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Dear Stan:

REVIEW OF CYPRUS ANVIL METALLURGICAL DATA AND  
CONCENTRATOR EXPANSION CALCULATIONS

In response to your request, I have reviewed the draft AFE for an expansion of the Cyprus Anvil mill, together with a substantial number of metallurgical reports. My assignment from you was to concentrate on the metallurgical aspects of this proposal, including the calculations rather than the detailed engineering design of the expanded concentrator.

SUMMARY

The proposed expansion of the Anvil mill for an increased capacity to 450 tonnes per hour vs. the present average of 400 tonnes per hour includes increasing primary grinding capacity by a factor of 2.21 and flotation cell capacity by a factor of 2.17. These changes are projected in the AFE to result in a significant increase in grade and recovery for both lead and zinc concentrates. The expected metallurgical results are based mostly on improved liberation in grinding and lowered pulp density in flotation.

A review of the pilot plant test results, together with the quantified flowsheet of the proposed plant, indicates that if the feed rate is held at the 450 tonnes per hour level, there is a strong probability of substantial losses of lead through overgrinding.

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Some additional plant tests in the present Anvil mill are recommended for a method of more precisely evaluating the proposed concentrator expansion.

In addition, an alternate flowsheet is suggested which should be considered and investigated to minimize overgrinding with consequent loss in lead recovery.

### I. Introduction

The basic assumption made in the CAMC proposal for a concentrator expansion is that by doubling the primary grinding capacity per ton of throughput, and thereby reducing the size of flotation feed from its present nominal 130 micron size to a 50 micron size, (i.e., 80% finer than 50 microns, or noted as P80), substantial increases in both concentrate grades and recoveries will be possible. The assumption includes a corresponding increase in flotation capacity and concentrate handling equipment for the finer product. The basic metallurgical flowsheet, however, will remain essentially the same as at present.

Since none of the pilot plant tests nor the limited plant test have produced metallurgical results matching the projected results, a further analysis has been made to assess the validity of the basic assumption, and subsequently to determine what changes in the proposed expanded concentrator or in its operating practices might be suggested to more nearly meet the projected goals.

All of the metallurgical test work on the Faro III, Grum and Vangorda ores show that a finer than 130 micron primary grind is beneficial for improved recovery and grade of concentrates. The exact size of the optimum grind, however, cannot be defined as yet due to the large number of interacting factors which must be considered. Operating experience at Faro indicates that the size of mineral liberation varies among the various ore types treated. In addition, the test work on Grum and Vangorda samples shows a similar variation, as well as demonstrating, in general, that these two ores require a somewhat finer grind for liberation than do the Faro ores.

Several factors place a practical economic limit on the degree of grinding for maximum liberation, namely, cost

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of additional grinding, efficiency of practical classification equipment, losses in flotation recovery due to over-grinding and/or sliming, and the cost and difficulty of handling finer concentrates and tailings. All these factors must be balanced against that of added metallurgical recovery to establish optimum grind.

## II. Analysis of Data on Faro Ore

A review of the data in Lakefield Research Progress Report No. 6, December 7, 1979 indicates that the tests, PP 28-35, on the Faro III "New Ore Sample" are perhaps the most valid for pilot plant response of an average Faro ore even though the sample tested was somewhat oxidized and lower in grade than normal mill heads. Although the chemical analysis shows an apparently significant oxide content, the amount of sodium carbonate required

<u>Sample</u>	<u>Assays, %</u>			
	<u>Pb</u>	<u>Pb Ox.</u>	<u>Zn</u>	<u>Zn Ox.</u>
Bulk Sample No. 1	2.35	0.40	3.85	0.25
Bulk Sample No. 2	2.40	0.35	4.34	0.22

does not seem excessive when compared with plant practice or with the roughly three times larger quantity used in the pilot plant tests of Bulk Sample No. 1. Similar oxide analyses of current Anvil mill feed on a periodic basis would be useful for confirming, with certainty, the comparative validity of the sample used in the pilot testing.

The ratio of other key reagent usages to the soda ash requirement appears to be about the same for the pilot plant tests and the Anvil mill. For example, the ratio of soda ash to other reagents for pilot plant tests 28 and 29 is roughly the same as the mill test of November 13-15.

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REAGENTS USED

Grams/Tonne

<u>REAGENT</u>	<u>BULK SAMPLE NO. 2 LR 2202 TESTS</u>			<u>ANVIL MILL TESTS (1978)</u>	
	<u>28, 29</u>	<u>31</u>	<u>33, 34</u>	<u>OCT. 16-22</u>	<u>NOV. 13-15</u>
Na <sub>2</sub> CO <sub>3</sub>	3279	4497	4589	946	1564
Na <sub>2</sub> SO <sub>3</sub>	--	--	--	226	--
NaCN	213	256	360	86	82
Z-11 (Pb)	110	152	213	58	66
Z-11 (Zn)	202	297	399	91	181
CuSO <sub>4</sub>	800	1312	1474	222	346

The increased quantities of activator and collector reagents used in the Lakefield Research pilot plant tests appear to be more a function of small pilot plant effects rather than the oxidized condition of the sample.

A comparison of metallurgical results at comparable grind for the Lakefield pilot plant with those of the Anvil mill for 1979 and October 16-22, 1978 period provides a measure of confidence that Sample No. 2 was not altered sufficiently to materially change its metallurgical response.

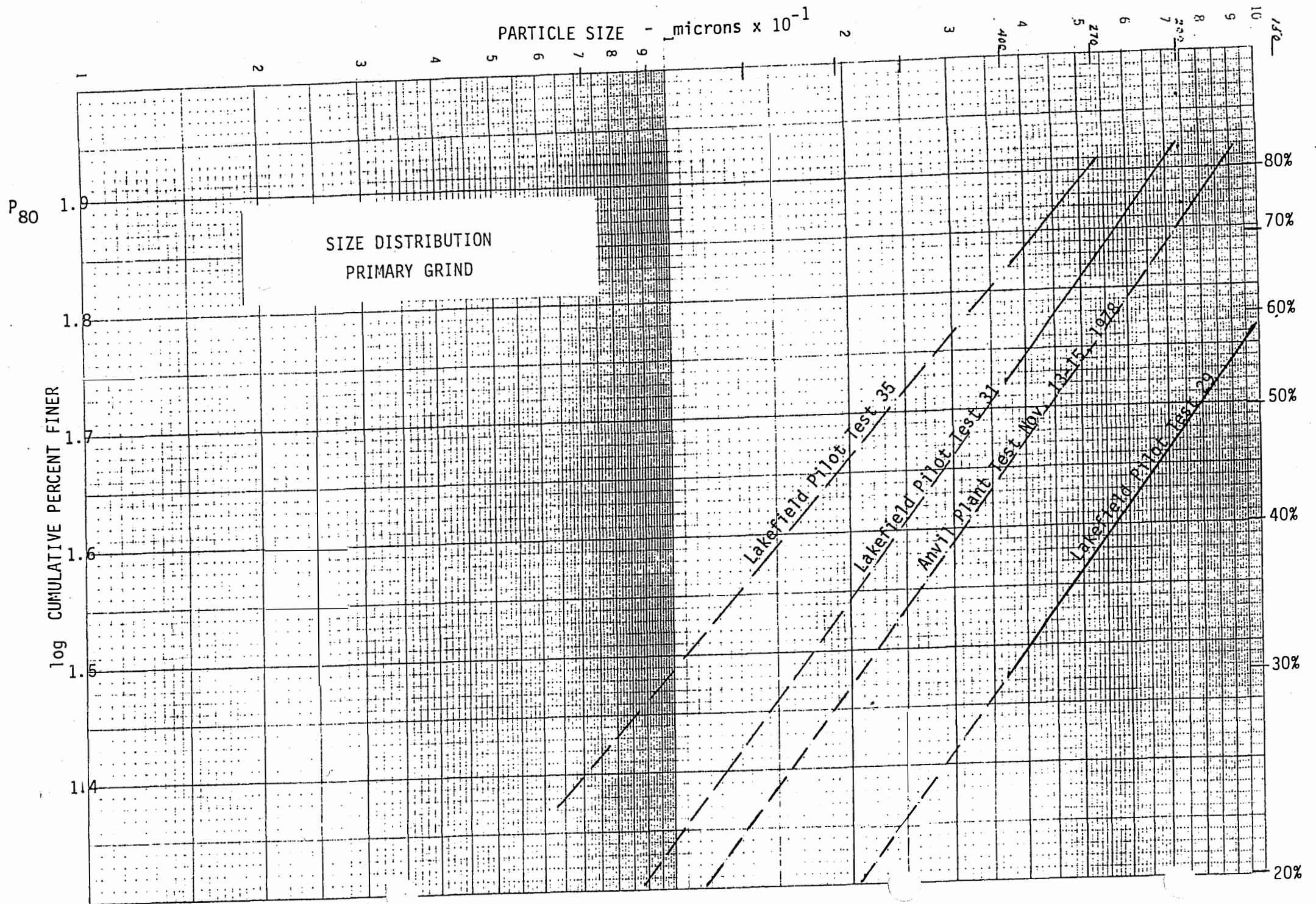
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		<u>PRIMARY GRIND P80</u>	<u>LEAD METALLURGY</u>		<u>ZINC RECOVERY</u>	
			<u>GRADE</u>	<u>RECOVERY</u>	<u>GRADE</u>	<u>RECOVERY</u>
Anvil Mill Tests						
Oct. 16-22, 1978	Feed	130*	3.0		5.6	
	Prod.		59.0	82.8	48.7	82.1
Anvil Mill Operation						
1979	Feed	130*	3.3		5.3	
	Prod.		61.4	83.7	50.4	80.5
Pilot Plant Tests						
28, 29	Feed	137	2.4		4.3	
	Prod.		71.6	79.1	54.2	83.6

\* Calculated with same method used in AFE.

The normal Anvil grind and the plant test grind for October 16-22, 1978, reported for the tailing at 60% minus 200 mesh, are close to the grind of tests 28 and 29. The grind for November 13-15, 1978 as recalculated from the tailings sizing of 74% minus 200 mesh is shown for comparison on the following graph giving the approximate size distributions of the flotation feeds for the tests tabulated above.

The Anvil plant test on November 13, 1978 and several of the Lakefield pilot plant tests were made at various degrees of primary grind. The metallurgical results for selected tests at various grinds are summarized as follows:



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	<u>P<sub>80</sub></u>	<u>LEAD METALLURGY</u>		<u>ZINC METALLURGY</u>	
	<u>GRIND</u>	<u>GRADE</u>	<u>RECOVERY</u>	<u>GRADE</u>	<u>RECOVERY</u>
Anvil Mill					
1979 Average	130	61.4	83.7	50.4	80.5
Oct. 16-22, 1978	130	59.0	82.8	48.7	82.1
Nov. 13 (30 hr.)	87	60.3	86.2	50.4	82.3
Lakefield Pilot Plant					
PP 28, 29 LR2202	137	71.6	79.1	54.2	83.6
PP 31	69	77.6	82.2	54.3	86.9
PP 33, 34	49	71.5	80.9	55.0	88.4
PP 35	50	74.7	82.3	55.5	89.1
PP 9, 10 LR2202	48	68.7	76.3	52.3	87.2
PP 11, 12 LR2202	31	66.3	76.3	51.9	84.2
AFE Projection	50	67.0	87.5	53.5	88.5

The grouping of the tests from the Lakefield Research pilot plant report LR2202 on Faro III ore is somewhat different from that used in the report summary. For example, tests 9 and 10 were duplicate runs at the same primary grind and, thus, have been averaged. Test 24 was run under different conditions. Tests 11 and 12, however, employed two stage classification in the primary grind circuit to attain the finer grind, thus, these sets, i.e., 9, 10 vs. 11, 12, appear to be a valid comparison. Similarly, the averaging of tests 31 and 35 as used by Lakefield Research is questionable since test 31 produced an intermediate grind of P<sub>80</sub> at 69 microns. Thus, the average results of tests 28 and 29, and tests 33 and 34 can be compared with test 31.

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Comparison of the three levels of grind in the pilot plant tests shows a decrease in lead recovery and grade at the 49 micron grind as compared to the coarser 69 micron grind. Test 35 is included since the Pb circuit conditions were reported to be a duplicate of test 31 and yet a 3% loss in lead grade was experienced. This raises the question of the deleterious effect of overgrinding the galena with resulting slime losses.

The results of these Lakefield Research pilot plant tests suggest that lead recovery might be maximized at a primary grind of about P<sub>80</sub> at 70 microns with zinc recovery maximized at the finer size of P<sub>80</sub> at 50 microns. This could be accomplished with a flowsheet similar to that of the Paragsha concentrator of Cerro de Pasco (4,5)\* discussed later in this report.

Since the existing Anvil mill, if operated at a rate of 236 tonnes/hour, would nearly duplicate the flowsheet and the proportional capacity at a 50 micron grind of the proposed expanded mill, then the November 13, 1978 plant test(1,2)\* should be a good indication of the benefits to be gained by lowering the primary grind from 130 microns to about 87 microns, but yields no clue on what would be obtained with a 50 micron grind. The fact that the capacity of the regrind mills will not be increased in the expansion must be noted, however, since this means that during the plant test on November 13-15, 1978, the regrind capacity apparently was proportionally greater than is proposed for the expansion.

Comparing the metallurgical results of November 13 with those of October 16-22, an increase of 1.25% in Pb concentrate grade and 3.44% in recovery is indicated for the finer grind at 87 microns. The Pb regrind was approximately comparable in the two periods since the decreased grade of the rougher concentrate, hence a greater quantity, would compensate for the lower relative quantity and higher grade during the October 16-22 test period.

\* References listed on Page 21.

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METALLURGICAL BALANCES (2)

October 16 to 22, 1978

<u>PRODUCT</u>	<u>D.S.T.</u>	<u>% ASSAY</u>			<u>% DISTRIBUTION</u>		
		<u>LEAD</u>	<u>ZINC</u>	<u>IRON</u>	<u>LEAD</u>	<u>ZINC</u>	<u>IRON</u>
Dry Feed	79,191	3.22	5.43	23.26	100	100	100
Lead Conc.	3,571	59.03	8.88	7.00	82.76	7.37	1.36
Zinc Conc.	7,250	2.10	48.70	12.44	5.97	82.14	4.90
Bulk Conc.	--	--	--	--	--	--	--
Tailings	68,370	0.42	0.66	25.26	11.15	10.42	93.74
TOTAL					82.76	82.14	

NOVEMBER 13, 0:00 - NOVEMBER 14, 0800, 1978

<u>PRODUCT</u>	<u>D.S.T.</u>	<u>% ASSAY</u>			<u>% DISTRIBUTION</u>		
		<u>LEAD</u>	<u>ZINC</u>	<u>IRON</u>	<u>LEAD</u>	<u>ZINC</u>	<u>IRON</u>
Dry Feed	11,329	4.01	5.88	30.96	100	100	100
Lead Conc.	649	60.29	7.20	8.40	86.20	7.00	1.60
Zinc Conc.	1,087	1.58	50.43	11.90	3.80	82.30	3.70
Bulk Conc.	--	--	--	--	--	--	--
Tailings	9,593	0.48	0.74	34.65	10.00	10.70	94.70
TOTAL					86.20	82.30	

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The higher grade feed<sup>(3)</sup> of 9.89% Pb + Zn and 30.96% Fe on November 13, compared with 8.65% Pb + Zn and 23.26% Fe averages for October 16-22 introduces the possibility of a recovery bias due to ore grade or type. To roughly evaluate this factor, the Anvil mill daily results for October and November 1978 were correlated with the finding that within this range of feed grades, roughly each 0.75% increase in feed grade, i.e., Pb + Zn, is accompanied by a 1% increase in lead recovery. Thus, it seems that only one-half of the 3.44% increase in lead recovery should be ascribed to the finer grind.

The extrapolation of these results to a proportional gain at a finer 50 micron grind is questionable in view of the Lakefield pilot test results. Furthermore, the analytical results for November 13 show some disquieting figures for the grades of the rougher concentrates and final tails, as noted below.

PLANT TEST, OCTOBER - NOVEMBER 1978<sup>(2)</sup>

X-R.F. HOURLY AVERAGES PER DAY

DATE	Pb CIRCUIT						Zn CIRCUIT					
	Pb FEED	RGHR. CONC.	SCAV. TAIL	FINAL CONC.	CLNR. TAIL	FINAL TAIL	Zn FEED	RGHR. CONC.	SCAV. TAIL	FINAL CONC.	CLNR. TAIL	FINAL TAIL
Oct. 16	3.2	30.3	0.26	61.5	2.7	0.43	6.3	32.7	0.60	49.3	2.6	0.80
Oct. 17	2.6	24.7	0.36	60.4	2.6	0.40	5.6	34.0	0.50	49.8	3.4	0.77
Oct. 18	2.8	25.0	0.44	61.7	2.7	0.42	5.4	31.6	0.70	49.5	2.3	0.73
Oct. 19	3.1	25.4	0.49	60.8	3.1	0.45	5.8	36.4	0.83	49.5	1.5	0.67
Oct. 20	2.7	23.6	0.35	56.4	2.0	0.43	5.3	37.6	0.77	49.0	1.2	0.63
Oct. 21	3.6	24.8	0.48	57.0	2.9	0.45	5.7	35.5	0.83	48.2	1.6	0.80
Oct. 22	3.2	27.9	0.46	59.5	3.0	0.48	5.3	34.2	0.60	49.0	2.3	0.70
AVERAGE	3.0	<u>26.0</u>	0.41	59.6	2.7	<u>0.44</u>	5.6	<u>34.6</u>	0.69	49.2	2.1	<u>0.73</u>
Nov. 13	4.2	<u>21.5</u>	0.60	61.2	5.6	<u>0.50</u>	6.0	<u>32.1</u>	0.70	49.7	2.8	<u>0.80</u>
Nov. 14	3.7	17.8	0.50	60.2	4.0	0.40	5.7	30.9	0.50	50.1	2.1	0.60
Nov. 15	3.4	18.6	0.50	64.2	3.7	0.40	5.6	36.4	0.50	51.1	2.2	0.60
AVERAGE	3.8	19.3	0.53	61.9	4.4	0.43	5.7	33.1	0.57	50.3	2.4	0.67

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Note that for November 13, the rougher concentrates were appreciably lower in grade than during the October 16-22, 1978 period and the tails for both lead and zinc were higher.

Zinc metallurgy during the 1978 plant test was essentially unchanged by the finer grind. The Lakefield tests on Sample 2, however, do show a progressive increase in zinc recovery of about 5%. The optimum balance of primary grind, lead regrind and zinc regrind does not emerge, nevertheless, as a clear cut picture for either the Faro or Grum ores in the results from the Lakefield Research pilot plant tests.

The mill feed rate during the period October 16-22, 1978 was apparently held at an average of about 427 tonnes per hour. During the first three weeks of December 1979, the Anvil mill feed rate varied from 490 to 360 tonnes per hour. The reported fineness of final tailings varied from 41% to 77% minus 200 mesh. This would be an apparent P<sub>80</sub> variation of about 200 microns to 80 microns. The lead and zinc recoveries for the same period do not appear to reflect a corresponding variation, although a slight correlation with zinc recovery can be found.

The above observations suggest that the design of any flowsheet for the expanded Anvil mill should have sufficient flexibility built in so that both the grinding and the flotation circuits can be optimized. They also suggest that a substantial metallurgical program be carried on to determine the controlling factors of ore mineralogy and flotation chemistry which affect recovery and grade. Finer grinding, to some degree, is obviously required; however, other factors could well be equally important.

## II. Grinding Plant Expansion Calculations

The justification for the Anvil mill expansion presents detailed calculations for attaining a 50 micron grind. These are based on accepted Bond calculation practice and are supported by documentation of widespread plant practice. For Faro III ore, a number of separate work index measurements have been tabulated and then a stated "conservative" work index of 13.224 has been employed in the expansion AFE.

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A review of the data available on the October through November 1978 Plant Test, as well as the operating report for December 1979, indicate that the Faro I ore milled was not nearly as difficult to grind as the "conservative" employed in the draft AFE. A revised work index of 11.2 can be shown to yield product sizes closely matching actual December 1979 plant results and the measured size for November 13, 1978 as can be seen on the attached tabulation (Columns 4 and 7). The grind for the October 16-22, 1978 period was calculated by the AFE technique and, assuming that the work index was the same as in the other two periods, is somewhat at variance with the "estimate based on technicians vs. operators." If, however, the ore treated during October 16-22 was appreciably softer (13% pyrite vs. 22% pyrite for November 13, 1978), then its work index would be about 10.48 to yield a primary grind of 130 microns and a tailing of about 115 microns as reported. (Column 10). If these calculations are valid, then the expanded plant would be able to grind 531 tonnes per hour to a 50 micron size, (Columns 11 and 14), when operating on ore similar to that of December 1979. These comments assume that the Faro I and Faro III ores are very similar, as stated by Anvil management and as shown in the AFE.

Since the operating plan for the expanded Anvil mill contemplates operating the existing grinding circuit at 200 tonnes per hour rather than its present design capacity of 378 tonnes per hour (or the operating average of 400 tonnes per hour), some careful consideration should be given to the danger of overgrinding at the AFE stated tonnage rates. Unless close control of the primary grind is maintained by varying tonnage throughput, losses resulting from overgrinding could easily offset the benefits of a finer grind.

The economic substantial advantage of milling more ore than called for in the design criteria (i.e., 450 tonnes/hour) rather than maximizing recoveries, is readily apparent. This approach has some known constraints beyond the scope of the present analysis.

If the Faro III ore is not greatly different in hardness from those ores milled in the latter periods of 1978 and 1979, it would appear, therefore, that the grinding plant design includes an 18% safety factor (531 vs. 450 tonnes/hour), assuming that the 50 micron grind is found to be that which actually does maximize recovery.

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AFE GRINDING CALCULATIONS  
 COMPARED WITH THREE PERIODS  
 OF ANVIL MILL OPERATION

	Existing Grinding Circuit											Proposed New Grinding Circuit		
	1	2	3	4	6	7	8	9	10	11	13	14	15	
	AFE	Lower Work Index	December 1979	December 1979	Nov 13, '78	Nov 13, '78	Oct 16-22 '78	Oct 16-22, '78	Oct 16-22, '78	Potential to 50 $\mu$	AFE	Potential to 50 $\mu$	Lower Work Index	
1 Feed rate Tonnes/hour	200	200	400	400	321	321	428	428	428	236	250	295	250	
2 Feed size P <sub>80</sub> = 12,500 microns														
3 Work Index - assumed	13.224	11.2	13.224	11.2	13.224	11.2	13.224	11.2	10.476	11.2	13.224	11.2	11.2	
5 Rod Mill Discharge - Calculated P <sub>80</sub> in microns	582	447	1579	1266	1176	928	1718	1385	1266	583	693	693	535	
7 Primary Ball Mills (Secondary Grind) "	165	122	530	406	371	281	589	452	405	165	120	120	89	
9 Secondary Ball Mills (Tertiary Grind) "	49 $\mu$	36 $\mu$	174 $\mu$	130 $\mu$	118 $\mu$	87 $\mu$	196 $\mu$	146 $\mu$	130 $\mu$	49 $\mu$	49 $\mu$	49 $\mu$	36 $\mu$	
12 Calculated Tailings P <sub>80</sub> - microns				115		80		137	115					
13 Corresponding % minus 200 mesh				60%		76%		53%	60%					
15 Measured Tailings P <sub>80</sub> equivalent				114		82		116	115					
16 Measured Tailings, % minus 200 mesh				63.5%		74%		60%	60%					
Notes:														
Col. 1 and 13 - AFE Calculations														
Columns 3, 6 and 8 are from Anvil Mill Records with AFE Calculations														
Columns 4, 7 and 9 assume a lower work index														
Line 12 assumes regrinding yields a reduction ratio of														
2:1 for Pb rougher conc.														
2.5:1 for Zn rougher conc.														
as used in AFE														

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#### IV. Mineralogy, Liberation and Flotation Response

Variations in ore grade and mineralogy on a day-to-day basis at the Anvil mill have a significant effect on lead recovery. For example, during the period from September to December 1978, the grade varied from a low of 7.3% Pb + Zn to a high of 14.6%. Lead recoveries, in general, correlate with grade and were 76% for the low grade feed and 90% for the high grade feed. Zinc recoveries, however, do not appear to fit this correlation. This suggests that the metallurgy is controlled by ore mineralogy and by ore chemistry to a greater degree than by size of primary grind.

Although the pyrite to pyrrhotite ratio is plotted on a daily basis in the mill records, no significant correlation can be found during the 1978 period nor during December 1979. On the other hand, the high pyrite ores appear to yield the higher recoveries.

One explanation of these observations is that the galena crystal size in the high pyrite and high grade ores is coarser and therefore more completely liberated at the normal mill grind. If such a condition occurs as frequently as these limited records indicate, then a constant primary grind of P<sub>80</sub> at 50 microns could result in severe losses of lead by overgrinding such more coarsely crystallized galena ores.

The relatively constant metallurgical response of the sphalerite indicates that, in general, it is of finer grain size and thus, the zinc regrind circuit compensates for ore grade changes. Furthermore, the primary function of the zinc regrind circuit seems to be that of liberation of sphalerite from pyrite, based on review of the Lakefield reports.

From the above cited 1978 Anvil mill records, two other observations can be made. In general, lead concentrate grade and recovery move together in the same direction. This would imply that the normal Anvil mill grind is adequate for galena liberation some of the time and that when finer grained ores are encountered, both recovery and grade suffer.

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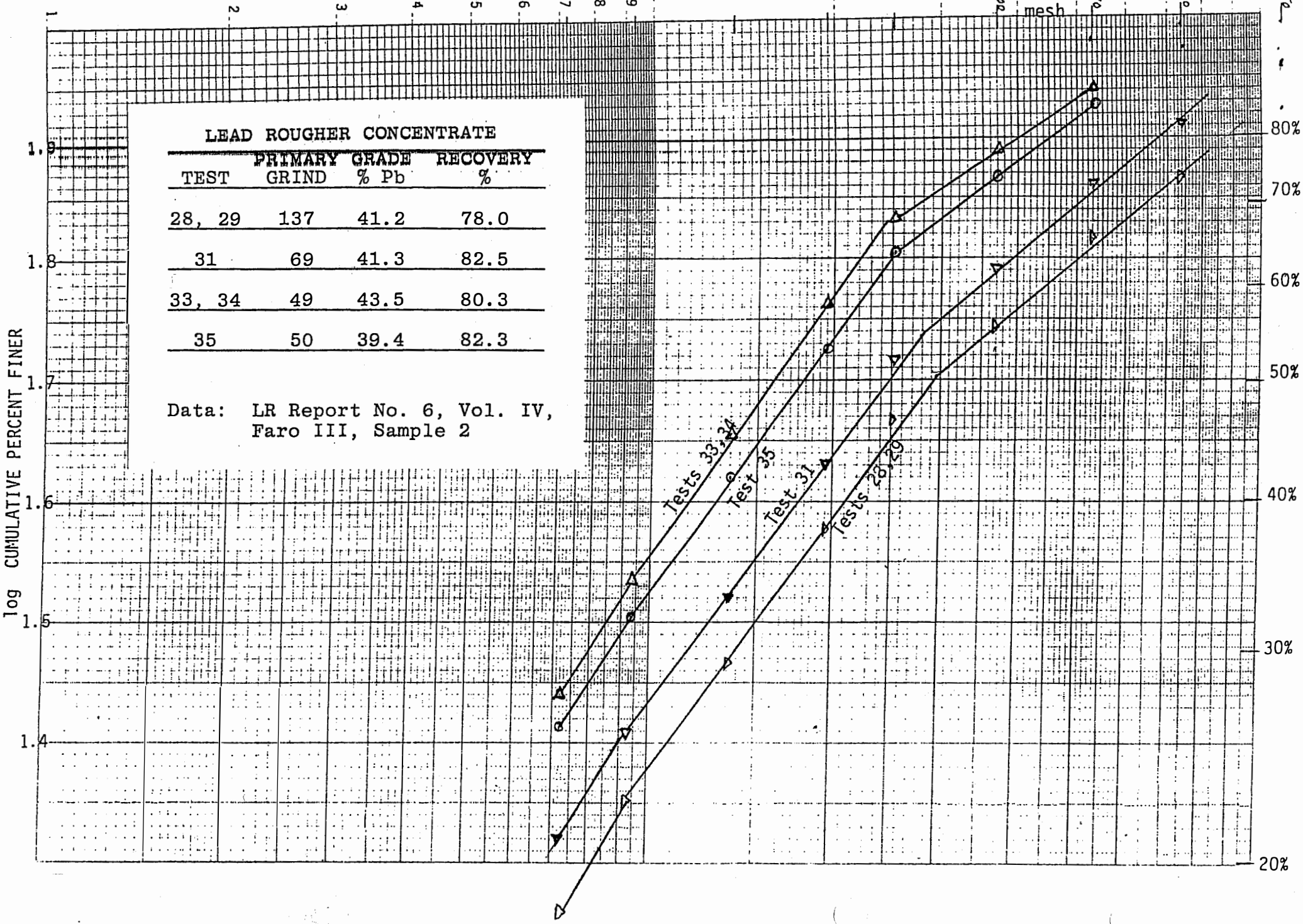
On the other hand, sphalerite grade and recovery appear to move in opposite directions, namely as grade goes up, recovery goes down. This can occur with poorly liberated ores or those in which two minerals of rather close flotation response are being selectively floated. Both conditions are present in the Anvil zinc circuit where the separation appears to be between sphalerite and pyrite/pyrrhotite. As the rougher concentrate regrind is taken to a finer size, zinc liberation is improved so that a higher grade concentrate is possible. At the same time, however, slime losses increase and selectivity between the liberated minerals decreases during flotation at the finer size, so grade suffers by contamination. Since additional zinc flotation cleaning capacity was not added during the November 1978 plant test, the grade of the final zinc concentrate increased only slightly but the recovery remained the same.

Further evidence of these effects, as well as some additional insight to grinding and flotation behavior is available by studying the Lakefield results on Sample No. 2 of Faro III ore. The following two graphs and tables show the size distribution and metallurgy for the rougher concentrates for tests PP 28 + 29, 31, 33 + 34 and 35.

As noted previously for this set of tests, the cleaner concentrate lead metallurgy was better for the intermediate primary grind of P<sub>80</sub> at 69 microns than with the finer 49 micron grind. The rougher concentrates show the same pattern. The metallurgical results illustrate the counterbalancing effects of improved liberation vs. increasing slime losses with finer grinding. Lead recovery increased by 4½ percentage points by changing the grind from 130 microns to 69 microns. The additional step to 49 microns lost either recovery and/or grade. Tests 33 and 34 compared to test 35, show a pattern of good liberation but poorer selectivity at a fine size, namely recovery and grade are counterbalancing. Below 10 microns, these distributions display a slightly increased downward slope which can be accounted for by increasing slime losses in this range.

The plots of the zinc rougher concentrates, when compared to those of the lead rougher concentrates, show that the galena is selectively ground to a finer size as would

PARTICLE SIZE - microns x 10<sup>-1</sup>



LEAD ROUGHER CONCENTRATE			
TEST	PRIMARY GRIND	% Pb	RECOVERY %
28, 29	137	41.2	78.0
31	69	41.3	82.5
33, 34	49	43.5	80.3
35	50	39.4	82.3

Data: LR Report No. 6, Vol. IV,  
Faro III, Sample 2

150  
100  
75  
50  
30  
20  
mesh  
270  
200  
150

PARTICLE SIZE - microns  $\times 10^{-1}$

mesh  
10  
9  
8  
7  
6  
5  
4  
3  
2  
1

log CUMULATIVE PERCENT FINER

1.9

1.8

1.7

1.6

1.5

1.4

80%

70%

60%

50%

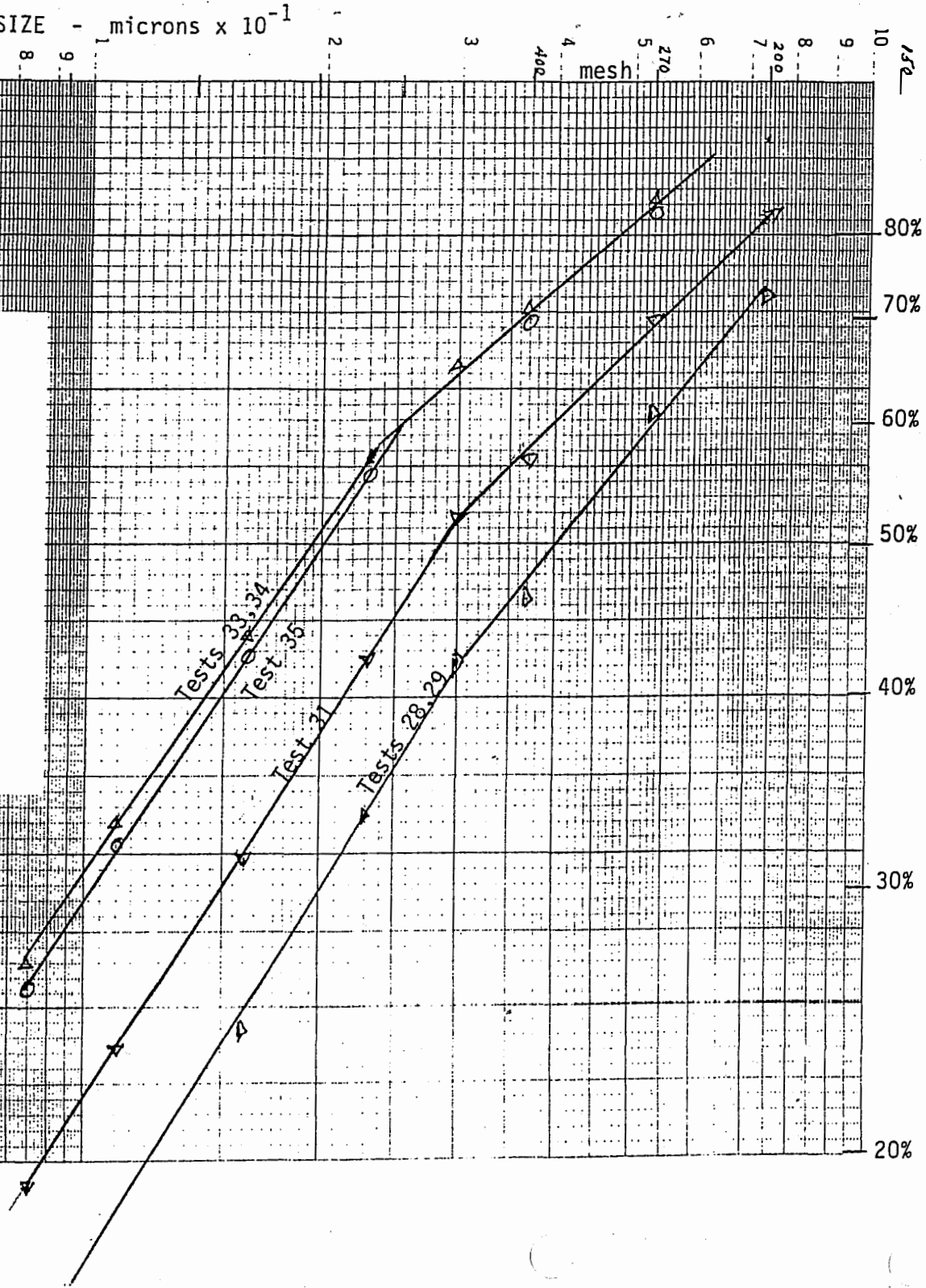
40%

30%

20%

ZINC ROUGHER CONCENTRATE				
TEST	PRIMARY GRIND	GRADE % Zn	RECOVERY %	
28, 29	137	34.2	85.6	
31	69	44.2	87.4	
33, 34	49	32.5	89.1	
35	50	32.2	90.0	

Data: LR Report No. 6, Vol. IV,  
Faro III, Sample 2



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be expected. The zinc plots are normal for increasing fineness. Furthermore, down to the  $8\frac{1}{2}$  micron limit of the sizing analysis, no noticeable break in the distribution occurs, thus, the sphalerite is indicated to be less susceptible to slime losses than the galena. The increase in zinc recovery with increasing fineness of grind, indicates improved zinc-pyrite liberation at the finer grind. The slight decrease in concentrate grade with increasing fineness is undoubtedly due to an increase in fine pyrite mechanical entrapment.

In plotting size distributions of essentially single mineral concentrates or of ball mill discharge products, for example, the log-log type of plot will normally result in a straight line from the finest size up to approximately the 90% passing point. This is the case if a consistent sizing method has been used throughout the plotted range. When two methods of sizing, namely screen sizing and sedimentation have been used as in the Lakefield tests, then two straight line portions of the plot are produced. This accounts for the angle at about the 25 micron point on the lead concentrate plots and the 30 micron point for the zinc concentrate plots.

Such size distribution plots are used in a number of ways to analyze mineral processing systems. For the lead, rougher concentrates within the screen size range of 200 to 400 mesh, the slopes of the distribution lines are essentially the same for tests 28, 29 and tests 31 and 35. This indicates that within this size range, both good liberation and good flotation recovery were experienced. The zinc rougher concentrate, on the other hand, displays a different response in that the slope for tests 33 to 35 is noticeably less than for tests 28, 29 in the 70 to 30 micron range, which is also well within the optimum flotation range. This means that the liberation of the sphalerite is better at 30 to 40 micron size than at 70 microns and greater.

The respective slopes of the distribution plots in the range of 25 microns down to 10 microns show the opposite pattern. Since the galena is obviously well liberated, the increased slope at the finer grind indicates an increasing loss of fine galena in flotation as its size is lowered by grinding. Conversely, the zinc concentrate

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curves within this range are parallel, indicating good liberation and good recovery.

These effects are a function of mineral liberation and normal flotation response. They are quite independent of the different effect which occurs when oxide minerals are present. On this basis, the somewhat oxidized condition of the Lakefield samples can be disregarded.

### Conclusion

The conclusions which can be drawn from this approach are that to produce an optimum lead recovery from Anvil ores, the mill should be designed and operated so as to grind the galena as closely as possible within the truncated size range of 100 to 20 microns. Considering that different types of Anvil ore have varying sizes of galena grains, a provision should be made for rougher flotation at an intermediate primary grind and then the rougher tailing should be ground to a finer size prior to a scavenger lead flotation step. Obviously, efficient cyclone classification is required to attain such a compression of particle sizes within the optimum recovery range.

It should be clearly understood that this analysis is based on very limited Anvil mill data combined with the rather detailed data from the Lakefield pilot plant. If a series of data accumulations for the Anvil mill at various ore grades and type were to be collected and analyzed, the potential problems and possible metallurgical improvements of an expanded mill would be much more clearly defined.

In addition, some well planned laboratory, locked-cycle flotation tests on newly collected Grum samples, both discretely and properly composited, would more adequately characterize that ore in comparison to the Faro I and III ores. An expensive and extended pilot plant campaign to try to find a unique reagent combination can only lead to the type of very confusing and conflicting results such as were produced by the earlier work.

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#### V. Proposed Additional Plant Tests

Since the basic metallurgical flowsheet of the expanded plant is proposed to be the same as at present, some additional plant tests at various levels of reduced tonnage appear to be needed for confirming the metallurgical projections. The cost in lost production which would be entailed by actually operating the Anvil mill at reduced feed rate, would appear to be a small price to pay for insurance that a workable fine grind and acceptable recoveries can be obtained. In addition, by conducting a concurrent laboratory program on selected plant test samples, some potentially useful flowsheet variations could well be developed.

The plan for the expanded plant is to operate the existing grinding circuit at 200 tonnes/hour rather than at 400 tonnes/hour as at present. The pulp density in the tertiary cyclones will seemingly be lowered to attain the finer 50 micron sizing and to give the desired lower pulp density feed to the lead rougher circuit.

Thus, by merely reducing the feed rate to 200 tonnes/hour for a test period at the Anvil mill, the grinding circuit projections can be evaluated. The most critical factor in the grinding circuit to be confirmed would be the operation of the tertiary cyclones.

If the grinding circuit will produce the fine sized product as projected, then the existing flotation plant, at reduced tonnage and pulp density, should be adequate to show whether or not the projected metallurgical results are realistic. During any test period, only separate lead and zinc concentrates should be produced.

The present lead flotation circuit at a feed rate of 200 tonnes/hour has the same volume to tonnage ratio as that proposed for the expanded plant. Thus, the significant effect of lowered pulp density in flotation can be verified. The factors of increased aeration, air dispersion and assumed increased efficiency of the proposed OK 38 flotation machines would obviously not be present; however, the relative effects of these factors is undoubtedly small compared to the projected major effects of finer grind and lower pulp density. In general experience, small flotation cells with close operator

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control, as should be the case with such a test run, will give metallurgical results closely matching those of new, larger flotation machines.

The present zinc flotation circuit at the halved feed rate would exceed the volume to tonnage ratio of the proposed plant by 50%. Thus, zinc recovery under these conditions might actually be better than that of the expanded plant. If necessary, some of the zinc cells for the test could be set at an idling pulp level.

The capacity of the lead regrind mill could be adjusted by reducing the ball charge or by changing the pulp density. Since the lead regrind is not presented in the AFE as a particularly significant factor in the proposed expanded plant, in comparison to the fine primary grind, the pulp density adjustment might well be sufficient.

The zinc regrind circuit would be closely comparable to the proposed plan, if one of the two mills were to be cut out of the circuit for the test.

Assuming, then, that the grinding calculations in the AFE are correct and that the Faro I ore to be processed in the test period would be similar to that milled during December 1979, the primary grinding circuit would produce a flotation feed at P<sub>80</sub> of about 36 microns. With such a fine grind, some serious difficulties could be expected in the tertiary cyclone classification and in the lead flotation circuit.

A second test period at a feed rate of about 236 tonnes per hour should more closely approximate a primary grind of P<sub>80</sub> at 50 microns, using the assumptions stated above.

To complete the plant test program, a third test at a P<sub>80</sub> grind of 70 to 80 microns is recommended. The feed rate (approximately 300 tonnes/hour) could be calculated on the basis of the results from the first two tests or from current normal operation. At this feed rate, the lead flotation circuit would have only two-thirds of the proportional volumetric capacity of the planned expansion, however, with the slightly coarser grind, metallurgical results for lead could be comparable.

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During such a plant test program, the opportunity to collect samples for laboratory flotation should be utilized. Since the secondary ball mill cyclone overflows in the proposed plant will have a P<sub>80</sub> size of about 120 microns, the potential benefits of a lead scalping flotation step before tertiary grinding should be investigated. This flowsheet variation has been employed at the Paragsha mill of Cerro de Pasco for the past several years. Since the Anvil mill now yields a lead recovery approaching 90% on days when higher than normal ore grades are experienced, such a modified flowsheet is likely to yield substantial advantages. Its primary function would be to eliminate the deleterious effects of slime from overgrinding the lead sulfides.

The Paragsha flowsheet appears to be more adaptable to the Anvil mill layout than that used at Mt. Isa. (6) For a finely interlocked lead-zinc ore, Mt. Isa employs a sand-slime separation and a too complex a flowsheet for consideration at Anvil to reduce the recognized problems of overgrinding galena.

#### VI. Suggested Flowsheet

The foregoing analysis and discussion leads to a suggested flowsheet which could be incorporated in the already designed expanded mill by the addition of one thickener and perhaps an additional pump or two to handle the changed pulp distribution. No attempt has been made to develop more than a rough approximation of grinding circuit capabilities and optimum pulp distribution. As based on the tabulation of revised grinding capacity, one of the three existing small rod mill-ball mill units would not be required and, thus, would be available as a standby unit or for further expansion. It could also be used, if additional lead rougher concentrate regrind capacity is found to be required.

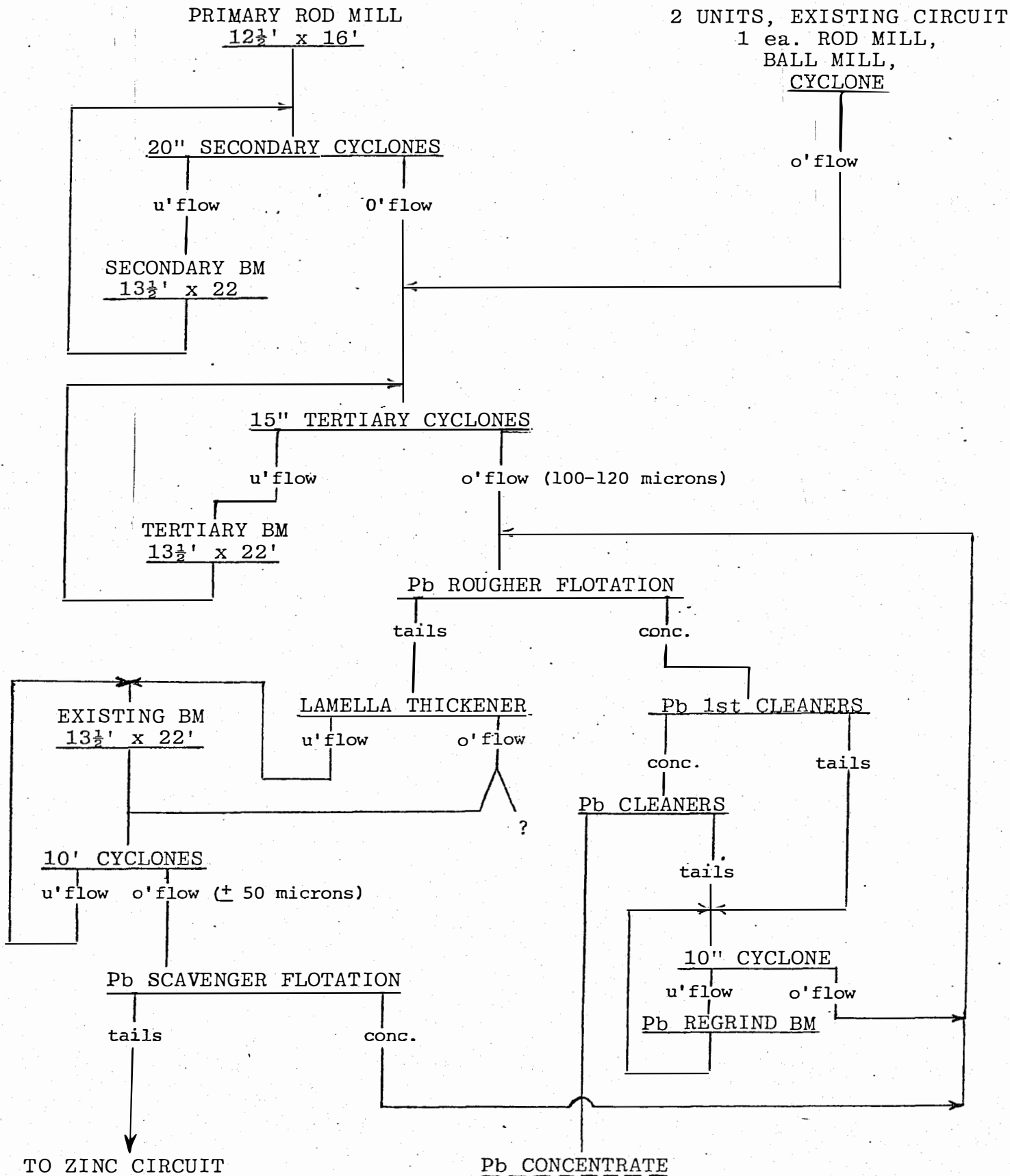
The added thickener on the rougher tails would provide operating flexibility to ensure the optimum grinding and classification in the lead rougher tails grinding circuit. For this application, a lamella thickener would probably be satisfactory with the main advantage being that of requiring very little space.

SUGGESTED ALTERNATE FLOWSHEET

EXPANDED ANVIL MILL

250 TONNES/HR.

200 TONNES/HR.



TO ZINC CIRCUIT

Pb CONCENTRATE

*MS*

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The use of a similar thickener in the zinc circuit might well be advantageous in providing operating flexibility to maintain low pulp densities for classification and flotation. No attempt has been made, however, to redesign the proposed AFE zinc circuit.

Although the concepts for the flowsheet modification were developed from this study, a subsequent review of the technical paper on the Paragsha concentrator shows many similarities between the metallurgical problems there and those encountered with the treatment of Faro, Grum and Vangorda ores. The ore at Paragsha is complex, variable, somewhat oxidized and slightly higher in grade than the Faro average. The ore is pyritic with about 25% Fe. Much of the galena has an average size of 100 microns but disseminated ores have locked pyrite and galena in the range of under 10 microns. Sphalerite occurs with the same pyrite locking and, in general, requires a finer grind than the galena for effective liberation.

The flowsheet, stressing close and careful cyclone classification and interstage lead rougher flotation, has now been followed for over ten years. The concepts may well be required at Anvil to attain the metallurgical goals expressed in the draft AFE.

#### VI. A Possible Added Metallurgical Improvement

Natural oxidation of the Anvil ore, as takes place in stockpiles, for example, has long been known to have an adverse effect on flotation recovery. This effect appears to exceed that which can be accounted for by the loss of the oxidized fraction of the value. It, therefore, is more probably related to flotation chemistry. On the other hand, in-circuit oxidation of pyrite and pyrrhotite in the presence of cyanide under alkaline conditions is known to be beneficial to flotation selectivity.

These observations suggest that either ferric sulfate or ferrous sulfate, which are produced under the natural, acidic oxidation conditions, react with enough of the galena surface to result in a surface coating thereon which, then, causes some of the galena to be depressed together with the pyrite in subsequent flotation.

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Conversely, when oxidation takes place under alkaline plus cyanide conditions in the grinding mills and aerating conditioners, then, as is well known, the ferro/ferricyanide ion does not act as a galena depressant within a well defined pH range.

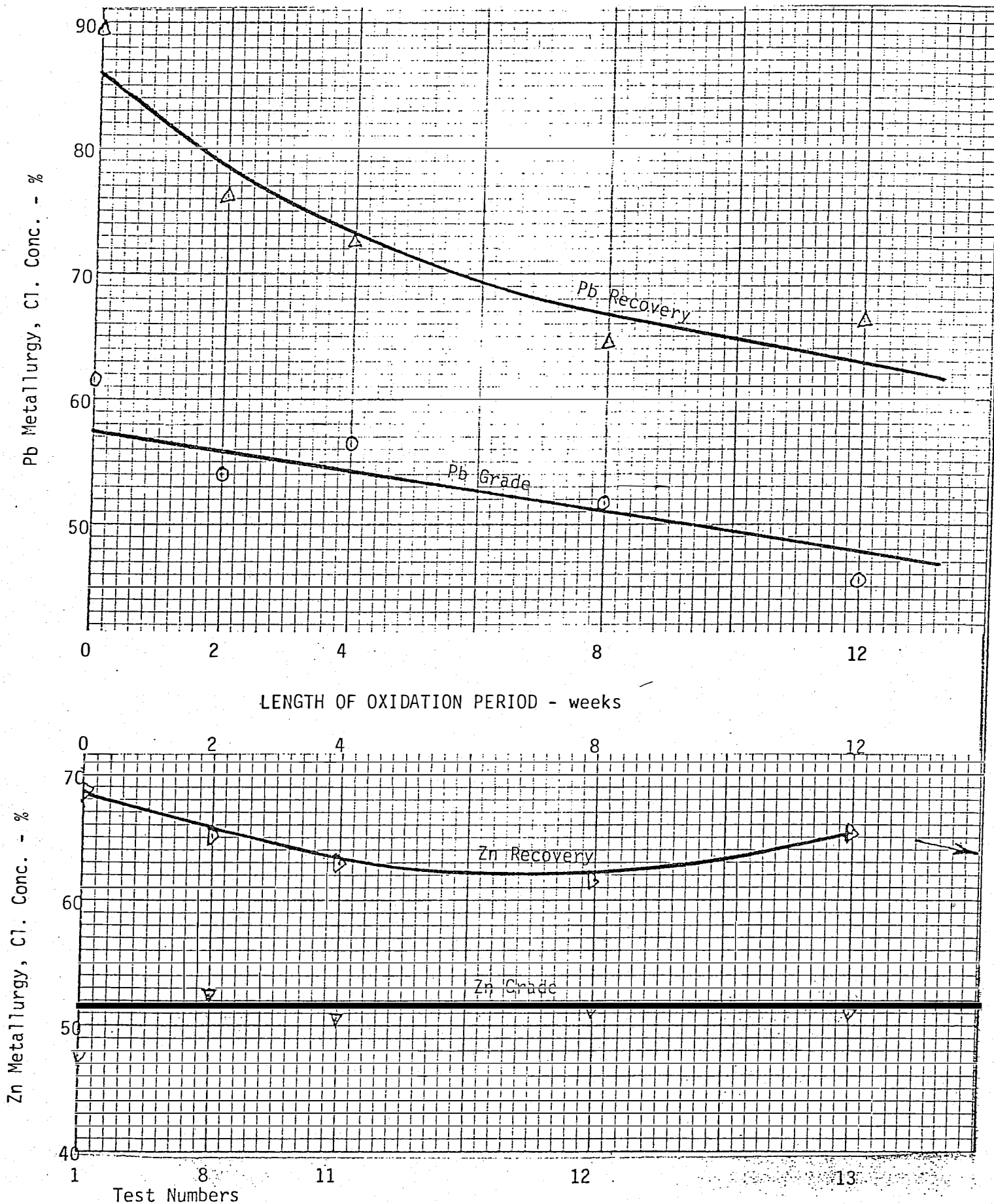
Some early (1968) test work done by Galligher on an Anvil sample showed that by allowing a moist sample to oxidize naturally for a short period, such as two weeks, both grade and recovery of lead suffered by 8 percentage points and 13 percentage points respectively. Zinc metallurgy was relatively unaffected. After 12 weeks of oxidation, when "about 22-25% of the lead minerals were converted to some non-sulfide form," lead grade was lowered by 18 percentage points and recovery by 22 percentage points.

These results lead to a suspicion that Anvil ore may undergo enough oxidation between the time of blasting in the pit to milling as to cause a measurable loss in lead recovery, even if stockpiling is not practiced.

The chemistry of natural oxidation has been studied extensively in the past few years. From this work, we know that rapid, catalyzed oxidation of pyrite (and pyrrothite) requires acidic conditions. Also, the development of these conditions will occur within five to ten days with very little moisture present. A potentially valuable metallurgical study, therefore, would be to establish the degree to which the addition of lime or other reagents to freshly broken ore from the pit would decrease the rate of natural oxidation and assist in achieving maximum lead recovery at an optimum grind. If the laboratory experiments confirm this hypothesis, then a relatively simple procedure of spraying an ore face in the pit with a lime slurry immediately after blasting might prove to be a valuable technique.

# OXIDATION EFFECTS ON FLOTATION METALLURGY

## Anvil Ore - Galligher Tests 1968



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References:

1. CAMC Interoffice Correspondence, December 11, 1978, "Plant Test: The Effect of Fineness of Grind on Lead/Zinc Metallurgy."
2. CAMC Interoffice Correspondence, January 11, 1979, "Supplementary Report."
3. CAMC Interoffice Correspondence, "Daily Standard Tests," Reports 1-9, September 15-November 17, 1978.
4. "The Beneficiation of Complex Lead-Zinc Ores at The Paragsha Concentrator of Cerro de Pasco," Chap. 23, AIME World Symposium on Mining and Metallurgy of Lead and Zinc, AIME, New York, 1970.
5. "Cerro de Pasco: Centromin's Oldest and Largest Mine," World Mining, April 1980, pp 42-48.
6. "Changes in Lead-Zinc Flotation Practice at Mount Isa Mines Limited, 1955-1970," Chap. 22, AIME World Symposium on Mining and Metallurgy of Lead and Zinc, AIME, New York, 1970.
7. "Laboratory Report on the Ageing Tests Conducted on Your Sample of Anvil Ore Designated as our Lot No. 1764," Letter Report to Cyprus Mines Corporation From The Galligher Company, July 1, 1968.

I appreciate the opportunity that this assignment has given me to review some of the voluminous data CAMC has developed on a most difficult, but interesting, metallurgical problem. The assistance of Peter Brown in spending many hours to add much pertinent information on the project is gratefully acknowledged.

Respectively submitted,



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