DEPARTMENT OF MINING AND METALLURGY

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FLOTATION TESTS ON TWO FRIENDS AND BANK OF ENGLAND ORE

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The tests discussed in the following pages were performed between October, 1943, and March, 1944, by L. J. Gall, Mining Student, and P. H. H. Hookings, H. M. Abbott, and M. D. E. Robinson, Metallurgy Students, in the Ore Dressing laboratories at the University of British Columbia.

The tests were performed under the able supervision of Professor G. A. Gillies; his many helpful suggestions are gratefully acknowledged. The authors also wish to thank Mr. W. R. Smith, Assistant in the Department of Mining and Metallurgy, and Dr. H. V. Warren, of the Department of Geology, for his information concerning the character of the ore.

TABLE OF CONTENTS

TABLES AND ILLUSTRATIONS

FLOTATION TESTS ON TWO FRIENDS AND BANK OF ENGLAND ORE

SUMMARY

1. High grade concentrates of sphalerite and galena, showing good recoveries of Zn and Pb, can be obtained by using ordinary flotation methods.

2. Iron can be reduced to below 1.5% in both concentrates without complicating the flotation circuit.

3. No feasible method was found to substantially reduce the high proportion of silver floating with the sphalerite.

4. Both the galena and sphalerite float more readily in an alkaline circuit.

5. The cycle of flotation giving the best results was roughly as follows:

(a) Depress sphalerite and pyrite with sodium cyanide.

(b) Float galena with sodium ethyl xanthate, using cresylic acid as a frother.

(c) Reactivate and float sphalerite with copper sulphate, using cresylic acid as a frother.

6. Further experimentation with grinding methods might result in freeing more silver from the sphalerite.

7. A comprehensive mineralogical study of the ore should be made prior to further investigations.

INTRODUCTION

The object of the tests outlined in this report was to determine practicable flotation methods for obtaining:

(a) a high grade, high recovery lead concentrate,

(b) a high grade, high recovery zinc concentrate,

and

(c) maximum separation of silver from the zinc, together with high recovery of silver.

Although the results of these tests are possibly somewhat mediocre, the authors feel that much of the information recorded in this report would prove valuable to anyone contemplating further investigations as to the "flotability" of the ore.

OUTLINE OF WORK

About 100 lb of representative samples of the coarse ore were picked for the tests and crushed to -10 mesh. A preliminary series of tests were then run to determine the best grinding time, pulp dilutions, and basic flotation circuit for this particular ore.

Grinding tests were carried out in conjunction with the preliminary flotation tests; grinds much under 10 minutes resulted in poor separation of galena and sphalerite, while those much over 10 minutes resulted in excessive sliming of the sphalerite and, hence, low zinc recovery in the sphalerite concentrates. All grinding was done in a laboratory rod-mill, 10" by 12" long, holding 24 steel rods weighing a total of approximately 50 lb, and revolving at 44 r.p.m.; pulp dilutions were 1:1 for each grind.

The preliminary flotation tests, to determine the best basic flotation cycle, were all carried out in a 1000-gm Ruth laboratory flotation cell at a pulp dilution of 3.5:1. Experimentation with different reagents finally resulted in a fair separation of galena and sphalerite with moderately high recovery of Pb and Zn in their respective concentrates; this particular test was used as a basis for the final tests.

Final tests were in two groups. The first tests were devoted to finding the best amounts of reagents to use to obtain the best concentrates; these tests were carried out in a 1000-gm Ruth laboratory flotation cell.

The second group of tests was devoted to obtaining good rougher concentrates and, from these, to produce high-grade, high-recovery concentrates; the roughers were obtained from a 2000-gm Denver sub-aeration cell and cleaned in a similar 1000-gm cell.

Skimmed concentrates and tailings were placed in the laboratory oven, dried 24 hours, and assayed for zinc, lead, iron, and silver.

At all times during the final tests, the operators attempted to standardize the following factors:

1. Grinding time.
2. Pulp densities.
3. Reagents and pH.
4. Pulp temperature.
5. Conditioning time of pulp with reagents.
6. Contact period of pulp with reagents.
7. Skimming practice.
8. Aeration of pulp in flotation cells.

REAGENTS USED

Promoters

Aerofloat 31, undiluted, cell
Aerofloat 208, 5% solution, cell
Sodium Aerofloat, dry, cell
Sodium Ethyl Xanthate, dry, cell.

Frothers

Pine Oil, undiluted, cell
Cresylic Acid, undiluted, cell
Frother 60, undiluted, cell.

Activators

Copper Sulphate, 5% solution, cell
Sulphuric acid, 1:1, cell.

Depressants

Sodium Cyanide, 5% solution, rod-mill and cell
Potassium Dichromate, 5% soln., cell
Zinc Sulphate, 5% solution, rod-mill
Starch, 10% solution, cell
Lime, dry, rod-mill and cell
Sodium Silicate, 10% solution, cell.

Modifying Agents

Lime, as above
Sodium Carbonate, 5% solution, cell
Sodium Silicate, as above
Sulphuric Acid, as above.

The best promoter for collecting the flotation products was sodium ethyl xanthate. The Aerofloat reagents did not prove successful in separating lead from zinc; in each case a heavy lead recovery was obtained in the zinc concentrate. This failure of the Aerofloats may not have been investigated thoroughly enough, but such good results were obtained by using the xanthate that it was considered a waste of time to continue further experimentation with the Aerofloats.

Cresylic acid proved to be the best frother, especially in conjunction with the sodium ethyl xanthate. The froth obtained with pine oil broke down too easily towards the end of the galena flotation, while that obtained with Frother 60 was too tough to allow for proper cleaning.

Copper sulphate was the natural choice for reactivation of the sphalerite.

Sodium cyanide was used to depress sphalerite and pyrite; zinc sulphate was used in the final tests to increase the effectiveness of the cyanide and to reduce the conditioning time. Both these reagents were added in the grinding circuit. Potassium dichromate was found to increase the grade of the sphalerite concentrate by depressing galena. Sodium silicate was used to disperse the gangue, to get a cleaner sphalerite concentrate. Starch and lime were also used in the preliminary tests, but were not required for the final tests.

Sodium carbonate was found to be the most successful agent for keeping the pH up to a practicable level. Lime was not quite as successful in this, although it would probably have been used had there been enough pyrite in the ore to warrant it.

DETAILS OF TESTS

The tests conducted on the ore were divided into two phases, preliminary and final. The preliminary tests were conducted to determine the correct grind and to find a basic flotation cycle to start the final tests. The final tests were conducted with a view to improving the grade and recovery of the concentrates obtained in the basic test.

Preliminary Tests

Twelve preliminary tests were run. From these, it was found that a grinding time of 10 minutes was the best balance to strike between oversliming the sphalerite and completely unlocking the galena and sphalerite. A screen analysis of this product ran 73.8% -200 mesh.

The authors consider that to record all these tests would be a waste of the reader's time, since the only results obtained which were worthy of note were those of test No. 9. For this reason, test No. 9 is the only test of the twelve to be recorded in this report. Suffice it to say that the balance of the tests were merely futile attempts to obtain even fair lead and zinc concentrates.

Test No. 9

1000 gm ore
Grind: 10 min., 0.3#NaCN/ton
Cell 1: 0.1#NaEX/ton, 2 drops Cresylic Acid.

Concentrate 1.

Cell 2: 0.05#Na_2SiO_3, 0.8#$CuSO_4$/ton, 1 drop Cresylic Acid.

Concentrate 2.

Concentrate 1 was a fair galena product, showing the first good galena recovery of the preliminary tests. Concentrate 2 was a good, clean sphalerite product, although recovery was low. This test was used as the basis on which to begin the final tests, and is recorded as test I in the table inside the back cover.

Final Tests

The twelve final tests, Nos. II to XIII, and test No. 9 of the preliminary tests, are recorded in the table inside the back cover.

Beginning with the results obtained in test 9, it was decided to vary the amount of cyanide in the rod mill, keeping the quantities of all other reagents constant. Conditioning time was eight minutes for each addition of reagents to cell.

Test I. See Test 9, Preliminary Tests.

Test II. 1000 gm ore
Grind: 10 min., 0.45#NaCN/ton
Cell 1: 0.1#NaEX/ton, 2 drops cresylic acid
Concentrate 1

Cell 2: 0.8#$CuSO_4$/ton, 1 drop cresylic acid.
Concentrate 2.

Test III. 1000 gm ore
Grind: 10 min., 0.6#NaCN/ton
Cell 1: same as test II
Cell 2: " " " II.

Test IV. 1000 gm ore
Grind: 10 min., 0.8# NaCN/ton
Cell 1: same as test II
Cell 2: " " " II.

Test V. 1000 gm ore
Grind: 10 min., 1.0#NaCN/ton
Cell 1: same as test II
Cell 2: " " " II.

Tests II to V completed a series of tests to determine the optimum amount of NaCN to add to the circuit. A graph was drawn (see Graph 1) showing per cent recoveries of lead and zinc against the amount of NaCN added to the mill; test IV was accordingly chosen as the basis for the following tests, since it showed maximum lead recovery in the galena concentrate and a fairly high zinc recovery in the sphalerite concentrate.

A second series of tests was then begun to determine the optimum amount of sodium ethyl xanthate. This was done by varying only the amount of NaEX used.

Test VI. 1000 gm ore
Grind: 10 min., 0.8#NaCN/ton
Cell 1: 0.4#NaEX/ton, 2 drops cresylic.
Concentrate 1.

Cell 2: 0.8#$CuSO_4$/ton, 1 drop cresylic.
Concentrate 2.

Test VII. 1000 gm ore
Grind: Same as test VI
Cell 1: 0.7#NaEX/ton, 2 drops cresylic.
Concentrate 1.

Cell 2: same as test VI.

After test VII concentrates were assayed, another graph was drawn (see Graph 2) to show per cent recoveries against the amount of NaEX added to the cell. Test VI showed the best results and was therefore used as a basis for the following tests. With the exception of test VIII, which was carried out to confirm the effect that lowering the pH had on recoveries of sphalerite, the remainder of the tests were run with 0.8#NaCN/ton added to the grinding circuit and 0.4# NaEX added to the cell.

Test VIII. 1000 gm ore
Grind: 10 min., 0.8#NaCN/ton
Cell 1: 0.3#NaEX-ton, 1 drop cresylic.
Concentrate 1.

Cell 2: 0.8#$CuSO_4$, 1.0#H_2SO_4/ton, 1 drop cresylic.
Concentrate 2.

Test VIII proved that lowering the amount of NaEX in cell 1 raised the zinc recovery in the lead rougher and confirmed that an acidic circuit was unsuitable for floating sphalerite.

Since sphalerite showed a tendency to increase slightly in the galena rougher as the amount of NaEX added to cell 1 was increased, it was decided to add an additional 0.15#NaCN/ton to cell 1, in order to depress the additional sphalerite; 2.0#$ZnSO_4$/ton was added to the grind to increase the effectiveness of the NaCN. It was also decided to double the size of batch, in order to have enough rougher for a 6:1 pulp dilution in the zinc and lead cleaner cells. Test VI was used as a basis.

Test IX. 2000 gm ore
Grind: 10 min., 0.8#NaCN, 2.0#$ZnSO_4$/ton.
Cell 1: 0.15#NaCN, 0.4#NaEX/ton, 3 drops cresylic.
Rougher 1 - Pb Concentrate
- Pb Middlings.
Cell 2: 0.25#Na_2SiO_3, 0.85#$CuSO_4$, 1.0#$K_2Cr_2O_7$/ton, 2 drops cresylic.
Rougher 2 - Zn Concentrate
- Zn Middlings.

Test X. 2000 gm ore
Grind: same as test IX.
Cell 1: 0.75#Na_2SiO_3, 0.15# NaCN, 0.4#NaEX/ton, 3 drops cresylic.
Rougher 1 - Pb Concentrate
- Pb Middlings.
Cell 2: 1.0# $CuSO_4$, 0.5# $K_2Cr_2O_7$/ton, 2 drops cresylic.
Rougher 2 - Zn Concentrate
- Zn Middlings.

Test XI. 2000 gm ore
Grind: 15 min., 0.8# NaCN, 2.0 # $ZnSO_4$/ton.
Cell 1: same as test X.
Cell 2: 0.5# $CuSO_4$, 0.25# Na_2SiO_3, 1# $K_2Cr_2O_7$/ton, 4 drops cresylic.
Rougher 2 - Zn Concentrate
- Zn Middlings.

Zinc recovery in the zinc rougher dropped considerably in test XI, so the original 10 minute grind was resumed.

Test XII. 2000 gm ore
Grind: same as test X.
Cell 1: " " " X.
Cell 2: " " " XI.

Test XIII. 2000 gm ore
Grind: same as test X.
Cell 1: 0.15# NaCN, 1.0# Na_2SiO_3, 0.5# NaEX/ton, 3 drops cresylic.
Rougher 1 - Pb Concentrate
- Pb Middlings.
Cell 2: 0.25# Na_2SiO_3, 1.1# $CuSO_4$, 1.0# $K_2Cr_2O_7$/ton, 3 drops cresylic.
Rougher 2 - Zn Concentrate
- Zn Middlings.

Refloatation of roughers in tests IX to XIII produced high grade lead and zinc concentrates, but only at the expense of recovery of Pb in the lead concentrates. Recovery of zinc in the zinc concentrates was almost as high as in the roughers, little cleaning being necessary.

CONCLUSIONS

Discussion of Results

In tests IX to XIII, lead recovery was increased from 90.0 to 91.3% in the lead roughers and zinc recovery from 87.4 to 92.5% in the zinc roughers. Cleaning the lead roughers gave only fair results; grades were increased to over 60%, but the resulting concentrates still contained over 5% zinc. Cleaning the zinc roughers was more successful; concentrates still recovered almost 90% of the zinc, while the grade was over 63% zinc in all five cases. Zinc recovery was not seriously affected by the retention of zinc in the lead concentrates, 5% Zn in the lead concentrates representing approximately only 1% of the total zinc in the ore.

An alkaline circuit proved best for floating both the galena and sphalerite. Recoveries were considerably lower in an acid circuit.

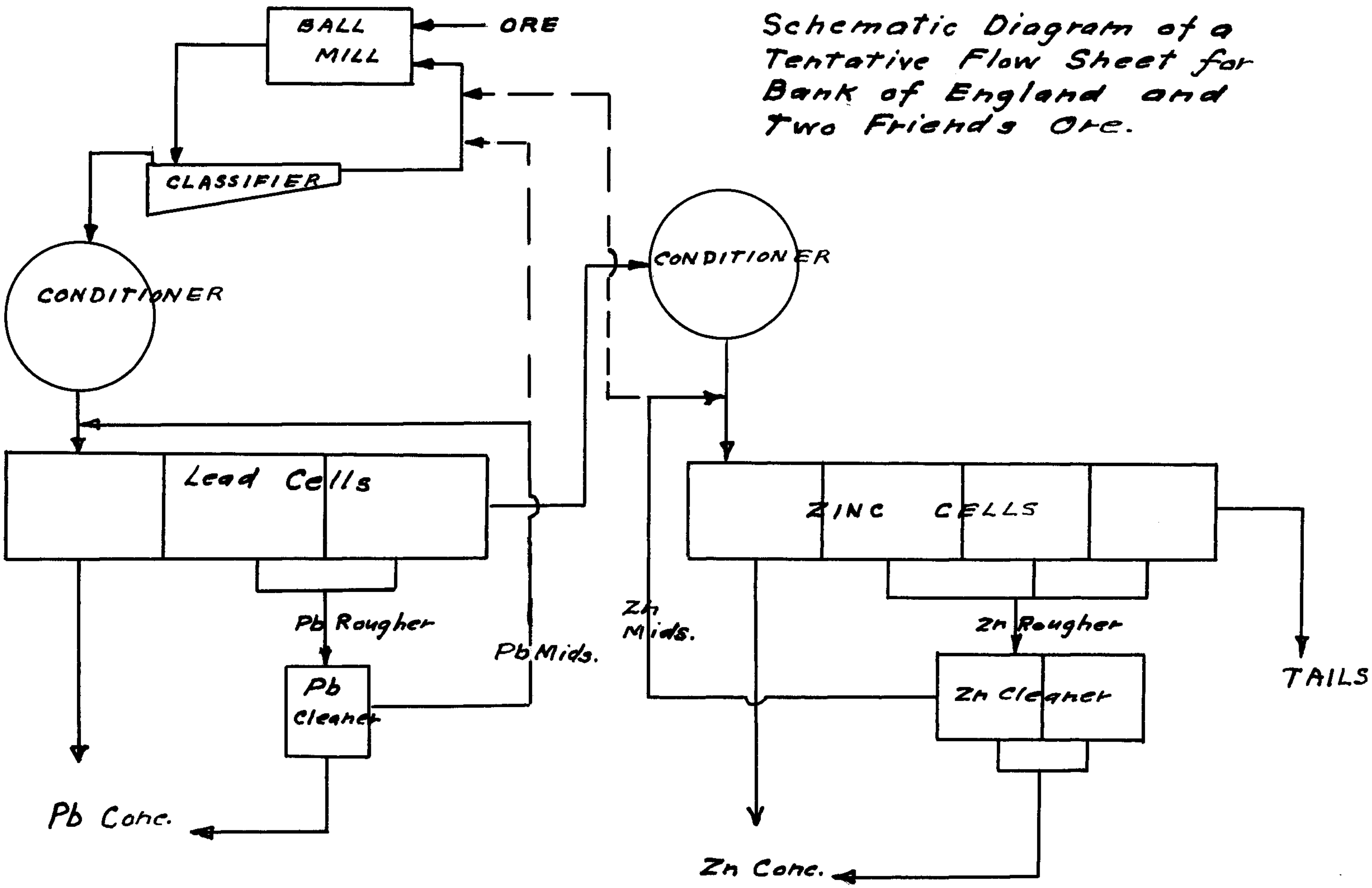
Silver values recovered were high in tests VI and IX, being divided approximately equally between the lead and zinc concentrates. No method was found to get most of the silver into the lead concentrates. A more comprehensive mineralogical study of the ore should be obtained before future work is attempted on separating the silver from the sphalerite. According to Dr. Warren[1], a good deal of the silver in the ore may be in the form of very fine particles of native silver in the sphalerite.

A report on the microscopic examination of the ore is at present being completed by one of the authors of this report, Mr. P. H. H. Hookings; this report should be obtainable from the Department of Geology after April, 1944. If the silver is indeed locked in the sphalerite in very fine particles, the problem of its extraction from the sphalerite is a difficult one.

Since time spent in the laboratory was necessarily short and since facilities were somewhat limited, it was found impossible to duplicate a circulating-load type of circuit in the laboratory. Due to the rapidity with which roughers oxidized it was found difficult to collect enough

[1] Dr. H. V. Warren, personal communication, Dept. of Geology, University of British Columbia.

Schematic Diagram of a Tentative Flow Sheet for Bank of England and Two Friends Ore.



of them for a representative cleaning flotation or a regrind. Pulp dilutions of 6:1 were obtained for cleaning the lead roughers and 4:1 for cleaning the zinc roughers.

A Tentative Flow Sheet

A schematic diagram (see facing page) and explanation of a suggested flow sheet for this particular ore is included at this point:

1. A fairly heavy circulating load and short grind in the ball-mill-classifier circuit would be used to obtain maximum unlocking of galena-sphalerite particles and, at the same time, to reduce excessive sliming of the sphalerite.

2. One cell would be used in each of the groups shown to pull off a good concentrate. The remaining cells would pull a rougher to be cleaned in the cleaner cells, the resulting concentrate to be added to the first.

3. Lead and (or) zinc middlings would be returned to the head of their respective circuits or, if the amount of locked particles warranted it, would be short-circuited through the ball mill for a regrind.

Attention is drawn to the fact that this flow sheet is only a tentative suggestion and has not been tested in any manner.

Recommendations for Further Research

1. Starting with test VI or one of tests IX to XIII, an attempt might be made to better the grades and

recoveries obtained, with special emphasis placed on effecting a cleaner separation of lead and zinc roughers.

2. To avoid excessive sliming of the sphalerite and, at the same time, to unlock as much of the Pb-Zn particles as possible, the following method of grinding is suggested:

(a) Grind in batch rod-mill for five minutes.

(b) Wet-screen pulp, saving particles below a chosen mesh, and put coarser particles back in the rod-mill.

(c) Regrind coarse material for five minutes.

(d) Wet-screen and repeat, if necessary, adding the screened material to the material obtained in (b) in each case.

A more evenly ground material for flotation tests would be the result. This material would more nearly approximate the classifier discharge of a closed-circuit grinding operation.

3. Regrinding the sphalerite concentrate might result in better separation of silver from sphalerite. As has already been pointed out, a further mineralogical study* of the ore should be made before attempting to further separate the silver from the sphalerite. There is evidence to support the theory that a large proportion of silver may lie close to the galena-sphalerite grain boundaries; an assay of the lead middlings of test XI ran 554 oz. Ag/ton, showing nearly 30% recovery.

*A detailed report on microscopic investigations of this ore is being made at the moment (March, 1944) and should be obtainable from the Dept. of Geology after April, 1944.

4. Further study of reagents, especially of sodium Aerofloat and Reagents 213 and 226, might improve the flotation of sphalerite. It is also suggested that lime could be used to raise the alkalinity of this circuit in order to further depress pyrite and improve the quality of the sphalerite rougher.

5. If the silver is present in the ore as native silver, blanketing might be tried after grinding. Probably some of the silver could be recovered before entering the flotation circuit.

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BANK OF ENGLAND - TWO FRIENDS   SILVER-LEAD-ZINC ORE

Test No.	WT. of Ore (gms)	WT. of Conc (gms)	Grind (mins.)	Conditioning (mins)	Skimming (mins)	Pulp Density	pH.	REAGENTS TO MILL: NaCN 5% cc.	#/T	$ZnSO_4$ 5% cc.	#/T	REAGENTS TO CELL: H_2SO_4 (1:1) cc	#/T	Na_2SiO_3 10% dr.	#/T	Na Eth Xan 5% cc	#/T	Na_2CO_3 5% cc	#/T	$CuSO_4$ 5% cc	#/T	$K_2Cr_2O_7$ gms	#/T	Cresylic Acid dr.	#/T	ASSAYS (Grade): Lead %	Zinc %	Iron %	Silver	RECOVERIES GRAMS: Lead	Zinc	Silver .OZS	PERCENT: Lead	Zinc	Silver
Heads																										3.65	23.6	8.5	oz/T			.0297			
I C1	1000	54	10	8	10	1/3.5	8.4	3	0.3							1	0.1	1						2		52.9	18.8	1.95	192	28.6	9.2	0.0114	79.0	3.9	38.3
C2		370			15															4	0.4			1		1.2	61.5	1.85	34.3	4.4	228.0	0.012	12.2	96.7	40.3
Tails		579																								.5	Low	12.8		2.9	Low		8.1	Low	
II C1	1000	58	10	8	10	1/3.5	8.5	4.5	0.45							1	0.1							2		46.4	23.2	3.2	161	26.9	13.5	0.0103	74.8	5.7	34.7
C2		372			15		7.5													8	0.8			1		1.0	57.0	2.7	29.5	3.7	219.0	0.0121	9.9	93.1	40.7
Tails		569																								1.0	.5	12.6		5.7	2.8		15.3	1.1	
III C1	1000	63	10	8	10	1/3.5	8.7	6	0.6							1	0.1							2		46.8	23.2	3.2	161	29.5	14.6	0.0112	80.8	6.2	37.7
C2		370			15		7.5													8	0.8			1		.7	58.6	2.6	30.1	2.6	217.0	0.0123	7.2	92.0	41.3
Tails		673																								.7	.7	12.4		4.0	4.0		11.1	1.7	
IV C1	1000	80	10	8	10	1/3.5	8.7	8	0.8							1	0.1							2		39.9	25.5	1.7	118	31.9	24.4	0.0104	87.5	8.6	35.0
C2		353			15		7.6													8	0.8			1		.45	60.9	4.4	21.3	1.6	215.0	0.008	4.4	91.1	26.9
Tails		577																								.4	.6	11.9		2.3	3.5		6.4	1.0	
V C1	1000	96	10	8	10	1/3.5	8.8	10	1.0							1	0.1							2		32.4	23.1	1.9	125	31.3	22.2	0.0132	86.5	9.4	44.4
C2		336			15		7.6													8	0.8			1		.2	62.8	3.1	32.2	.7	216	0.0119	2.0	89.4	40.0
Tails		578																								.7	.6	12.6		4.0	3.5		11.2	1.0	
VI C1	1000	84	10	8	10	1/3.5	8.9	8	0.8							4	0.4							2		38.6	28.6	2.9	149	32.4	24.0	0.0138	90.0	10.2	46.4
C2		346			15		7.8													8	0.8			1		.7	59.5	2.4	28.3	2.4	206.0	0.0140	6.7	87.4	47.2
Tails		577																								.2	1.0	12.9		1.2	6.0		3.3	2.5	
VII C1	1000	88	10	8	10	1/3.5	9.1	8	0.8							7	0.7							2		37.2	21.8	4.1	149	32.8	19.2	0.0144	91.1	8.2	48.4
C2		317			15		8.0													8	0.8			1		.5	56.0	3.1	29.7	1.7	178.0	0.0104	4.7	75.4	35.0
Tails		596																								.2	6.5	12.3		1.2	38.6		3.3	16.3	
VIII C1	1000	136	10	8	10	1/3.5	8.4	8	0.8							3	0.3							1		22.6	38.8	4.8	97.5	30.8	53.8	0.0146	85.6	22.8	49.2
C2		302			15		6.7					1	1											1		.53	59.6	1.7	24.9	1.6	180.0	0.0083	4.4	76.4	27.9
Tails		568																								.5	.5	12.2		2.8	2.9		7.8	1.2	
																																0.0594			
IX Pb c	2000	149	10	6	8	1/6	8.4	16	0.8	40	2.0	NaCN cc 3.0	0.15			8	0.4							3		39.9	27.3	2.9	159	59.5	40.7	0.0261	82.7	8.62	43.9
Zn c		501			12	1/4	7.3							5	0.25					17	0.85	1	1.0	2		.2	66.8	.6	65.4	1.0	335.0	0.0361	1.4	70.8	60.7
Pb M		118																								6.9	8.9	5.1		8.2	10.5		11.4	2.22	
Zn M		144																								.8	53.1	3.6		1.2	76.5		1.7	16.2	
Tails		1188																								Low	.9	12.5		Low	10.3		—	2.18	
X Pb c	2000	74	10	6	8	1/6	8.4	16	0.8	40	2.0	3	0.15	15	0.75	8	0.4							3		68.9	8.5	1.7	139	51	15.6	0.0113	70.8	2.88	19.0
Zn c		647			12	1/4	7.2													20	1.0	1/2	0.5	2		2.1	63.6	1.2	29.0	13.6	412	0.0207	18.9	87.3	34.9
Pb M		45																								7.0	35.5	4.0		3.2	16.0		4.5	3.39	
Zn M		58																								2.1	22.2	5.3		1.2	12.9		1.7	2.73	
Tails		1174																								Low	2.8	13.4		Low	32.9		—	6.97	
XI Pb c	2000	74	15	6	8	1/6	8.4	16	0.8	40	2.0	3	0.15	15	0.75	8	0.4							3		68.4	6.5	1.4	146	50.5	4.8	0.0119	70.2	1.02	20.1
Zn c		390			12	1/4	7.3							5	0.25					20	1.0	1	1.0	4		.4	64.5	1.12	45.7	1.6	252	0.0196	2.2	53.4	33.0
Pb M		29																								74.8	30.4	7.35	554	4.3	8.8	0.0177	6.0	1.87	29.8
Zn M		65																								.5	42.8	7.04	18.5	.3	28.5	0.0013	.4	6.03	2.19
Tails		1430																								1.1	11.1			15.7	159.		21.8	33.7	
XII Pb c	2000	83	10	6	8	1/6		16	0.8	40	2.0	3	0.15	15	0.75	8	0.4							3		62.6	6.3	1.3	126	52	5.2	0.0126	72.2	1.1	21.2
Zn c		610			12	1/4								5	0.25					20	1.0	1	1.0	3		.6	64.8	1.5	34.6	3.7	407.	0.0232	5.1	86.3	39.1
Pb M		58																								16.5	39.7	6.9		9.6	23.1		13.3	4.9	
Zn M		61																								4.7	23.5	7.5		2.9	14.3		4.0	3.03	
Tails		1171																								Low	2.7			Low	31.7		—	6.72	
XIII Pb c	2000	85	10	6	8	1/6		16	0.8	40	2.0	3	0.15	20	1.0	10	0.5							3		65.7	5.2	1.1	113	55.8	4.4	0.0124	77.5	0.93	20.9
Zn c		627			12	1/4								5	0.25					22	1.1	1	1.0	3		.5	65.4	1.4	34.6	3.1	422.	0.0257	4.3	88.4	40.2
Pb M		63																								15.8	35.2	7.8		9.9	22.2		13.8	4.71	
Zn M		62																								4.5	23.8	6.9		2.8	14.8		3.9	3.14	
Tails		1163																								Low	1.			Low	11.6		—	2.46	