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O T T A W A   July 2nd, 1943.

R E P O R T

of the

ORE DRESSING AND METALLURGICAL LABORATORIES.

Investigation No. 1438.

Pilot Plant Tests on a Lead-Zinc Ore from the Property of New Calumet Mines Limited, Calumet Island, Campbell's Bay, Quebec.

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(Copy No.____.)

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SECTION A. - GENERAL.

Shipments:

During April and May, 1943, two shipments of lead-zinc ore were received from the New Calumet Mines Limited, Campbell's Bay, Quebec. Lot No. 1 weighed approximately 18 tons and was considered representative of the ore body. Lot No. 2 consisted of approximately 5 tons and was a sample of the low-grade section of the mine.

Location of Property:

The property is located on Calumet Island, in the Ottawa river adjacent to Campbell's Bay, Pontiac County, Quebec. It is served by the Waltham branch of the Canadian Pacific railway.

General Characteristics of the Ore:

The ore is a lead-zinc ore carrying appreciable values in silver and minor values in gold.

The lead mineral is galena and the zinc minerals are sphalerite and marmatite(?). The silver minerals are associated with the galena.

Pyrite and pyrrhotite are present, the latter being the more abundant.

Considerable mica is present, and calcite was noted, particularly in the low-grade ore.

Details of the microscopic examination of drill-core samples are given later in the report.

Results of Tests:

No difficulty was experienced in the concentration of the economic minerals of this ore. Results were uniformly good on both lots of ore. The following table gives the results to be expected from the concentration of this ore:

Metal	LOT NO. 1		LOT NO. 2	
	Grade	Recovery, per cent	Grade	Recovery, per cent
Lead	65 to 66 per cent	97-98	63 to 64 per cent	95-96
Zinc	54 to 55 "	94-95	48 to 50 "	93-94
Gold (in lead conc.)	0.10-0.15 oz./ton	60-70	0.10-0.15 oz./ton	45-50
Silver (in lead conc.)	105-110 "	87-92	120-130 "	77-80

Tbe lower recoveries in Lot No. 2 can be attributed almost entirely to the lower grade of the feed, as the tailings assays gave almost the same results as were obtained in the higher-grade ore in Lot No. 1.

The grade of zinc concentrate obtained from Lot No. 2 would probably be higher in operation than was shown by the tests.

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Sampling and Analysis:

The ore in Lot No. 1 was crushed to minus 1 inch and a head sample cut by the automatic sampler. This sample was further reduced by crushing and riffling to an appropriate size for analysis.

The ore in Lot No. 2 was crushed to minus $\frac{1}{4}$ inch and an operational sample was taken of the ball mill feed at 15-minute intervals during the run of this ore. This sample was also further reduced by standard methods to obtain the final assay sample.

The analyses of the head samples are as follows:

Determination	Lot No. 1	Lot No. 2
Lead (Pb)	6.21 per cent	2.30 per cent
Zinc (Zn)	14.66 "	7.45 "
Iron (Fe)	5.71 "	6.93 "
Copper (Cu)	0.14 "	0.10 "
Cadmium (Cd)	0.016 "	Not determined.
Arsenic (As)	0.008 "	Not determined.
Antimony (Sb)	0.03 "	Not determined.
Lime (CaO)	6.84 "	9.30 per cent
Silica (SiO_2)	27.18 "	Not determined.
Alumina (Al_2O_3)	4.78 "	Not determined.
Magnesia (MgO)	10.49 "	Not determined.
Acid insoluble	46.11 "	45.00 per cent
Gold (Au)	0.02 oz./ton	0.01 oz./ton
Silver (Ag)	10.85 "	6.52 "

The mill was operated for six hours daily, the samples of the products being taken during the last $3\frac{1}{2}$ hours of the run. This allowed the circuit to become adjusted to changes of reagents, etc.

A bulk sample of both the lead and zinc concentrates

(Sampling and Analysis, cont'd) -

was obtained from Run No. 5. These samples were taken during one 15-minute interval only but are considered indicative of what is to be expected from the mill operation. A complete analysis of these two products is given below:

Determination	Lead concentrate	Zinc concentrate
Lead (Pb)	63.94 per cent	0.31 per cent
Zinc (Zn)	3.83 "	55.69 "
Iron (Fe)	3.39 "	8.48 "
Copper (Cu)	0.45 "	Not determined.
Sulphur (S)	14.10 "	32.13 per cent
Cadmium (Cd)	Not determined.	0.12 "
Arsenic (As)	0.02 per cent	None detected.
Antimony (Sb)	0.18 "	Trace.
Lime (CaO)	2.17 "	0.28 per cent
Silica (SiO_2)	6.82 "	0.70 "
Alumina (Al_2O_3)	Not determined.	0.24 "
Magnesia (MgO)	Not determined.	0.44 "
Acid insoluble	11.34 per cent	1.09 "
Gold (Au)	0.16 oz./ton	0.01 oz./ton
Silver (Ag)	103.86 "	2.02 "
Iron (HCl soluble)	Not determined.	7.88 per cent

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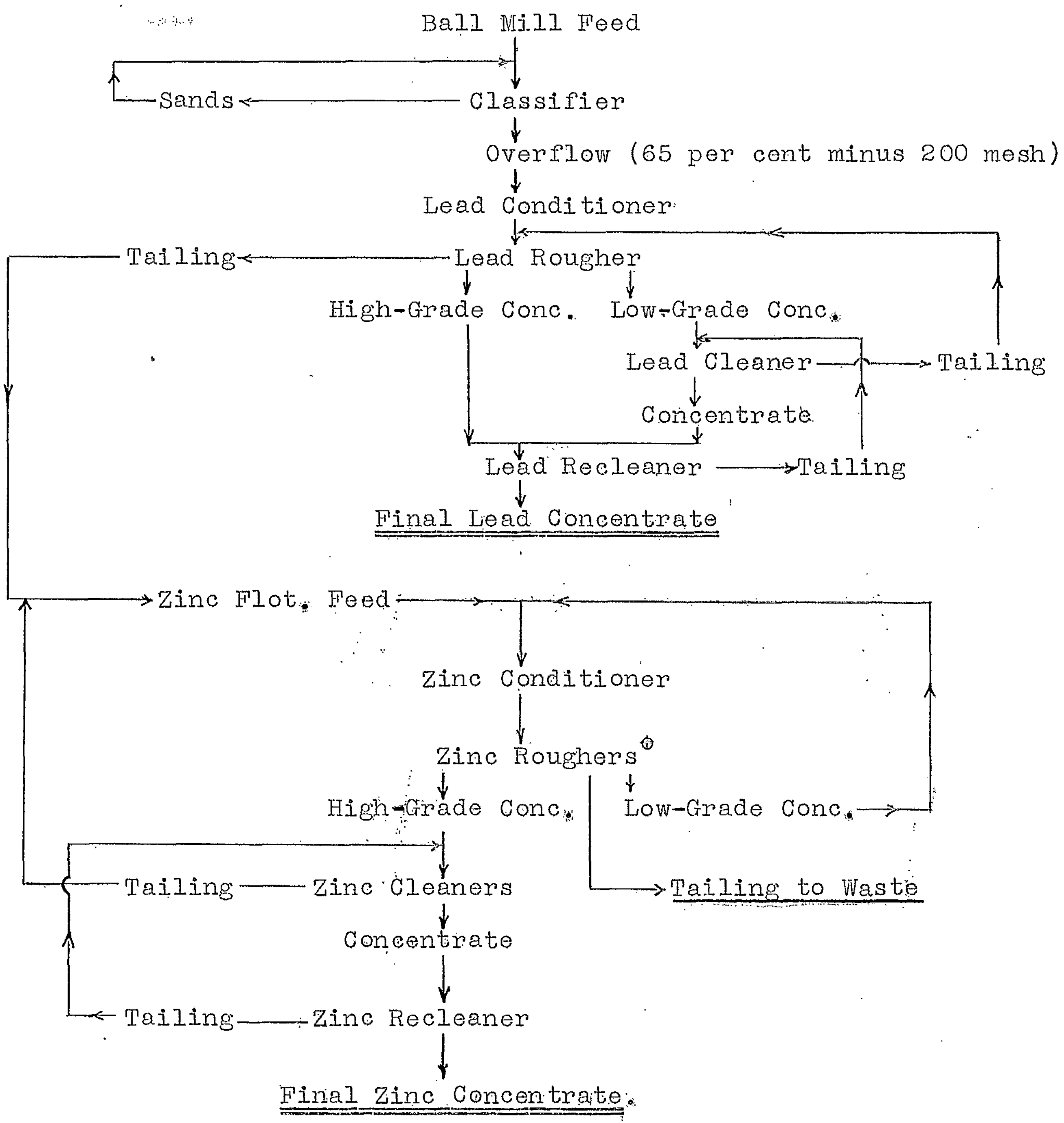
RECOMMENDED FLOW-SHEET:



⊕ Note: A double concentrate launder for the zinc roughers is desirable so that the operator can vary the amount of concentrate returned to the zinc rougher conditioner according to the grade of zinc in the zinc rougher feed. The flow-sheet actually used for these tests is shown diagramatically on Page 15.

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Discussion of Flow-Sheet:

The flow-sheet recommended on Page 5 was used in the lead circuit for all tests succeeding Test No. 7. Due to lack of sufficient equipment we were unable to follow that section of the flow-sheet which is recommended for the zinc circuit, and all the zinc rougher concentrates were sent to the zinc cleaners in every test.

In the lead circuit, while one cleaning step gave a good grade of lead, the tests indicate that another cleaning step would help in increasing the zinc recovery in the zinc concentrates, by decreasing the recovery of zinc in the lead concentrate. While a satisfactory grade of lead could be obtained in one cleaning step, it might lead to short-circuiting of the coarser lead minerals with the subsequent loss of lead and silver in the lead rougher tailing. The first cell of the lead rougher circuit (total of 8 cells) was of sufficiently high grade to be sent direct to the second cleaner circuit, with consequent lowering of the burden on the lead cleaner circuit.

The use of a double launder in the zinc rougher circuit, so that a variable portion of the zinc rougher concentrates could be returned to the head of the zinc circuit, is indicated. This would serve to obtain a fairly even grade of feed to the zinc rougher circuit and would keep some of the gangue minerals from reporting in the zinc cleaning circuit. When a low-grade zinc rougher concentrate is obtained, as in the case of Lot No. 2, the gangue minerals, particularly mica, appear in the final zinc concentrate. It is advantageous to maintain the grade of the zinc rougher concentrates above 48 per cent zinc and the use of this type of circuit will give

(Discussion of the Flow-Sheet, cont'd) -

this, along with the maximum recovery of the zinc minerals.

While the recoveries and grades of both lead and zinc were good at grinds of about 54 per cent minus 200 mesh, there was some indication that the coarser minerals were having a tendency to build up in the circuit. There was a steady increase in the grade of zinc in the final tailing at the coarser grinds, which increase appears to be attributable to the coarseness of the grind rather than to changes in reagents.

The galena appears to be free and to float quite easily at the coarser grinds, but the recovery of zinc in the lead concentrate appears to be higher at the coarse grind than at the finer grind. More mica also appears in the lead concentrate. This may be due to the larger quantity of frother required to keep the coarser galena particles from dropping out.

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SECTION B. - DETAILS OF REPRESENTATIVE TESTS (Test No. 14, High Grade; Test No. 11, Low Grade).

Test No. 14. - High-Grade Ore.

General:

Feed rate: 500 pounds per hour.

	Per cent solids	pH value
Classifier overflow -	33	10.0
Lead rougher tailing -	25	9.7
Zinc rougher feed -	20	11.3
Zinc cleaner tailing -	-	11.3

Lead Circuit -

The conditioning time was 17 minutes and the lead rougher flotation time was 18 minutes. The lead rougher circuit comprised eight No. 7 Denver flotation cells. The first cell of the lead roughers, along with the concentrate from the single cleaner cell, went to the lead recleaner (one cell) from which a final concentrate was removed. The concentrate from the remaining seven cells of the lead rougher circuit, was cleaned in a single cell. The lead recleaner tailing joined the lead cleaner feed and the lead cleaner tailing joined the lead rougher feed. The lead rougher tailing went to the zinc circuit.

Zinc Circuit -

The zinc rougher feed was conditioned for 9 minutes then floated for 18 minutes in ten No. 7 Denver cells. The zinc rougher concentrates went to four cleaner cells and the zinc cleaner concentrates went to two recleaning cells. The zinc recleaner tailing joined the zinc cleaner feed and the zinc cleaner tailing joined the lead rougher tailing and was

b

(Test No. 14, cont'd) -

pumped to the zinc rougher conditioner. The zinc rougher tailing was discarded as the final tailing.

Reagent Consumption:

Place of addition	Reagent	Pounds per ton of feed
Ball mill:	Soda ash -	2.0
	Sodium silicate -	2.8
	Zinc sulphate -	1.0
	Sodium cyanide -	0.10
	Cresylic acid -	0.07
	Potassium ethyl xanthate -	0.05
Lead conditioner:	Potassium ethyl xanthate -	0.05
	Cresylic acid -	0.07
Zinc conditioner:	Lime -	3.0
	Copper sulphate -	1.0
	Potassium ethyl xanthate -	0.20
	Cresylic acid -	0.14
Zinc cleaner:	Lime -	0.50

(Continued on next page)

Insert from bottom

Results:

	Weight, per cent	ASSAYS							RECOVERIES, per cent						
		Per cent					Oz./ton								
		Pb	Zn	Fe	Cu	Insol.	Au	Ag	Pb	Zn	Fe	Cu	Insol.	Au	Ag
Feed		6.10	14.93	5.81			0.015	10.04							
Ball mill discharge		7.30	14.12	6.86											
Classifier sands		7.30	13.70	6.86											
" overflow	100.00	6.10	14.12	5.35	0.13	45.89	0.014	11.31	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Lead recl. conc.	9.06	66.00	3.29	2.50	0.88	13.12	0.10	110.58	98.1	2.1	4.2	60.6	2.6	64.3	88.6
" " tailing		19.30	5.02	4.88											
Lead cleaner tailing		12.05	7.31	5.40											
Lead rougher conc.		44.00	6.90	4.18											
" " tailing	90.94	0.13	15.19	5.63		49.15	0.005	1.42							
Zinc recl. conc.	24.47	0.13	55.60	8.03	0.13	1.28	0.01	2.83	0.5	96.4	36.8	24.2	0.7	14.3	6.1
" " tailing		0.13	21.60	13.60											
Zinc cleaner conc.		0.13	54.60	8.14											
" " tailing		0.13	5.85	15.90											
Zinc rougher conc.		0.13	50.70	8.65											
" " tailing	66.47	0.13	0.32	4.75	0.03	66.78	0.005	0.90	1.4	1.5	59.0	15.2	96.7	21.4	5.3

(Test No. 14, cont'd) -

(Test No. 14, cont'd) -

Screen Tests:

Mesh	Weight, per cent: Classifier overflow	Lead concentrate	Zinc concentrate	Final tailing
+ 28	0.2			0.3
- 28+ 35	0.3			0.5
- 35+ 48	1.2			1.8
- 48+ 65	2.8		0.1	3.9
- 65+100	5.4		0.7	7.8
-100+150	11.4	0.5	2.6	16.1
-150+200	13.0	0.9	4.7	17.8
-200	65.7	98.6	91.9	51.8

This run can be considered as typical of the results to be expected from this ore. The flotation time in the zinc rougher circuit appears to be about right and the grade of zinc going to the cleaners was good. In this case all of the zinc rougher concentrates should go to the cleaning circuit.

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Test No. 11. - Low-Grade Ore.

Reagent Consumption:

The reagent consumption on this test was identical with that used in Test No. 14, except that potassium cyanide to the amount of 0.10 pound per ton of feed was fed to the recleaner circuit as a further addition.

The same pH values were maintained as in the previous tests.

(Continued on next page)

Results:

	Weight, per cent	ASSAYS Per cent Pb	Zn	Fe	Insol.	Oz./ton Au	Ag	RECOVERIES, Per cent Pb	Zn	Fe	Au	Ag
Feed		2.60	7.69	6.70		0.01	6.90					
Ball mill discharge		2.65	7.48	7.10								
Classifier sands		2.75	8.02	7.70								
" overflow	100.00	2.80	7.37	6.40		0.01	6.62	100.0	100.0	100.0	100.0	100.0
Lead recl. conc.	4.18	64.00	2.49	5.10	6.00	0.10	134.80	95.6	1.4	3.3	43.3	78.7
" " tailing		12.30	6.39	9.40								
Lead cleaner conc.		52.00	3.09	7.50								
" " tailing		10.60	6.23	10.45								
Lead rougher conc.		28.40	4.77	12.40								
" " tailing		0.13	7.58	6.46		0.005	1.86					
Zinc recl. conc.	14.13	0.13	48.60	11.95	3.64	0.01	4.45	0.6	93.2	26.4	14.4	8.8
" " tailing		0.80	4.12	27.45								
Zinc cleaner conc.		0.13	46.40	13.10								
" " tailings.		1.28	42.92	23.80								
Zinc rougher conc.		0.15	41.20	13.70								
" " tailing	81.69	0.13	0.49	4.73		0.005	1.095	3.8	5.4	70.3	42.3	12.5

(Test No. 11, cont'd) -

Screen Test - Classifier Overflow.

Mesh size		Weight, per cent
+ 20	-	0.1
- 20+ 28	-	0.4
- 28+ 35	-	1.9
- 35+ 48	-	6.0
- 48+ 65	-	6.3
- 65+100	-	10.7
-100+150	-	13.4
-150+200	-	11.2
-200	-	50.0

The results of Test No. 11 were quite good considering the lower grade of the feed as compared with that in Test No. 14. In comparing this test with Test No. 14 the lower percentage recovery obtained in this test is due to the lower grade of the heads, as the assays of the final tailings compare favourably with those in Test No. 14. The effect of the lower grade is most noticeable in the case of the gold and silver, the value of the gold and silver in the final tailing being almost exactly the same in the two tests, but the recovery of gold is 20 per cent less and that of the silver is 10 per cent less in Test No. 11.

In this test the lead froth was quite heavy and slow-floating. This appears to be due mainly to the coarseness of the grind, as the previous run (No. 10) on this ore was quite satisfactory in this respect and the only change (apart from a slight change in the value of the feed) was in lowering the grind from 65 per cent minus 200 mesh to 50 per

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(Test No. 11, cont'd) -

cent minus 200 mesh.

The grade of the zinc products showed a considerable drop over that obtained in Test No. 14. Both the iron and insoluble showed a distinct increase in the final concentrates. It is considered that as good a grade could have been obtained on this ore as on the higher-grade ore if a portion of the zinc rougher concentrates could have been returned to the head of the zinc rougher circuit. This would have resulted in a higher grade of feed to the cleaner circuit and would, in turn, have given a better grade of zinc in the zinc concentrates. Although the grind was comparatively coarse in this test the fact that a similar grade of concentrate was obtained in Test No. 10 (65 per cent minus 200 mesh) would seem to indicate that the cleaner circuit was being somewhat crowded with gangue minerals.

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Flow-Sheet Used for These Tests.

On the following page is given a diagram of the flow-sheet actually used in this investigation.

(See diagram on Page 15)

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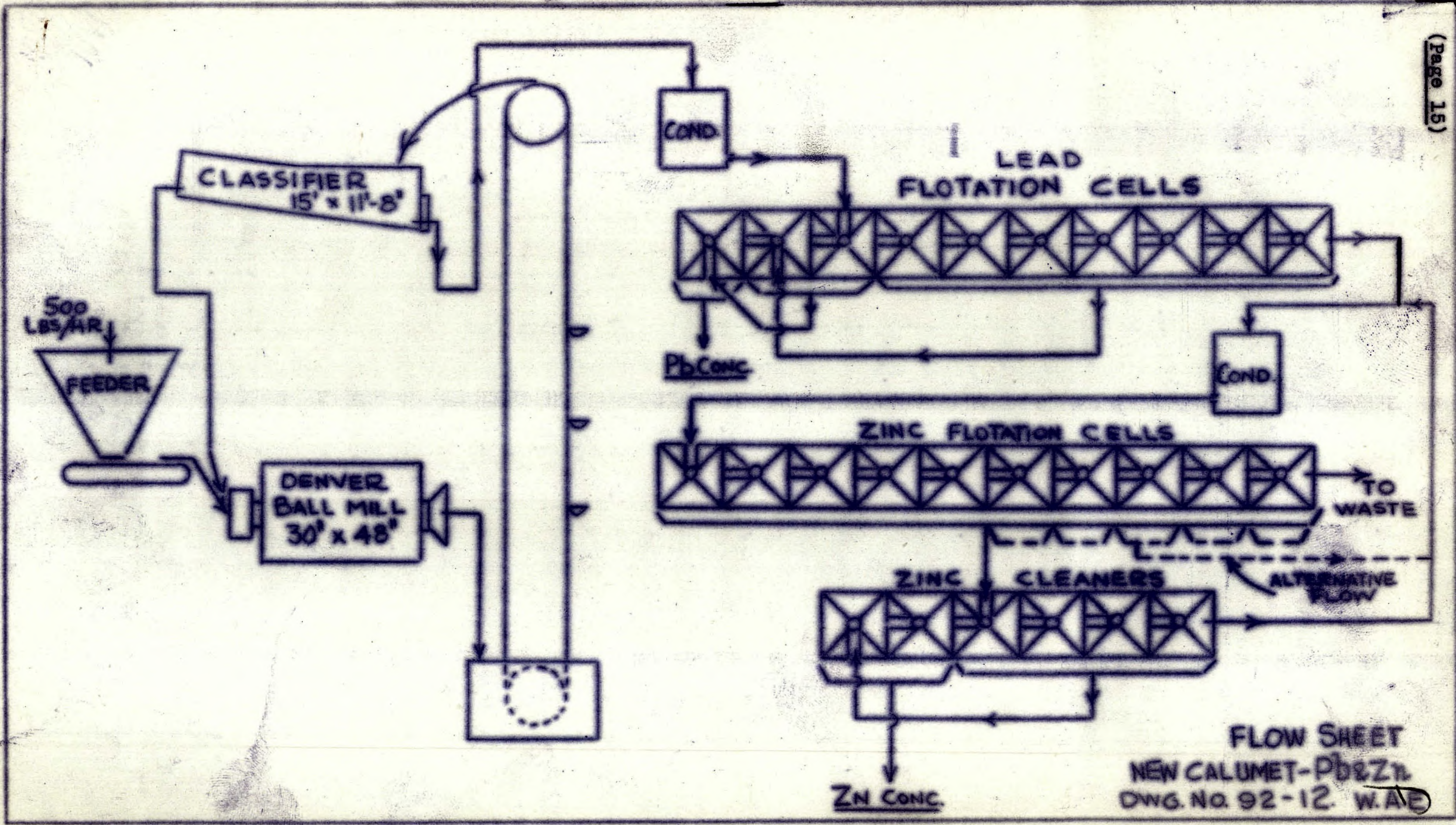
FLOW-SHEET AS ACTUALLY USED IN TESTS NOS. 14 AND 11.



SECTION C. - MICROSCOPIC EXAMINATION.

The ore is a lead-zinc ore carrying appreciable values in silver and minor values in gold. It also contained pyrite, pyrrhotite and chalcopyrite. Noticeable among the gangue minerals was muscovite mica.

In general, the metallic minerals are distributed through the ore as masses, grains, and stringers. Massive pyrrhotite, sphalerite and galena are common.

Lead Minerals -

The lead mineral is galena and occurs in quite coarse crystals. With the galena are associated the silver minerals and tetrahedrite.

Zinc Minerals -

The main zinc mineral is sphalerite but evidence obtained from flotation also indicates the presence of marmatite. Sphalerite commonly contains grains of galena.

The presence of marmatite was suspected when the iron content of the zinc concentrate appeared to reach a minimum of 8 per cent iron. Chemical analysis of the concentrates obtained in Test No. 5 gave 8.48 per cent total iron and 7.80 per cent hydrochloric-acid-soluble iron. Although pyrrhotite is present in the ore, the zinc concentrate when tested under the magnet appeared to be completely free of this mineral. The only way that this high amount of acid-soluble iron could be accounted for is by the presence of marmatite, a zinc-iron sulphide. The presence of this iron limits the maximum grade that can be expected from this ore to 57 to 58 per cent zinc.

Silver Minerals -

Microscopic examination indicates that the silver

(Microscopic Examination, cont'd) -

content of the ore is distributed (1) as argentite inclusions in galena, (2) in solid solution in galena, and (3) in tetrahedrite.

Gold -

No evidence has been found as yet to show any particular relation of the gold with any of the other minerals. Flotation tests indicate that if there is any association it is with the silver and lead minerals. No evidence of free gold was noted. The classifier bed and the ball mill sands were assayed at the conclusion of the tests, but no unusual concentration of gold was observed there.

Minor Sulphides -

There are three main iron sulphides present: pyrrhotite, pyrite, and marcasite. Pyrrhotite is probably the most abundant, and occurs often in masses. Pyrite is usually present as disseminated grains, often somewhat corroded where they are included in massive sulphides. Marcasite appears to be an alteration product of pyrite and pyrrhotite.

The copper occurs as tetrahedrite, which is associated with galena, and chalcopyrite, which is associated with the sphalerite and pyrrhotite.

Gangue Minerals -

The main gangue minerals that affect the flotation are mica and calcite. Considerable muscovite mica is present in the ore. Calcite was quite prominent, particularly in Lot No. 2 from the low-grade section. The gangue appears to be an altered amphibolite. Feldspar, tremolite, altered pyroxene, and the alteration products chlorite and serecite are present.

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For the geology, reference should be made to "Zinc-Lead Deposits of Calumet Island," by W. W. Moorehouse, Bulletin of the Geological Society of America, 1941.

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SECTION D. - DESCRIPTION OF INDIVIDUAL TESTS.

The following section of the report covers the individual tests made. These tests, with the exception of Test No. 5, are not given in complete detail. Certain analyses are picked from various tests for comparison but beyond that only the results of the main products are given.

The weights and recoveries are all calculated from the bi-metallic formula, using the classifier overflow as being the true head sample. Due to the large circulating load carried in the grinding-classification circuit there is often a discrepancy between the head sample analysis and the classifier overflow sample analysis.

The feed rate in all of the tests was 500 pounds per hour.

In Tests Nos. 1 and 2 the Denver unit cell was in the ball mill-classifier circuit which accounts for the lower grade of the lead in the classifier overflow.

In Tests Nos. 1 to 7, inclusive, only one stage of cleaning was used in the lead circuit. In the remaining tests the flow-sheet as previously described for Test No. 14 was used.

With the exception of Test No. 8, two stages of cleaning were used on the zinc circuit in all the runs. In Test No. 8 only one stage of cleaning was used.

Tests Nos. 9, 10 and 11 were made on the low-grade sample, the remainder being on Lot No. 1 ore.

(Continued on next page)

Reagent Consumption - Pounds per ton of ore.

	TESTS NOS.													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14
To Ball Mill														
Soda ash	2.5	2.5	1.0	1.0	2.4	2.5	2.5	2.5	2.5	2.5	2.5	5.5	2.0	2.0
Sodium silicate	2.0	2.0	4.0	4.0	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8
Zinc sulphate	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Amyl xanthate	0.05	0.05												
Barrett No. 4	0.20													
Cyanide	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
Cresylic acid		0.14	0.14	0.10	0.10	0.10		0.12	0.12	0.12	0.12	0.14	0.07	0.07
Pot. ethyl xanthate			0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
Aerofloat 25							0.10							
To Lead Conditioner														
Pot. ethyl xanthate			0.05	0.05	0.05	0.05		0.05	0.05	0.05	0.05	-	0.05	0.05
Cresylic acid	0.19	0.10	0.02	0.07	0.14	0.14		0.10	0.10	0.12	0.12	0.12	0.07	0.07
Aerofloat 25							0.05							
Amyl xanthate	0.05	0.10												
To Zinc Conditioner														
Lime	2.00	2.00	2.00				3.2	4.1	4.1	4.1	4.1	4.1	3.0	3.0
Copper sulphate	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	0.50	1.00	1.00
Ethyl xanthate	0.20	0.15	0.15	0.15	0.15	0.15	0.15	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Cresylic acid	0.24	0.11	0.07	0.10	0.02		0.04	0.10	0.10	0.14	0.14	0.14	0.10	0.10
Pine oil						0.06								
To Zinc Cleaner														
Lime	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
To Zinc Recleaner														
Lime				2.5	2.5	2.5								
Cresylic acid					0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02		
Lead Recl. (Cyanide)									0.10	0.10	0.10	0.10		

Results, Test Nos. 1 to 14:

Test No.	Product	Weight, per cent	ASSAYS Per cent Pb	Zn	Fe	Insol.	Oz./ton Au	Ag	RECOVERIES, per cent Pb	Zn	Fe	Au	Ag	MESH, % -200
1.	Feed	100.00	4.15	16.94	5.70		0.021	8.45	100.0	100.0	100.0	100.0	100.0	69.8
	Lead concentrate	9.44	42.80	6.69	7.55	19.76	0.13	80.84	97.4	3.7	12.5	57.5	90.3	
	Zinc "	29.56	0.10	54.45	8.20	1.00	0.01	1.37	0.7	95.0	42.5	14.0	4.8	
	Final tailing	61.00	0.13	0.35	4.20		0.01	0.68	1.9	1.3	45.0	28.5	4.9	
2.	Feed	100.00	4.50	15.92	5.98		0.018	8.15	100.0	100.0	100.0	100.0	100.0	61.2
	Lead concentrate	8.75	50.00	5.73	4.13	18.05	0.14	84.12	97.3	3.2	6.0	67.2	90.4	
	Zinc "	27.58	0.15	54.85	8.41	1.87	0.01	1.37	0.9	95.0	38.8	15.3	4.6	
	Final tailing	63.67	0.13	0.46	5.18		0.005	0.64	1.8	1.8	55.2	17.5	5.0	
3.	Feed	100.00	5.80	14.41	5.75		0.022	10.55	100.0	100.0	100.0	100.0	100.0	54.9
	Lead concentrate	9.06	62.80	4.12	3.15	12.60	0.14	103.96	98.1	2.6	5.0	58.3	89.4	
	Zinc "	28.67	0.10	46.84	9.80	8.00	0.01	2.33	0.5	93.2	48.8	13.3	6.1	
	Final tailing	62.27	0.13	0.98	4.26		0.01	0.74	1.4	4.2	46.2	28.4	4.5	
4.	Feed	100.00	6.24	13.94	5.96		0.017	10.10	100.0	100.0	100.0	100.0	100.0	71.0
	Lead concentrate	7.73	76.40	1.73	1.50	8.64	0.14	122.72	94.6	1.0	1.9	65.1	85.5	96.5
	Zinc "	24.62	0.77	53.85	8.30	3.24	0.01	3.63	3.0	95.1	34.3	15.1	8.0	
	Final tailing	67.65	0.22	0.81	5.62		0.005	1.06	2.4	3.9	63.8	19.8	6.5	
5.	Feed	100.00	5.70	13.72	5.95		0.026	10.46	100.0	100.0	100.0	100.0	100.0	54.2
	Lead concentrate	9.70	58.00	4.39	3.53	13.66	0.17	97.88	98.8	3.1	5.9	61.5	90.7	81.0
	Zinc "	23.48	0.08	55.59	8.57	1.40	0.01	1.92	0.3	95.1	33.8	7.7	1.8	95.5
	Final tailing	66.82	0.08	0.36	5.38		0.01	1.17	0.9	1.8	60.3	30.8	7.5	
6.	Feed	100.00	6.00	13.78	5.49		0.017	11.62	100.0	100.0	100.0	100.0	100.0	52.6
	Lead concentrate	10.52	56.00	4.68	3.34	17.06	0.11	95.44	98.2	3.6	6.4	67.4	86.4	
	Zinc "	22.98	0.10	55.10	8.28	1.30	0.01	2.26	0.4	91.9	34.7	13.4	4.5	
	Final tailing	66.50	0.13	0.94	4.87		0.005	1.59	1.4	4.5	58.9	19.2	9.1	
7.	Feed	100.00	6.50	15.00	5.50		0.023	12.57	100.0	100.0	100.0	100.0	100.0	52.6
	Lead concentrate	15.57	41.20	7.18	4.33	26.10	0.11	72.19	98.7	7.5	12.3	75.7	89.4	72.4
	Zinc "	25.51	0.10	52.10	9.05	4.17	0.01	2.46	0.4	88.7	42.0	11.5	5.0	65.6
	Final tailing	58.02	0.10	1.00	4.33		0.005	1.21	0.9	3.8	45.7	12.8	5.6	

(Continued on next page)

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Results, Test Nos. 1 to 14, cont'd.:

Test No.	Product	Weight, per cent	ASSAYS Per cent Pb	Zn	Fe	Insol.	Oz./ton Au	Ag	RECOVERIES, per cent Pb	Zn	Fe	Au	Ag	MESH, % -200
8.	Feed	100.00	6.40	15.18	5.61		0.021	11.39	100.0	100.0	100.0	100.0	100.0	53.2
	Lead concentrate	9.97	63.20	4.28	2.90	9.44	0.15	102.42	98.5	2.8	5.2	72.5	89.7	
	Zinc "	25.16	0.13	54.72	8.65	2.20	0.01	1.77	0.5	90.7	38.8	12.1	3.9	
	Final tailing	64.87	0.10	1.52	4.85		0.005	1.13	1.0	6.5	56.0	15.4	6.4	
9.	Feed	100.00	5.70	13.10	5.40		0.017	10.59	100.0	100.0	100.0	100.0	100.0	55.6
	Lead concentrate	9.97	56.00	4.34	3.40	15.53	0.11	93.75	98.0	3.3	6.3	65.8	88.4	76.1
	Zinc "	22.93	0.13	53.20	9.40	2.80	0.01	2.09	0.5	93.1	39.9	13.8	4.5	68.8
	Final tailing	67.10	0.13	0.70	4.33		0.005	1.12	1.5	3.6	53.8	20.4	7.1	
10.	Feed	100.00	3.35	8.89	5.15		0.016	7.38	100.0	100.0	100.0	100.0	100.0	65.5
	Lead concentrate	5.65	56.80	3.47	4.65	12.86	0.15	104.47	95.8	2.2	5.1	54.2	80.1	
	Zinc "	16.54	0.13	50.25	11.90	3.65	0.02	2.62	0.7	93.5	38.2	21.0	5.9	
	Final tailing	77.81	0.13	0.49	3.75		0.005	1.33	3.5	4.3	56.7	24.8	14.0	
11.	Feed	100.00	2.80	7.37	6.40		0.01	6.62	100.0	100.0	100.0	100.0	100.0	50.0
	Lead concentrate	4.18	64.00	2.49	5.10	6.00	0.10	134.80	95.6	1.4	3.3	43.3	78.7	86.0
	Zinc "	14.13	0.13	48.60	11.95	3.64	0.01	4.45	0.6	93.2	26.4	14.4	8.8	
	Final tailing	81.69	0.13	0.49	4.73		0.005	1.10	3.8	5.4	70.3	42.3	12.5	
12.	Feed	100.00	2.55	7.20	5.99		0.0084	6.63	100.0	100.0	100.0	100.0	100.0	50.0
	Lead concentrate	5.23	46.80	4.25	5.25	19.18	0.06	102.94	96.0	3.1	4.6	36.9	81.2	
	Zinc "	11.23	0.15	48.57	13.44	3.35	0.01	3.53	0.7	75.8	25.2	13.1	6.0	
	Final tailing	83.54	0.10	1.82	5.04		0.005	1.02	3.3	21.1	70.2	50.0	12.8	
13.	Feed	100.00	5.70	14.09	5.57		0.015	11.41	100.0	100.0	100.0	100.0	100.0	71.7
	Lead concentrate	8.38	66.80	3.13	2.20	8.50	0.11	118.62	98.3	1.9	3.3	61.7	87.2	
	Zinc "	24.43	0.13	54.39	8.30	2.00	0.01	2.54	0.6	94.3	36.4	16.1	5.4	
	Final tailing	67.19	0.10	0.80	5.00		0.005	1.26	1.1	3.8	60.3	22.2	7.4	
14.	Feed	100.00	6.10	14.12	5.35		0.014	11.31	100.0	100.0	100.0	100.0	100.0	65.7
	Lead concentrate	9.06	66.00	3.29	2.50	13.12	0.13	110.58	98.1	2.1	4.2	64.3	88.6	98.6
	Zinc "	24.47	0.13	55.60	8.03	1.28	0.01	2.83	0.5	96.4	36.8	14.3	6.1	91.9
	Final tailing	66.47	0.13	0.32	4.75		0.005	0.90	1.4	1.5	59.0	21.4	5.3	

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SECTION E. - DISCUSSION OF METALLURGY.

Effect of Various Grinds:

In the tests, grinding to 54-55 per cent minus 200 mesh gave good recoveries of lead, silver, gold and zinc. The grind of 65 per cent minus 200 mesh is recommended because of apparent building-up of the coarse minerals in the zinc circuit and their subsequent loss in the tailing.

While the galena floats rather easily at the coarse sizes, results indicate that the finer grind is beneficial in causing less zinc to report in the final lead concentrates. **For the coarser grind slightly more frother is required,** which may be the cause of the floating of more sphalerite in the lead circuit.

It was thought that the finer grinds would be detrimental to the recovery of silver, but this does not appear to be the case.

Comparison of Lead Concentrate at Various Grinds.

Test No.	% -200 mesh in classifier overflow	Recovery of zinc, per cent	Recovery of silver, per cent	Ore
4	71.0	1.0	93.9	Lot No. 1 - High Grade.
13	65.5	1.9	87.2	"
14	65.5	2.1	88.6	"
5	54.2	3.1	89.7	"
6	52.4	3.6	86.4	"
8	53.2	2.8	89.7	"
10	65.5	2.2	80.1	Lot No. 2 - Low Grade.

Conditions were similar with regard to reagents and circuit in Tests Nos. 4, 5 and 6. In Tests Nos. 10, 11, 13 and 14, a recleaning stage was used. From the above table it appears fairly certain that the finer grind should be used.

On examining the tests, it was noted that a steady

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(Effect of Various Grinds, cont'd) -

increase of the zinc content of the zinc rougher tailing was occurring in the case of the coarser grinds. Trouble was experienced in Test No. 13 due to sanding-up in the conditioner. This condition was, most likely, due to the coarse grind obtained in Tests Nos. 11 and 12. It was necessary to add air to the conditioner to relieve this condition and consequently coarse sand was released to the circuit, causing a higher tailing in the zinc circuit than was normal.

The following table shows the increasing losses of zinc in the final tailing, due to coarse grinding.

Effect of Grind on Final Tailings.

Test No.	% -200 mesh in classifier overflow	Per cent zinc in final tailing	Per cent distribution of zinc in final tailing
5	54.2	0.36	1.8
6	52.6	0.94	4.5
7	52.6	1.00	3.8
8	53.2	1.52	6.5
14	65.5	0.32	1.5

Although other factors were involved in these tests, such as change in frothers, the steady increase of the zinc in the final tailing appears to be attributable mainly to the building-up of coarse zinc in the circuit and its eventual elimination in the final tailing.

Also, it will be noted from the table of reagents that it was necessary to add additional frother to the zinc recleaner circuit where the coarser grinds were used.

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Action of Frothers:

The use of four different frothers was tested during the runs, Barrett No. 4 (Test No. 1), Aerofloat 25 (Test No. 7), Pine oil (Test No. 6), and cresylic acid.

Cresylic acid gave the best results of any of the

(Action of Frothers, cont'd) -

frothers used.

In Test No. 1, Barrett No. 4 oil was used in conjunction with cresylic acid. It was thought that improved recovery of the precious metals would result but results were so uniformly good with cresylic acid that, despite the fact that the test was unsatisfactory due to starting up, the test was not repeated. In view of the results obtained with cresylic acid it is doubtful whether these would be improved through the use of Barrett No. 4.

Aerofloat 25 was also tested to see whether improved recovery of the precious metals would be obtained. The strong collecting properties of this reagent, however, resulted in a greater loss of zinc in the lead circuit, which more than offset any possible gain in recovery of the precious metals. This reagent is not recommended for use on this ore.

Pine oil was used in the zinc circuit only as a substitute for cresylic acid (Test No. 6). While there was an increase of zinc loss in the final tailing, this appears to be attributable to the coarse grind rather than to the use of pine oil. Under the circumstances, it is believed that pine oil could be used in place of cresylic with comparable results.

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<u>Action of Collectors</u>:

Three different collectors were used; Aerofloat 25 (Test No. 7), amyl xanthate (Tests Nos. 1 and 2), and potassium ethly xanthate.

The action of Aerofloat 25 has been discussed under frothers and it appears to be too powerful a promoter for both zinc and mica to be applicable on this ore.

Amyl xanthate was used to improve recovery of the

(Action of Collectors, cont'd) -

precious metals but did not appear to give any better results than potassium ethyl xanthate.

Potassium ethyl xanthate gave excellent results on the recovery of all the economic minerals in this ore.

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Action of Depressants:

Potassium Cyanide: This reagent was used in the lead circuit as a depressor for the iron and zinc minerals.

Zinc Sulphate: This reagent was used in the lead circuit in conjunction with cyanide as a depressor for the zinc minerals. It is possible that excessive use of this reagent might also have a depressing effect on the silver minerals, although results from using 1.0 pound per ton of ore were very satisfactory.

Sodium Silicate: The use of this reagent is considered necessary as a depressor for the micaceous minerals, which are relatively abundant in the ore. The optimum results appear to be obtained at around 3.0 pounds per ton of ore.

Lime: This reagent was used both as pH regulator and as depressor for iron in the zinc circuit. The amount of this reagent and its place of addition were tested. Tests were run, (1) adding all the lime to the recleaners and (2) adding the lime to the zinc rougher conditioner and to the cleaners. The addition of lime to the zinc rougher conditioner appeared to be the more beneficial. The following table of assays gives a comparison of the results:

(Continued on next page)

(Action of Depressants, cont'd) -

	ADDED TO RECLEANER		TO ZINC CONDITIONER	
	Test No. 5	Test No. 6	Test No. 13	Test No. 14
	(Iron, per cent)			
Zinc recl. conc. -	8.57	8.28	8.30	8.03
Tailing -	11.39	12.40	17.00	13.60
Zinc cl. conc. -	10.68	9.66	8.50	8.14
Tailing -	23.60	23.39	16.20	15.90
Zinc rghr. conc. -	14.75	12.25	9.20	8.65
Tailing -	5.38	4.87	5.00	4.77

Although the iron in the zinc recleaner concentrates was uniformly good it would appear from the assays that the cleaner and recleaner circuits were having an undue amount of work. A circulating load of iron minerals had developed which probably would, in time, result in a decrease in the grade of zinc concentrates due to contamination from iron minerals. One other thing was noticed in these two circuits: the zinc recleaner tailing in the case of Tests Nos. 5 and 6 assayed 47.02 and 39.35 per cent zinc respectively as against 13.68 and 21.60 per cent zinc for Tests Nos. 13 and 14. This would indicate a crowding of the zinc cleaner circuit which might result in an eventual loss in recovery of the zinc.

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Action of pH Regulators:

Two pH regulators were used: soda ash in the lead circuit, and lime in the zinc circuit.

In Tests Nos. 3 and 4, 1.0 pound soda ash per ton of ore was used. While Test No. 3 appeared quite normal in every respect, Test No. 4 gave a very high lead concentrate with a heavy "dead" froth. This also resulted in a greater loss of lead and silver in the lead rougher tailing, which would undoubtedly have increased if this condition had persisted. Returning the soda ash to 2.0 pounds per ton of ore the following day, however, cleared up this condition and, except in the case

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(Action of pH Regulators, cont'd) -

of the low-grade ore, this condition did not appear again. The best apparent conditions obtained in the lead circuit were in Tests Nos. 13 and 14, where pH values of 9.8 and 9.7 were obtained in the lead rougher tailing by the use of 2.8 pounds of soda ash per ton of ore.

The use of lime in the zinc circuit is fairly thoroughly discussed, under the heading of "Depressants", on Page 25. The pH in the zinc rougher circuit was maintained at 11.3, and sufficient lime was added to the zinc cleaners to maintain this pH in the zinc cleaner tailing. A slightly higher pH than this would be safer, as it was noticed that when the pH dropped below 11.2 the froth level

in turn, might

in the cells dropped, which, in turn, might cause a loss in the recovery of the zinc.

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Action of Activators:

The only activator used was copper sulphate in the zinc circuit. The use of 1.0 pound per ton appears satisfactory.

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Lead Metallurgy:

A grade of 65 per cent lead, with a recovery of 97 per cent of the lead, can be easily obtained. Although it is not necessary to obtain so high a grade the lower zinc recovery in this concentrate appears to warrant seeking it.

In the first series of tests (Nos. 1 to 7), the lead was cleaned only once, while in the remaining tests the lead was given a second cleaning.

The grade of lead to be expected from one cleaning is 56 to 58 per cent, with a recovery of 3.6 to 3.1 per cent of the zinc in the lead concentrate. A grade of 65 per cent

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(Lead Metallurgy, cont'd) -

lead, however, would show only 2.0 per cent of the zinc recovered in the lead concentrate, a gain of 1 per cent in the zinc recovery. This is shown in Test No. 14, where the recovery of zinc in the zinc concentrates was 96.4 per cent, whereas the best previous recovery with a comparable tailing was 95.1 per cent (Test No. 5).

While the lead mineral is freed at a fairly coarse mesh size, the flotation of the galena at this coarse size is only obtained by the use of more frother, which in turn tends to cause both the slime sphalerite and coarse mica to report in the concentrates. It is desirable, therefore, that the grind be at least 60 per cent, and preferably about 65 per cent, minus 200 mesh.

The use of the Denver unit cell was attempted in Tests Nos. 1 and 2, with unsatisfactory results. Two things were accountable, in large part, for the poor performance: (1) the high circulating load in the ball mill-classifier circuit, and (2) the necessity of maintaining a high density (63 per cent solids) going to the classifier to obtain the desired density of overflow. The cell worked only intermittently, with occasional flooding due to sanding in the discharge. Consequently, no sample was taken. It was noted, however, that when the concentrate was being removed satisfactorily it contained a very large amount of sphalerite. From this it would appear that the concentrates would require cleaning for the removal of the sphalerite. On this account it was felt that the use of the unit cell did not improve the metallurgy in any way. A slight concentration of the gold was obtained in the hutch concentrate but not enough to warrant the use of the cell for this purpose alone. This hutch product assayed 7.40 per cent lead, 16.52 per cent zinc, 0.10 ounce per ton gold, and 12.21 ounces per ton silver.

Gold Metallurgy:

The recovery of gold in the lead concentrate averaged about 68 per cent. The gold in the lead rougher tailing assayed less than 0.01 ounce per ton and more than 0.005 ounce per ton. The recovery of gold was more dependent upon the amount of gold found in the heads than upon any other factor, as the final tailing assay remained very uniform.

No indication of free gold was obtained during the run and examination of the ball mill sand and the classifier bed at the conclusion of the run did not show an abnormal concentration of the gold in either of these products.

No indication of whether the gold was associated with any other mineral was observed. If anything, the gold appeared to be associated with the lead and silver minerals. It was thought that the gold might be associated with the pyrite, but this was not borne out by the tests. The recovery of the gold remained fairly uniform in spite of comparatively wide variations in the recovery of iron in the lead concentrates. The final tailings were tabled to make an iron concentrate but at no time did the gold assay go above 0.02 ounce per ton in the table concentrate, and the usual figure was 0.01 ounce per ton in this concentrate.

Silver Metallurgy:

The recovery of silver in the lead concentrate was uniformly good, averaging about 90 per cent. It was thought that grinding finer than 55 per cent minus 200 mesh might cause sliming of the silver minerals, but this does not appear to be the case. The flotation reagents used for the lead flotation gave as good recoveries of the silver as did special reagents used in an effort to increase the recovery of the silver minerals.

Zinc Metallurgy:

A recovery of 94 to 95 per cent of the zinc, at a grade of 55 per cent zinc, should be readily obtained. The zinc minerals float fairly rapidly and a low tailing is easily made. Chamical analysis indicates that marmatite is present in the ore and that the limiting grade to be expected from the pure minerals would be approximately 57 per cent zinc. Care must be taken in this circuit to keep the mica from floating in the zinc concentrates. A grade of 48 to 50 per cent zinc should be maintained on the zinc roughers to ensure obtaining a high-grade final concentrate. This can be done by returning the lower-grade portion of the zinc rougher concentrate, which is high in iron and mica minerals, to the zinc rougher conditioner as shown in the recommended flow-sheet. In Test No. 8, only one stage of cleaning was used and, while the tailings were quite high, this high tailing can be attri- attributed to the coarse grind rather than too short-circuiting due to only one stage of cleaning (refer to Page 23, "Effect of Grind on Final Tailings"). It is recommended, however, that two stages of cleaning be adopted.

Iron Metallurgy:

The two main iron minerals present in the ore are: pyrrhotite and pyrite. There is no particular difficulty in depressing them and while the zinc concentrate assays over 8 per cent in iron, 7.9 per cent of this is soluble in hydrochloric acid, which would indicate that it is tied up as marmatite in the zinc. This is further borne out by the fact that the grade of iron in the zinc concentrate is quite uniform.

Cyanide in amounts of 0.10 pound per ton act as a sufficient depressor of the iron minerals, and 3 pounds per ton

(Iron Metallurgy, cont'd) -

of lime to the zinc circuit gives a satisfactory depressing action on the zinc minerals.

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Tonnage Figure Calculations,
Using Test No. 5 as Example:

Test No. 5 has been worked out, to give a complete set of tonnage figures. These figures will give some indication of the tonnages in the various parts of the circuit. Unfortunately, it was not possible to calculate the tonnage figures for Test No. 14, due to the complexity of the lead circuit.

Tonnages were all calculated by formula. The screen analysis figures were used for the circulating load, and assay results for all other tonnage figures.

(The results of Test No. 5)
(appear on the next page,)
(in tabulation form.)

Results of Test No. 5:

Product	Per cent weight	ASSAYS Per cent Lead	Zinc	Iron	Insol.	Oz./ton Gold	Silver	Distribution, per cent Lead	Zinc	Iron	Gold	Silver	MESH, per cent -200
Ore	100.00	5.32	14.28	6.05	.	0.02	9.60						
Ball mill discharge	1350.00	5.98	14.24	7.18									
Classifier sands	1250.00	6.00	14.28	7.28									
" overflow	100.00	5.70	13.72	5.95		0.026	10.46	100.0	100.0	100.0	100.0	100.0	54.2
Final lead conc.	9.70	58.00	4.39	3.53	13.66	0.17	97.88	98.8	3.1	5.9	61.5	90.7	81.0
Lead cleaner tailing	4.82	10.40	7.85	7.16				8.8	2.7	5.9			
Lead rougher feed	104.82	5.91	13.44	6.01				108.8	102.7	105.9			
" " conc.	14.52	42.20	5.54	4.82				107.6	5.8	11.8			
" " tailing	90.30	0.08	14.72	6.21		0.011	1.07	1.2	96.9	94.1	38.5	9.3	
Final zinc conc.	23.48	0.08	55.59	8.57	1.40	0.01	1.92	0.3	95.1	33.8	7.7	1.8	95.5
Zinc recl. tailing	70.44	0.08	47.02	11.39				1.1	241.4	134.8			
Zinc cleaner feed	110.36	0.09	42.26	12.60				1.8	340.0	233.8			
" " conc.	93.92	0.08	49.16	10.68				1.4	336.5	168.6			
" " tailing	16.44	0.13	3.47	23.60				0.4	4.2	65.2			
Zinc rougher feed	106.74	0.09	12.98	8.89				1.6	101.0	159.3			
" " conc	39.92	0.10	33.87	14.75				0.7	98.5	99.0			
" " tailing	66.82	0.08	0.36	5.38		0.01	1.17	0.9	1.8	60.3	30.8	7.5	

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Per cent weight

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SECTION F. - SUMMARY AND CONCLUSIONS.

1. An excellent recovery of the lead, gold, silver and zinc minerals can be expected from this ore.

2. The standard lead-zinc flotation reagents were satisfactory for use on this ore. However, the presence of micaceous minerals makes the use of sodium silicate, in addition to the standard reagents, desirable as an important factor in obtaining a high-grade concentrate.

3. The gold and the silver follow the lead mineral, and special reagents to benefit their recovery did not show any improvement over the standard reagents used.

4. The use of the Denver unit cell did not appear to be beneficial on this ore.

5. The use of the recleaner circuit in the lead flotation is recommended because of the resultant lower recovery of zinc in the final lead concentrate.

6. The return of a portion of the zinc rougher concentrate to the zinc feed conditioner would have the effect of "levelling off" the grade of zinc rougher feed and concentrate. This would tend to give better results in the cleaner-recleaner circuit, by keeping to a minimum the burden of gangue minerals in this circuit.

7. Close control of the zinc circuit will have to be maintained in order to obtain a grade of 55 per cent zinc, as the maximum grade of zinc theoretically obtainable appeared to be about 57 per cent zinc.

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