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ESSEX

ESSEX INTERNATIONAL, INC.
1704 WEST GRANT RD., TUCSON, ARIZONA 85705
PHONE (602) 624-7421

January 22, 1971

Mr. Lynn Burr, President
Associated Smelters
324 S. Third St., Suite 15
Las Vegas, Nevada 89101

Dear Mr. Burr:

We are sending you 3 copies of the report "A Technical Evaluation of the Continuous Copper Leaching and Precipitation Plant at Hurricane, Utah Operated By Dixie Basin Smelters."

Our economic conclusions are set forth in an attachment to this letter, together with our analysis of the costs of operation.

The unfavorable cost picture is largely due to factors associated with the ore being treated, not with the operation of the plant itself. It is obvious from the Table of Costs that cost items such as mining, hauling, crushing, and acid result in very large costs compared to the value of the ore being treated. It should be emphasized that the Reef ore that was treated during the 17-day run is about double the usually-desired acid consumption. The actual performance of the leaching plant itself was reasonable, although improvement is desirable in solution recovery and materials handling. An increase in percent operating time plus increased throughput would improve the cost picture but the essential problem lies with the ore.

Very truly yours,

ESSEX INTERNATIONAL, INC.

Clement K. Chase (TD)

Clement K. Chase
Operations Manager

CKC:td
attachments

TABLE OF COSTS

Breakdown of Costs for the Continuous
Copper Leaching and Precipitation Plant at
Hurricane, Utah

COST ITEM	Cost Rate, $/Unit	TOTAL COST: Dollars for 17-Day Operation	TOTAL COST: $/Ton Ore	TOTAL COST: $/lb. Cu Recovered
Direct Costs				
Mining*	$4/ton	1210.80	4.000	0.190
Hauling (*Mine-Crusher-Plant)	$3/ton	908.10	3.000	0.143
Milling	---	---	---	---
Crushing*	$1/ton	302.70	1.000	0.048
Labor (*Operating & Maintenance)	---	991.70	3.357	0.156
Acid (*Includes Hauling)	$30.75/ton	1200.48	4.064	0.189
Plate Scrap*	$47.71/ton	112.37	0.380	0.017
Power*	$0.032/kwhr	72.10	0.244	0.011
Water*	$0.24/1000 gal.	18.47	0.062	0.003
Operating** Supplies	$15/day	255.00	0.863	0.040
Maintenance Supplies**	$18/day	306.00	1.036	0.048
Total Direct Costs	---	5377.72	18.006	0.845
Indirect Costs				
General Overhead (at 12.3% of Direct Costs)***	---	661.46	2.239	0.104
Payroll Overhead (at 15% of Labor Cost)**	---	148.76	0.504	0.023
Total Indirect Costs	---	810.22	2.743	0.127
Amortization of Capital Cost	$20,000/yr	931.43	3.153	0.146
Mine Property Payments	$12,000/yr	558.96	1.892	0.088
Grand Total		$7678.33	$25.794	$1.206

COST EXPLANATION

Labor - Normal labor at $3.45/hr; overtime labor at $5.18/hr; and supervision at $5.00/hr.

Acid - As 93.9% pure sulfuric acid.

Scrap Iron - At $22.50/ton at the supplier, $13.68/ton for hauling, and $11.53/ton for preparation

SOURCE OF COST INFORMATION

* Costs furnished by Dixie Basin Smelters

** Costs estimated by C.E. Osborn

*** Overhead costs from June 8, 1970 booklet, Organization and Planning, Associated Smelters, Inc.

REPORT ON

A TECHNICAL EVALUATION OF THE CONTINUOUS COPPER LEACHING AND PRECIPITATION PLANT AT HURRICANE, UTAH OPERATED BY DIXIE BASIN SMELTERS

by

J. RANDALL BURKE

of

ESSEX INTERNATIONAL, INC.

January 20, 1971

SUMMARY

The following results were obtained during the evaluation of the Dixie Basin leach plant at Hurricane, Utah over an operating period of 17 days:

1. Overall copper recovery $\frac{\text{(Cement produced)}}{\text{(Copper in heads)}}$ 77.0%
2. Iron consumption in precipitation $\frac{\text{lb. iron}}{\text{lb. copper precipitated}}$ 0.74
3. The average grade of cement copper produced was 86.5%
4. The average treatment rate for a 9-hr per day run was 29.5 tons

In operation of the plant, the use of electric power in the cementation step appears to result in lowered scrap iron consumption. It would be preferable, however, to study any effects of the use of electric power in leaching and cementation in the laboratory where all conditions and measurements can be carefully controlled. The difficulties inherent in making these studies in an operating plant without undue interference to production are obvious. In addition, in this plant, the common practice of re-circulating solutions back to the head of the circuit makes evaluation especially difficult.

The use of the large new settling pond has improved the continuity, hence the efficiency, of the plant operation. It also appears to have improved the recovery of soluble copper but the degree of this improvement is difficult to quantify. In fact, though a strong effort was

made to get accurate data for the evaluation, few data are exact and some are perforce only estimates. However, the figures reported in the above summary are those in which reasonable confidence may be placed.

INTRODUCTION

The continuous leaching and precipitation pilot plant of Dixie Basin Smelters, Inc. at Hurricane, Utah leaches oxidized copper ores with sulfuric acid and recovers the resulting soluble copper on scrap iron. A description of the features of the plant was presented in a previous technical report. Unique features of the system include the application of electric power through electrodes to both the leach solution in the leach trough and to the pregnant solution in the cementation cells.

The performance of the plant was monitored over 17 days of operation that extended from August 18 to September 5, 1970. This was done in order to determine the overall operating performance and costs of the plant. Figure 1 presents a flowsheet of the plant as it existed at the time of the plant test.

PROCEDURES

The following procedures were employed in monitoring the performance of the plant:

1. The plant was cleaned out prior to and following the 17-day test.

2. All materials which entered and left the plant during the test were either sampled and/or accounted for, insofar as was possible.

3. The labor required to run the plant during the test was determined.

FLOWSHEET OF HURRICANE LEACHING AND PRECIPITATION PLANT

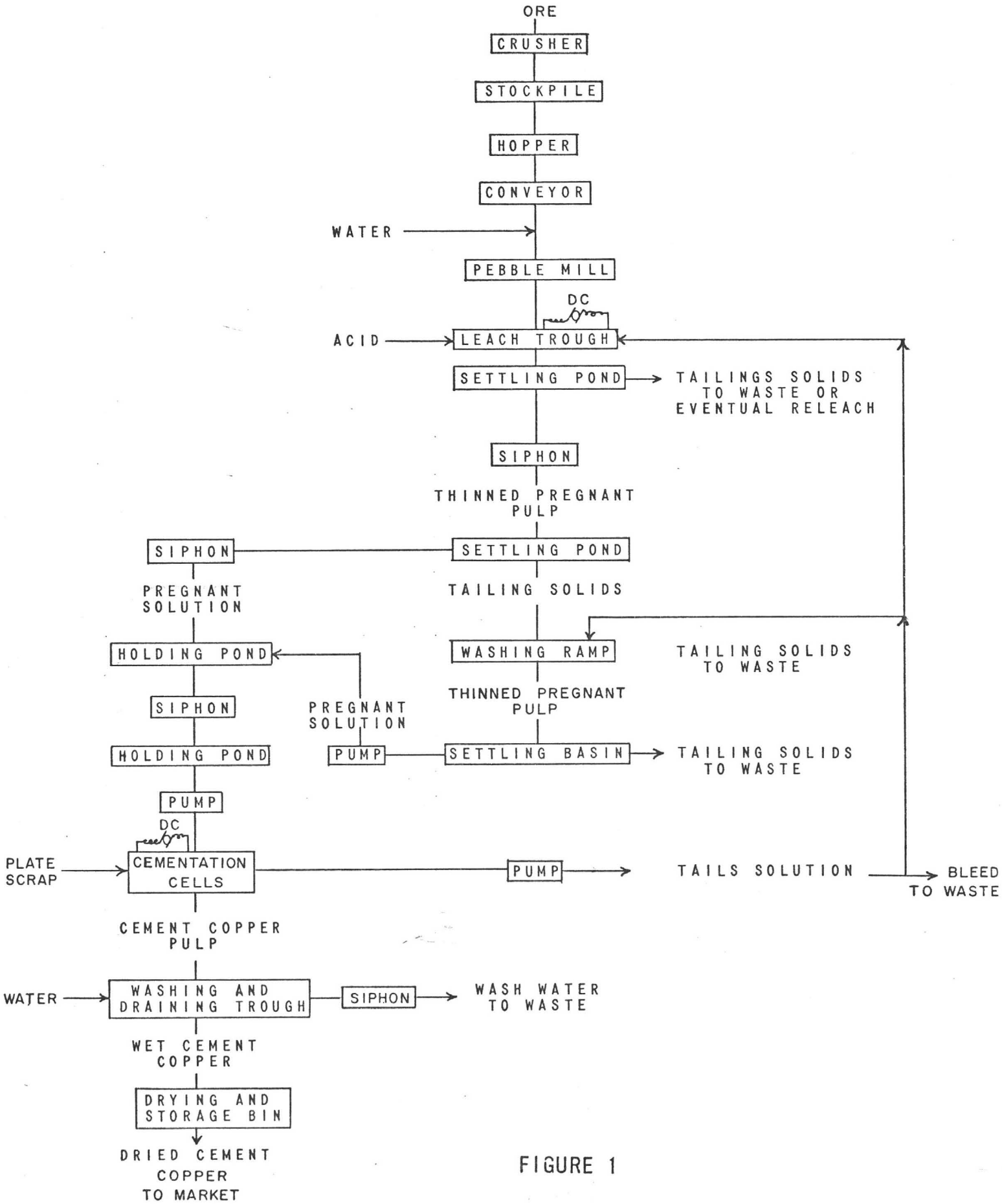

FIGURE 1

During the test, the plant was operated according to the following procedures in an effort to duplicate normal operating practice:

1. Leaching and cementation were conducted for an average of 9 hrs/day.

2. Whenever possible, leaching and cementation were run simultaneously. This was the case for 9 days following the first day of leaching. Seven additional days involved only cementation, with most of these following the 10-day leaching period.

3. Electric power was applied to the solution in both the leach trough and cementation cells.

4. Both the leaching and cementation systems were operated under dynamic conditions involving continuous solution circulation.

5. The cementation tailing solution was recirculated to the leach trough whenever enough pregnant solution had accumulated to permit continuous cementation. Fresh water was continuously added to the leach trough whenever cementation tailing solution was not being recirculated.

The following recent changes were adopted as a part of normal operating practice at the beginning of the plant test:

1. A large circular asphalt-lined settling pond was constructed and put into operation to achieve a more efficient separation of solubulized copper from the leach tails.

2. Plate scrap was adopted for use in the cementation cells in place of the wire scrap that had been used in the past. It was noted that the use of plate scrap produced a cleaner cement that contained less unconsumed metallic iron.

OPERATING CHARACTERISTICS

The ore leached in the plant during the test came from the Reef Mine near Hurricane. It consisted of a malachite-and azurite-bearing sandstone of the grade indicated in Table I. The ore head sample was taken around the surface of the cone of stockpiled plant feed at a level one third the height of the cone from the base. The moisture sample of the ore heads was taken from the interior of the same feed stockpile. Note that the average of several composite assays of the pebble mill discharge solids checks closely with the assay of the ore heads.

The following methods were employed in collecting the other samples listed in Table I:

1. A 500-ml sample of the pebble mill discharge solids was taken every 2 hours during leaching and combined with similar samples to form a daily composite.

2. A 200-ml sample of the leached pulp was taken every 2 hours during leaching and immediately filtered. The resulting solids and filtrate plus washings were combined with other similar samples into daily composites.

3. Samples of wet cement copper were taken by spatula from each lot of cement after the tailings water had been allowed to drain off. The samples from each lot were then combined to form a composite sample that was later oven-dried prior to assay. Moisture, iron, and copper assays were all determined on the same sample.

4. The sample of cement pulp was taken and treated in the manner described in the previous report, to determine grade of product.

TABLE I

The Composition of Samples Collected
During Leaching and Cementation

Sample Description	Cu %	Cu g/l	ASCu(1) %	Fe %	Fe g/l	Moisture %	SO_4 %	Acid Insol.,%
Ore Heads	1.40	---	1.39	---	---	2.4	---	---
Pebble Mill Discharge Solids*	1.39	---	---	---	---	---	---	---
Leached Pulp Solids*	0.13	---	0.11	---	---	---	---	---
Wet Cement Copper, Lot A	88.98	---	---	1.71	---	26.0	---	---
Wet Cement Copper, Lot B	89.96	---	---	2.11	---	28.3	---	---
Wet Cement Copper, Lot C	88.51	---	---	2.17	---	32.8	---	---
Wet Cement Copper, Lot D	88.36	---	---	2.23	---	32.3	---	---
Wet Cement Copper, Lot E	73.98	---	---	5.54	---	32.0	---	---
Cement Copper Pulp, Lot D**	88.95	---	---	2.10	---	---	4.98	1.21
Barren Tails Solution***	---	0.56	---	---	20.60	---	---	---
Barren Waste Solution***	---	0.54	---	---	21.43	---	---	---
Pregnant Head Solution***	---	1.39	---	---	19.82	---	---	---

*Average of several composite assays.
**Vacuum dried prior to assay.
***A composite sample.
(1) Acid-soluble copper assay: 2 hour room temp. leach with occasional agitation in 20% H_2SO_4 solution saturated with SO_2

5. A 100-ml sample of the cementation tailing solution from the last cementation run of the test was taken every 2 hours and combined with similar samples to form a composite.

6. A 100-ml sample of waste solution that was discarded near the end of the test was taken every 10 minutes and combined with similar samples to form a composite.

Additional details of how these samples were taken and treated were given in the previous report.

The operating characteristics of the plant test are presented in Table II.

DISCUSSION

The Fe/Cu ratio which was achieved in cementation during the plant test is lower than the theoretical value of 0.88 and is approximately half that normally attained in efficiently operated cementation plants. A portion of this reduction in the Fe/Cu ratio is undoubtedly due to the production of electrowon copper. The effect of this, however, is small, for the 1000 kw of power estimated to have been consumed by the cementation cells during the plant test is equivalent to the production of only 1000 lbs. of copper. Unsuspected cathode corrosion of any metallic fixtures of the cementation cells could account for the reduction of the Fe/Cu ratio, but not from a value of 1.5, for this would require the consumption of several thousand pounds of metallic fixtures. Such an effect must, therefore, be small. The reduction of the Fe/Cu ratio through the application of electric power to the cementation cells has not been adequately explained.

TABLE II

Operating Data Characteristics of the Plant Test

	Average	Test Total
Length of leaching test	8.8 hrs/run	10 days
Ore throughput rate:		
(1) Daily Run	29.54 tons/day	295.4 tons
(2) Hourly	3.36 tons/hr	
(3) 24-hr capacity (full-time operations)	81 tons/day	
Pulp density of pebble mill discharge	57.4%	
Water consumed (leaching & cementation)	7696 gals/day	76,960 gals
Power consumed (leaching & cementation)	176 kw hr/day	2253 kw hr
Acid consumed (leaching & cementation)	7004 lbs/day	73,318 lbs.
Labor expended (leaching & cementation):		
(1) Normal	---	120 hrs
(2) Overtime	---	15 hrs
(3) Supervision	---	100 hrs
Copper content of cement produced	---	6365 lbs
Average grade of cement copper	---	86.5%
Overall copper recovery = $\frac{\text{Cement produced} \times 100}{\text{Copper in heads}}$	---	77.0%
Iron consumption in precipitation = $\frac{\text{lb iron used}}{\text{lb copper precipitated}}$		0.74

The continuous recirculation of the cementation tails through the plant apparently contaminates the cement copper with a large quantity of soluble ferrous sulfate, for the iron content of the different lots of cement copper increased during the plant test as the iron concentration of the tails solution increased (see Table I). More thorough washing of the cement could reduce the soluble iron content.

Table III presents data concerning the overall metallurgical copper balance resulting from the test run.

J. Randall Burke

J. Randall Burke

TABLE III

Metallurgical Copper Balance

	Weight of Contained Copper, lbs.	% of the Copper in the Heads
Cement copper	6365	77.0
Leach tails (unleached copper)	753	9.1
Solubilized copper in partial leach tails	160	1.9
Cementation tails	88	1.1
Waste solution	45	0.5
Total accounted for in test	7411	89.6
Ore heads	8271	100.0

$$\text{Accountability} = \frac{7411 \times 100}{8271} \qquad 89.6\%$$

For Clem

(1)

% Cu Extn.

$$\% \text{ Cu Extn.} = \frac{\text{Vol. Preg. (g/l Preg.)}}{\text{Vol. Preg. (g/l Preg.)} + \text{Wt. tails (Cu assay)}}$$

2nd. Day $\% \text{Extn.} = \dfrac{2.356\ (2.35)}{2.356\ (2.35) + 979.8\ (.001+)}$

$= \dfrac{\quad}{+} = \quad = \underline{80.1\%}$

3rd. Day $\% = \dfrac{2.84\ (1.60)}{2.84(1.60) + 958.3\ (.0010)}$

$= \dfrac{\quad}{+} = \quad = \underline{82.6\%}$

4th. Day $\% = \dfrac{2.53\ (1.86)}{2.53\ (1.86) + 962.6\ (.0008)}$

$= \dfrac{\quad}{+} = \quad = \underline{85.9\%}$

5th. Day $\% = \dfrac{2.678\ (1.94)}{2.678\ (1.94) + 973.1\ (.0013)}$

$= \dfrac{\quad}{+} = \quad = \underline{80.4\%}$

(2)

7th. Day % $= \dfrac{0.58\ (1.90)}{0.58(1.90) + 247.0(.0015)}$

$= \dfrac{\quad}{+} = \dfrac{\quad}{\quad} = \underline{74.8}\ \%$

8th. Day % $= \dfrac{1.46\ (1.97)}{1.46(1.97) + 468.4(.0020)}$

$= \dfrac{\quad}{+} = \dfrac{\quad}{\quad} = \underline{75.4}\ \%$

9th. Day % $= \dfrac{0.728\ (1.61)}{0.728(1.61) + 226.2(.0023)}$

$= \dfrac{\quad}{+} = \dfrac{\quad}{\quad} = \underline{69.3}\ \%$

10th. Day % $= \dfrac{0.765\ (1.45)}{0.765(1.45) + 243.1(.0012)}$

$= \dfrac{\quad}{+} = \dfrac{\quad}{\quad} = \underline{79.2}\ \%$

— % Acid soluble Cu in ore heads $= \dfrac{1.39}{1.40} = \underline{99.3\%}$

(3)

Weighted Avg. Cu Extn.

Day No.	No. of Leach Spls	% Cu Extn	Extn. x No. Spls
1	—	—	—
2	4	80.1	320.4
3	4	82.6	330.4
4	4	85.9	343.6
5	4	80.4	321.6
6	—	—	—
7	1	74.8	74.8
8	2	75.4	150.8
9	1	69.3	69.3
10	1	79.2	79.2
Total	21	Total	1690.1

$$\text{Weighted Avg. Cu Extn.} = \frac{1690.1}{21} = \underline{80.5\ \%}$$

% Pulp Density (from pebble mill)

$$\% \text{ P.D.} = \frac{\text{Wt. dry cake}}{\text{Wet wt.} + \text{Filt. vol. in mls.}}$$

1st. Day $\text{P.D.} = \frac{891.7}{1106.7 + 458} = 57.0\ \%$

2nd Day $\text{P.D.} = \frac{1801.5}{2167.7 + 999} = 56.9\ \%$

3rd. Day $\text{P.D.} = \frac{1823.4}{2123.8 + 875} = 60.8\ \%$

4th. Day $\text{P.D.} = \frac{1831.5}{2297.4 + 969} = 56.1\ \%$

5th. Day $\text{P.D.} = \frac{1735.6}{2127.2 + 1040} = 54.8\ \%$

6th. Day $\text{P.D.} = \frac{834.4}{1024.6 + 495} = 54.9\ \%$

7th. Day $\text{P.D.} = \frac{467.6}{570.7 + 193} = 61.2\ \%$

8th. Day $\text{P.D.} = \frac{934.1}{1170.5 + 460} = 57.3\ \%$

9th. Day P.D. $= \frac{457.1}{557.9 + 195} = \underline{60.7\%}$

10th. Day P.D. $= \frac{492.3}{590.4 + 230} = \underline{60.0\%}$

Weighted Avg. Pulp Density of Mill Pulp

Day No.	No. of Mill Pulp Spls.	% P.D.	P.D. x No. Spls.
1	2	57.0	114.0
2	4	56.9	227.6
3	4	60.8	243.2
4	4	56.1	224.4
5	4	54.8	219.2
6	2	54.9	109.8
7	1	61.2	61.2
8	2	57.3	114.6
9	1	60.7	60.7
10	1	60.0	60.0
Total	25		1434.7

Weighted Avg. P.D. $= \frac{1434.7}{25} = \underline{57.4\%}$

P.D. in leach trough is, however, less than this because either launder tails or water is added to the leach trough.

Daily Mill Pulp Cake Assays

— Get these direct from assay result sheet.

Weighted Avg. Mill Pulp Cake Assay

Day No.	No. of Mill Pulp Spls.	Assay, % Cu	Assay x No. Spls.
1	2	1.36	2.72
2	4	1.38	5.52
3	4	1.41	5.64
4	4	1.43	5.72
5	4	1.41	5.64
6	2	1.37	2.74
7	1	1.34	1.34
8	2	1.40	2.80
9	1	1.38	1.38
10	1	1.35	1.35
Total	25		34.85

Weighted Avg. Assay $= \frac{34.85}{25} = \underline{1.39\ \%\ Cu}$

Assay of Head Ore Sample = 1.40% (or 28 lbs Cu/ton ore)

Weight of Ore Charged

30.27 tons or 60,540 lbs daily (1 − 2.4% moist.) =
30.27 (0.976) = 29.54 tons of dry ore/day
295.40 tons or 590,800 lbs total for 10-day test.

Acid Used

Total vol. of acid tank — (tank is 84" in diam.)

Case I

Empty depth — 24" ; % depth — 28.6 ; % Vol. — 23.61
Full depth — 74" ; % depth — 88.1 ; % Vol. — 93.28
Quantity of acid — 54,370 lbs or $\frac{54370\,(454)\,(.2642\ \text{gal/l})}{1.83\,(1000)}$
= 3564 gals.

% of tot. vol. rep. by 3564 gals acid = 93.28 − 23.61 = 69.67%

% Vol. (V) = gals acid

$$V_{TOT.} = \frac{\text{gals acid}}{\%\,\text{Vol.}} = \frac{3564}{0.6967} = 5116$$ gals. to in a full tank.

(8)

Case II

Empty depth — 18" ; % depth — 21.4 ; % Vol. — 15.69
Full depth — 67" ; % depth — 79.8 ; % Vol. — 85.55

Quantity of acid — 54,014 lbs or $\frac{54014\ (454)(.2642)}{1.83(1000)}$

$= 3540$ gals

% of tot. vol. rep. by 3540 gals acid $= 85.55 - 15.69 = 69.86\%$

% Vol. (V) = gals acid.

$V_{TOT.} = \frac{\text{gals acid}}{\%\ \text{Vol.}} = \frac{3540}{.6986} = \underline{5068}$ gals total in a full tank

Avg. gals. in a full tank $= \frac{5116 + 5068}{2} = \underline{5092}$ gal

(9)

Day No.	Acid Depth, inches	% Depth	% Vol.	Vol. Acid, gals	Acid Used gals*	Acid Used lbs**
0	74	88.1	93.3	4751	—	—
1	65.5	78.0	83.7	4262	489	7005
2	59	70.2	75.0	3819	443	6346
3	51.5	61.3	64.3	3274	545	7808
4	42	50.0	50.0	2546	728	10429
5	35	41.7	39.5	2011	535	7664
6	30	35.7	32.0	1629	382	5472
7	25	29.8	25.0	1273	356	5100
8	18	21.4	15.7	799	474	6790
0	67	79.8	85.6	4359	—	—
9	59.5	70.8	75.7	3855	504	7220
10	53.5	63.7	67.2	3422	433	6203
15	50.5	60.1	62.7	3193	229	3281 for cementation pH control.

* gals of impure acid
** lbs of pure acid

(10)

Acid Used

Day No.	Acid Used, lbs
1	7005
2	6346
3	7808
4	10429
5	7664
6	5472
7	5100
8	6790
9	7220
10	6203
Total	70,037 lbs.

Avg. acid used daily $= \frac{70,037}{10} =$ 7004 lbs acid

Water Used

Day No.	Water Used, gals
1	7950
2	8300
3	9700
4	6820
5	6520
6	6550
7	7920
8	6750
9	10170
10	6280
Total	76,960 gals

Avg. daily water used $= \frac{76,960}{10} = \underline{7696 \text{ gals}}$

Power Used

Day No.	Power Used, Kwhr
1	160
2	160
3	240
4	160
5	160
6	160
7	160
8	240
9	160
10	160
Total	1760 Kwhr.

Avg. daily power used $= \frac{1760}{10} =$ 176 Kwhr for leaching & cementation

Power used during 10 days of leaching = 1760 Kwhr
Power used during cementation (8/29–9/3) = 493
Total = 2253 Kwhr.

$\frac{\text{Kwhr}}{\text{ton}} = \frac{2253 \text{ Kwhr}}{295.4 \text{ tons}} =$ 7.63 Kwhr/ton

Calcn. of Power Used in Cementation from 8/29 to 9/3

Cementation —

Power — 0236 x 80 of 8/29

Power — 0243 x 80 of 9/3

Ball mill & leach trough consumed power during cem.

9/2 Time on — 9:15 PM
9/3 Time off — 1:30 AM
} 4 hrs. run

Power for cem. only from 8/29 — 9/3 = (Tot. power from diff.) — (Power for mill & leach on 9/3)

Power used by 4-hr. operation of ball mill & leacher —

B = Power used = $\frac{160 \text{ kwhr}}{9.5 \text{ hrs}}$ (4 hrs) = 67 kwhr by ball mill & leach trough on 9/3.

Power for cem. only from 8/29 – 9/3 = 7 (80) – 67

= 560 – 67 = 493 kwhr for cementation only

Labor

Date	Normal Labor* Time, hrs.	Normal Labor* Wages, $	Overtime Labor** Time, hrs.	Overtime Labor** Wages, $	Supervision*** Time, hrs.	Supervision*** Wages, $	Total Wages, $
8/18	4	13.80	—	—	8	40	53.80
8/19	12	41.40	2	10.36	4	20	71.76
8/20	8	27.60	1	5.18	8	40	72.78
8/21	8	27.60	2	10.36	8	40	77.96
8/22	8	27.60	2	10.36	8	40	77.96
8/23	6	20.70	—	—	—	—	20.70
8/24	8	27.60	2	10.36	8	40	77.96
8/25	8	27.60	2	10.36	8	40	77.96
8/26	8	27.60	2	10.36	8	40	77.96
8/27	14	48.30	—	—	8	40	88.30
8/28	8	27.60	2	10.36	8	40	77.96
8/29	6	20.70	—	—	4	20	40.70
8/31	6	20.70	—	—	4	20	40.70
9/1	4	13.80	—	—	4	20	33.80
9/2	6	20.70	—	—	4	20	40.70
9/3	4	13.80	—	—	4	20	33.80
9/5	2	6.90	—	—	4	20	26.90
Totals	120 hrs	$414.00	15 hrs	$77.70	100 hrs	$500	$991.70

* At $3.45/hr.

** At $5.18/hr.

*** At $5.00/hr.

(15)

Total Costs

1) Crushing —
Tot. Cost = 302.7 tons ($1/ton) = $302.70

2) Hauling (mine — crusher — leach plant) —
Tot. Cost = 302.7 tons ($3/ton) = $908.10

3) Plate scrap —
a) Tot. Costs
(1) Cost — $22.50/ton (2.36 tons) = 53.00
(2) Labor in prep. of scrap
18 hrs ($3.45/hr) = $62.10 for 5.39 tons
For 2.36 tons, cost = $\frac{\$62.10\ (2.36)}{5.39}$ = 27.15

Cost/ton = $\frac{\$62.10}{5.39}$ = $11.53/ton

(3) Hauling — Cost = $0.06/ton-mi. (228 mi.)(2.36 tons) = 32.22
Cost/ton = 0.06 (228) = $13.68/ton
b) Total Cost/ton = $\frac{\$53 + \$27.15 + \$32.22}{2.36}$ = $47.71/ton

4) Acid (includes hauling) —
lbs. pure acid used = 70037 + 3281 = 73318 lbs. total

Tons impure acid used = $\frac{73318}{2000\ (.939)}$ = 39.04 tons

Tot. cost = 39.04 tons ($30.75/ton) = $1200.48

(16)

5) Water —
Tot. cost = 76,960 gals (\$0.00024/gal) = \$18.47

6) Power —
Tot. cost = 2253 Kwhr (\$0.032/Kwhr) = \$72.10

7) Labor (operating and maintenance) —
Tot. cost = \$991.70

8) Mining —
Tot. cost = 302.7 tons (\$4/ton) = \$1210.80

9) Supplies —
a) Operating supplies
Tot. cost = 17 days (\$15/day) = \$255
b) Maintenance supplies
Tot. cost = 17 days (\$18/day) = \$306

Tot. cost of supplies = \$561.00

Total Direct Costs = \$5377.72

10) Indirect Costs
a) General overhead = 0.123 (\$5377.72) = \$661.46
b) Payroll overhead = 0.15 (\$991.70) = \$148.76
11) Amortization of capital cost = \$931.43
12) Mine property payments = \$558.96

Grand Total = \$7678.33

(17)

Weight of Cu in Cement Produced

Wt. dry cement = wt. wet cement (1 − % Moist.)

Wt. Cu in cement = Assay (wt. dry cement)

Wt. Cu = Assay (Wet. Wt.)(1 − % Moist.)

Lot CC3A

Wt. Cu = .8898 (2007 lbs) (0.74) = 1322 lbs.

Lot CC3B

Wt. Cu = .8996 (2668) (0.717) = 1721 lbs

Lot CC3C

Wt. Cu = .8851 (3165) (0.672) = 1882 lbs

Lot CC3D

Wt. Cu = .8836 (1700) (0.677) = 1017 lbs

Lot CC3E

Wt. Cu = .7398 (840) (0.68) = 423 lbs

Total = 6365 lbs. Cu

Cu Recovery

$$\% \text{ Cu recovery} = \frac{\text{Tot. Wt. Cu in Cement}}{\text{Wt. Cu in Ore Heads}}$$

$$\% \text{ Cu recovery} = \frac{6365 \text{ lbs.}}{590{,}800 \text{ lbs}(0.014)}$$

$$= \frac{6365}{8271} = \underline{77.0} \text{ \% recovery}$$

Washing Efficiency

$$\% \text{ Washing efficiency} = \frac{\% \text{ Cu recovered}}{\% \text{ Cu extracted}} = \frac{77.0}{80.5} = \underline{95.6 \text{ \%}}$$

Fe/Cu Ratio

$$\text{Fe/Cu ratio} = \frac{4711 \text{ lbs. Fe}}{6365 \text{ lbs. Cu}} = \underline{0.74}$$

(this is the effect of power in cementation).

Leached Copper Held by Slimes

Wt. Cu in slimes = Wt. Cu extd − Wt. Cu discarded − Wt. Cu in cement

Wt. of Cu discarded —

(1) In 10,000 gal (assumed vol.) of 0.54 g/l Cu solution sent to waste:

$$\text{Wt. Cu} = \frac{10{,}000\ \text{gal}\ (3.785\ \text{l/gal})\ (0.54\ \text{g/l})}{454\ \text{g/lb.}} = \underline{45}\ \text{lbs.}$$

(2) In final cementation tails solution:

Data from timing of flow into launders —
(Preg. flow in equals tails flow out)

Gals in 10 sec.	Flow, gpm	Time Pump On, mins	Soln. Vol., gals
8.75	52.5	105	5512
8.50	51.0	11	561
8.75	52.5	30	1575
9.00	54.0	51	2754
8.75	52.5	59	3098
8.50	51.0	55	2805
8.00	48.0	52	2496

Total 18,801 gals tails containing 0.56 g/l Cu

$$\text{Wt. Cu} = \frac{18801\ (3.785)\ (0.56)}{454} = \underline{88\ \text{lbs}}$$

Tot. Wt. Cu discarded = 45 + 88 = <u>133 lbs</u>

Wt. Cu in slimes = 590,800(.014)(.805) − 133 − 6365

= 6658 − 133 − 6365 = 160 lbs Cu in soluble form tied up in slimes.

% of leached Cu held in slimes = $\frac{160}{6658}$ = 2.4 %

Acid Consumption

— In lbs acid/ton ore —

(1) By assay (4-hr. leach of −65 mesh ore with 1.02 N H_2SO_4) — 238.4 lbs/ton

(2) From plant test — $\frac{70{,}037 \text{ lbs}}{295.40 \text{ tons}}$ = 237.1 lbs/ton

— In lbs acid/lb Cu recovered — $\frac{73318 \text{ lbs}}{6365 \text{ lbs}}$ = 11.5 lb/lb.

— In lbs acid/lb. Cu extracted — $\frac{70{,}037 \text{ lbs}}{6658 \text{ lbs}}$ = 10.5 lb/lb

Total Costs/Unit

— Value/ton ore — $\frac{590800(.014)(\$0.40)}{295.40} = \$11.20$/ton ore

— In \$/month — \$20.749/ton(81 tons/day)(22 days/mo.) = \$36,974.72/m

Item	Total Cost, \$	Cost, \$/lb Cu Recovered	Cost, \$/ton ore
Crushing	302.70	0.048	1.000
Hauling	908.10	0.143	3.000
Plate scrap	112.37	0.017	0.380
Acid	1200.48	0.189	4.064
Water	18.47	0.003	0.062
Power	72.10	0.011	0.244
Labor	991.70	0.156	3.357
Mining	1210.80	0.190	4.000
Supplies	561.00	0.088	1.899
Indirect Costs	810.22	0.127	2.743
Amort. of Capital	931.43	0.146	3.153
Mine Prop. Payments	558.96	0.088	1.892
Total	\$7678.33	\$1.206	\$25.794

Weighted Avg. Tails Assays

Day No.	No. of Leach Spls	Assay, % Cu	Assay x No. Spls	Assay, % ASCu	Assay x No. Spls
1	—	—	—	—	—
2	4	0.14	0.56	0.11	0.44
3	4	0.10	0.40	0.08	0.32
4	4	0.08	0.32	0.06	0.24
5	4	0.13	0.52	0.12	0.48
6	—	—	—	—	—
7	1	0.15	0.15	0.13	0.13
8	2	0.20	0.40	0.18	0.36
9	1	0.23	0.23	0.21	0.21
10	1	0.12	0.12	0.10	0.10
Total	21		2.70		2.28

Weighted Avg. Tails Assay $= \frac{2.70}{21} = \underline{0.13}$ % Cu

Weighted Avg. ASCu Tails Assay $= \frac{2.28}{21} = \underline{0.11}$ % ASCu

% of Cu in tails that is acid soluble $= \frac{0.11}{0.13} = \underline{84.6}$ %

Ore Throughput Rate

Day No.	Length of Leach Run, hrs
1	9.5
2	10
3	9
4	9
5	8.75
6	8.5
7	8.25
8	8.75
9	7.25
10	9
Total	88.0 hrs

Avg. Length of Leach Run $= \frac{88.0}{10} = \underline{8.8}$ hrs.

Throughput rate:

(1) $\frac{29.54 \text{ tons}}{8.8 \text{ hrs}} = \underline{3.36}$ tons/hr.

(2) 3.36 tons/~~hr~~ (24 ~~hrs~~/day) = $\underline{81}$ tons/day

Metallurgical Copper Balance

— Copper in leach tails:

Assume tonnage of tails at 98% of head ore tonnage.

Wt. of tails = 295.4 tons (0.98) = 289.5 tons

Wt. of Cu in tails = 289.5 (2000) (0.0013) = 753 lbs of Cu

— Solubilized copper tied up in leach tails:

Soluble Cu in tails = Cu extd. − Cu produced − Cu discarded

= 6658 − 6365 − 133 = 160 lbs of Cu

— Metallurgical balance:

Item	Weight of Copper, lbs	% of Total in Ore
Copper in cement	6365	77.0
Copper in leach tails	753	9.1
Solubilized copper in leach tails	160	1.9
Cementation tailings	88	1.1
Waste solution	45	0.5
Total Accounted For	7411	89.6
Ore heads	8271	100.0

Miscellaneous Costs

— Amortization of Capital Cost

$100,000 capital cost of leach plant as per Mr. Spendlove.

Over a 5-year amortization period:

$$\frac{\$100,000}{5\text{ yr.}} = \$20,000/\text{yr.}$$

$$\frac{\$20,000/\text{yr.}}{365\text{ days/yr.}} = \$54.79/\text{day}$$

For a 17-day operating period, amortization is: ($54.79/day)(17 days) = $931.43

— Mine Property Payments

Payments are at $12,000/yr. or $\frac{\$12,000}{365} = \$32.88/\text{day}$

For a 17-day operating period, the payment is: ($32.88/day)(17 days) = $558.96