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THE TREATMENT OF OXIDIZED STOCKPILE ORE AT ANVIL

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SUMMARY

A three day plant test was carried out in early September, when some 20,000 S.D.T. of oxidized stockpiled ore were treated using a new sodium carbonate/sodium sulphite reagent scheme. The test results confirmed laboratory results and demonstrated that there appeared to be no physical reason why the oxidized ore could not be successfully treated in the expanded Anvil concentrator circuits.

Despite the new reagent scheme which would incur an estimated additional cost of $1.10 to $1.20/ton milled, the metallurgy for the stockpiled ore remains substandard when compared to normal ore metallurgy. Lead recovery especially is very poor and is not expected to exceed a total of 70% under normal operating conditions, with the new reagent scheme, and the expanded mill circuits.

An interesting observation made during the plant test was that zinc metallurgy even with normal ore, appeared to be enhanced when using sodium sulphite. A recommendation was made to further investigate this phenomena with a view to a further plant test to probe the value of sodium sulphite on normal ore.

INTRODUCTION

The original plan for the development of the Anvil orebody was based on the concept of processing the highest grade ore zone first. An integral part of this plan was that during the first four years of operation between two and three million tons of lower grade ore would be stockpiled in order to extract the high grade core of the orebody. Present estimates indicate that by January 1, 1974, this phase of the mining operation will be completed, the high grade core depleted and some 2.75 million tons of lower grade ore (averageing 8.0% combined lead and zinc) stockpiled.+ The plan then called for the treatment of the stockpiled ore during the years 1974 to 1975.

By late 1972, however, it was apparent that the stockpiled ore was undergoing rapid oxidation; iron oxides could be seen staining the stockpile and the internal temperature of the stockpile was obviously relatively high. Since oxidized sulphide ores are notoriously difficult to process, some samples of the stockpiled ore were subjected to preliminary metallurgical tests. The results of these tests showed that severe oxidation of the ore had indeed occurred, and that the metallurgical response of the ore would probably be substandard. (1)

A test program designed to investigate the effects of oxidation on process metallurgy was submitted and approved in principle by management in December, 1972. (2) Samples of the stockpile ore were taken initially by churn drilling and later by trenching and channel sampling the stockpiles. (3) Laboratory work commenced in January of this year and was initially directed to provide an investigation on a broad front covering the most promising avenues of attack. (4, 5) After about four months work, sufficient data was

+ Actually there are two separate stockpiles designated Red and Yellow. See Appendix I for details.

available to permit a detailed investigation into the most promising area of test work, centered at that time on the use of sodium sulphite and sodium carbonate. (6, 7, 8, 9)

The detailed investigation of the sulphite/carbonate reagent scheme took the form of open circuit cleaning tests (10); in these test attempts were made to produce target selective concentrate grades at best recoveries, thus permitting process grade-recovery curves to be constructed. (11) As an adjunct to the Anvil laboratory work, the Galigher Company, Salt Lake City, was asked to develop independently a scheme for treating the oxide ore; their results, based on a lime circuit were not reproducible, and did not equal the results from the sulphite/carbonate tests. (12, 13)

As the laboratory test program was concluded, a recommendation was made to perform a plant test to verify the laboratory findings. (14) This test was scheduled to take place in August but poor sulphite and carbonate deliveries delayed the test until September. The analysis of results of the plant test, conclusions drawn, and the effect of processing the oxide ore on the concentrator operation are examined in detail in the text of this report.

THE PLANT TEST

During the three day period from September 1 to September 3 inclusive, approximately 20,000 S.D.T. of partially oxidized "Red" stockpile ore were processed utilizing a reagent scheme developed in the laboratory. Throughout the test period, the milling rate was adjusted to closely simulate the various operational parameters (expected in the expanded mill configuration), which will be used eventually to treat the oxidized ore. (See Appendix II). The operation of the plant was closely monitored and the metallurgy was aimed at producing target selective concentrate grades+ and measuring the reduction in recovries.

Since the laboratory test work had indicated that the fine material in the stockpile was probably more oxidized than the coarse material, the plant test was designed to investigate the three possible plant conditions which would occur in normal operations:

(a) A natural mixture of fine material and crushed coarse material. This condition would be typical of the 'average' metallurgical response of the ore.

(b) Fine material only direct from the primary crusher. This condition should exhibit the most adverse metallurgical response of the ore, and would occur when the primary crusher only was in operation.

(c) Crushed coarse material only. This condition should, in theory, have provided the best oxide ore metallurgy, and represents the condition which would occur with the secondary crusher circuit only in operation.

The results of the test by phase are summarized below in Table 1. Note the allowence for a 'flush out' period to remove all traces of normal ore and minerals from the ore storage and flotation circuits.

+ Throughout the laboratory and plant tests, the target concentrate grades remained the same: Lead selective concentrate grade 65.0% Pb and Zinc selective concentrate grade at 51.5% Zn.

Table 1

Metallurgy by Shift +

Test Phase	Lead: Total Recovery	Lead: Selective Concentrate Grade	Zinc: Total Recovery	Zinc: Selective Concentrate Grade
Normal Ore-Flush System	83.0	68.7	83.1	52.0
(a) Oxidized Fines & Coarse	71.7	65.6	78.2	51.6
(b) Oxidized Fines Only	73.8	63.0	70.0	52.0
(c) Coarse Oxidized Metallurgy	80.4	64.6	83.8	53.2

+ See Appendix III for details of calculations

RESULTS OF THE PLANT TEST (Refer to Table 1)

The period required to remove all traces of normal mineralization from the circuit was longer than anticipated. In fact, about four shifts passed before a significant decline in the plant metallurgy was noted. The long response time of the circuit was later explained when the laboratory flotation tests showed that the "oxidized" ore milled during the first three shifts of the test was not typical of stockpile, and was only mildly oxidized.

The next three shifts of the test period were devoted to the treatment of the mixture of fines and crushed coarse material, hopefully forming a composite mill feed which would represent average metallurgy for the oxide ore treatment. The metallurgy deteriorated rapidly at first and then leveled off with total lead and zinc recoveries significantly reduced at the target grade levels. The physical appearance of the flotation circuits was near normal although it was apparant from the high reagent consumption (See Appendix IV), that the ore being treated was oxidized. Of special interest was the dependence of the plant metallurgy on the flotation feed pH; the addition rate of sulphite and carbonate was varied to maintain this pH in the target range of 9.8 to 10.0. (See Appendix IV)

The next phase of the test commenced at 0800 hours on September 3 when primary crusher fines were treated. The appearance of the circuit changed in a subtle manner and the sulphite and carbonate additions were increased to maintain the flotation feed pH in the target range. Rather surprisingly, the lead metallurgy did not change appreciably but the zinc metallurgy declined markedly, almost a reversal of the laboratory results.

The final phase of the test commenced at 1600 hours on September 3 when the crushed oxidized coarse material was treated. As expected the metallurgy improved but did not quite reach the levels exhibited by fresh ore.

Analysis of Results

At the conclusion of the test the metallurgical results were analysed by Anvil personnel assisted by Mr. H. L. Ames. It was agreed that the test had been performed in a satisfactory manner and that there was no doubt that the oxidized ore could be treated by using sulphite/carbonate reagent scheme.

For the purpose of deciding the basic metallurgy to be used when treating oxide ore, the plant test results for phases (a) and (b) were averaged+ and corrected (using normal process grade-recovery curves), to the target selective concentrate grades. These corrected results were then compared to the Anvil laboratory estimate and the Galigher Laboratory estimates. The results of the comparison are show in Table 2.

Table 2

Comparison of Plant Test and Laboratory Results

	Anvil Laboratory Estimate	Galigher Laboratory Estimate	Corrected Plant Test Results	Oxide Ore Metallurgy
Total Lead Recovery at 65% Lead Sel. Conc. Grade	63	55	71	70
Total Zinc Recovery at 51.5% Zinc Sel. Conc. Grade	69	77	74	74

Notes:

The Anvil estimate was based upon numerous laboratory flotation tests with 10.0 lb/ton soda ash and 7.5 lb/ton sodium sulphite. It was assumed that some of the intermediate cleaner products could be used in the manufacture of bulk concentrates.

The Galigher results shown were the results of one test and were based on a lime circuit. Neither Anvil nor Galigher personnel have since been able to reproduce these results.

+ Because normal testing operations would provide mixed ore, or primary fines most of the time, the rather unusual condition of undiluted coarse oxidized ore was omitted from the averaged results.

In the final review of the plant test results the corrected plant test data were scrutinized and, after considerable discussion, it was agreed that the lead recovery be reduced by 1% to average 70%. The rational for this reduction was based on the belief that during the plant test, conditions the cleaner circuit were such that the capacity/ton metal was anomalously high, (See Appendix II). The basic metallurgy for the treatment of oxidized stockpile ore was thus generated for use in the asesssement of production schedules for the next three years of operations. (15)

Table 3

Basic Metallurgy for Oxide Ore Treatment

	Assays		Distribution	
	Pb	Zn	Pb	Zn
Lead Conc.	65.0	6.0	57.5	3.1
Zinc Conc.	1.5	51.5	3.0	62.5
Bulk+ Conc.	18.9	29.8	12.5	11.5

CONCLUSIONS

1. The oxidized stockpile ore can be treated in the expanded mill at the 10,000 S.D.T./day rate. Although the metallurgy of the process will be substandard compared with normal ore metallurgy, it will be possible to treat the oxidized ore during 1974 and 1975 and produce the target concentrate requirements. (The fact that the plant did physically handle the slightly different froth characteristics occasioned by the new reagent balance was a source of considerable relief to Anvil personnel.)

+ Bulk concentrate production is assumed to be at the rate of approximately 250 SDT/calendar day.

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2. Using averaged reagent quantities employed during the test periods (a) and (b), and comparing them with the 1973 year to date average consumptions, permitted an estimate to be made of the additional reagent requirements for oxidized ore treatment. Using average reagent prices and additional treatment cost of between $1.10 to $1.20/ton milled was estimated for oxidized ore processing. (Appendix IV, Table IV – 2)

3. Based on observations made during the test it is believed that the use of sodium sulphite at about 2 lb/ton may improve metallurgy even with normal ore. Further, it is probable that close control of the lead circuit feed pH may be critical to process metallurgy at any time and should be further investigated.

RECOMMENDATIONS

1. It is apparent that the sulphite/carbonate reagent scheme, while offering a partial solution to the oxidized ore treatment problem, is expensive. (The high cost of the process is, however, of small improtance when compared with the very tight supply situation for carbonate. Indeed, the search for an additional 4,000 S.D.T. of sodium carbonate, the minimum needed for the oxide ore treatment program, has proved completely unsucessful.)

   Work has already started on the quest for a cheaper, more readily available alternative (16), and currently is centered on the use of hydrated ammonia as a possible partial substitute for sulphite and carbonate. (17, 18, 19) Another alternative may be to consider screening out the primary crusher fines from oxidized ore and discarding them. The relative economics of discarding the fines from the ore, or modifying the present crushing circuit, are under review now, but will require more study to effect a reliable con – clusion.

   The remaining avenue of investigation, as yet relatively unexplored, is an extensive literature research, augmented by plant visits to observe other operations engaged in treating similar ores. (20) It is recommended that these phases of work proceed immediately and concurrently utilizing input from management, consultants, and Anvil operating personnel.

2. It is recommended that an extended plant test be carried out to determine the effect of sodium sulphite on normal ore metallurgy. Further, it is recommended that the present plant pH system be augumented with more sophisticated and reliable monitoring and recording equipment, and that the maintenance of the equipment be given top priority.

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July 19, 1973

APPENDIX I

Stockpile Status From Engineering Department Records August, 1973

Table I - 1 Estimated Stockpile Contents

It is interesting to note that an estimated 30% of all material destined for stockpile has been moved or will be moved subsequent to May 31, 1973. The stockpiling was considered necessary in order to maintain an acceptable grade of zinc in the mill feeds particularly during the last half of this year.

In view of recent discussions regarding the reliability of the diamond drill assay data it should be understood that the assays shown below are based exclusively on blast hole sample analysis, and should therefore be reasonable accurate.

Period	Yellow Stockpile			Red Stockpile		
	S.D.T.	Pb	Zn	S.D.T.	Pb	Zn
To May 31, 1973	1,494,266	2.7	4.3	432,942	3.4	5.8
From May 31 to Dec. 31, 1973 (est)	460,864	2.2	4.3	368,749	4.4	5.5
Total Estimated Tons Stockpiled	1,955,130	2.6	4.4	801,691	3.9	5.7

Av. 3.2 - 5.1

APPENDIX II

Plant Test September 1 to 3, Inclusive

Table II – 1 Comparison of Operating Parameters

	Average Plant Test Conditions	Expanded Mill Conditions
Grind		
% +100#	6.0	5.5
% –200#	72.6	72.0
Rougher Capacity Feet^3 / Ton Solids / Hour		
Lead	143	137
Zinc	149	215
Cleaner Capacity Feet^3 / Ton Metal / Hour		
Lead	409	471
Zinc	268	336

Notes:

1. The grind expected in the new mill was based upon an average work index figure for the last two years.

2. The normal convention of using residence times for comparison in flotation feed capacities was not applicable here since a decrease in flotation feed pulp density is envisaged as the result of the expansion.

3. The cleaner capacity comparison was effected by using design data for A.F.E. 72 – 15.

APPENDIX III

Details of Plant Metallurgy During the Plant Test

September 1 to 3 Inclusive

Table III - 1 Metallurgy by Shift Summary

Date	Test Phase	Shift	Lead Rec.	Lead Grade	Zinc Rec.	Zinc Grade
Sept. 1		12 to 8	88.2	72.7	88.60	51.7
		8 to 4	82.1	67.2	77.40	51.8
		4 to 12	81.3	67.4	87.70	51.2
Sept. 2		12 to 8	80.3	67.6	78.8	53.6
	(a)	8 to 4	68.2	63.6	73.8	52.9
	(b)	4 to 12	68.0	69.4	78.8	52.2
Sept. 3	(a)	12 to 8	78.8	63.8	82.0	49.6
	(b)	8 to 4	73.8	63.0	70.0	52.0
	(c)	4 to 12	80.4	64.6	83.8	53.2

Table III - 2 Metallurgy By Phase

Test Phase	Lead Rec.	Lead Grade	Zinc Rec.	Zinc Grade	%
Normal Ore Flush System	83.0	68.7	83.1	52.0	
(a) Oxidized Fines & Coarse	71.7	65.6	78.2	51.6	
(b) Oxidized Fines Only	73.8	63.0	70.0	52.0	35
(c) Coarse Oxidized Material Only	80.4	64.6	83.8	53.2	65

Detailed Shift Metallurgical Balances – September 1 to 3 Inclusive

Table III – 3 METALLURGICAL BALANCE September 2 12 to 8

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2136	2.64	5.17	100.00	100.00
Pb Conc.	58.6	67.6	5.95	69.7	3.1
Zn Conc.	144.2	0.87	53.6	0.7	69.1
Bulk	33.0	18.2	32.9	10.6	9.7
Tailings	1900.2	0.57	1.06	19.0	18.0

Table III – 4 METALLURGICAL BALANCE September 2 8 to 4

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2241	2.20	5.46	100.00	100.00
Pb Conc.	45.5	63.6	6.5	58.6	2.4
Zn Conc.	143.1	1.36	52.9	3.9	61.8
Bulk	33.0	14.4	44.5	9.6	12.0
Tailings	2019.4	0.68	1.44	27.8	23.8

Table III – 5 METALLURGICAL BALANCE September 2 4 to 12

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2208	2.35	5.13	100.00	100.00
Pb Conc.	33.8	69.4	4.85	44.9	1.4
Zn Conc.	158.4	1.30	52.2	3.9	73.1
Bulk	33.0	36.6	19.5	23.1	5.7
Tailings	1982.8	0.74	1.13	28.1	19.8

Table III – 6 METALLURGICAL BALANCE September 1 12 to 8

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2220.6	3.45	6.16	100.00	100.00
Pb Conc.	74.1	72.7	4.7	70.3	2.6
Zn Conc.	203.2	1.5	51.7	4.0	76.9
Bulk	33.0	18.0	48.3	7.8	11.7
Tailings	1910.3	.72	.63	17.9	8.8

Table III – 7 METALLURGICAL BALANCE September 1 8 to 4

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2205.8	3.20	5.72	100.00	100.00
Pb Conc.	70.1	67.2	5.8	66.8	3.2
Zn Conc.	163.9	1.8	51.8	4.2	67.3
Bulk	33.0	32.6	38.4	15.3	10.1
Tailings	1938.8	.50	1.26	13.7	19.4

Table III – 8 METALLURGICAL BALANCE September 1 4 to 12

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2124.7	2.97	5.85	100.00	100.00
Pb Conc.	64.4	67.4	5.4	68.7	2.8
Zn Conc.	193.8	1.1	51.2	3.3	79.8
Bulk	33.0	24.1	29.9	12.6	7.9
Tailings	1833.5	.53	.64	15.4	9.5

Table III - 9 METALLURGICAL BALANCE September 3 12 to 8

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2321.0	2.53	5.16	100.00	100.00
Pb Conc.	67.0	63.8	6.1	72.6	3.4
Zn Conc.	172.0	1.06	49.6	3.1	71.3
Bulk	33.0	11.0	38.8	6.2	10.7
Tailings	2049.0	.52	.85	18.1	14.6

Table III - 10 METALLURGICAL BALANCE September 3 8 to 4

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2309.4	2.66	5.14	100.00	100.00
Pb Conc.	66.9	63.0	5.8	68.5	3.3
Zn Conc.	134.2	1.18	52.0	2.6	58.7
Bulk	33.0	9.8	40.6	5.3	11.3
Tailings	2075.3	.70	1.53	23.6	26.7

Table III - 11 METALLURGICAL BALANCE September 3 4 to 12

		Assay %		Distribution	
	S.D.T.	Pb %	Zn %	Pb	Zn
Feed	2354.5	2.51	5.54	100.00	100.00
Pb Conc.	70.4	64.6	6.2	77.0	3.3
Zn Conc.	176.7	.84	53.2	2.5	72.1
Bulk	33.0	6.1	46.4	3.4	11.7
Tailings	2065.4	.49	.81	17.1	12.9

APPENDIX II

Reagent Consumption During Oxidized Ore Test

Table IV

Date	Shift	Z - 11	NaCN	$CaSO_4$	Na_2CO_3	Na_2SO_3	CaO
		Reagents lb/Ton Milled					
Sept. 1	8 - 4	0.53	0.22	1.52	4.37	2.63	3.20
	4 - 12	0.55	0.23	1.78	6.82	4.55	3.20
Sept. 2	12 - 8	0.53	0.22	1.91	6.62	4.42	3.50
	8 - 4	0.49	0.22	2.08	6.41	6.27	3.50
	4 - 12	0.58	0.22	2.04	6.51	4.34	3.50
Sept. 3	12 - 8	0.57	0.21	1.57	8.25	4.12	3.50
	8 - 4	0.62	0.26	1.50	10.4	6.21	3.50
	4 - 12	0.65	0.27	1.90	10.2	6.10	3.50
Test Average +		0.57	0.23	1.80	7.89	4.74	3.50 ++
1973 Y.T.D. Average		0.27	0.16	1.18	2.55	0.00	3.20
Net Increase in Consumption		0.30	0.07	0.62	5.34	4.75	0.30

\+ The Test Average was calculated on the data generated during Phase II and Phase III of the test.

++ Estimate Only.

APPENDIX IV - 2

Table IV - 2 Estimated Cost of Oxide Ore Treatment Reagents

	Reagents					
	Z - 11	NaCN	$CuSO_4$	Na_2CO_3	Na_2SO_3	CaO
Increase in Consumption lb/ton	0.30	0.07	0.62	5.34	4.75	0.30
Reagent Cost per pound delivered to minesite $/lb	0.262	0.223	0.260	0.048	0.127	0.031
Increase in Treatment Cost $/lb	0.079	0.016	0.161	0.256	0.603	0.009

Minimum Increase in Reagent Cost to Treat Stockpiled Low Grade Oxidized Ore — $ 1.124 / ton