness of Grind

I do not understand Pete's chart on Pg. 6 when compared to that on Pg. 1 I would like to see the estimated improvement in metallurgy for

(1) doing nothing
(2) screen out fines
(3) get grind to 65 u
(4) (2) & (3) together

John will need this for his economics.

How quickly do you expect we can get up to the finest grind after you start milling? Obviously if there is a delay it must be reflected in the net balances which in turn affects the economics of screening.

Will we be re-installing the two tertiary screens? What are the implications of doing or not doing so?

.../2

(3) Process Controls

Please provide me with details of exactly what is being done or is planned to be done to ensure -

(1) reliable reagent feeding systems

(2) dependable on-stream analysis

(3) dependable level control systems

(4) mechanical and electrical deficiencies are eliminated. What are, say, the six or seven most frequent types of mechanical/ electrical failures? Can they be corrected? Have we reviewed past operational reports to identify the most common problems with the process circuit?

(5) clear operating procedures and parameters are in place - metallurgical, process, mechanical, electrical etc.

(6) do we need on-shift metallurgical assistance? If so, what, who? Will test work suffer? Are more technicians required?

(7) do we need 4th stage lead cleaning of the oxide ore? If so, what is involved? Benefits?

While the organizing to get answers to these questions is your responsibility, the details and assistance can be provided by others. Hence, I am sending copies of this to the other operating departments in order that they can start preparing their input.

Please let me know as soon as possible your schedule for these events.

JKC:vjw

Cyprus Anvil Mining Corporation

Inter-office Correspondence

file 273

To Distribution

Copy to

From P. Taggart

Date December 20, 1982

Subject OXIDE METALLURGY

In response to requests from T. Biggs and R. Vissagi, the following notes have been prepared which suggest a means by which the metallurgical responses of oxidized ores can be improved. The additional capital and operating costs required to effect these improvements have not been calculated due to limitations of time and data in Vancouver.

It is important to note that the attached data had not been reviewed with the Mill personnel at the time it was sent for typing.

P. Taggart

PT/mw

Distribution: R. Vissagi - Dome
T. Biggs - Dome
R. McCallum - Faro
J. Carrington - Faro
J. Levanaho - Faro

CYPRUS ANVIL

NOTES RE: THE METALLURGICAL PERFORMANCE OF OXIDE STOCKPILE ORES

1. SUMMARY

Among the many factors which effect the metallurgical performance of oxidized ores the following must rate amongst the most important.

1. the nature of the ore
2. the fineness of grind
3. the degree of process control and
4. the technical expertise of the flotation operating crews.

The amenability of oxidized ores can be enhanced by removing the fines (minus 12 mm) from the rod mill feed. The economic viability of this plan should be assessed quickly, since little time remains to carry out the work required if the plan is financially attractive.

The finer grinds expected as a result of recent plant and laboratory work will result in improved metallurgical response.

The time available prior to the plant start-up should be used to maximize our ability to control the difficult oxide flotation circuits.

No reasonable alternative should be ignored in an effort to maximize the technical expertise of the flotation operators prior to start-up and during normal operations.

On the assumption that the mill would process oxidized ores at 11,000 DMT per operating day the following metallurgical performance could be achieved if the fines were removed from the mill feed. The proposed metallurgical parameters are compared with those included in case P04.

A COMPARISON OF LEAD & ZINC RECOVERIES

P04 vs POTENTIAL METALLURGY

Pb & Zn Concentrate Grades Constant 60% Pb & 48% Zn Respectively

Period	P04 Lead Rec. %	P04 Zinc Rec. %	Potential Lead Rec. %	Potential Zinc Rec. %	Variance Lead Rec. %	Variance Zinc Rec. %
Month 1	67.7	68.4	72.0	71.5	4.3	3.1
2	68.2	69.4	72.5	72.5	4.3	3.6
3	68.2	69.4	73.0	73.5	4.8	4.1
4	69.6	72.2	74.0	76.0	4.4	3.8
5	70.0	73.0	75.0	76.5	5.0	3.5

The data above is considered attainable. In rèsponse to the question which was asked "What is the most optimistic improvement in metallurgy which MIGHT be achieved?" I would suggest, for this hypothetical case adding 2.0%, 1.0% and 1.0% to the respective values of lead recovery, zinc recovery and zinc grade.

In all cases it is expected that silver recoveries would increase in direct proportion to increases in lead recoveries.

2. ORE SUPPLY

The most predictable characteristic of the material in the oxide stockpile is its distinct lack of homogeneity. Visually some of the coarser ore appears to be minimally oxidized while much of the fines displays a closer resemblance to yellowish-brown clay than ore. It is this material which causes some material handling difficulties and for metallurgical reasons, does not constitute ore in the true sense of the word. Not surprisingly the degree of oxidation is much higher in these fines of higher specific areas.

To modify the present metallurgical forecasts without changing some of the important variables would be dangerous and based upon uninformed optimism since nobody knows the composition of the remaining oxide stock-pile. Positive adjustments could be made to predicted metallurgical values and be estimated with some confidence if the fines (minus 12 mm) were removed from the rod mill feed.

In 1973 plant tests were conducted which included a comparison of results obtained from the "primary fines" alone, a normal mixture of fines and coarse material and the coarse, fine crushing product only. The results are tabulated below.

Material	Duration of Test	Lead		Zinc	
		Grade	Recovery	Grade	Recovery
Fines	2 Shifts	63.0	73.8	52.0	70.0
Fines & Coarse(Blend)	2 Shifts	65.6	71.7	51.6	78.2
Coarse	1 Shift	64.6	80.4	53.2	83.8
Improvement coarse vs blend		(1.0)	8.7	1.6	5.6

While considering the limitations of relatively short, plant test programs and the difficulties in obtaining representative mill feed samples the following points are worthy of note:

a) The results obtained when milling the blend of coarse and fine are better than those currently achieved, probably due to the lesser degree of oxidation of this ore in 1973, and the more amenable ores stockpiled during the earlier years.

b) Lead grades appear to remain relatively constant.

c) Significant improvements in lead and zinc recoveries are attained when the fines are removed.

d) A small improvement in zinc grade could be expected when milling coarser material.

It is reasonable to expect that the differences shown would be the same were the test to be repeated today.

This information could be analysed in conjunction with some of the most recent recovery by size work carried out on samples of rod mill feed originating from the oxide stockpile. These data, which are reproducible illustrate that the recoveries of lead as PbO are approximately 20 units lower than the overall lead recovery in any size fraction.

RECOVERY BY SIZE DATA - KM095-2

Size mm.	% Weight		Assays % Pb		% Pb	Assays % Zn		% Zn	Assays % PbO		% PbO
	Head	Tail	Head	Tail	Rec	Head	Tail	Rec	Head	Tail	Rec
+106	2.3	4.9	.77	.28	63.6	1.45	.19	86.9	.18	.12	33.3
+ 75	6.6	15.6	.86	.22	74.4	2.62	.16	93.9	.17	.11	35.3
+ 33	34.5	25.1	3.13	.22	93.0	4.23	.14	96.7	.39	.12	69.2
+ 23	19.1	18.8	3.13	.36	88.5	6.37	.16	97.5	.49	.16	67.3
+ 16	10.6	9.4	3.03	.41	86.5	6.68	.19	97.2	.66	.24	63.6
+ 11	7.1	6.7	3.17	.53	83.3	7.09	.25	96.5	.95	.35	63.2
+8.6	3.8	3.8	3.35	.73	78.2	6.86	.30	95.5	1.26	.53	57.9
-8.5	16.0	15.7	6.09	3.04	50.1	6.60	1.81	72.6	NC	NC	NC

* NC = Not Calculated

Based on this information it may be reasonable to forecast improved metallurgical performance based on the supposition that the fines will be removed from the oxide stockpile. For this purpose one could conservatively assume that approximately one half of the previously measured improvements could be achieved at the same concentrate grades; this representing increases in lead and zinc recoveries of 4.5% and 3.0% respectively.

In addition to the metallurgical benefits shown above it is highly probable that savings in reagent costs would accrue as a result of minimizing the oxidized fines in the rod mill feed. Grinding and flotation circuit control would be enhanced through the expedient of providing a more uniform feed to the plant. Thus, not only would the fluctuations in fines, and therefore degrees of oxidation be reduced, but also the "coarse material would be crushed, screened and restockpiled in such a way as to maximize homogeneity.

The removal of fines from the oxide stockpile would have some actual and possibly incorrectly perceived disadvantages. Since it is possible that the fines are of slightly higher grade than the average for the stockpile there would be a reduction in grade in addition to the reduction in stockpile reserves. This reduction in tonnage could amount to 25-30% of the total reported tonnage. It must be remembered however that much of this material should not be considered as ore, particularly at prevailing, or even higher metal prices.

Naturally the expenditure of some capital funds and increases in operating costs will be incurred if the fines are to be removed. It should be noted that long term advantage at subsequent higher levels of mill throughput could be enjoyed as a result of some of the proposed changes.

The removal of fines can physically be achieved by taking the following steps on the assumption that the entire oxide stockpile would be screened prior to the commencement of milling operations.

1. Temporarily remove #12 conveyor tail pulley and construct a new TEMPORARY stacking conveyor to carry #11 conveyor discharge product to the East. To do this the cladding can be removed from the East end of the transfer tower.

2. Install upper decks on the two primary screens and complete the installation of level probes and controls in the crusher discharge pocket and primary screen surge bins.

3. Remove the cladding from the East end of the coarse ore storage.

4. Install additional conveying capacity to provide the capability of stockpiling coarse ore to the East of the existing coarse ore storage. This coveyor could be an extension of the #4 tripper conveyor, or a new unit fed by the existing tripper. The extension to the East may be in the order of 60-100 feet maximum.

Having made these modifications the entire oxide stockpile would be delivered to the primary crusher. The fines from the primary screens would report as usual to #11 conveyor and hence, via the new temporary conveyor to a stockpile due East of the transfer tower. The fines would be removed by loader and truck to a suitable dump site. (This represents an additional operating cost). Coarse ore would automatically report to #4 conveyor. Having filled the coarse ore storage,ore would be stockpiled using the new conveyor. The ore would be moved as short a distance as possible consistent with providing adequate room for further stockpiling. (This stockpiling, and subsequent reclaiming constitute additional operating costs.) Upon commencement of normal milling operations the tail pulley of #12 conveyor would be replaced and the coarse ore reclaimed through the coarse ore storage feeders and the primary crusher if the location of the stockpiles renders this cost effective. (The reclaiming costs would constitute additional operating costs.)

When developing and analysing the costs incurred the following points are worthy of note.

1. The new conveyor East of the transfer is strictly a temporary unit to be removed after 3 months operation. Budget estimates must be the cheapest consistent with safe operating practice.

2. The extension of stockpiling capabilities to the East of the coarse ore storage shed will be an invaluable asset when considering future increases in mill throughput levels under times of fiscal constraint.

Given the time available, and a complete lack of any relevant drawings, no meaningful estimates of capital costs could be developed for inclusion in these notes. In any event I would recommend that this work be done by the Mill and Mechanical Departments under clearly defined terms of reference. I am sure that all design and construction can be readily carried out using Cyprus Anvil employees. If such a scheme were proven to be economically viable, time, like money, is a resource which is in very tight supply!! The time required to crush the oxidized stockpile will be approximately 2 months. (see appendix)

3. THE FINENESS OF GRIND

Our level of understanding of grinding performance has increased substantially as a result of laboratory and plant testwork. On the basis that the mill resumes operation with the recommended changes in grinding media and pulp densities proposed by the Faro metallurgical group there is little doubt that finer grinds will be achieved.

Current metallurgical predictions for oxide ores vary with grind as shwon below.

Grind P_{80} (μ)	LEAD % Grade	LEAD % Recovery	ZINC % Grade	ZINC % Recovery
83	60.0	69.6	48	72.2
75	60.0	70.0	48	73.0
65	60.0	70.6	48	74.0

It is reasonable to expect a 0.3% and 0.5% improvement in lead and zinc recoveries at the same concentrate grades in anticipation of the improved fineness of grind which will be achieved.

It should be noted that the simple expedient of changing new and smaller balls will not immediately yield the desired results since grinding effeciency depends upon the presence of a size-graded ball charge. Adjustments to cyclone parameters will also take one or two months. Thus advantages from the finer grind will be realized over a definite period of time.

In view of the nature of the proposed mill feed it may be considered advantageous to re-install the two tertiary screens.

4. PROCESS CONTROLS

Unquantifiable benefits will be reaped by improving our control of the process through the use of instrumentation systems and the sound application of good milling practise.

Instrumentation can help immediately in the control of the process by providing reliable reagent feeding systems, dependable on-stream chemical analysis, dependable flotation cell and pump box level control systems, etc. This work is proceeding well and could be accelerated prior to the resumption of operations in some cases to assist personnel in the operation of the circuit.

Mechanical and electrical deficiencies also exist which adversely effect the operators ability to control the circuit. Wherever possible these items should be corrected prior to start-up again in an effort to stabilize the sensitive oxide flotation circuits. The need to install a fourth stage of lead cleaning when treating oxide ore could very quickly be established in the Faro metallurgy lab. If improved results are forthcoming the existing cleaner section can be readily modified to provide the additional circuits.

5. PERSONNEL

It is known that the best flotation operator has terminated. At this time it is not known how many other personnel trained as flotation operators have left. Recognizing that 12 hour shift schedules and the possibility of a shorter week initially will reduce the number of people required it is still probable that there will be a reduction in the overall level of technical expertise available on the flotation floor to handle one of the most difficult ore types - the oxide.

Since no training personnel remain in the present work force serious thought should be given to alternatives. For example staff personnel from the Metallurgical Department may be required to work on shift to work with the operators to permit a form of on-the-job training. Naturally, metallurgical testing activities will be compromised in the short term but justified at least on the basis that the plant will start-up in 1983. It remains for the Mill Staff, possibly in conjunction with the Personnel Department to ensure that effective, well-trained flotation operators are on shift as soon as possible, even to the point of hiring externally if necessary. In short, all the work involved in screening the feed, achieving a finer grind and improving the control of the process will be completely wasted if competent people are not available on the flotation floor to use these tools to the advantage of all.

APPENDIX

CALCULATION OF TIME REQUIRED TO CRUSH THE OXIDE STOCKPILE

In the hypothetical case considering five months milling of oxide ores assume:

1. Haulage truck average load 100 W.M.T.
2. Haulage truck delivery to primary crusher 12 trucks/hour
3. 10.5 operating hours/12 hour shift, 7 days/week operation
4. 2 shifts per week required for crushing plant maintenance (85% plant availability)

Therefore average daily crushing rate will be:

12 X 100 X 10.5 X 2 X 0.85 = 21,420 WMT/calendar day

Oxide milling for 5 months, say 88 operating days @ 11,000 M.T./cal. day.

Total tonnage oxide to be milled = 11,000 X 88
= 968,000 M.T.

Time required to crush pile will be: $\frac{1,210,000}{21,420}$

= 56.5 days

i.e. 2 months required in which to crush the oxide stockpile.

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2	68.2	69.4	72.5	72.5	4.3	3.6
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5	70.0	73.0	75.0	76.5	5.0	3.5

b) Lead grades appear to remain relatively constant.

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RECOVERY BY SIZE DATA - KM095-2

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+106	2.3	4.9	.77	.28	63.6	1.45	.19	86.9	.18	.12	33.3
+ 75	6.6	15.6	.86	.22	74.4	2.62	.16	93.9	.17	.11	35.3
+ 33	34.5	25.1	3.13	.22	93.0	4.23	.14	96.7	.39	.12	69.2
+ 23	19.1	18.8	3.13	.36	88.5	6.37	.16	97.5	.49	.16	67.3
+ 16	10.6	9.4	3.03	.41	86.5	6.68	.19	97.2	.66	.24	63.6
+ 11	7.1	6.7	3.17	.53	83.3	7.09	.25	96.5	.95	.35	63.2
+8.6	3.8	3.8	3.35	.73	78.2	6.86	.30	95.5	1.26	.53	57.9
-8.5	16.0	15.7	6.09	3.04	50.1	6.60	1.81	72.6	NC	NC	NC

* NC = Not Calculated

THE MILLING OF OXIDIZED ORES

The following notes should be read in conjunction with the memorandum "Oxide Metallurgy, P. Taggart to Distribution, December 20, 1982" and relate specifically to the "Asset Preservation" Case.

Problems

Poor metallurgical performance of oxidized ores due to:

1) the nature of the ore,

2) deficiencies in circuit control,

3) a shortage of well-qualified flotation operators, and

4) relatively coarse flotation feed grinds.

Solutions

1) Remove the "fines" from the oxide stockpile prior to the flotation to effectively reduce the degree of oxidation in the mill feed.

2) Continue work currently in progress to complete the fine tuning of the new mill circuitry. Some additional systems may be required.

3) Ensure that the current deficiencies in operator expertise are rectified by:

   a) thorough pre-start-up training and subsequent on-the-job instruction.

   b) recruiting well qualified flotation and grinding operators as necessary, and

   c) utilizing available staff expertise as required.

4) Improve the flotation feed grind using the information generated by recent plant and laboratory test programs.

5) Pursue the test programs currently in progress to discover other means by which the metallurgical performance of oxidized ores can be improved.

Results

Metallurgical forecasts have been modified to reflect the implementation of the programs identified above. For comparison purposes, the production statistics shown below are those forecast to be achieved during the third month of operation. The PO_4 figures are shown as the BASE CASE. The adjusted data represents the REVISED CASE. It should be stressed that the values used are best estimates based upon old data. It is strongly recommended that a short laboratory test program be conducted to confirm the forecast data. The Net Smelter Return (F.O.B. Load-out) values are predicated upon milling 10,000 D.M.T. of oxidized ore. The calculations are appended hereto.

	LEAD		ZINC		N.S.R.
	Grade % Pb	Recovery %	Grade % Zn	Recovery %	F.O.B. Load-Out ($)
Base Case (PO_4)	60.0	68.2	48.0	69.4	175,780
Revised Case	60.0	75.0 + 6.8	49.0	74.5 + 5.1	195,946

+ 8.7
+ 5.6
+ 1.6
- 1.0

NB Improvements in cash flow based upon enhanced silver recoveries are not taken into account in these calculations.

The results indicate an increase of approximately $20,000 N.S.R., F.O.B. load-out per 10,000 DMT milled if the oxidized stockpile were to be screened prior to the flotation process.

Economic Viability of Screening The Stockpile Normal Plant Feed

The following calculations are based upon rough production cost estimates to compare the BASE CASE with the REVISED CASE. For this comparison G + A costs have not been included.

COMPARISON OF PRODUCTION COSTS

Cost Centre	Base Case $/DMT	Revised Case $/DMT
Mining	0.50	1.00
Milling	13.00	12.00
Power	3.00	3.00
Coal + Environment	1.00	1.00
TOTAL	17.50	17.00

The mining costs in the BASE CASE are those costs incurred in feeding the oxide stockpile ores to the primary crusher. The costs are doubled in the REVISED CASE to reflect the additional material handling costs.

The milling costs are reduced in the REVISED CASE to recognize the overall reduction in reagent consumptions which will occur as a result of removing the highly oxidized fines.

Power costs are calculated assuming an overall energy consumption of 45 Kwhr/DMT and 7¢/Kwhr energy price. The unit cost shown is somewhat lower than that calculated to reflect a favourable volume variance.

a) BASE CASE

			$(000)
Costs:	1,300,000 DMT X $17.50/DMT	=	22,750
Revenues:	130 X $175,780 (N.S.R. FOB load-out/ 10,000 DMT)	=	22,851
	NET CASH FLOW		101

Since no capital costs are required the relative positive cash flow is $101,000.

b) REVISED CASE

Assume 65% of the stockpile is available for milling following screening i.e. 65% of 1,300,000 = 845,000 D.M.T.

			$(000)
Costs:	845,000 D.M.T. X $17.00/DMT	=	14,365
Revenues:	84.5 X $195,946 (N.S.R. FOB load-out/ 10,000 DMT)	=	16,557
	NET CASH FLOW		2,192

The capital cost required to screen the oxide stockpile will be in the order of $500,000. Based on these rough numbers therefore it would appear beneficial to screen the fines from the stockpile since the comparable, relative cash flow will be in the order of $1,500,000.

General

Preliminary indications suggest tha revenues generated by the milling of oxidized ores in accordance with the "Asset Preservation" Plan will be minimal at best, particularly if screening of the ore prior to grinding is not carried out. Cash flows could be improved by maximizing the instantaneous milling rate of the oxidized material. Thus, in a normal 4 day work cycle:

		Hrs
	Maximum hours available	96
minus	Provision for start-up and shutdown	8
minus	4% operating time (96% availability)	4
	Effective operating hours	84

84 hours X 550 DMT/hour = 46,200 DMT/work cycle.

The number of effective operation hours can be increased by the judicious use of start-up and shutdown teams which may consist of one or two operators, a shift mechanic and shift electrician. The prime responsibility of these teams is to start all equipment motors prior to the start of the first regular shift of operations in a work cycle. By so doing, start-up delays are minimized. Conversely, the use of a shutdown team permits normal operations to continue dor a longer period of time during the last regular shift.

NET SMELTER RETURN CALCULATIONS

The calculations are based upon the following assumptions.

1. A mill feed of 10,000 DMT assaying 2.8% Pb and 4.8% Zn.

2. Exchange rate $1.23 Cdn ≡ $1.00 U.S.

3. Metal Prices

	U.S. ¢/lb	Cdn ¢/lb
Lead	31.0	38.1
Zinc	42.0	51.7

4. Transportation Costs

to tidewater:	$60.00	Cdn
ocean freight:	$17.00	Cdn
TOTAL	$77.00	Cdn

5. Treatment Charges

	U.S. $/DMT	Cdn $/DMT
Lead:	174.0	214.0
Zinc:	176.0	~~204.4~~ 216.3

6. Moisture Contents

Lead: 6.0% H_2O

Zinc: 6.0% H_2O

7. No taxes or royalties paid.

8. Metallurgy

Based on hypothetical case, 3rd month operation per attached memo.

BASE				REVISED			
Lead		Zinc		Lead		Zinc	
Grade % Pb	Rec. %	Grade % Zn	Rec. %	Grade % Pb	Rec. %	Grade % Zn	Rec. %
60.0	68.2	48.0	69.4	60.0	75.0	49.0	74.5

ITEM			BASE	REVISED
Concentrate Production				
Lead	concentrates produced	(D.M.T.)	318.3	350.0
		(W.M.T.)	337.4	371.0
	metal recovered	(lbs)	420,653	462,546
	payable metal	(lbs)	399,620	439,419
Zinc	concentrates produced	(D.M.T.)	694.0	729.8
		(W.M.T.)	735.6	773.6
	metal recovered	(lbs)	733,730	787,654
	payable metal	(lbs)	611,442	659,057
TOTAL concentrate Production		(W.M.T.)	1073.0	1144.6
Costs ($)				
	Transportation		82,621	88,134
	Lead treatment charge		68,116	74,900
	Zinc treatment charge		141,854	149,171
	TOTAL Costs		292,591	312,205
Value of Sales ($)				
	Lead concentrate		152,255	167,419
	Zinc concentrate		316.116	340,732
	TOTAL Value of Sales		468,371	508,151
NET SMELTER RETURN F.O.B. LOAD-OUT ($)			175,780	195,946

Copy in File B-0-510

MET ENGINEERS LTD.

(403) 463-1513 5404 - 40 Avenue Edmonton, Alberta T6L 1B2 (403) 892-3038

May 31, 1982

MEMO TO: Mr. W. N. Wallinger

FROM: Mr. P. J. Brown

SUBJECT: Site Visit - May 24th - May 29th, 1982

Following our discussion on this subject, I have attempted to indicate in this report pertinent data regarding those areas of the operation to which you directed my attention. Please let me know if this new format better serves your requirements and if there is anyway in which further modications can be made.

Plant Operation

1. Grinding

Although there has been much progress in optimizing the grinding circuit operation, average grind still remains at the 70 μm level. The reason for the coarse grind is still the low pulp densities in some of the grinding mills. Shown below in Table 1 are averaged data taken from recent grinding circuit studies.

TABLE 1

Critical Grinding Circuit Parameters

Unit	Density Target	Density % Average	Comments
Rod Mills 1, 2, 3 Rod Mill 4	86 86	80-82 80-83	Density control improved since recycled water removed from rod mills
Ball Mills 1, 2, 3, 6	83	84-87	Improvements in 1, 2, 3 since 20" cyclones installed
Ball Mills 4, 5	83	80-81	These units still are not functioning reliably despite 15" ø cyclone installation
Lead Regrind	85	60-70	Cyclopac D63 operational but not optimized
Zinc Regrind	70	L60	Cyclopac D10B not optimized

L means "Less than"

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I have recommended that further work with the adjustable apexes be halted and a suitable range of fixed apexes purchased. For the critical D15B cyclones, sets of ceramic apexes of 2.5", 3.0" and 3.5" should be purchased. Also, at least one spare D15B should be purchased on a most urgent basis.

Grinding mill power meters are still not receiving the attention they merit. If the electrical department cannot repair these units then this work should be contracted out.

The calculations on grinding rod consumption are most interesting. However, before a decision is made regarding a change in media size, I recommend that the rod mill rejects be weighed over a period of several days. Probably, you will find that steel rejection from the rod mills is of the order of ten percent or less, and certainly not as high as indicated by the theoretical calculations. (None-the-less, it is possible that the use of 4" rods throughout the mill may result in some cost savings.)

2. Flotation Plant

General Operation

There has been, in recent weeks, an increase in the incidence of pH probe failures with significant deleterious effects on metallurgy. I strongly recommend that these concerned meet and formulate a plan to solve this problem. The same group could most usefully address the phenomenon of reagent delivery system failures too.

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The Courier system seems to be functioning well and, as far as I could judge, the operators are most enthusiastic about using the data outputs. Now that the system is operational it may be worth considering some major remodelling of the hourly assay sheet. (I have mentioned to Jakko that Highmont do have an 8 - 24 hour trend processor with C.R.T. on key XRF streams: Cost quoted was less than $ 2,000.)

The valving systems, installed to permit changes in flowsheet are largely inoperative. This is most unfortunate because with different ore types, throughputs and metal loads, it may be most useful to be able to manipulate the circuits by simply changing a valve set.

I concur with the planned conversion of the remaining lead conditioner to rougher duty. It would be most useful, however, to have some before and after data regarding mass flows - possibly a plant sampling campaign might yield useful results.

Sodium Sulphite Test Program

I have reviewed the data available regarding the use of sodium sulphite. Shown below in Table 2 are data extracted from Stan Chemelyk's recent report.

TABLE 2

Metallurgical Comparison - Sulphite Effects

	Grade		Recovery	
	Lead	Zinc	Lead	Zinc
Standard Reagents - Lead	56.6(5.2)		69.3	
Standard Reagents - Zinc		48.1(2.4)		68.0
Sulphite Circuit - Lead	59.6(2.8)		71.8	
Sulphite Circuit - Zinc		48.6(1.2)		70.4

The numbers in parentheses in Table 2 are the calculated standard deviations of the data and show clearly the relatively greater scatter in the data for standard circuits. I suggest that this data be reviewed again and results beyond the standard deviation limits "culled" as being non-representative. Probably, the results will then show a somewhat less optimistic picture regarding sulphite effects.

However, despite these reservations, the sulphite effects are interesting and even reworking of the data may still show a significant advantage for sulphite usage. I concur with the idea of continued sulphite testing on non-oxidized ore species.

Treatment Schemes for Graphitic Quartzites

A synopsis of the work carried out at Kamloops aimed at the development of treatment schemes for graphitic quartzites was presented to the metallurgical group. Briefly, the best results appear to be generated by using starch to depress galena and graphite and then reactivating the galena using xanthate.

A review of test data now being processed will be available by month end: At that time a decision will be made regarding the direction and extent of future work. Probably the work will be focussed on starch or starch/SO_2 combinations either at the head of the circuit or as a post-flotation treatment scheme.

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Laboratory Flotation Studies

The development of laboratory techniques has improved markedly since my last visit. At least now the laboratory has cleaner test capability. I have the following comments regarding the progress currently in progress.

a) The advantage of R241/242 as a lead collector is well demonstrated in the laboratory. However, our plant experience suggests that the use of this material may present insurmountable operational difficulties. If 241/242 are considered for plant operation they should be introduced with great caution.

b) Although final results are not yet available, preliminary indications are that the lead regrinding effects tests did show, as anticipated, considerable metallurgical gains with finer regrinding.

c) The use of Depramin/Dextrine as a post-flotation treatment is to be tested in the laboratory. Tests will consist of conditioning plant lead concentrate pulps with depressants and then attempting to refloat the galena.

3. Oxide Ore Treatment

A review of the oxide ore metallurgy during the last three months indicates that metallurgy is remarkably uniform. The recent decline in zinc metallurgy being attributable to slightly lower zinc feed grades.

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TABLE 3

Oxide Ore Metallurgy*

Period	Lead		Zinc		Reagents g/tonne			
	Grade	Rec**	Grade	Rec**	NaCN	CaO	$CuSO_4$	Z-11
March 11th - 31st	59.2	69.2	48.4	74.1	90	5000	800	400
April 1st - 30th	59.4	69.8	47.9	72.3	100	6700	810	370
May 1st - 20th	57.7	68.9	49.3	69.7	91	6800	736	420

* D.P.R. data

** Recovery

An examination of the tailings assays for the same period indicates remarkable consistency. These consistent results, especially with regard to zinc assays are surprising since in the laboratory zinc tails are seldom above 0.5% zinc. This phenomenon is worthy of the most detailed investigation - all data suggests that even the most severely oxidized material should produce significantly higher zinc recoveries.

TABLE 4

Oxide Ore Tail Assays

Period	Lead	Zinc
March 11th - 31st	0.78	1.20
April 1st - 30th	0.75	1.18
May 1st - 20th	0.78	1.21

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4. Long Range Planning Options

The long range planning options available to the mill have been detailed in my notes to you on this subject. The key data is summarized below to serve as a reference. I recommend that Jakko and Ron re-calculate the K_{80} values when you finally select a case of more detailed examination.

TABLE 5

Comparison of Options

Power, Grinding Media and Metallurgy

Option	Tonnes/yr 10^6	KwH/year 10^6	Steel Used Tonnes	Change from Base Met Lead	Change from Base Met Zinc
Base Case	3.75	65.5	7157	ø	ø
1. High Tonnage	3.06	49.2	5416	0.5	1.0
2. Maximum Tonnage	4.73	63.3	6964	1.5	3.0
3. New Circuit Only - 7 days	3.40	37.8	4155	2.5	5.0
4. New Circuit Only - 5 days	2.45	26.4	2908	2.5	5.0

NOTES: Power is for primary grinding circuit only

Steel is based on 110 g/KwH - 1982 Y.T.D.

Metallurgy assumes recovery units gained at constant grade, of 0.10 and 0.20 for lead and zinc, per %-325 Mesh increase in primary grind.

Yours sincerely,

Peter Brown, P. Eng.
Consulting Metallurgist

PJB/rl

cc: Peter Taggart - Vancouver

R. Taggart

CYPRUS ANVIL MI NG CORPORATION INTER ICE CORRESPONDENCE

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To [redacted]	c.c. G.D. Biles
From S. Chmelyk	
Date April 26, 1982	

Subject OXIDE METALLURGY IN THE PLANT COMPARED TO PLAN

Discussion:

The following is the metallurgical balance for the period March 11th to April 18th during which we milled oxidized ore.

Product	D.M.T.	ASSAYS Lead	Zinc	Iron	g/t Ag	DISTRIBUTION Lead	Zinc	Iron	Ag
Dry Feed	416,011	2.99	4.87	26.69	34.78	100.00	100.00	100.00	100.00
Pb Conc.	14,289	61.04	5.26	8.01	514.82	70.21	3.71	1.03	50.84
Zn Conc.	31,636	2.56	47.91	11.97	59.70	6.51	74.74	3.41	13.05
Tailings	370,086	0.78	1.18	28.67	11.12	23.28	21.55	95.56	36.11

1982 FEB PLAN - Pb Gr. 60 ; Pb Rec. 75 ; Zn Gr. 50 ; Zn Rec. 75

As you can see the zinc metallurgy in the plant is quite close to plan. The only shortfall is about 2 units in zinc grade. We are quite able to achieve our forecast of lead grades. The largest shortfall is in the lead recovery - we are about 5 units below plan.

I have reviewed the information I have available on oxide ore and the following is the summary of this information.

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		Pb Gr.	Pb Rec.	Zn Gr.	Zn Rec.
Plant Test October 1973		65.6	71.7	51.6	78.2
Plant Test March 1979 [illegible]		55.6	78	47.5	57
Plant Test January 1982		64.4	71	46.9	64.6
October 1982 Plan		55	70	52	78
February 1982 Plan		60	75	50	75
Plant Test Estimate January		60	75	49	73
Kamloops Testwork Drill Cuttings Sample June 1981		50	65	52	75
Testwork 1980 Cyprus Anvil Sample representative of upper third of Oxide Stockpile	at	60	69	at 50	74

Note: The testwork at Cyprus Anvil on the drill cuttings sample consisted solely of rougher tests. Altough no estimate could be made of final concentrate grade the final tails assays were comparable to the Kamloops testwork so recoveries would be much the same.

The data from the plant test conducted in 1979 was not viewed as being indicative. Due to many problems encountered during the test such as an inadequate soda ash and hence a low lead rougher pH (9.1) the zinc metallurgy was very poor. Even with the many problems though the lead metallurgy was reasonably good. Some very bad problems were encountered during the January plant test such as virtually no lead rougher pH control and an inadequate supply of xanthate. At no time during the test was lead grade a problem and it was felt that had we been able to maintain a consistent lead rougher pH lead recovery would have been much better than the 71% achieved. The lead metallurgy during the test was very good considering the large number of problems we encountered.

It was felt that drill cuttings sample the Kamloops group did their testwork on, was not a representative sample due to its; 1. extreme oxidation and 2. the lead assays were lower than the plant test results and other laboratory testwork conducted on different samples. Based strongly on the results of the January plant test when we had very promising results on the lead circuit the lead grade was increased 5 units to 60 from the October 1982 plan of 55. The recovery, again based strongly on the promising results of the January plant test, was increased 5 units to 75 from the October plan of 70.

In retrospect we may have been optimistic in our forecast on the lead recovery as the sample in January was not a representative block of oxidized ore. The following is the comparison between head grades from the January plant test and the heads to date on oxidized ore:

	Pb	Zn	Fe
January	2.93	4.65	32.75
Mar.11-Apr.18	2.99	4.87	26.69

The thing to note is the differenc in the iron assays.

We are coming quite close to plan with our zinc metallurgy. Recoveries are very close to plan while grades are down about 2 units from plan. Based on the considerable problems encountered with zinc metallurgy during the January test the October forecast was adjusted downwards.

	Pb Gr.	Pb Rec.	Zn Gr.	Zn Rec.
October Forecast 1982	55	70	52	78
January Plan Test Estimate	60	75	49	73
February Plan 1982	60	75	50	75

It was felt that with better pH control on the lead roughers and an adequate supply of reagents we would be able to improve upon the results of the January plant test. Taking these factors into account the zinc metallurgical forecast was adjusted downwards from the October forecast by 2 units in zinc grade and 3 units in zinc recovery.

Steve Chmelyk

S. Chmelyk
Plant Metallurgist

SC/ct

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Monica
Please hold for filing
Thanks
Pete.

To	L.P. Taggart, G.D. Biles, W. Kleinschrot, Met Techs, P.J. Brown
From	S. Chmelyk — c.c. G. Piwowar, Shifters, Flotation Operators
Date	January 16, 1982
Subject	OXIDE ORE PLANT TEST JAN 4 - JAN 8 / 82

Discussion

In 1982 the mill is scheduled to treat 958,200 tonnes of material from the oxide stockpile. In an effort to establish the capabilities of the mill, metallurgical results, reagent scheme, etc and problem areas with particular attention to the lime system when treating this ore an oxide plant test was conducted over a period of 5 days 4-8 of January.

Results

The test as whole was inconclusive. A large number of factors contributed to the problem, the major ones were:

a) Inadequate control system for lime addition to the rod mills resulting in wide fluctuations in lead rougher pH.

b) Inadequate xanthate supply restricting the ability to try increased dosages to improve metallurgy.

c) The plugging of the soda ash loop removed our ability for sodium sulphite addition.

d) Problems with the primary crusher handling the fine wet muck resulted in an inconsistent blend between the primary fines and material from the secondary crushing circuit. This caused the oxidation level of the ore to the grinding circuits to vary dramatically.

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Results (cont'd)

Delivery of oxide ore began midnight January 4th to the primary crusher. The flotation circuit began to see the effects at 0800 hours on the 4th. The decision to terminate the test prematurely was made on dayshift January 8th when the primary crusher plugged for the second time. It was felt that we would only run into serious problems by continuing to run the wet muck in the extreme cold weather. The following is the metallurgical breakdown for the period that we were milling the oxide ore: Jan 4 - Jan 8 / 82

		Assays			Distribution		
Product	D.M.T.	Lead	Zinc	Iron	Lead	Zinc	Iron
Dry Feed	31,526	2.93	4.65	32.75	100.00	100.00	100.00
Pb Conc.	1,019	64.44	5.52	6.92	71.01	3.83	0.68
Zn Conc.	2,019	2.86	46.87	12.60	6.25	64.58	2.46
Tailings	28,487	0.74	1.63	34.15	22.73	31.59	94.22

It must be stressed that due to the many problems encountered during the test these values should not be used as the absolute. Suprisingly the lead circuit under controlled conditions performed above expectations. The most problems occurred with zinc recovery and grade. Despite enormous increases in reagent usage, especially copper sulphate and mechanical changes in order to pull the circuit harder we were unable to lower the zinc rougher scavenger tails below 1.1-1.2% Zn. Lab tests indicated considerable room for improvement on the zinc roughers. The largest problem with lead metallurgy was lack of rougher pH control. If the pH on the roughers dropped too low (below 9.5) massive frothing would occur and the lead scavenger tailings would climb alarmingly high. Too high a pH $>$ 10.5 and the lead rougher concentrate would become very high and again the scavenger losses would become excessive. These fluctuations in pH on the lead circuit would carry over and affect the zinc metallurgy as well. Based on the operation of the circuit when it was most stable the following are estimates of metallurgy on the oxide ore compared to plan.

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Results (cont'd)

	Jan Plant Test	1982 Plan
Pb Grade	60	55
Pb Recovery	75	70
Zn Grade	49	52
Zn Recovery	73	78

Approximate reagent usage during the test are as follows:

	gm/MT
NaCN	50 maximum due to environmental problems
$CuSO_4$	600-800
Z-11	270
CaO	5.8 (kg/MT)

Conclusion

Although the results of the test are inconclusive there were several problems established that will require attention before we mill the oxide ore and some observations that we can make:

1) The lime system to the rod mills must be revised to assure proper pH control on the lead circuit and adequate volume.
2) Two to three times the normal amount of copper sulphate may be required. We must ensure that CIL is able to supply us.
3) We will not be able to exceed a cyanide dosage of 50 gm/MT due to environmental concerns.
4) Problems in the crushing area oan be expected due to the fine wet nature of the ore.
5) Lead metallurgy is significantly better than forecast while zinc metallurgy is somewhat below that forecast.

S. Chmelyk
Plant Metallurgist
SC/ct