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PRELIMINARY METALLURGICAL TESTING OF FIREWEED PROPERTY SAMPLES

Prepared for:

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Attention: Robert Holland

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1.0 INTRODUCTION

Preliminary metallurgical testwork has been conducted on samples from Canadian-United Minerals' Fireweed property at the request of Mr. Robert Holland. Three composites of the lead-zinc-silver ore were provided for laboratory investigation. The laboratory study focused on the flotation characteristics of the ore and included some minor optimization of a differential flotation procedure that produced both a lead concentrate and a zinc concentrate.

Only four flotation tests were conducted. The information provided by these tests is insufficient to make definite conclusions regarding flowsheet design. Instead, the four tests provide a significant data base on the general floatability of the ore and suggest several avenues that should be investigated in future metallurgical evaluations of the property.

Throughout the testwork, communications were maintained with Mr. Robert Holland of Canadian-United Minerals. All test results were discussed before planning and proceeding with the next phase of the investigation.

2.0 SUMMARY

2.1 Flotation

Three flotation tests were conducted on composite FW88-26. The tests started from a "standard" procedure for lead-zinc differential flotation and evolved slightly with each new test. Some of the procedural changes included extra collecting reagents in bulk flotation, regrinding of the bulk concentrates, and extra depressing reagents in cleaning flotation. The third test (F3), incorporating all these modifications, yielded the best results:

Final (2nd Cleaner) Lead Concentrate:
- recovered 87.8% of the lead and 64.5% of the silver
- concentrate assayed 48.0% Pb plus 17.7 oz/ton Ag

Final (2nd Cleaner) Zinc Concentrate:
- recovered 70.8% of the zinc plus a further 12.5% of the silver
- concentrate assayed 50.1% Zn plus 3.7 oz/ton Ag

This test also involved a finer primary grind than the first two tests. It is believed that this finer primary grind (78.3% minus 200 mesh) is partly responsible for the relatively high recoveries of this test.

It should be noted that the recirculation of cleaner tails in a continuous circuit would be expected to increase lead and zinc recoveries to their respective concentrates.

While the zinc concentrate is acceptable from a marketing point of view, the lead concentrate requires additional upgrading.

A fourth flotation test was conducted on a separate ore composite (composite FW88-25). This test used almost the identical flotation procedure of previous tests and was intended to investigate whether a common flowsheet for all ore types was a realistic goal for this property. The results for this test show that despite significant head grade variation between tests, the ore behaved similarly.

2.2 Microscopy

The first attempt at cleaning the bulk lead and zinc concentrates (F2) did not work very effectively. Polished sections of each concentrate were made and examined microscopically. This analysis indicated one of the main reasons for the poor cleaning efficiency to be the presence of middling particles; the degree of liberation was insufficient to allow proper cleaning. This observation prompted the decision to regrind the bulk concentrates before cleaning. When regrinding was incorporated, cleaning flotation improved dramatically.

A second phenomenon observed during microscopic examination of the flotation concentrates was that the silver values occurred (almost exclusively) as inclusions within pyrite grains. Not all the pyrite grains had this silver content and those that did had it in varying amounts. This observation has metallurgical consequences: any attempts to depress pyrite will involve a loss in silver recovery. Finer primary grinding and/or regrinding would help alleviate this problem only to a minor degree.

It should be noted that polished sections were examined only for lead and zinc concentrates from Test F2, composite FW88-26. Microscopy is a very powerful tool in the development of any metallurgical flowsheet. We recommend microscopic analysis of the following flotation products as a prelude to further laboratory investigation:

1. Final (2nd Cleaner) Lead and Zinc Concentrates from Test F3
   - to determine what minerals account for the estimated 36% and 23% gangue content (neither galena nor sphalerite).
   - to determine why 7.6% of the zinc is still floating with the lead.
   - to assess the liberation of the silver values.
2. Final (2nd Cleaner) Lead and Zinc Concentrates from Test F4
   - to compare microscopic observations between composite FW88-26 and composite FW88-25.

2.3 Recommendations

Further laboratory investigation of the Fireweed property should include a flotation test that attempts to produce a bulk pyrite concentrate in addition to lead and zinc concentrates. Theoretically, this will allow more aggressive flotation techniques in the lead and zinc stages (without fear of silver loss) as silver will be recovered by subsequent pyrite flotation. The bulk pyrite concentrate can be upgraded by cleaning and can then be reground and cyanided to recover the silver values.

It is important to establish and compare the floatability of each ore composite. Significantly different behaviour between composites is problematic and can lead to major changes in the design of the entire project. We therefore recommend individual metallurgical testing of all major variations in ore type before proceeding with detailed optimization.

Finally, a blend of all ore types should be prepared that is representative of the average characteristics of the deposit. Detailed metallurgical studies should be conducted on this material. This detailed study should <u>not</u> be conducted until all previously mentioned factors have been analyzed and the results warrant further investigation.

3.0 DISCUSSION

3.1 Sample Description

A series of samples from the Fireweed property was received on February 24, 1989 (see Appendix I). Instructions from the client specified the preparation of three composites from the samples received:

1. Composite Sample FW 88-08 - (Sample No's 3951-3954)
   Client description: relatively high grade Ag, low grade Pb-Zn from eastern end of mineralized horizon.

2. Composite Sample FW 88-25 - (Sample No's 17549-17559)
   Client description: moderate grade Ag, low grade Pb-Zn from middle portion of the mineralized horizon.

3. Composite Sample FW 88-26 - (Sample No's 17599-17605)
   Client description: low grade Ag, higher grade Pb-Zn from the deeper, eastern area of the mineralized horizon.

A summary of the assayed and calculated heads of each composite is presented in Table 3.1. The grades expected by the client are presented for comparison.

Table 3.1
HEAD ASSAY SUMMARY

Test No.	Composite	Ag oz/ton	Pb %	Zn %
Client Assay	FW88-08	19.070	1.02	2.14
Client Assay	FW88-25	9.240	0.77	1.11
BD&A Assay	FW88-25	8.832	0.56	1.24
F4	FW88-25	8.008	0.45	1.15
Client Assay	FW88-26	1.380	3.21	3.27
BD&A Assay	FW88-26	1.110	2.36	2.80
F1	FW88-26	1.068	2.30	3.07
F2	FW88-26	1.212	2.41	2.90
F3	FW88-26	1.243	2.25	2.75

The specific gravities of the composites were also calculated.

These are as follows:

Composite	Specific Gravity (g/cm^3)
FW88-08	2.78
FW88-25	2.83
FW88-26	2.88

A further point should be mentioned here in a general description of the samples received for this investigation. The two ore composites that were actually tested in the lab (FW88-25 and FW88-26) had high work indices. The work indices were not formally calculated, but experience with the BD&A grinding mill indicates them to be in the range of 17 to 20. These values classify the ore as being relatively hard.

3.2 Bulk Flotation

Four flotation tests were conducted in this investigation. The first three tests, all on composite FW88-26, were intended to assess the floatability of the ore in general and to optimize the flotation procedure as much as possible. This composite was chosen for the initial testwork because it was felt that its relatively high Pb and Zn content would facilitate flotation testwork and the subsequent interpretation of results. The fourth test was conducted on composite FW88-25 to compare the floatability of the two composites. A summary of the rougher flotation results is presented in Table 3.2

Before proceeding with a discussion of results, it should be noted that past experience with other lead-zinc-silver ores has shown that if often requires upwards of 20 flotation tests before definite patterns emerge regarding "average" results and fine tuning of flotation procedures. The four tests conducted in this study should not be over-analyzed. There are several specific points that can be concluded, but for the most part the tests provide some general flotation information and suggest avenues for future investigation.

All flotation tests were based on a fairly standard lead-zinc flotation procedure:

1. Reagents added to primary grind
   - Soda Ash (for pH control)
   - Sodium Cyanide (for depression of iron sulfides and sphalerite)
   - Zinc Sulphate (for depression of sphalerite)

2. Two minutes conditioning in flotation cell
   - Approximately 30% solids
   - pH maintained at 9.2 (with Soda Ash)
   - Add A343 (collector)
   - Add A242 (promoter)

3. Lead Rougher Flotation
   - DF250 (frother)
   - Skim froth until barren

4. Two Minutes conditioning in flotation cell
   - pH maintained at 10.5 (with Lime - high pH to depress pyrite)
   - Add $CuSO_4$ (to activate sphalerite)
   - Add A343 (collector for newly activated zinc minerals)

5. Zinc rougher flotation
   - DF1012 (frother - stronger than DF250)
   - Skim froth until barren

Only small procedural changes were made between tests:

Test F1: Base procedure

Test F2:
1. Extra A343 in lead rougher conditioning
2. Extra A242 in lead rougher conditioning
3. Longer lead rougher flotation time

Test F3: 1. Slightly finer primary grind
2. Shorter lead rougher flotation time

Test F4: 1. Different Composite
2. Much finer primary grind

An examination of the results summary in Table 3.2 indicates that most of the procedural changes did not produce a definite effect on the test results.

The extra reagents added in Test F2 were used in the hope of increasing the lead content of the lead rougher concentrate. In a plant situation, any lead that does not report to the lead concentrate is lost; either reporting as a loss to the tails or as a loss <u>and</u> a smelter penalty to the zinc concentrate. Therefore, despite seeing no noticeable improvement in results for F2, the extra reagent addition was maintained in subsequent tests.

The changing flotation times for the lead rougher were a result of "on-line" decisions. In Test F2 the froth did not appear to go barren until twice the flotation time of F1. In Test F3 it went barren sooner than F2 but later than F1. In any event, the results were not noticeably affected by changing the flotation time. This is as expected since the very small amount of slow floating lead-silver minerals recovered at the end of the test would not be significant compared to the rest of the test.

The only procedural improvement was the finer primary grinding of Test F3. It resulted in significant improvements in the Ag and Pb recoveries to the lead rougher concentrate. The zinc rougher concentrate was also improved. Test F3 produced the most promising results of the first three tests. The lead rougher concentrate contained 73.2% of the silver, 92.6% of the lead and assayed 9.210 oz/ton Ag and 23.19% Pb. The zinc concentrate contained 83.9% of the zinc and a further 22.8% of the silver. The zinc concentrate assayed 15.72% Zn and it had the lowest contamination of lead (0.58% Pb) of the first three tests.

Table 3.2
Rougher Flotation Summary

TEST NO	COMPOSITE	GRIND % -200#	PRODUCT	PRODUCT WEIGHT %	ASSAYS Ag oz/ton	ASSAYS Pb %	ASSAYS Zn %	DISTRIBUTION Ag %	DISTRIBUTION Pb %	DISTRIBUTION Zn %
F1	FW88-26	69.1	Pb Rougher Conc	7.94	9.443	25.20	3.88	70.2	86.9	10.0
			Zn Rougher Conc	16.22	1.736	1.12	16.48	26.4	7.9	87.0
			Tails	75.83	0.048	0.16	0.12	3.4	5.3	3.0
F2	FW88-26	70.2	Pb Rougher Conc	9.75	8.478	19.95	4.14	65.9	85.7	14.5
			Zn Rougher Conc	13.92	2.253	1.22	16.33	25.2	7.6	82.7
			Tails	76.43	0.145	0.20	0.10	8.9	6.8	2.8
F3	FW88-26	78.3	Pb Rougher Conc	9.63	9.210	23.19	4.29	73.2	92.6	14.3
			Zn Rougher Conc	15.46	1.786	0.58	15.72	22.8	3.7	83.9
			Tails	74.91	0.065	0.12	0.07	4.0	3.7	1.9
F4	FW88-25	95.3	Pb Rougher Conc	11.50	52.987	3.36	2.30	76.4	86.8	23.2
			Zn Rougher Conc	9.20	12.564	0.30	9.30	14.5	6.1	74.8
			Tails	79.22	0.923	0.04	0.03	9.1	7.1	2.1

The fourth test, on composite FW88-25, corroborated the general findings of the first three tests. Overall recovery and the distribution of metal values between lead and zinc concentrates were similar. As this composite had a different assay head, the resulting flotation concentrates have significantly different grades. With only one test on this composite, it is not possible to determine if the extremely fine primary grind used in this test was necessary.

3.3 Cleaner Flotation

Once the results of Test F1 were compiled, it was apparent that rougher flotation had been reasonably successful in terms of total metal recovery and it now remained to be seen if the concentrates could be upgraded by cleaning to produce marketable flotation concentrates. Cleaner flotation was conducted on all subsequent tests. The results are summarized in Table 3.3.

The basic cleaning procedure of Test F2 involved:

1. <u>1st Lead Cleaner</u>
   - NaCN (for depression of iron sulfides and sphalerite)
   - $ZnSO_4$ (for depression of sphalerite)

2. <u>2nd Lead Cleaner</u>
   - No reagents
   - Float until barren to produce final Pb concentrate

3. <u>1st Zinc Cleaner</u>
   - Lime to pH = 11.0 (for depression of pyrite)

4. <u>2nd Zinc Cleaner</u>
   - Lime to pH = 11.5 (for further depression of pyrite)
   - Float until barren to produce final Zn concentrate

Table 3.3
Cleaner Flotation Summary

TEST NO	COMPOSITE	GRIND % -200#	PRODUCT	PRODUCT WEIGHT %	ASSAYS Ag oz/ton	Pb %	Zn %	DISTRIBUTION Ag %	Pb %	Zn %
F2	FW88-26	70.2	Pb 2nd Cleaner Conc	6.51	11.682	28.80	3.68	61.2	83.4	8.7
		70.2	Zn 2nd Cleaner Conc	7.51	3.355	1.56	29.40	20.3	5.2	80.3
F3	FW88-26	98.1	Pb 2nd Cleaner Conc	4.41	17.699	48.00	5.00	64.5	87.8	7.6
		98.3	Zn 2nd Cleaner Conc	4.09	3.688	1.04	50.13	12.5	1.8	70.8
F4	FW88-25	~99	Pb 2nd Cleaner Conc	1.64	305.621	22.00	7.24	62.4	80.5	10.3
		~99	Zn 2nd Cleaner Conc	1.70	44.404	0.80	44.80	9.4	3.1	66.4

This procedure experienced some success. Final lead concentrate was upgraded to 11.682 oz/ton Ag and 28.80 % Pb (from 8.478 oz/ton Ag and 19.95 % Pb in the bulk concentrate). The upgrading came with only minor losses in Ag and Pb recovery. Some zinc impurities were cleaned out, but the final lead concentrate still assayed 3.68 % Zn. The zinc concentrate assayed 29.40 % Zn plus 3.355 oz/ton Ag (upgraded from 16.33 % Zn and 2.253 oz/ton Ag in the bulk concentrate).

Despite some minor improvements in grade, both the lead concentrate and the zinc concentrate of Test F2 were far from being marketable flotation concentrates. The problems with cleaning the concentrates were not believed to be the fault of the procedure itself, which was fairly standard. Polished sections were made of the lead and zinc concentrates from Test F2. These were examined microscopically and provided some important information:

Pb Concentrate

- Lots of gangue particles (non-sulfides)
- Lots of middlings: Galena/gangue, Galena/sphalerite
- Chalcopyrite occluded in sphalerite
- Silver mineral (argentite) occurrence is almost exclusively as inclusions in pyrite
- Silver minerals make up an average of less than 20% of those pyrite grains in which they occur (visual volume estimate)

Zn Concentrate

- Lots of liberated pyrite
- Less gangue than Pb conc
- Some middlings : sphalerite/gangue
- Pyrite contains significantly less silver minerals than pyrite in Pb conc
- Chalcopyrite occluded in sphalerite

To summarize, it appeared that both concentrates were too coarse for effective cleaning. Regrinding was proposed and this would be combined with increased reagent additions to make up for the extra mineral surface created. Of special interest was the observation that the silver minerals occurred (almost exclusively) as inclusions within pyrite grains. Obviously, any attempts to depress pyrite in either of the concentrates would lower the recovery of silver. Regrinding would likely help alleviate this problem somewhat.

Test F3 incorporated regrinding of both the rougher concentrates before cleaning. For lead cleaning the additions of NaCN and $ZnSO_4$ were doubled and a small amount of collector (A343) was added to the first cleaning stage. The second lead cleaner still contained no reagents. In addition to regrinding, zinc cleaning added 0.02 lb/ton NaCN to both cleaning stages.

The results for F3 show a definite improvement. The lead concentrate was upgraded to 17.699 oz/ton Ag and 48.00 % Pb (from 9.210 oz/ton Ag and 23.19 % Pb in the rougher concentrate). The zinc concentrate was upgraded to 50.13 % Zn (from 15.72 % Zn in the rougher concentrate).

A closer look at the details of Test F3 (Appendix II) shows that the most effective cleaning took place in the first cleaner stages. In the second stages both the lead and zinc concentrate lost significant amounts of valuable metal recovery (compared to the amount and weight % of the total material removed). Assays of the final concentrates indicate that there is still a significant percentage of material present that is neither lead minerals nor zinc minerals; approximately 36% in the Pb concentrate and 23% in the Zn concentrate (based on the assumption that all Pb values are galena and all Zn values are sphalerite). Some (or all) of this extra material is pyrite, and it has already been established that it is within the pyrite that the silver values occur. Therefore, there will likely be a trade-off between upgrading the concentrates further and maximizing total recovery of silver. Microscopic examination of the concentrates from this test would help to identify the mineral components and the metallurgical problems they may pose.

A possible alternative to the current flowsheet that may solve these problems is to float a lead concentrate then a zinc concentrate as done previously, and then add a bulk pyrite float. If this were done, it would be possible to clean the lead and zinc concentrates more aggressively as tailings would eventually be recycled to pyrite flotation. The resulting bulk pyrite concentrate could be upgraded by flotation and/or cyanided to recover the silver values. A few lab-scale tests will quickly indicate if this procedure is viable.

The final flotation test (F4) was conducted on composite FW88-25. This composite was higher in silver values while lower in lead and zinc. The results of cleaning indicate a significant amount of upgrading was accomplished in both the lead and zinc concentrates. The detailed results (Appendix II) show both cleaning stages participated fairly equally in the upgrading. This fact suggests that a third cleaning stage could probably be added before reaching the recovery/grade trade-off of previous tests. Nonetheless, despite significant head grade differences, this composite behaved very similar to the previous one and it seems likely that a common flowsheet for all composites could be developed. A microscopic examination of these products would also be informative.

APPENDIX 1

Samples Received

Samples Received

COMPOSITE	CLIENT I.D.	GROSS WEIGHT (g)
FW88-08	3951	647
	3952	1006
	3953	986
	3954	1094
FW88-25	17549	1344
	17550	1188
	17551	1476
	17552	1280
	17553	1322
	17554	1911
	17555	1661
	17556	1596
	17557	1621
	17558	1297
	17559	1825
FW88-26	17599	1148
	17600	1458
	17601	1838
	17602	1369
	17603	972
	17604	1327
	17605	1141

APPENDIX 2

Flotation Results

TESTWORK PROCEDURE

Test No. M89-114 F1 **Date February 27, 1989**

COMPOSITE FW88-26

STAGE	TIME (min)	ADDITIONS lb/ton	ADDITIONS REAGENT
Grind	26	0.58 0.10 0.50	Soda Ash NaCN $ZnSO_4$
			pH = 7.4
Condition	2	0.05 0.012 1.02	A 343 A 242 Soda Ash to pH = 9.2
Lead Rougher	8		DF 250
Condition	2	1.57 0.05 0.10	Lime to pH = 10.5 $CuSO_4$ A-343
Zinc Rougher	15		DF 1012

TEST NUMBER: M89-114 F1   Composite: FW88-26

PRODUCT	WEIGHT GMS	WEIGHT %	ASSAYS Ag oz/ton	Pb %	Zn %	% DIST Ag %	Pb %	Zn %
Pb Rougher Conc	156.5	7.94	9.443	25.20	3.88	70.22	86.85	10.03
Zn Rougher Conc	319.7	16.22	1.736	1.12	16.48	26.37	7.89	87.01
Tails	1494.4	75.83	0.048	0.16	0.12	3.41	5.27	2.96
CALC HEAD	1970.6	100.0	1.068	2.304	3.073	100.00	100.00	100.00

SIZE DISTRIBUTION

SAMPLE NO. M89-114 F1 HEAD

Ground 26 min at 65% solids

Size Fraction (mesh)	Individual Percentage Retained %	Cumulative Percentage Passing %
+ 65		100.0
- 65 + 100	0.8	99.2
- 100 + 150	8.4	90.8
- 150 + 200	21.7	69.1
- 200 + 325	22.6	46.5
- 325 + 400	46.5	

TESTWORK PROCEDURE

Test No. M89-114 F2 Date March 7, 1989

COMPOSITE FW88-26

STAGE	TIME (min)	ADDITIONS	
		lb/ton	REAGENT
Grind	26	0.58 0.01 0.50	Soda Ash NaCN $ZnSO_4$
Condition	2	0.73 0.06 0.025	Soda Ash to pH = 9.2 A 343 A 242
Lead Rougher	16		DF 250
Condition	2	1.24 0.50 0.10	Lime to pH = 10.5 $CuSO_4$ A 343
Zn Rougher	15		DF 1012
1st Lead Cleaner	10	0.02 0.10	NaCN $ZnSO_4$
2nd Lead Cleaner	5		
1st Zinc Cleaner	6	0.38	Lime to pH = 11.0
2nd Zinc Cleaner	5	1.09	Lime to Ph = 11.5

TEST NUMBER: M89-114 F2   Composite: FW88-26

PRODUCT	WEIGHT GMS	WEIGHT %	ASSAYS Ag oz/ton	Pb %	Zn %	% DIST Ag %	Pb %	Zn %
Pb 2nd Cleaner Conc	128.8	6.51	11.682	28.80	3.68	61.21	83.43	8.72
Pb 2nd Cleaner Tail	11.7	0.59	3.045	3.60	5.68	1.45	0.95	1.22
Pb 1ST CLEANER CONC	140.5	7.1	10.963	26.70	3.85	62.66	84.38	9.94
Pb 1st Cleaner Tail	50.4	2.55	1.553	1.12	4.96	3.18	1.27	4.60
Pb ROUGHER CONC	190.9	9.7	8.478	19.95	4.14	65.85	85.65	14.53
Zn 2nd Cleaner Conc	148.5	7.51	3.355	1.56	29.40	20.27	5.21	80.28
Zn 2nd Cleaner Tail	34.1	1.72	1.841	1.20	1.64	2.55	0.92	1.03
Zn 1ST CLEANER CONC	182.6	9.2	3.072	1.49	24.22	22.82	6.13	81.31
Zn 1st Cleaner Tail	92.7	4.69	0.640	0.68	0.81	2.41	1.42	1.38
Zn ROUGHER CONC	275.3	13.9	2.253	1.22	16.33	25.24	7.55	82.69
Tail	1511.7	76.43	0.145	0.20	0.10	8.92	6.80	2.78
CALC HEAD	1977.9	100.0	1.243	2.248	2.750	100.00	100.00	100.00

SIZE DISTRIBUTION

SAMPLE NO. M89-114 F2 HEAD

Ground 26 min at 65% solids

Size Fraction (mesh)	Individual Percentage Retained %	Cumulative Percentage Passing %
+ 65		100.0
- 65 + 100	1.1	98.9
- 100 + 150	7.4	91.5
- 150 + 200	21.3	70.2
- 200 + 325	23.4	46.8
- 325 + 400	46.8	

TESTWORK PROCEDURE

Test No. M89-114 F3 Date March 22, 1989

COMPOSITE FW88-26

STAGE	TIME (min)	ADDITIONS	
		lb/ton	REAGENT
Grind	29.5	0.58	Soda Ash
		0.10	NaCN
		0.50	$ZnSO^4$
Condition	2	1.60	Soda Ash to pH = 9.2
		0.60	A 343
		0.025	A 242
Lead Rougher	11		DF 250
Condition	2	2.18	Lime to pH = 10.5
		0.50	$CuSO_4$
		0.10	A 343
Zinc Rougher	15		DF 1012
Regrind Pb Rougher Conc	20		
1st Lead Cleaner	7	0.01	A 343
		0.04	NaCN
		0.20	$ZnSO_4$
2nd Lead Cleaner	5		
Regrind Zinc Rougher Conc	20		
1st Zinc Cleaner	7	0.037	Lime to pH = 11.0
		0.02	NaCN
2nd Zinc Cleaner	5	0.24	Lime to pH = 11.5
		0.02	NaCN

TEST NUMBER: M89-114 F3 Composite: FW88-26								
			ASSAYS			% DIST		
PRODUCT	WEIGHT GMS	WEIGHT %	Ag oz/ton	Pb %	Zn %	Ag %	Pb %	Zn %
Pb 2nd Cleaner Conc	87.4	4.41	17.699	48.00	5.00	64.48	87.79	7.62
Pb 2nd Cleaner Tail	11.1	0.56	4.394	5.20	9.12	2.03	1.21	1.76
Pb 1ST CLEANER CONC	98.5	5.0	16.200	43.18	5.46	66.51	89.00	9.38
Pb 1st Cleaner Tail	92.2	4.66	1.742	1.84	3.04	6.69	3.55	4.89
Pb ROUGHER CONC	190.7	9.6	9.210	23.19	4.29	73.20	92.55	14.27
Zn 2nd Cleaner Conc	81.0	4.09	3.688	1.04	50.13	12.45	1.76	70.75
Zn 2nd Cleaner Tail	36.5	1.85	2.490	0.92	14.40	3.80	0.70	9.18
Zn 1ST CLEANER CONC	117.5	5.9	3.315	1.00	39.01	16.24	2.47	79.93
Zn 1st Cleaner Tail	188.5	9.52	0.832	0.32	1.20	6.54	1.26	3.94
Zn ROUGHER CONC	306.0	15.5	1.786	0.58	15.72	22.78	3.73	83.87
Tail	1483.3	74.91	0.065	0.12	0.07	4.02	3.72	1.86
CALC HEAD	1980.0	100.0	1.212	2.413	2.897	100.00	100.00	100.00

SIZE DISTRIBUTION

SAMPLE NO. M89-114 F3 HEAD

Ground 29.5 min at 65% solids

Size Fraction (mesh)	Individual Percentage Retained %	Cumulative Percentage Passing %
+ 65	0.3	99.7
- 65 + 100	0.6	99.1
- 100 + 150	3.3	95.8
- 150 + 200	17.5	78.3
- 200 + 325	26.3	52.0
- 325 + 400	52.0	

SIZE DISTRIBUTION

SAMPLE NO. M89-114 F3 Pb CONC.

ReGround 20 min at 70% solids

Size Fraction (mesh)	Individual Percentage Retained %	Cumulative Percentage Passing %
+ 65		
- 65 + 100		100.0
- 100 + 150	0.2	99.8
- 150 + 200	1.7	98.1
- 200 + 325	15.6	82.5
- 325 + 400	82.5	

SIZE DISTRIBUTION

SAMPLE NO. M89-114 F3 ZN CONC.

ReGround 20 min at 70% solids

Size Fraction (mesh)	Individual Percentage Retained %	Cumulative Percentage Passing %
+ 65		
- 65 + 100		100.0
- 100 + 150	0.3	99.7
- 150 + 200	1.4	98.3
- 200 + 325	14.4	83.9
- 325 + 400	83.9	

TESTWORK PROCEDURE

Test No. M89-114 F4 **Date April 5, 1989**

COMPOSITE FW88-25

STAGE	TIME (min)	ADDITIONS	
		lb/ton	REAGENT
Grind	40	0.58	Soda Ash
		0.10	NaCN
		0.50	$ZnSO_4$
Condition	2	0.82	Soda Ash to Ph = 9.2
		0.06	A 343
		0.025	A 242
Lead Rougher	9		DF 250
Condition	2	2.37	Lime to pH = 10.5
		0.50	$CuSO_4$
		0.10	A 343
Zn Rougher	11		DF 1012
Regrind Pb Rougher Conc	20		
1st Lead Cleaner	10	0.04	NaCN
		0.20	$ZnSO_4$
		0.05	A 343
2nd Lead Cleaner		0.05	NaCN
Regrind Zn Rougher Conc	20		
1st Zinc Cleaner	6	0.31	Lime to pH = 11.0
		0.02	NaCN
		0.02	A 343
2nd Zinc Cleaner	5	0.86	Lime to pH = 11.5
		0.02	NaCN

TEST NUMBER: M89-114 F4 Composite: FW88-25

PRODUCT	WEIGHT GMS	WEIGHT %	ASSAYS Ag oz/ton	Pb %	Zn %	% DIST Ag %	Pb %	Zn %
Pb 2nd Cleaner Conc	32.2	1.64	305.621	22.00	7.24	62.43	80.54	10.31
Pb 2nd Cleaner Tail	25.1	1.28	50.354	0.84	4.80	8.02	2.40	5.33
Pb 1ST CLEANER CONC	57.3	2.9	193.802	12.73	6.17	70.45	82.93	15.64
Pb 1st Cleaner Tail	169.9	8.63	5.496	0.20	1.00	5.92	3.86	7.52
Pb ROUGHER CONC	227.2	11.5	52.987	3.36	2.30	76.38	86.80	23.16
Zn 2nd Cleaner Conc	33.5	1.70	44.404	0.80	44.80	9.44	3.05	66.39
Zn 2nd Cleaner Tail	27.1	1.38	12.613	0.28	5.12	2.17	0.86	6.14
Zn 1ST CLEANER CONC	60.6	3.1	30.187	0.57	27.06	11.61	3.91	72.52
Zn 1st Cleaner Tail	121.2	6.16	3.753	0.16	0.42	2.89	2.20	2.25
Zn ROUGHER CONC	181.3	9.2	12.364	0.30	9.30	14.49	6.11	74.77
Tail	1559.3	79.22	0.923	0.04	0.03	9.13	7.09	2.07
CALC HEAD	1968.3	100.0	8.008	0.447	1.149	100.00	100.00	100.00

SIZE DISTRIBUTION

SAMPLE NO. M89-114 F4 HEAD

Ground 40 min at 65% solids

Size Fraction (mesh)	Individual Percentage Retained %	Cumulative Percentage Passing %
+ 65		100.0
- 65 + 100	0.4	99.6
- 100 + 150	0.9	98.7
- 150 + 200	3.4	95.3
- 200 + 325	25.9	69.4
- 325 + 400	69.4	