MS-002
Box 139
Folder 25

L. F. Engle December 21, 1966

Ash Pump at Ahmeek Mill

On December 19 we discussed the possibility of bringing all effluent streams at the Ahmeek Mill to one point of discharge from which an accurate sample can be taken for copper analysis and density, and where the volume of pulp leaving the mill could be measured with reasonable accuracy. One difficulty with collecting a good sample on a regular interval basis from material leaving via the ash pump in the basement of the mill has been the presence of ashes from the boiler house in the effluent stream a good portion of the time.

In investigating why the boiler house ashes are pumped first to the mill ash pump and then pumped into the lake, no valid reason has been uncovered. This is just a hangover practice from when the tailings, etc. had to be pumped a considerable distance out on the tailing bank. Since this area has all now been dredged out, there seems no reason why the ashes from the boiler house cannot be pumped directly from the boiler house to the lake, or to a pile behind the boiler house from where they can be taken for use as fill by people desiring this material. If a pile were made on land, the water could be made to drain into the Hungarian creek or to the lake.

The ashes are pumped through a six-inch cast iron line through the mill, under the floor, and frequent choke-ups are experienced which have to be cleared by mill maintenance personnel. Apparently all maintenance costs on this line and on the ash pump are charged to the mill.

We have asked the Engineering Department to investigate the possibility of discharging these ashes by a different means, with the thought in mind that with a likely arrangement for collecting mill spills, etc., a few further changes can be made whereby the ash pump could be shut down or eliminated, with a considerable saving in power and maintenance costs, and all mill effluents would be discharged through one line from which acceptable sampling could be done.

It is my understanding that R&D has terminal responsibility for sampling process design based on B. C. Peterson's instructions to you last week. Suggestions for one or more schemes of centralizing the discharges and taking samples from the mill will be presented as soon as the investigation is complete.

LCK/wmh L. C. Klein
cc: BCP
JA
TWK
JK
BW
RWK
✓RS

Preliminary Investigation

Ahmeek Mill Boiler House

Proposed Ash Handling Equipment Vs Present Hydraulic System

February 23, 1967

To: T.W. Knight

A recent request concerning the disposal of the Ahmeek Boiler House ashes, employing other methods than now used, are indicated as follows with relative cost:

Scheme I - Flush System direct to lake.

Ash disposition by 8" C.I. pipe directly to the lake involves 1200 feet of 8" C.I. pipe crossing under 3 railroad tracks, with 3 points of jetting along the line.

Total cost of material and labor.... $17,100

Scheme II - Belt Conveyor System - horizontal and inclined.

(a.) Belt conveyor, 100 feet horizontal X 16" wide, including excavation for belt, construction of concrete conveyor trench, and installation of conveyor, including complete drive.
Cost of this conveyor, $6,700.

(b.) Belt conveyor, 36' wide X 16" wide, inclined at 20° to outside storage pile.
Cost includes material and labor, drive, and housing as shown by Sketch Scheme II attached.
Cost of this conveyor, $4,610.

Cost for Scheme II above a and b.... $11,310

Scheme III - Belt Conveyor plus Elevator.

(a.) 100 feet of 16" width horizontal belt conveyor.
Same as Scheme II a. above, $6,700.

(b.) Suitable 50 feet center-center belt bucket elevator, material and labor, $7,700.

Total cost Scheme III above a and b... $14,400

Comments:

(a) If it is desirable to change from the present ash disposal system, Scheme I, appears to be the most suitable of the three systems indicated above. Only an extention of the present pipe line system would be required.

(b) Scheme III appears to be the second best, affording more storage. However, stacking of ashes may result in undesirable dust problems when strong winds prevail. It is to be noted that Scheme III ash storage pile would have a storage capacity for 45 days. Higher maintenance cost should not be overlooked for Schemes II and III.

(c) In reviewing a report to L.F. Engle by L.C. Klein, (attached) dated December 21, 1966, it is to be noted that the existing Ahmeek Boiler House ashes are pumped to the Ahmeek Mill and then pumped to the lake---actually the ashes are flushed from the Ahmeek Boiler House by high pressure water and are pumped out by the Mill 10" ash and tailing pump. This method appears to be the best and most common practice for ash disposal. Tests made during October, 1966, disclosed that the 10" ash pump solids averaged 1.20% solids with an average tonnage of 335 tons per 24 hours. This included 7 to 10 tons of ashes per 24 hours or about 2 to 3% ash. The ashes are discharged once on each shift for about 1 to 1 1/2 hour duration.

Since automatic sampling and instrumentation shown by drawing No. 12702 indicates a good possibility of measuring density, flow and tailing tonnage, it is apparent that if the ashes were allowed to be included, it would not be too detremental. Also, if the ashes were not wanted in the sampling circuit, the sampler could be shut off during ash disposal periods. Since this is a Research and Development terminal responsibility, and in view of the fact that several other testing problems to include changes and tests required for diversion and treatment of elevator shovel wheel overflows, as well as possible diversion of slime thickeners overflows and pending the requirement of several tests; more time is required to finalize a more definite conclusion.

We are attaching the following sketches and drawings for your study of this problem.

1. Proposed ash handling arrangement - Scheme II.

2. Proposed ash handling arrangement - Scheme III.

3. Existing overflow diagram No. 17331.

4. Automatic sampling and instrumentation arrangement, drawing No. 12702.

John J. Vitton
Industrial Engineering Dept.

Approved:

R.L. Ellison
Industrial Engineering Manager

JJV:11

cc: LE
LK,
JA,
JK,
KE,

Attachments (5)

MS-002
Box 78
Folder 10

RECEIVED
DEC 22 1966
REFERRED TO | REPLY BY

L. F. Engle

December 21, 1966

Ash Pump at Ahmeek Mill

On December 19 we discussed the possibility of bringing all effluent streams at the Ahmeek Mill to one point of discharge from which an accurate sample can be taken for copper analysis and density, and where the volume of pulp leaving the mill could be measured with reasonable accuracy. One difficulty with collecting a good sample on a regular interval basis from material leaving via the ash pump in the basement of the mill has been the presence of ashes from the boiler house in the effluent stream a good portion of the time.

In investigating why the boiler house ashes are pumped first to the mill ash pump and then pumped into the lake, no valid reason has been uncovered. This is just a hangover practice from when the tailings, etc. had to be pumped a considerable distance out on the tailing bank. Since this area has all now been dredged out, there seems no reason why the ashes from the boiler house cannot be pumped directly from the boiler house to the lake, or to a pile behind the boiler house from where they can be taken for use as fill by people desiring this material. If a pile were made on land, the water could be made to drain into the Hungarian creek or to the lake.

The ashes are pumped through a six-inch cast iron line through the mill, under the floor, and frequent choke-ups are experienced which have to be cleared by mill maintenance personnel. Apparently all maintenance costs on this line and on the ash pump are charged to the mill.

We have asked the Engineering Department to investigate the possibility of discharging these ashes by a different means, with the thought in mind that with a likely arrangement for collecting mill spills, etc., a few further changes can be made whereby the ash pump could be shut down or eliminated, with a considerable saving in power and maintenance costs, and all mill effluents would be discharged through one line from which acceptable sampling could be done.

It is my understanding that R&D has terminal responsibility for sampling process design based on B. C. Peterson's instructions to you last week. Suggestions for one or more schemes of centralizing the discharges and taking samples from the mill will be presented as soon as the investigation is complete.

L. C. Klein

LCK/wmh
cc: ✓BCP
JA
TWK
JK
RW
RWK
RE

For Inter-Office Correspondence Only — B. C. PETERSON — April 17, 1967

To Mr.

cc: RJW, RWK, JA, TWK, JK, LCK, Engr. Dept., Ind. Engr. Dept., #1913, Reading #1

Subject: Ahmeek Mill Sampling and Effluent Handling

Form 022010

Some time ago, you asked R & D to study sampling and effluent handling at the Ahmeek Mill. The results of that study, as reported in L. C. Klein's memo to me of April 14, 1967, are attached. There was some delay in presenting this information since, during the course of the investigation, Mr. Klein did feel that test work on cyclones of shovel-wheel overflows could contribute greatly to the overall design. I am pleased to report that this test work did show that cyclones will conservatively add some 200 to 300 lbs/day of copper to mill recovery.

Various units of the Engineering Division and the Mill operators have, of course, contributed a great deal to the work presented here. In my opinion, there is sufficient information and justification for you to authorize the proper paper work from the Divisions concerned to implement the recommendations.

We are well aware of techniques of measuring mass flow from a mill of this sort. Use of nagnetic flow meter and density gauges is quite common for the job, and under certain situations, can perhaps be justified. Becoming more and more common also is the use of on-stream continuous chemical analysis instrumentation. We do feel, at this time, that the $20,000-30,000 expense for such devices is unjustifiable. We propose that the system attached be installed, and as further operating experience and needs are defined, we can re-evaluate the inclusion of sophisticated measurement devices.

Detailed equipment and design data pertinent to Mr. Klein's report are available at the Engineering Division, and are not attached here.

If there are any questions concerning the recommendations, we, of course, would be pleased to assist in clarification.

L. F. Engle

L. F. Engle

LFE/bv

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APR 18 1967
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2

Calumet & Hecla

CALUMET
DIVISION
CALUMET, MICHIGAN 49913

For Inter-Office Correspondence Only

To Mr. L. F. ENGLE Date April 14, 1967

Form 022010 SUBJECT Proposed System for the Sampling and Disposal of Ahmeek Mill Taili

An investigation was made of the present method of disposing of tailings and other wastes from the Ahmeek Mill to determine a reliable method of automatically sampling and handling these waste streams. Presently, wastes from the mill leave the plant at three different points. Each of these streams contains wastes from different areas of the plant, so that even if a good sample of each stream were obtained, it would be difficult to pinpoint the source of abnormally high levels of copper in any one stream. Such samples would be practically worthless to the mill operators for the purpose of controlling the operation.

Wastes now leave the mill by way of the general tailings pump, the ash pump, or by a launder under the mill basement floor, by gravity (See attached drawing No. 17331-A). The general tailings pump handles the tailings from the eight primary flotation machines, and the overflows from the six slime thickeners. The capacity of this pump is overtaxed when more than six of the eight mill units are operating, and the pump sump overflows to the ash pump sump. This situation will be worse when two additional slime thickeners are activated to handle the additional amount of slime and water from the spill recovery system that will soon be in operation.

The ash pump disposes of ashes from the boiler house, the tailings from the slime flotation machine in the mill basement, spills from the basement of the mill, overflow from the general tailings pump sump, and at times, part of the shovel wheel overflows. Recently, the shovel wheel overflows have been diverted entirely to this pump for sampling purposes.

The gavity discharge launder normally handles the shovel wheel overflows and water overflowing from the mill constant head water tank.

Before attempting to set up a reliable sampling system, it is recommended that some re-routing of the effluent waste streams be made so that samples of these streams would be more valuable for the purpose of mill control, and so that these wastes be eliminated from the mill at only two points instead of three. All flotation tailings would be handled by the main tailings pump. All other "unprocessed" wastes would leave the plant by gravity through the basement floor launder. Automatic samplers would be installed to sample each stream in a conventional manner. A major change will be necessary in the general tailings discharge lines in order to install a sampler that will collect reliable samples. (See attached Drawing No. 17331-B).

A system is being installed in the mill to collect all spills. These will be returned to process entirely, except in very rare instances; such as a power failure or major spill when there is a possibility of some loss, but as a rule, this material will have no outlet from the plant.

The only materials leaving the plant will be flotation tailings and overflows from the thickeners and shovel-wheels. In the present flowsheet, the thickener overflows are combined with the primary flotation machine tailings and removed from the plant by the general tailings pump. These overflows represent over half of the total volume handled by the pump. The proposed scheme woul divert these overflows to the basement floor launder where they would flow out of the plant by gravity, relieving the load on the general tailings pump.

The tailings from the slime flotation machine in the mill basement which are now discharged from the mill by the ash pump would be pumped into the general tailings sump to be discharged with the primary flotation tailings. The reduced volume handled by the pump is more than sufficient to allow the pump to handle all possil flotation tailings from an eight stamp operation, with no overflow going to the ash pump.

Recent tests, using a cyclone to recover coarse material and enriched copper bearing fines from the shovel-wheel overflows have indicated that a substantial amount of copper bearing material can be recovered for reprocessing, and an essentially minus 200 mesh slime rejected. It is recommended that an additional cyclone be installed to process all of this material, and that the cyclone overflows be combined with the thickener overflows for sampling and discharging through the basement floor launder. It is conservatively estimated (based on test results) that from 200-300 lbs./day of copper can be recovered with this equipment.

The proposed system will have the following advantages in addition to those already mentioned: The ash pump will no longer need be operated except for the infrequent periods that ashes from the boiler house are being flushed. There will be a saving in power used to operate the main tailings pump because of the reduced load The "unprocessed wastes" will be deposited in an area away from the general tailings pile where the settled portion will probably be enriched in copper and would be accessible to the dredge for reprocessing at the Tamarack Reclamation plant at some future date

Following is a summary of what must be done to make the recommende changes in the Ahmeek Mill flowsheet and install adequate sampling stations:

1. The 16" general tailings pump discharges into a 22" pipe-line that goes directly to the tailings dump, approximately 300 feet east of the mill. It will be necessary to raise the pipe about three feet where it goes through the mill. This pipe would then terminate inside the mill at the east end and flow

into a short launder, on the end of which would be mounted an automatic sampler. The pulp would then discharge into a large box, from which two 22" pipes would take the tailings by gravity to the tailings dump. The 12" pipe from the ash pump would also discharge into the box that the tailings do. A second 22" pipe would replace the 12" pipe going to the tailings dump and this can be mounted along side the present 22" pipe on the same bents. A secondary sampler would be used to reduce the size of the sample collected by the primary sampler to a workable volume. Samples would be taken at ten minute intervals and analyzed for each shift. Equipment required for this includes two samplers, 300 feet of 22" pipe, launder, box, and supports. The estimated cost is $12,045.

2. To divert the tailings from the basement slime flotation machine to the general tailings pump sump will require a pump and motor installation and piping. A 4" pump and 10 hp motor are available. Total cost of installation is estimated at $440.

3. One 20" Krebs cyclone is now installed experimentally for processing shovel-wheel overflows. Two will be required to process the entire flow. These cyclones are currently available at the mill and could be used for possibly a year before they will be required for installation in the ball mill circuit. To reclaim the cyclone underflow product, a 2-1/2" pump with motor, a screen, and piping is required. Estimated cost, including new pump and motor will be about $1,540. Replacing the cyclones at a later date will cost approximately $3,400.

4. To combine the thickener overflows with the shovel-wheel overflows, install an automatic sampler and divert this flow to the basement floor launder will require an approximately 35 ft. extension of the floor launder, plus a catch box, and short launder with sampling station, and an automatic sampler. Estimated cost is $3,103.

Much of the work required on Item 1 can only be done when the mill is down. This work could presumably be done during the regular two weeks vacation shutdown. Work on the other three items could be mostly completed when the mill is operating, requiring, at the most, one day each to connect into the different systems when the mill is not operating.

Attached is a summary of labor and materials estimates for each of the items listed. Also attached are flow diagrams, 17331-A showing the present arrangement, tailings and overflow streams, and 17331-B showing the proposed arrangement with changes indicated by heavy lines.

L C Klein

LCK/bv L. C. Klein

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COST ESTIMATES

	Labor	Materials	Total
Item 1. Raise 22" tailings pipe 3'. Procure and install launder, box, and sampler. New tailings pipe plant to tailings dump	$4,550.00	$6,400.00	
Contingencies	455.00	640.00	
	$5,005.00	$7,040.00	$12,045.00
Item 2. Pump and pipe to pump slime flotation tailings to main tailings sump. 4" pump and motor available.	200.00	200.00	
Contingencies	20.00	20.00	
	220.00	220.00	440.00
Item 3. Cyclone installation, including chip screen, pump and motor, and pipeline for returning recovered values for reprocessing. *	1,000.00	400.00	
Contingencies	100.00	40.00	
	1,100.00	440.00	1,540.00
Item 4. Slime thickener and shovel-wheel overflow sampling system, including sampler and relocation of outlet.	1,260.00	1,561.00	
Contingencies	126.00	156.00	
			3,103.00
Totals	$7,711.00	$9,417.00	$17,128.00

* No purchase of cyclones necessary for about one year. Cyclones awaiting installation in ball mill circuits can be used. Replacement cost approximately $3,400.00.

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Box 78
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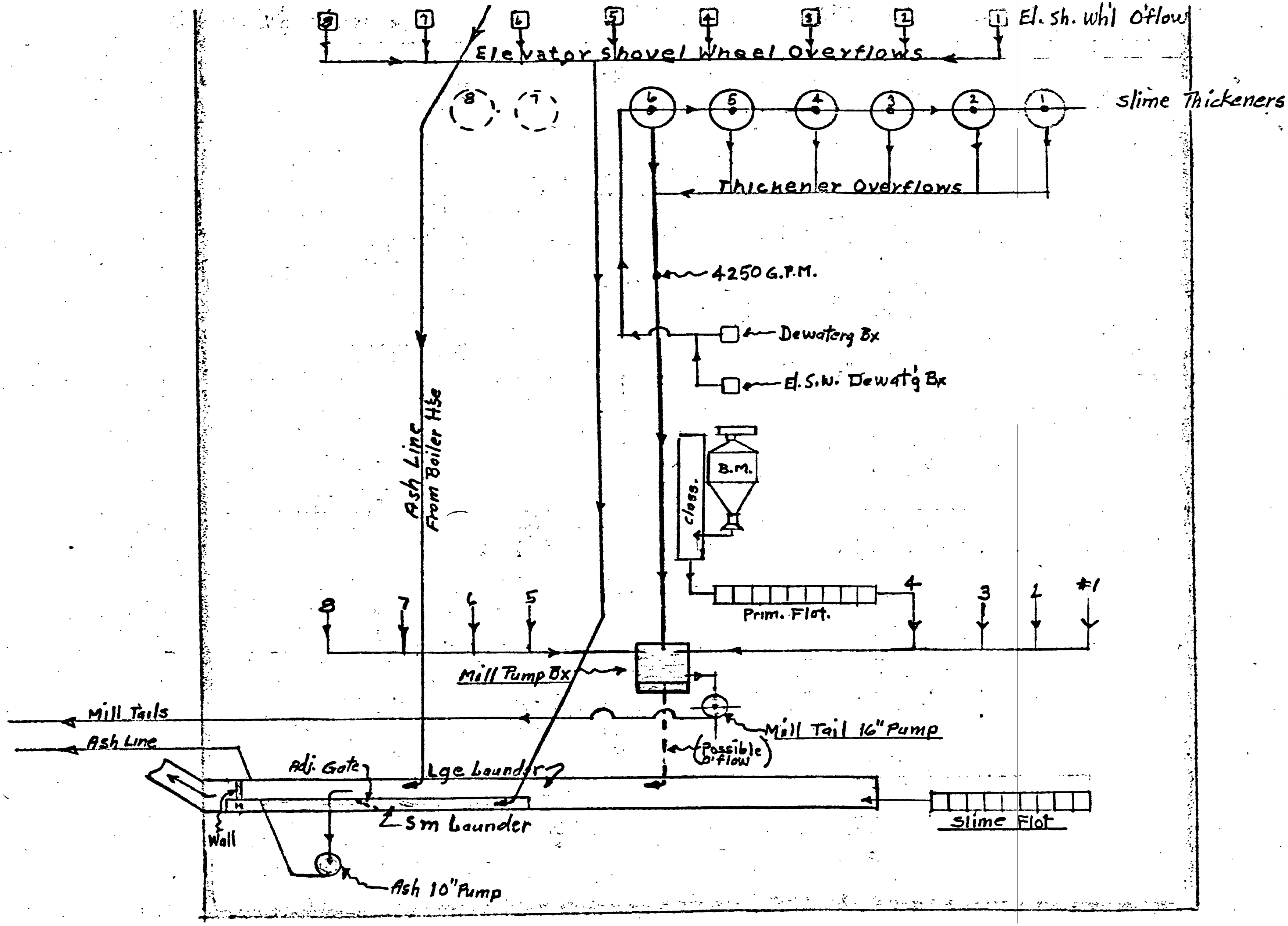
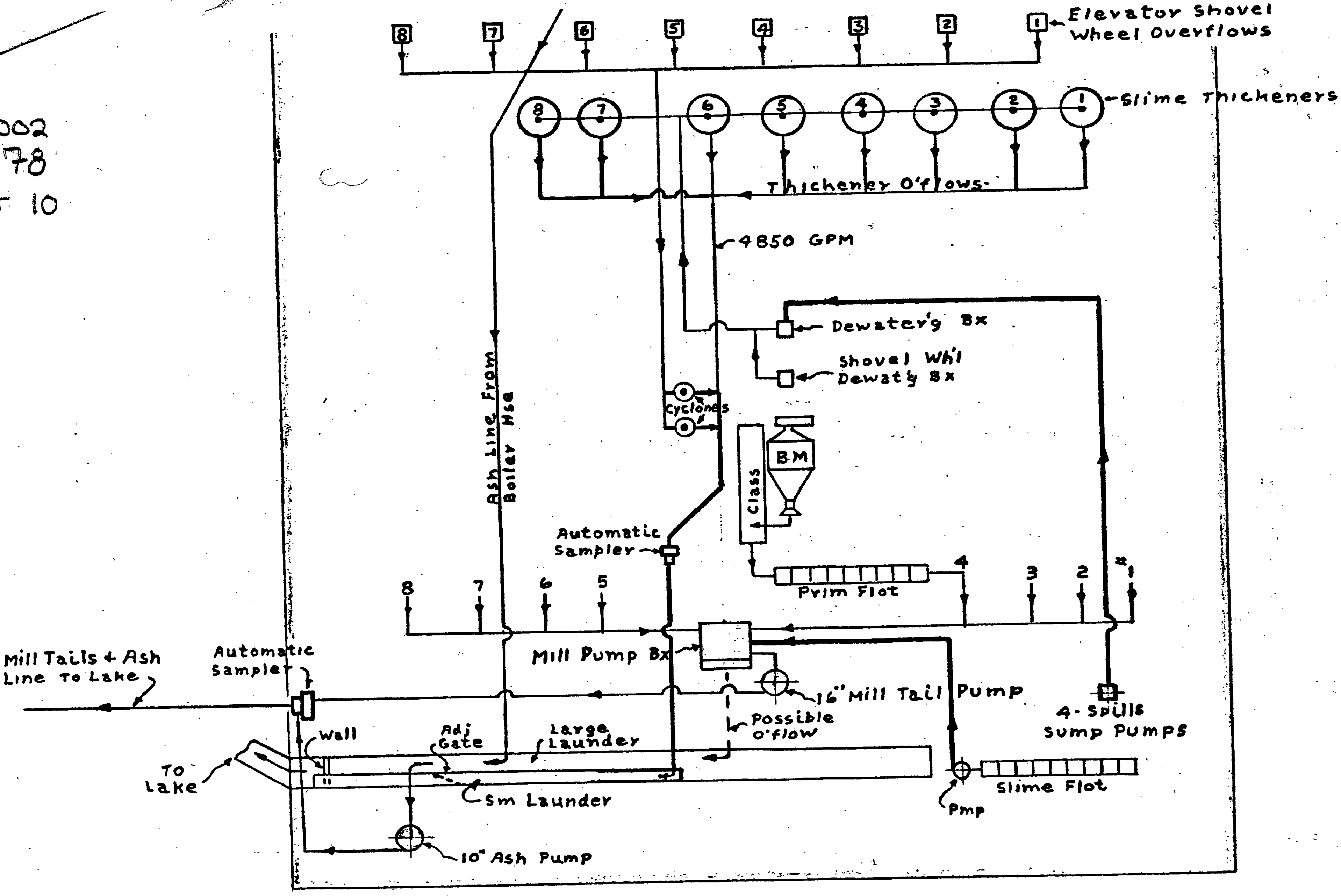


Calumet Division
Calumet & Hecla Inc
Ahmeek Mill
Exist'g O'flow Diagram
Date 2-8-67 By J.J.V. No 17331-A REV 4-12-67 EM

REF:
For Proposed Changes To Diagram see Dwg 17331-B Date 4-13-67

MS-002
Box 78
Folder 10



Calumet Division
Calumet & Hecla
Ahmeek Mill
Proposed O'flow Diagram
Date 4-13-67 EM

REF:
For Existg Diagram see Dwg 17331-A

№ 17331-B

341

MS-002
Box 151
Folder 27

Calumet & Hecla

CALUMET
DIVISION
CALUMET, MICHIGAN 49913

For Inter-Office Correspondence Only B. C. PETERSON

April 17, 1967

To Mr.

cc: RJW JK
RWK LCK
JA Engr. Dept.
TWK Ind. Engr. Dept.
~~#1913~~
Reading #1

Subject: Ahmeek Mill Sampling and Effluent Handling

Form 022010

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We are well aware of techniques of measuring mass flow from a mill of this sort. Use of magnetic flow meter and density gauges is quite common for the job, and under certain situations, can perhaps be justified. Becoming more and more common also is the use of on-stream continuous chemical analysis instrumentation. We do feel, at this time, that the $20,000-30,000 expense for such devices is unjustifiable. We propose that the system attached be installed, and as further operating experience and needs are defined, we can re-evaluate the inclusion of sophisticated measurement devices.

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L. F. Engle

L. F. Engle

LFE/bv

342

Calumet & Hecla

CALUMET
DIVISION
CALUMET, MICHIGAN 49913

For Inter-Office Correspondence Only

To Mr. L. F. ENGLE Date April 14, 1967

Form 022010 SUBJECT Proposed System for the Sampling ~~and Disposal of Ahmeek Mill Taili~~

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343

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The tailings from the slime flotation machine in the mill basement which are now discharged from the mill by the ash pump would be pumped into the general tailings sump to be discharged with the primary flotation tailings. The reduced volume handled by the pump is more than sufficient to allow the pump to handle all possil flotation tailings from an eight stamp operation, with no overflow going to the ash pump.

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344

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into a short launder, on the end of which would be mounted an automatic sampler. The pulp would then discharge into a large box, from which two 22" pipes would take the tailings by gravity to the tailings dump. The 12" pipe from the ash pump would also discharge into the box that the tailings do. A second 22" pipe would replace the 12" pipe going to the tailings dump and this can be mounted along side the present 22" pipe on the same bents. A secondary sampler would be used to reduce the size of the sample collected by the primary sampler to a workable volume. Samples would be taken at ten minute intervals and analyzed for each shift. Equipment required for this includes two samplers, 300 feet of 22" pipe, launder, box, and supports. The estimated cost is $12,045.

2. To divert the tailings from the basement slime flotation machin to the general tailings pump sump will require a pump and motor installation and piping. A 4" pump and 10 hp motor are available. Total cost of installation is estimated at $440.

3. One 20" Krebs cyclone is now installed experimentally for processing shovel-wheel overflows. Two will be required to process the entire flow. These cyclones are currently available at the mill and could be used for possibly a year before they will be required for installation in the ball mill circuit. To reclaim the cyclone underflow product, a 2-1/2" pump with motor, a screen, and piping is required. Estimated cost, including new pump and motor will be about $1,540. Replacing the cyclones at a later date will cost approximately $3,400.

4. To combine the thickener overflows with the shovel-wheel overflows, install an automatic sampler and divert this flow to the basement floor launder will require an approximately 35 ft. extension of the floor launder, plus a catch box, and short launder with sampling station, and an automatic sampler. Estimated cost is $3,103.

Much of the work required on Item 1 can only be done when the mill is down. This work could presumably be done during the regular two weeks vacation shutdown. Work on the other three items could be mostly completed when the mill is operating, requiring, at the most, one day each to connect into the different systems when the mill is not operating.

Attached is a summary of labor and materials estimates for each of the items listed. Also attached are flow diagrams, 17331-A showing the present arrangement, tailings and overflow streams, and 17331-B showing the proposed arrangement with changes indicated by heavy lines.

L C Klein

LCK/bv L. C. Klein

345

AR#3 E&P

COST ESTIMATES

AR#1 Cap.

	Labor	Materials	Total
Item 1. Raise 22" tailings pipe 3'. Procure and install launder, box, and sampler. New tailings pipe plant to tailings dump	$4,550.00	$6,400.00	
Contingencies	455.00	640.00	
	$5,005.00	$7,040.00	$12,045.00
Item 2. Pump and pipe to pump slime flotation tailings to main tailings sump. 4" pump and motor available.	200.00	200.00	
Contingencies	20.00	20.00	
	220.00	220.00	440.00
Item 3. Cyclone installation, including chip screen, pump and motor, and pipeline for returning recovered values for reprocessing. *	1,000.00	400.00	
Contingencies	100.00	40.00	
	1,100.00	440.00	1,540.00
Item 4. Slime thickener and shovel-wheel overflow sampling system, including sampler and relocation of outlet.	1,260.00	1,561.00	
Contingencies	126.00	156.00	
	1,386.00	1,717.00	3,103.00
Totals	$7,711.00	$9,417.00	$17,128.00

AR#1 CAP

AR#2 Cap

AR#2 CAP

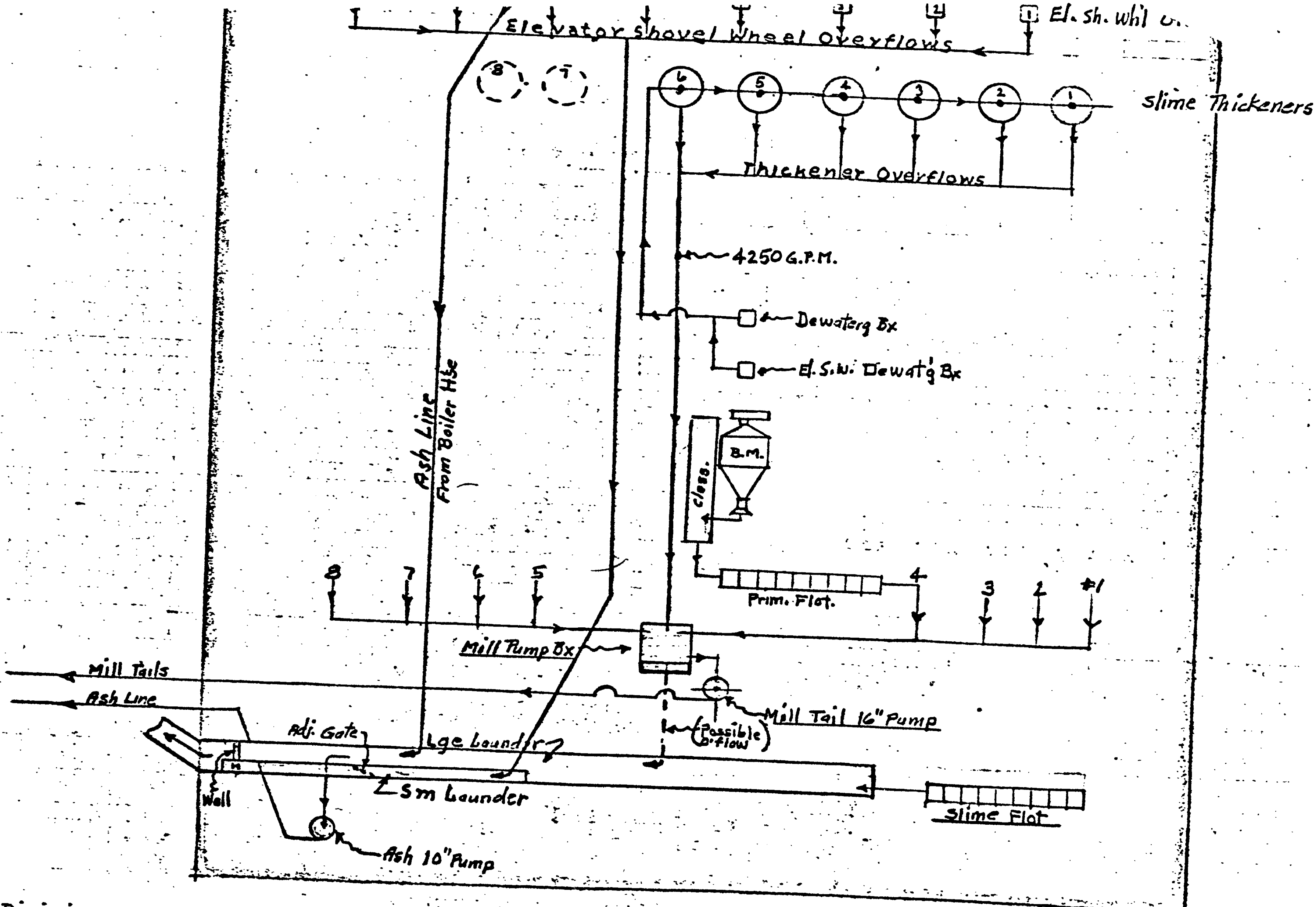
* No purchase of cyclones necessary for about one year. Cyclones awaiting installation in ball mill circuits can be used. Replacement cost approximately $3,400.00.

17,128.00
3,400.00
$20,528.00
12,377.00
8,151.00

AR#1 7,260.00
AR#2 5,117.00
12,377.00

E&P - AR#3 = $17,128 + $3,400.00 - (AR#1 + AR#2)

346



Calumet Division
Calumet & Hecla Inc

Ahmeek Mill

Exist'g O'flow Diagram

Fe 2-8-67 By J.J.V.

REF:
For Proposed Changes To Diagram see Dwg 17331-B Date 4-13-67

No 17331-A REV 4-12-67 EM

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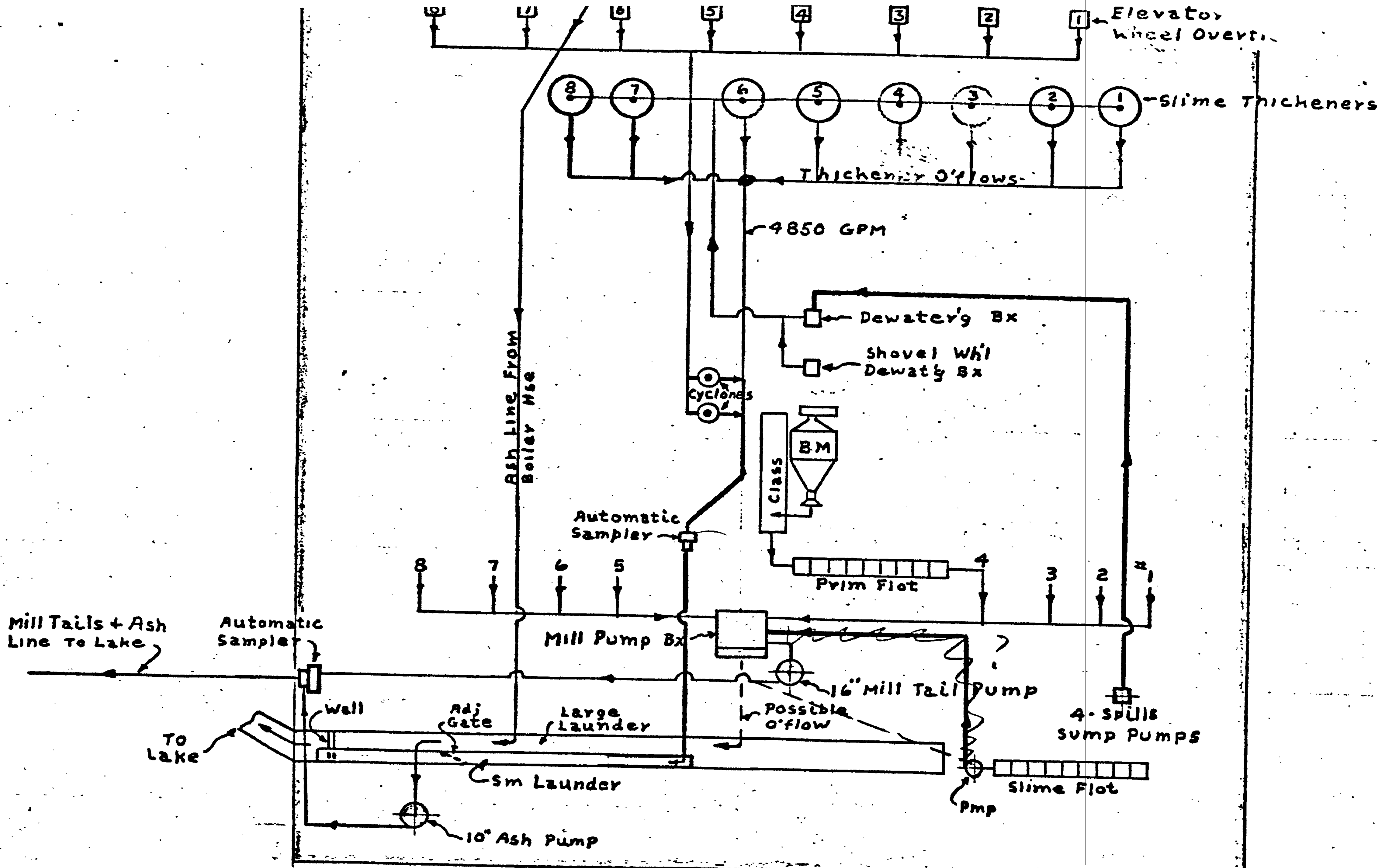


Calumet Division
Calumet & Hecla
Ahmeek Mill
Proposed O'flow Diagram
ate 4-13-67 EM № 17331-B

REF:
For Existg Diagram see Dwg 17331-A

MS-002
Box 40
Folder 8

Tamarack Reclamation – Steam Main

STEAM LINE FROM AHMEEK MILL TO TAMARACK RECLAMATION

Length of Post Supports.
Number of Supports required.

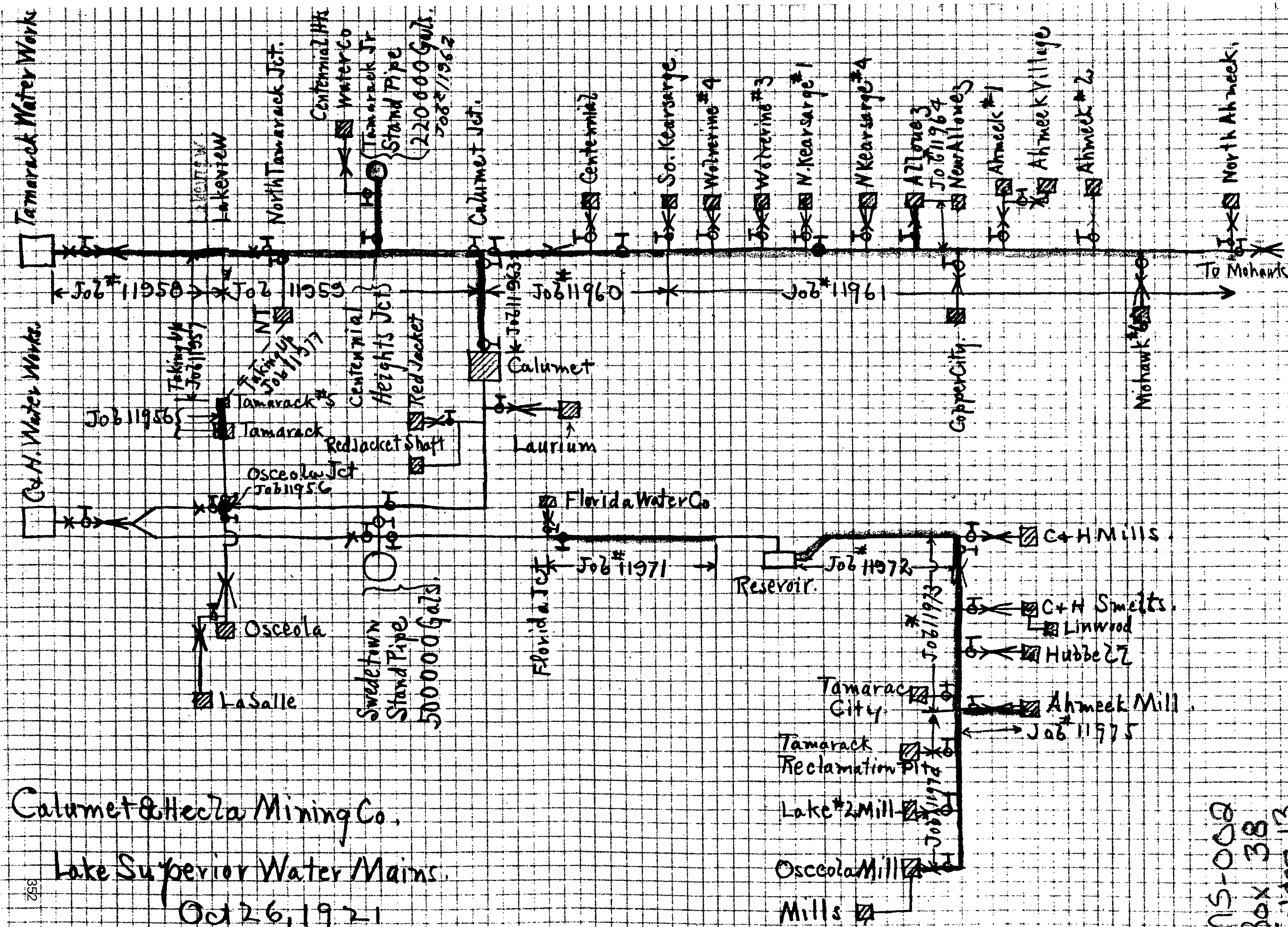
Length given is from ground to center line of 8" steam pipe.

Post Number	Length	Post Number	Length	Post Number	Length	Number of Posts	Length
1	10½'	40	12½'	79	8'	16	3'
2	10½	41	12	80	8	20	4
3	10½	42	12	81	8	19	5
4	3	43	11½	82	7½	17	6
5	3	44	11	83	7½	17	7
6	3	45	10½	84	7	9	8
7	3	46	10½	85	7	1	10
8	3	47	10	86	6½	6	11
9	3	48	3	87	6	3	12
10	3	49	3½	88	6	4	13
11	3½	50	3½	89	6	1	14
12	3½	51	4	90	6	1	15
13	4	52	4	91	6	1	24
14	4½	53	4	92	5½	1	25
15	4½	54	5	93	5½	116	
16	4½	55	5	94	5½		
17	5	56	5½	95	5½		
18	4½	57	5½	96	5		
19	4½	58	5½	97	5		
20	4½	59	5	98	5		
21	4	60	4½	99	5		
22	3½	61	4½	100	5		
23	3½	62	4	101	4½		
24	3	63	4	102	4		
25	3	64	4	103	4		
26	2½	65	3½	104	7		
27	2½	66	3½	105	7		
28	2½	67	3½	106	7		
29	2½	68	4½	107	7		
30	2½	69	5½	108	7		
31	3	70	6	109	7		
32	3½	71	6	110	7		
33	23½	72	6	111	7		
34	24½	73	6	112	7		
35	14½	74	6½	113	7		
36	13½	75	6½	114	7½		
37	13	76	7	115	7½		
38	13	77	7	116	8		
39	13	78	8				

No 1 nearest to Ahmeek Mill Boiler House

Req. #11744
Job #C-1297

TAMARACK RECLAMATION FACILITIES



Calumet & Hecla Mining Co.

Lake Superior Water Mains.

Oct 26, 1921

MS-003
Box 38
Folder 13

352

Six-cent Copper from Calumet & Hecla Tailings

Over 50,000,000 Tons of Sand, Accumulated During Half a Century, Being Treated by Tabling, Flotation, and Leaching

By C. H. Benedict
Metallurgist, Calumet & Hecla Consolidated Copper Co.,
Lake Linden, Mich.

THE CALUMET & HECLA MINING CO. was organized in 1871 as a consolidation of four companies, two of minor importance, together with the Calumet Mining Co., formed in 1865, and the Hecla Mining Co., organized about a year later. Both these latter companies found profitable ground at once and began the erection of stamp mills. The Calumet Mining Co. first erected its mill at the mine, but in a few years moved to its present site on Torch Lake adjoining the site on which the Hecla Mining Co. had already built its original mill and which is about four miles from the mine. These mills have been in existence then, at the same site, for upward of fifty years and have been depositing their tailings continuously into Torch Lake during that period.

Torch Lake is one of a series of inland lakes in the Keweenaw peninsula of upper Michigan, but is distinguished by being on the same level as Lake Superior, with which it has navigable connection; and it is distinguished further by the fact that it is quite deep relative to its area. At the time the mills were erected on the shore of this lake there was no thought of anything except the desire to find room for the sand tailings, but considering the development of the mine and of the art of metallurgy in the fifty years following the foundation of the mills, no better site could have been found anywhere. The shore adjacent to the lake is relatively flat, so that all buildings are at approximately the same level, but the sloping hills running down toward the shore permit of a uniform grade to a gravity railroad, and a trestle of moderate length gives the necessary elevation for gravity run to the ore bins and stamps.

The mills were built close to the shore of Torch Lake, and for some years there was sufficient elevation so that the tailings ran into the lake by gravity, but as the deposit increased in area the shore gradually receded and it was necessary to provide some means of giving the tailing sufficient elevation to reach the lake. For this purpose a device was developed unique in this country at that time, and even yet but few are in use outside of the Lake Superior copper district. This is

Dredge and portion of pontoon line

Shore plant, showing storage pool and swinging suction line

known as a sand wheel, a slowly revolving wheel with depressions or buckets in the periphery which take their load at the bottom and discharge tangentially on approaching the top. Starting with a 30-ft. wheel the diameter of successive wheels was increased to 40, then to 50, and finally to 65 ft., to reach the desired elevation.

Originally two mills were built, known as the Calumet mill and the Hecla mill, and there are two distinct tailing piles with their centers about three-quarters of a mile apart and with clear water between, the two almost inclosing a bay from which is drawn the water for the pumping station. These tailing piles covered an area of about 152 acres at one time and vary in depth from nothing at the shore line to 120 ft. The reclamation plants are erected centrally to the north of the Calumet pile, and it is this pile that is now being reclaimed. The Hecla bank was all of it conglomerate up to about 1900, but since that date the south or extreme end has been mostly amygdaloid tailings of much lower grade, and there will be a boundary line between the conglomerate and the amygdaloid, which, as dredged, will be a mixture of the two. When this mixed material is to be reclaimed it will result either in the inclusion of low-grade material which would not pay by itself to reclaim or in the exclusion of some conglomerate tailing which by itself would be profitable.

At the beginning of mining operations in the late 60's the ore was running better than 100 lb. to the ton by assay and the metallurgical methods were naturally crude compared to present-day standards. The ore increased in richness for some years after the opening of the mines, and tailing losses of 20 lb. to the ton and more were not out of the ordinary. In those years the smelters required a very high-grade product, and as the fine copper could not be concentrated profitably to smelter requirements, practically no effort was made to save the slimes until about 1884, when buddles or circular slime tables were introduced. Neither was any effort made, except spasmodically, to do any grinding other than that by the original stamp, which crushed all material to pass through a $\frac{3}{16}$-in. round opening screen. This made for very rich tailings, and it was not until

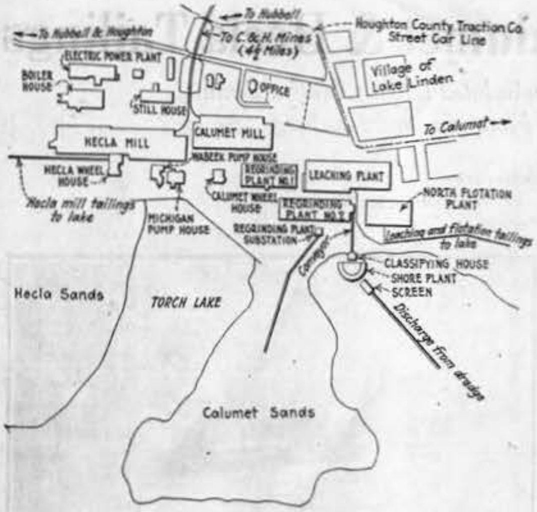


Fig. 1—Map of Calumet & Hecla units at Lake Linden

about 1898, with the introduction of Wilfley tables, soon followed by fine grinding in Chilean and later in Hardinge mills, that any serious attempt was made to reduce the losses of fine inclosed or attached copper. The introduction of this more modern machinery was coincident with a rapid falling off in the quality of the ore sent to the mill, owing partly to the lessening in grade of the deposit underground and partly to the fact that lower-grade ore could be treated economically. A further decided drop in the copper losses followed the installation of a regrinding plant in 1908 and made it appear that the tailings subsequent to that date might not be profitably reworked, although they still contained about 9 lb. of copper to the ton. These tailings were accordingly segregated on one portion of the Calumet pile. With the discovery of the leaching process in 1912 and the adaptation of flotation a few years later it became evident that these apparently worthless tailings were of economic value. In the year just passed the work was almost entirely on these later fine, low-grade tailings, and the curious fact is that while the recovery per ton was less, the cost per pound of copper recovered was as low as on the richer coarse tailings of previous operations. This is because the regrinding cost is entirely eliminated.

That these tailings had commercial possibilities was recognized for a great many years before any effort at recovery was begun. Until the development of modern fine-grinding machinery, coincident as it was in the Lake Superior district with the introduction of the low-pressure turbine and consequent cheap electric power, it was not felt that the time was right for beginning operations. Then, at first, the plans were only for finer grinding and Wilfley table treatment, and it was the expectation that the recovery might not exceed 40 per cent. Even this was attractive, and work was begun in 1912. The regrinding plant was not yet in operation, however, before the leaching process was developed, which promised to double the anticipated recovery. Construction work was started in 1914 on the leaching plant, designed to treat sand, and experimentation was continued on the treatment of the slime by the same process. The rapid development of flotation, however, and its adaptation at the Calumet & Hecla to native copper, made it advisable to discontinue work on the leaching of slime, and a flotation plant was erected for this material. This again increased the recovery, so that about 85 per cent of the values contained in these tailing piles will probably be obtained by the present process. Inasmuch as the recovery on the ore originally was about 75 per cent, over 95 per cent of the metal contents of the Calumet &

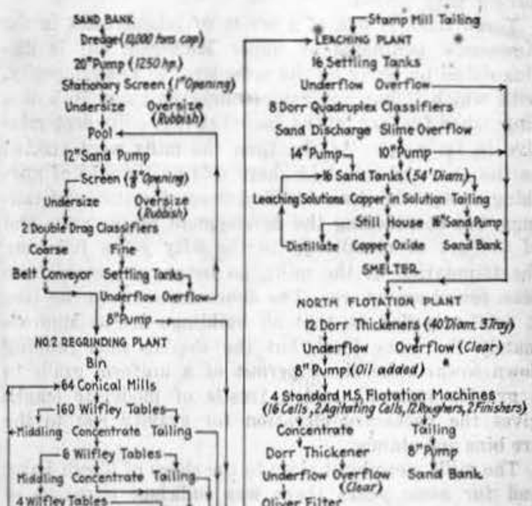
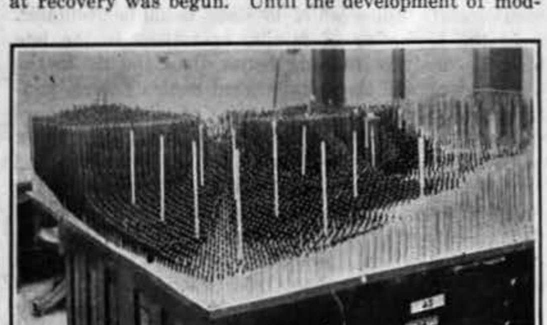


Fig. 2—Flow sheet of reclamation plant

Hecla conglomerate lode as mined will be obtained as refined copper, a record probably unique in the history of any copper-mining operation.

Although the commercial possibilities of these deposits had been recognized long before recovery plants were erected, their treatment was not sufficiently imminent to prevent surface contamination. All the rubbish of the stamp mills and from the adjacent towns was deposited on top of these sand banks, and although this rubbish does not constitute a large percentage of the total weight, it amounts to thousands of tons of every conceivable kind and size of material, and any plan for the reworking of the sand had to take into account a mixture of ashes, hoop iron, wire cable, launder plates, and submerged logs. After five years of operation the suction dredge originally chosen has been in commission without at any time having had serious difficulty in operation.



Model showing status of sands for reclamation plant. The white represents lake bottom, and the dark, reclaimable sands.

The reclamation plant as at present constituted consists of five units separately housed as follows:

1. Dredge.
2. Shore pumping plant and classifying house.
3. Regrinding plant.
4. Leaching and distillation plant.
5. Flotation plant.

Fig. 1 shows a diagram of the units of this plant in connection with the other mill buildings. Fig. 2 is a flow sheet.

1. *Dredge*—It was recognized at once that a suction dredge would be the only possible means of reclaiming these tailings, because of the depth of the deposit and the severity of the climate. It was not feasible to develop a dredge by experimentation on a small scale, because from the very nature of the problem large-scale operation only was possible. No dredge had ever been designed to dig 110 ft. below the water level. A bucket dredge was considered entirely unfit, because of the depth, and also because of the fact that although a bucket dredge might handle rubbish to good advantage, the sharp coarse sand would be very hard upon the innumerable bearings. Further, it was thought impossible to operate such a dredge in winter time in the Lake Superior district, with the thermometer frequently reaching 20 deg. below zero.

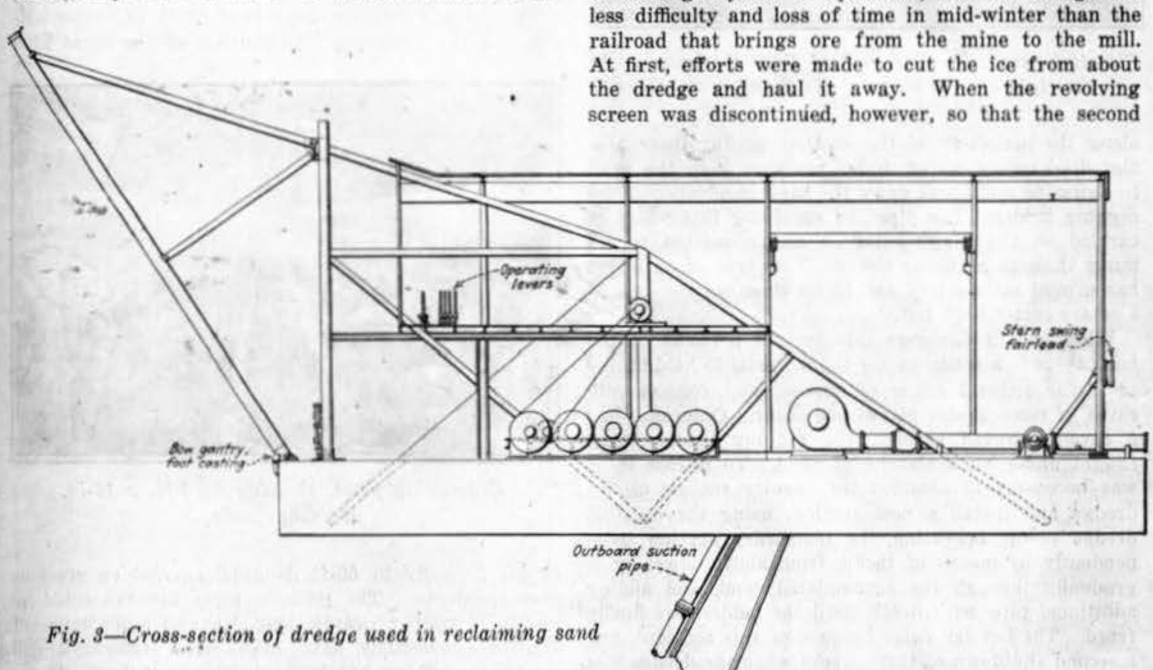


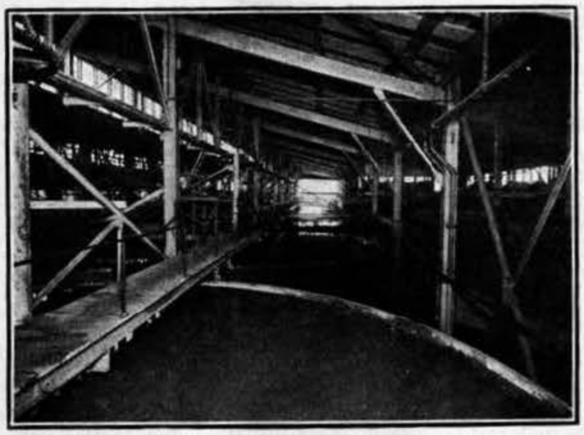
Fig. 3—Cross-section of dredge used in reclaiming sand

The dredge finally adopted was developed with the aid of the Bucyrus Company, of South Milwaukee. It is a steel-hull dredge 56 ft. wide and 110 ft. long, with an overhanging deck 8 ft. wide. As originally installed, having in mind the amount of oversize rubbish to be handled, the dredge was provided with two dredging pumps. The first discharged into a 1½-in. revolving screen for removing rubbish, the undersize going to a sump from which a second pump elevated the material and discharged it through the pontoon line.

After experiencing considerable trouble with the elimination of the oversize rubbish in this way, a method was finally devised of using but a single pump for dredging the sand directly from the pile and discharging it through the pontoon line to a stationary screen at the shore plant. This eliminated entirely the revolving screen on the dredge and the second dredge pump, and resulted in a tremendous saving of power and maintenance. It is surprising what large pieces of timber, rope, and even steel plate can be picked up by the pump and carried through 3,000 ft. of the 20-in. discharge pipe to the receiving pool on shore without choking pump or pipe.

In winter time 2 ft. of ice on the lake is usual, 20 deg. below zero is not uncommon, and usually for two to three weeks the temperature is continuously below zero. However, these conditions have caused no delay in the eight years of operation, and the dredge has less difficulty and loss of time in mid-winter than the railroad that brings ore from the mine to the mill. At first, efforts were made to cut the ice from about the dredge and haul it away. When the revolving screen was discontinued, however, so that the second dredging pump was not required, this pump was connected to a pipe surrounding the dredge and discharged water through a series of nozzles about 10 ft. above the deck of the dredge. The agitation from this water was sufficient to keep the ice from forming freely and also to melt such ice as had been formed adjacent to the dredge. For the last four years it has not been necessary to break or remove ice by other means, and the cost of operating this pump for a few hours each day is small.

Fig. 3 shows a cross-section of this dredge. The dredge pump itself has a 20-in. diameter inlet and outlet, with impellers 55 in. in diameter, operating at 360 r.p.m., and is equipped with a 1,250-hp. motor. The pump casing is split vertically and is lined throughout. Various types of material for liners have been used, and although the best results are obtained with manganese steel, chilled cast iron made in the local foundry is much the cheapest material per ton dredged. In addition to the main dredge pump, and an auxiliary pump for supplying water to prevent ice formation, there is a 4-in. centrifugal pump for service water and an 8-in. centrifugal pump for supplying water at 75-lb. pressure for agitating the sand. The dredge suction is supplied with nozzles reaching out in all directions



Flotation plant, showing row of Dorr thickeners. At the right are the backs of the flotation machines.

along the periphery of the suction mouth; these nozzles discharging water under pressure keep the sand in agitation and break down the bank in advance of the digging ladder. The pipe for supplying this water is carried on the digging ladder and connected to its pump through a rubber sleeve. This type of agitation has proved satisfactory and at no time has the lack of a rotary cutter been felt.

When the dredge gets into certain portions of the bank there is a tendency for the material to hold up beyond the natural angle of repose, and consequently caves of considerable magnitude occur. Only once has a cave-in proved serious, the digging ladder being caught under an avalanche of sand. To release it, it was necessary to abandon the regular suction of the dredge and install a new suction, using the original dredge pump, operating the temporary suction independently by means of tackle from above, lowering it gradually through the accumulated sand, and adding additional pipe with depth until the ladder was finally freed. The ten-day delay because of this accident, and a second shutdown of three weeks when the dredge was dry-docked and the hull scraped and painted, have been the only interruptions to continuous service during eight years of operation.

The suction ladder consists of two longitudinal latticed girders thoroughly fastened and well braced. It is 141 ft. from center of suspension to the end of suction, and will permit of dredging to a depth of 110 ft. It carries the outboard suction of lap-welded pipe and also an 8-in. water pipe for supplying water under pressure for breaking down the sand. The center of the dredge pump is on the same center line as the pivot of the suction ladder, and the suction pipe is connected to the pump by an elbow, swiveled on this same center line. Near the outer end of the suction ladder the lower block of the hoisting tackle is attached, and the upper block is attached to the bow gantry, forming part of the hull.

The dredge is not self-propelling, but is operated by swinging lines fastened to anchors in the water and deadmen along the shore. There are two four-drum winches for operating the lines, two drums of which are for the ladder swing, two for the bow-line swing, two for the stern-line swing, one for the stern line, and one for hoisting the suction ladder. The bow swing lines are ¾ in. diameter, the ladder and stern lines are ⅞ in. and the ladder hoist rope is 1 in. diameter, all plow steel. All pumps and winches are electrically driven, but there is a small boiler on the dredge to provide steam for heat and also for operating capstans when the electric power is cut off.

Electricity is supplied to the dredge at 2,300 volts, the power lines being supported on towers attached to the pontoons carrying the discharge pipes. The main dredging pump motor is 1,250 hp. rating and consumes about 900 kw. when working through its maximum discharge pipe of 3,000 ft. length. An additional 200 kw. is consumed by the 8-in. and 4-in. water pumps. Water rheostats control the speed of the pump with a variation of about thirty revolutions, from a minimum of 335 r.p.m.

Inboard and outboard pipe is of 21 in. diameter outside and the discharge line consists of the same kind

Regrinding plant, showing 64 8-ft. x 18-in. Hardinge mills

of pipe, mostly in 60-ft. lengths, carried on steel or wood pontoons. The pontoon pipes are connected by means of rubber sleeves, and although other types of flexible connections have been tried, especially ball joints, everything has been abandoned in favor of the rubber sleeve. The sleeves are held on by split steel bands, which arrangement, in connection with the beaded ends of the pipe, gives a simple flexible joint, and one that does not cause much trouble.

The pontoon line discharges at a fixed point on the shore into a stationary screen 16x20 ft., with round openings 1 in. in diameter. This discharge is placed so that the sand runs from the screen into a pond or reservoir which supplies the pumps in the shore plant about to be described. Not much rubbish accumulates on the screen, but it may consist of large pieces, including many relics of the early days. A chain 6 ft. long was once removed from the screen; also pieces of cast iron, innumerable rocks, and logs up to 16 in. in diameter.

The dredge has a rated capacity of 10,000 cu.yd. per day, which has been realized. No facilities have ever been at hand to make a definite test of its efficiency, and there is so much variation in the size and nature of the sand to be pumped, the length of discharge line, the condition of the impeller and pump casings, and depth of suction, that a test would be of little value. In the winter about sixteen hours, including delays, are required to do the necessary work, and sometimes during the summer, when the pontoon line is 3,000 ft.

long and the sand is fine and scattered, continuous operation is necessary.

No effort is made to synchronize the operation of the dredge with the plants on shore, which must run uniformly twenty-four hours per day. The capacity of the reservoir in front of the shore plant is such that the dredge can shut down two or three days at a time without the plants being short of sand. A dredge is necessarily intermittent in its operation, even when working under the best of conditions, and under the varying conditions met in this operation there is no possibility of supplying sand in either uniform quantity or dilution.

The general method of operating the deposit and the measurement and control of the work is carefully studied from day to day. Fortunately the United States Government had made accurate soundings in Torch Lake before the mills were operating for any length of time, so that the extent of the deposit both laterally and in depth is accurately known. A sectional model (shown in the illustration) has been made, consisting of vertical wooden pegs fastened into a horizontal board representing a base line. These pegs indicate conditions for each 50-ft. station and by means of different colors show lake bottom, lake level, and recoverable sand. Frequent soundings keep this model up to date, and operators on the dredge know at all times at what depth to expect sand and to what extent. Ranges along the shore and floating buoys assure accurate knowledge as to the position of the dredge, and a section of the model in the pilot house provides complete information as to conditions of deposit. The original lake bottom is fortunately a compact sand that resists dredging with this type of machine, and much of the area is as clear of tailing after dredging as a surface operation could be. In addition to this model, which is changed to show existing conditions, there is a cross-section for every 100 ft. which shows the progress from month to month.

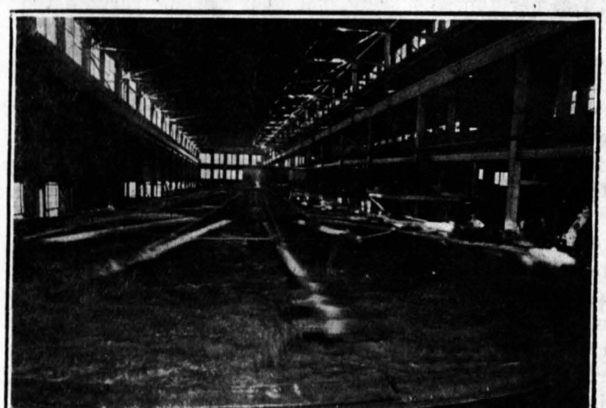
Row of tanks in leaching plant. First tank has launder and distributor in place. Second tank is uncovered. Others have covers in place.

A problem at first was to find room for current tailing. For this reason the outer and more recent deposit has been dredged up to this time. In winter when the climatic conditions are at their worst the dredge is moved close to shore to shorten up the pontoon line and minimize operating difficulties; in summer more distant material is attacked. The operation is from its nature a "one-stope mine," but the quality of the sand does not vary greatly from day to day, although the proportion of coarse to fine does show a wide fluctuation. This results in operating difficulties felt more particularly in the flotation plant.

The cost of this operation for 1923, including all replacements and renewals, was 6.30c. per ton dredged. This is made up roughly of 32 per cent for labor, 14 per cent for pump renewals, 40 per cent for power, and 14 per cent for other supplies. These costs are for an average discharge line of about 2,000 ft., and vary somewhat with the size of sand, length of line, and other factors. The costs are showing a downward tendency from year to year as capacity is being increased and as greater experience is being acquired.

2. *Shore Pump Plant and Classifying House*—This plant is built upon a concrete dock constructed by driving piles through the sand down into the original lake bottom and upon these piles putting a cap of concrete 3 ft. 6 in. thick. The shore plant contains a 12-in. Morris centrifugal pump for the elevation of the sand, stationary screens for removing fine rubbish, drag-belts for separating the coarse sand that requires regrinding, from the fine, pumps for handling this fine sand, and a belt conveyor for conveying the coarser sand from this plant to the top of the regrinding plant. In the pond or reservoir in front of the shore plant into which the dredge sand is discharged is a semicircular row of piles having a radius of 35 ft. on which is constructed a track. Supported on this track and pivoted in front of the plant is a structural steel bridge 55 ft. long, which carries the suction pipe of the pump in the shore plant. This suction pipe works on swivel joints, so that it can travel through an angle of about 150 deg. and its outer or suction end can be raised or lowered vertically through an angle of 90 deg. The effect of this is to get a storage capacity for this suction pump in the shape of a "V" section along about one-half of the circumference of the 55-ft. circle and to a depth of 30 ft. below the water line. Thus is obtained a storage reservoir equivalent to about 20,000 tons' sand capacity, from any part of which the sand may be reclaimed, depending upon the position of the suction pipe carried on this swinging bridge. In reality it is a stationary dredge working under uniform conditions as to length of suction line, so that uniform capacity is obtained and the re-treatment plants are kept operating under uniform conditions, a necessity for efficient metallurgical practice.

To guard against delays in the shore plant, the original design called for two pumps so placed that



Leaching plant, showing tank filling

by swinging a suction elbow through 180 deg., one pump could be replaced by the other. Delays have been so infrequent, however, that such changing of parts as is necessary can be done during the regular six-hour shutdown once a week, so that only a single pump has been necessary. This pump has a 12-in. suction and discharge, is split horizontally and lined throughout. The impeller is 40 in. in diameter, running 375 r.p.m. direct connected to a 200-hp. motor. The discharge of this pump goes into a large receiving box fitted with an overflow for the excess water, and as this water carries considerable fine sand it is returned to the storage pool. Circulation of water is very useful in winter, as the agitation caused thereby aids in keeping the pool free from ice. The receiving box is provided with four openings, gate controlled, through which the sand and water flow to stationary screens with $\frac{3}{8}$-in. diameter openings. These screens are for the purpose of removing the finer particles of rubbish. The undersize of these stationary screens is fed to two double drag-classifiers, each fitted with two 20-in. belts, these belts being provided with 6x4-in. angle

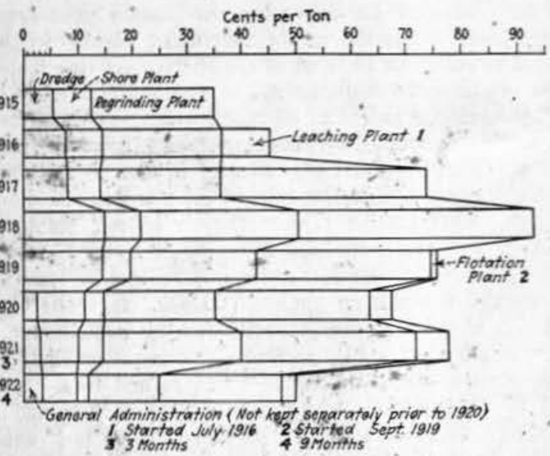


Fig. 4—Reclamation plant costs, 1915 to date

irons, 24 in. long, as drags. The discharged product of these drag belts is fed by chutes to a 22-in. belt conveyor, and the overflow, after suitable dewatering, is pumped directly to the same sand- and slime-classifying system that takes also the tailing of the regrinding plant.

The cost of operation of this plant for 1923 was 2.47c. per ton, including belt conveyor delivery to the regrinding plant. Of this cost 45 per cent was for labor, 30 per cent for power, and 25 per cent for supplies, chiefly pump repair parts and pipe.

3. *Regrinding Plant*—The material treated in the regrinding plant is the coarse sand classified out by the drag-belts in the shore plant and fed to the top of the plant by means of a belt conveyor. This belt conveyor is 275 ft. between centers, runs at a speed of 500 ft. per minute, and has an inclination of $2\frac{1}{2}$ in. to the foot. It discharges into a receiving bin, which has sufficient storage to supply the mills for about thirty minutes. In the bottom of this receiving bin are discharge openings, from which the relatively dry sand is run at a uniform rate and fed by means of water jets into launders running down either side of the plant and discharging into dewatering boxes for feeding the conical mills.

The building itself is 122x431 ft. and contains 64 Hardinge mills, 8 ft. x 18 in., driven individually by 40-hp. motors connected by means of flexible couplings to herringbone pinions driving corresponding herringbone gears. Each mill revolves at 26 r.p.m., is lined with either Belgian silex or domestic quartzite, and uses flint pebbles for the grinding medium. The conical mills were among the first built by by the Hardinge Company and are low in capacity. At a corresponding plant now being built for the Tamarack sand, Hardinge mills are being used as before, but they have a cylindrical length of 6 ft. instead of 18 in., are driven with 100-hp. motors and give about three times the capacity of the shorter mill. These conical mills and motors are carried on a structural steel framework or floor about 12 ft. above the Wilfley table floor. The mills are in two rows of thirty-two each and are served by a fifteen-ton traveling crane which can pick up a full mill. All relining is done at the end of the plant, where piers are provided for this purpose. By the use of two extra mills it is usually possible to take a worn-out mill and replace it within an hour by a newly lined spare.

The mills are fed by a gate in the overhead launder handled by a lever from the conical-mill floor. The material cut out in this manner is run into a dewatering box, from which a plug discharges the thickened product into the feed scoop of the conical mill. The discharge of the mill is run into a distributing launder and fed directly to Wilfley tables, the product of two mills going to five Wilfleys. The concentrates from the Wilfleys are reconcentrated on other tables, and the final concentrate is pumped into elevated bins for dewatering, afterward fed by gravity into concentrate cars. Table middlings are returned to the conical mills for finer grinding. The tailing from the Wilfley tables joins the fine product from the drag-belts in the shore plant, and this combined material is pumped by means of a 16-in. centrifugal pump to a classifying section in the leaching plant where sand for leaching is separated out from the slime for flotation.

Special Considerations Govern Grinding Practice

This regrinding plant has a capacity of about 3,000 tons per twenty-four hours. The capacity per mill is low and the grinding efficiency is not the best, but efforts to improve conditions have not been successful. All grinding is single-pass, which would not seem to be according to best practice, but such experimentation as has been done with closed-circuit grinding on native copper ores, both in this plant and in other sections of the district, has not met with great success. A little thought as to the difference between native copper ore with its flat metallic particles compared to more friable crystalline ores met with in other metallurgical fields will be enlightening. With ordinary ores, in closed-circuit work the particles finally overflow the classifier or pass through a screen of the size adopted and thus are ready for further metallurgical treatment. On native copper ores, however, the very particles which are to be eliminated from the circuit as soon as possible are the ones that resist comminution. The result is that the native copper builds up in the circuit to an alarming extent—so much so, in fact, that the abrasive loss due to the sliming of this concentrated copper becomes a serious question in subsequent treatment, particularly as flotation recovery is not so satisfactory as it might be.

The use of steel liners and balls also comes to mind, but they have not proved advisable. The conglomerate ore probably resists comminution to a greater extent than any other ore treated in this country. Tests indicating this have been made by various independent investigators, notably Lennox, who found for this ore a "comparative crushing resistance" of 1.33 as against a low of 0.37 on Ray ores and 0.38 on Utah. For this reason consumption of pebbles is large—about five pounds to the ton—and such steel and cast iron balls as have been tried show a loss almost as great. With pebbles delivered at ¾c. a pound, amounting to 4c. per ton of sand crushed, and with steel costing 3½c. per pound, or 14c. per ton of sand crushed, there has never been any inducement to add the necessary equipment for changing over to steel balls in view of an increased operating cost of 10c. per ton.

The cost of operation for this plant, in cents per ton, for 1923 was as follows:

General expense		1.82
Sand conveying and distribution		1.86
Grinding		25.27
Attendance	1.37	
Power	13.63	
Pebbles and lining	9.38	
Other supplies	.89	
Table treatment		4.00
Total		32.95

Metallurgical results for the same year follow:

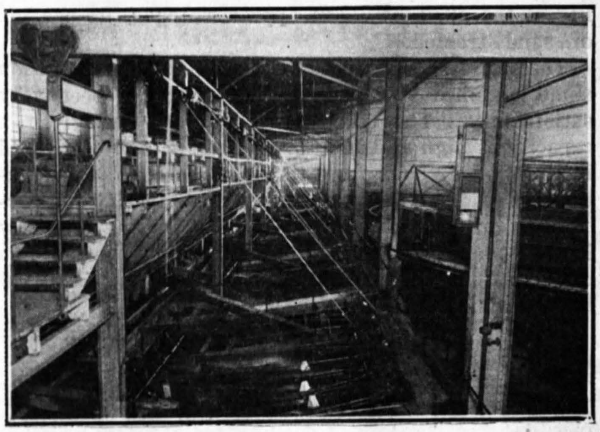
Tons treated	866,524
Assay feed, per cent copper	0.723
Assay tailing, per cent copper	0.468
Pounds refined copper	4,458,100
Pounds copper per ton sand	5.14
Cost per pound copper, excluding smelting and selling, cents	6.40

4. *Leaching Plant*—An article describing this plant appeared in the *Engineering and Mining Journal* of July 14, 1917, written at a time when the plant had been in partial operation for about a year. Since the publication of that article the plant has been doubled in size and the cycle time decreased, so that a daily capacity of 6,000 to 7,000 tons has been reached for months at a time, this tonnage depending more upon the capacity of the plants preceding the leaching process than of the leaching plant itself.

Time of Leaching Now Reduced

As at present installed, the leaching plant has sixteen leaching tanks in two rows of eight each, each tank having a capacity of 1,000 tons of sand. Originally the cycle was four days in length, but changes in strength of leaching solution and plant improvements have cut this down to as low as forty-eight hours, which would permit of a tonnage of 8,000 tons per twenty-four hours.

The material entering this plant consists of a combination of the fine material classified out by the drag belts at the shore plant, of the tailing of the regrinding plant, and the current fine tailing from the stamp mills. As this material enters the plant it is led into sixteen "V" shaped settling tanks each 19½ ft. long, 10½ ft. wide, and 6½ ft. deep. The overflow from these settling tanks contains upward of 95 per cent minus-200-mesh material. The thickened product from these tanks is drawn off by means of plugs to eight quadruplex Dorr classifiers the slime overflow of which joins the original overflow from the "V" tanks after thickening and is treated by flotation. This overflow contains about 93 per cent minus-200-mesh material. The sand discharge from the classifiers is treated by leaching, and although it contains about 15 per cent of minus-200-mesh product, is comparatively free from

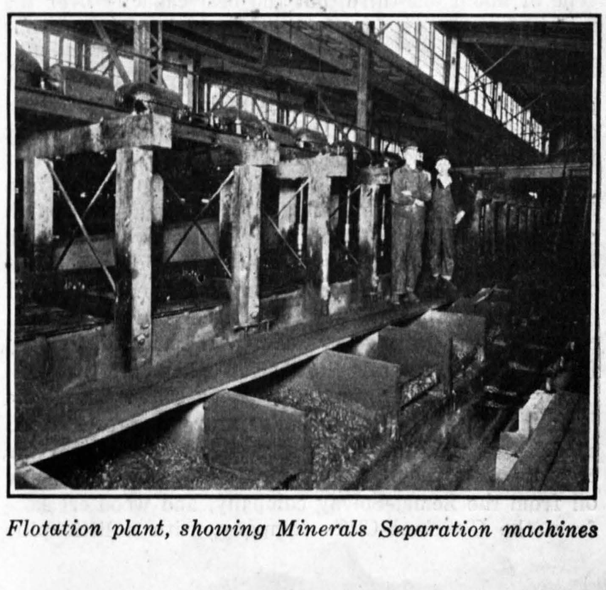


Leaching plant, showing "V" settling tanks, Dorr quadruplex classifier, and leaching tanks with covers in place. Colored lights against column in foreground are signals flashed from plant to plant to show load conditions.

colloids. A characteristic sizing of the feed and tailing of this plant with assays is given below:

	Per Cent Total Material	Assay Feed, Per Cent Copper	Assay Tailing, Per Cent Copper	Per Cent Recovery
On 28 mesh	2.3	0.561	0.283	49.6
On 48 mesh	11.4	0.499	0.140	71.9
On 100 mesh	39.0	0.466	0.099	78.8
On 200 mesh	29.4	0.454	0.092	79.7
Through 200 mesh	17.9	0.966	0.139	85.6
Total	100.0	0.558	0.111	80.1

Originally it was believed that the feed to this plant would not contain over 10 lb. of copper per ton. This may possibly be the average over the entire life of the deposit in question, but occasionally the feed has run as high as 14 lb. per ton. The solutions originally used were very dilute because it was felt that the loss of ammonia would be directly proportional to the strength of solutions used or at least the stronger solutions would require considerable washing in order to free the sand of the dissolved copper and absorbed ammonia. This general fact was found to be true, but, on the other hand, it was possible to increase the strength of solution and thereby decrease the volume required—in other words, decrease the time of the cycle—and by other means to keep the ammonia loss at a low figure. The consumption of ammonia over the



Flotation plant, showing Minerals Separation machines life of the plant has been approximately ½ lb. per ton of sand treated.

The classification between leaching and flotation slime, obtained by the Dorr classifiers, has been very satisfactory. The quadruplex classifier has given a capacity as high as 1,100 tons of sand actually delivered in twenty-four hours, and it is not sensitive to fluctuations of load. When working at this capacity the percentage of plus-200-mesh material in the overflow increases and the Dorr thickeners give considerable difficulty, but the classification for leaching is at all times satisfactory.

The cost of this leaching operation for 1923 in cents per ton follows:

General expense	5.09
Sand classification and distribution	3.49
Leaching	13.93
Distillation	11.73
Total	34.24
Ammonia (included in above)	8.68

The metallurgical results for 1923 were:

Tons treated	1,664,130
Assay feed, per cent copper	0.513
Assay tailings, per cent copper	0.105
Assay oxide, per cent copper	82.32
Pounds refined copper	13,625,000
Pounds copper per ton sand	8.19
Recovery, per cent	79.5
Cost per pound copper, excluding smelting and selling, cents	4.18

5. *Flotation Plant*—The feed to the flotation plant consists of the overflow of the "V" tanks mentioned above and of the Dorr classifiers. It is very dilute and subject to wide fluctuations in quantity and accordingly in dilution. The Dorr thickeners do not respond readily to fluctuations in feed, and overloading, with its attendant difficulties, is frequent and a source of inefficient operation.

The plant comprises settling units consisting of twelve three-tray Dorr thickeners with diaphragm pumps, four 16-cell 24-in. impeller Minerals Separation flotation machines, a 25-ft. Dorr thickener for concentrates, an 8x8-ft. Oliver filter, and the necessary incidental pumps and compressor.

The thickeners are in two rows of six each, with the feed launder in the center and an adjustable gate in the launder at each machine. The first eight thickeners were of the open type, but four more were required and these are of the connected type. The latter are much heavier in construction and far superior for the fluctuating conditions to which they are subjected. They show an increased capacity over the open type of about one-third, but the best capacity over the twenty-four hours for the twelve thickeners is about 1,800 tons.

The pulp is thickened to a consistency of about three parts of water to one of solids and pumped by means of an 8-in. centrifugal pump to a distributing box feeding the four flotation machines. The flotation oils are added at this same pump. The pulp is fed into the third cell of the machine, and this and the fourth cell are used for agitation only. Cells five to sixteen inclusive make middlings, which is returned to cells one and two for final concentration, the tailings of these joining the original feed at cell three. A final cleaning up of tailings is made by a series of air cells following the Minerals Separation machines.

For flotation a mixture of various coal-tar products is found most effective. The mixture at present used consists of coal tar from a local gas plant, coal tar creosote from the Barrett company, a residual coal tar oil from the Semet-Solvay company, and wood creosote from the Cleveland-Cliffs company, with a little pine oil added as required for frothing. Special flotation reagents have shown no advantage over the oils mentioned. Neither heat nor acid is found necessary, and the consumption of oil is about 1½ lb. per ton of slime treated.

The plant is very compact and efficiently operated, two men only being required for shift work. The extraction is low, about 65 per cent, but native copper does not float so readily as the sulphide, and little that is coarser than 200 mesh is recovered. A characteristic sizing of feed and tailing is as follows:

	Per Cent Total Material	Feed Assay, Per Cent Copper	Tailing Assay, Per Cent Copper	Per Cent Recovery
On 200 mesh	5.97	0.289	0.243	15.90
Through 200 mesh	94.03	0.516	0.157	69.60
Total	100.00	0.502	0.162	67.80

The cost of flotation, in cents per ton, for 1923, with metallurgical data, follows:

General expense	2.78
Slime conveying and distribution	2.80
Flotation	5.47
Royalty	4.49
Total	15.54
Feed to machines, per cent copper	0.453
Tailing of machines, per cent copper	0.164
Concentrates, per cent copper	28.58
Pounds refined copper	2,135,500
Pounds copper per ton slime	5.76
Recovery, per cent	63.89
Cost per pound, excluding smelting and selling, cents	2.70

Summary of Results

The details of operation of the various plants as given above are for the individual units, and the costs and metallurgical data for the leaching and flotation plants include figures pertaining to current stamp-mill product treated in those plants. The portion of the product from these plants to be credited to current mine production is arrived at by difference of assay of feed and tailing. All material is treated finally by leaching or flotation, and as the leaching is a batch operation, each tank containing 1,000 tons, the weight of this product is accurately determined. The weight of material treated by flotation is determined by sampling, the difference between feed and tailing assay, divided into copper recovered, giving the tonnage.

In addition to the cost of the individual units of the reclamation plant as given above, this department bears its proportion of total administrative costs at mine and mill based on number of men employed. For 1923 the complete cost of this operation in cents per ton of sand treated was as follows:

General administration and miscellaneous	4.4
Dredge	6.3
Shore plant	2.5
Regrinding	16.4
Leaching	25.1
Flotation	3.4
	58.1

Tons treated	1,743,100
Assay feed, per cent copper	0.608
Assay tailing, per cent copper	0.124
Copper produced, pound	16,901,200

The total tonnage reclaimed from the beginning of operation up to Jan. 1, 1924, was 7,955,500 tons with a copper recovery of 82,102,924 lb., being 10.32 lb. to the ton, obtained at an operating cost of 6.32c. per lb. As these tailing piles were constituted at the beginning of operation they contained 46,683,000 tons of conglomerate tailing, of which 34,470,000 tons was estimated as available for treatment, with the probabilities that the final figures would exceed this estimate. It is evident that this deposit will constitute a profitable operation for many years and an important source of revenue to the Calumet & Hecla Consolidated Copper Co.

Figure 1. 40 ft. thickeners in north flotation plant, Lake Linden.

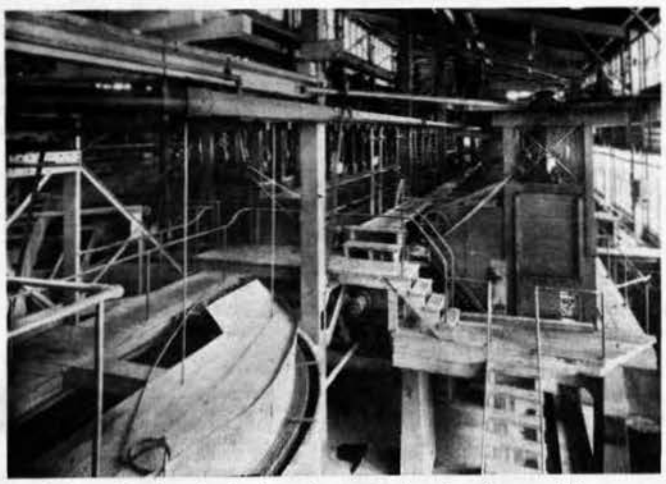


Figure 2. Standard minerals separation flotation machines, north flotation plant, Lake Linden.

Figure 4. Fahrenwald flotation machines, Ahmeek mill.

FLOTATION

at the

By

FLOTATION was introduced at the Calumet & Hecla in 1918, at which time the first plant was put in operation to treat the fines from the stamps in both the Hecla and Calumet mills. The following year the North Flotation plant was completed and began handling the slimes from the reground lake sand together with that from the mills. Five years later two more flotation machines were added to those originally in this plant and the installation treating the primary slimes was then shut down.

The Tamarack reclamation plant was put in operation in 1925 to treat the old Tamarack sand. This plant is equipped with four 40-ft. diameter three-tray Dorr thickeners and two flotation machines to handle the slimes. Xanthate was adopted about this time in place of the coal tar oils theretofore used.

Three years later the Fahrenwald flotation machine was tried out and adopted for treating the amygdaloid sands which were too coarse for the old standard Minerals Separation machine to keep in suspension. Fahrenwald machines were installed at the Ahmeek, Isle Royale, and Lake Milling Company mills to treat the fines from the stamp together with the reground jig middling. Last year at the Ahmeek mill the practice of regrinding and floating the entire stamp product was begun.

Conglomerate Flotation Practice

The ores treated in the Calumet & Hecla stamp mills are of two very different types, (1) conglomerate and (2) amygdaloid. The former comes from the Calumet conglomerate and the latter from the Kearsarge and Osceola amygdaloid lodes. The conglomerate ore is noted for its hardness and carries a large part of its copper in such finely disseminated form that grinding through 200 mesh does not liberate it very thoroughly from the gangue, while on the amygdaloid, crushing through 48 mesh is sufficient to give a satisfactory extraction. Curves plated from the screen analyses of flotation tailings from these ores show this clearly.

As a consequence the conglomerate tailings are separated into plus 200 and minus 200 mesh products and only the later is floated, the coarser sand being leached with ammonia.

The north flotation plant handles the conglomerate slimes. The feed is derived mainly from two sources, (1) primary slime from the stamps, (2) secondary slime from the pebble mills in the regrinding plants. Most of the latter comes from the reclaimed sand from Torch Lake. About 1,000 tons per 24 hours of primary slime is treated and 1,400 tons of reground sand together with a highly variable amount of reclaimed slime from the lake. The pri-

* Metallurgist.

PRACTICE

Calumet & Hecla

Robert M. Haskell*

mary slime when treated by itself gives a decidedly higher tailing and richer concentrate than the reground sand (—200 mesh), but treating the combined product has given a better final result than handling them separately, partly because the feed is more steady and uniform in quality as well as in quantity.

The secondary slime which is derived from the leaching plant classification system comes into the plant at a density of approximately 2 percent solids and is divided between twelve 40-ft. diameter, three-tray Dorr thickeners, of both the open and connected types. (*Figure 1*). The thickened underflow at 25 percent solids is plugged off into an 8 in. direct connected Morris sand pump from the four thickeners nearest it and by means of 4 in. quadruplex diaphragm pumps from the others. To avoid the use of additional reagent feeders and obtain maximum conditioning, Xanthate, lime and pine oil are added at the feed pump. In addition to this feed, the slime from each head in the stamp mill is thickened in a 25 ft. diameter three-tray tank and the combined slime pumped over to the 8 in. feed pump which elevates the pulp to a distributor located between the flotation machines and is split between four of them. There are a total of six 16-cell, 24-in. standard Minerals Separation machines of the splash type, two usually being held as spares. (*Figure 2*). The feed enters the fourth cell, which is used for mixing only. From the remainder of the machine a rough concentrate is skimmed off which is elevated in a bucket elevator to the head of the machine and cleaned in the first three cells. This concentrate from all four machines is combined in another elevator and sent to the latter part of that machine nearest the concentrate thickener, for recleaning. This portion of the machine is blanked off from the rest and consists of five cells. The final concentrate flows by gravity to a 25-ft. diameter concentrate thickener. Two pneumatic machines of the Inspiration type, each consisting of seven cells 3 ft. by 4 ft. 6 in. in size, act as scavengers on the tailing from the Minerals Separation machines and make a low grade middling which returns to the feed pump. This operation lowers the tailings about .01 percent copper, although with the old coal tar reagents this saving was twice as great. The tailing flows to a second 8 in. pump which elevates it to the tailing bank in Torch Lake. This flow sheet is shown in figure 3. (B. P. 9602).

The underflow from the concentrate thickener at 60 percent solids is pumped by means of a 2 in. pressure diaphragm pump to an 8 ft. by 8 ft. Oliver filter. The filter cake carrying 13 percent moisture drops through a chute into a 50-ton hopper-bottom concentrate-car for shipment to the smelter.

The summary of operations at the North Flotation plant for the year 1930 follows:

Tons treated	625,760 tons
Copper recovered	5,427,000 lbs.
Assay feed	.527
Assay tailing	.105
Assay concentrate	35.55
Recovery	80.3

Costs

General		1.2c
Pumping and thickening		1.9c
Flotation:		
Attendance	.6c	
Power	1.3c	
Reagents	1.8c	
Repairs	.6c	4.3c
Royalty & Miscellaneous		1.1c
Total		8.5c

Reagents

In the beginning a crude pyridine oil obtained as a by-product from the manufacture of ammonia from certain by-product coke oven plants proved to be the best collector. A few gas-plant coal-tars and creosotes were satisfactory, more especially as stiffeners for the pyridine. Wood creosote together with some pine oil were used as frothers. The recovery was about 70 percent and grade of concentrate 25 to 30 percent copper. Xanthate increased 10 to 15 percent over previous results and the grade of concentrate to 35 to 40 percent copper. The reagent consumption is as follows:

Sodium xanthate	.05 lbs. per ton
Pine oil	.15 lbs. per ton
Lime	.30 lbs. per ton

The pine oil used is a mixture of 75 percent G. N. S. No. 5 and 25 percent Cleveland Cliffs No. 2 wood creosote at the feed pump. A. T. & T. No. 11, a destructively distilled pine oil, is used in place of wood creosote in the mixture for the drip cans on the machines. The drip cans are allowed to stand a day or so before use, which allows any sediment to settle out, thus preventing choke-up troubles.

The ores of this district are all more or less alkaline due to the presence of considerable lime and no sulfides. Even the lake water has a pH content of 7½ and the water in the flotation feed is usually about 8, so little alkali is needed. As the lime drops the iron oxide, which the wood creosote tends to float, and also decreases the pine oil required, a little of it is still used, although more than half of the total is added to the thickeners to aid in settling the slime. The lime is slacked in boiling water at 4 to 1 in an altered form of Pachuca tank with a coarse screen over the bottom, and then diluted to 10 to 1 before use. This gives a very high percentage of lime in soluble form. Diaphragm pumps are used for lime feeders.

Of all the different xanthates so far tried butyl-xanthate alone has proved superior to ethyl-xanthate, but the additional recovery is hardly enough to warrant the extra cost of reagent. Potassium-ethyl-xanthate was used at first, but has been superseded by sodium-xanthate, which is equally efficient. Amyl-xanthate is decidedly inferior. Phosphocresylic acid and some of its compounds are fairly satisfactory.

Preliminary work in the laboratory on amygdaloid ore indicated that soda-ash was better than lime, but after experimenting with it in various amounts in the mills its use was found to be unnecessary and it was discarded with no ill effects. The only reagents used are:

Sodium-ethyl-xanthate	.05 lbs. per ton
Pine oil	.15 lbs. per ton

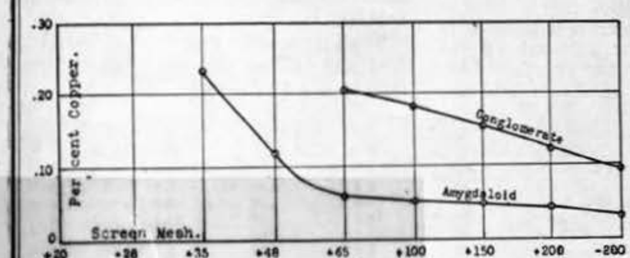
G. N. S. No. 5 pine oil containing 3 percent A. T. & T. No. 11 pine oil is used. The oil consumption is higher here than in western practice, as it is impractical under local conditions to settle the mill tailings and reuse the water.

The ordinary pulley-type reagent feeder is used in the mills because it is simple, foolproof, and inexpensive. The summers are cool and without great temperature variation between day and night as in the southwest and the mills are steam-heated for about seven months in the year so that the viscosity of reagents is not changed sufficiently to alter the rate of oil feed and affect the flotation circuit.

Flotation on Amygdaloid

With the amygdaloids it is relatively easy to separate the copper from the gangue so that crushing through 48 mesh gives a final tailing assay of .02 to .05 percent, depending upon the lode and its grade.

Where regrinding and flotation of the entire stamp product is practiced the



Screen analysis of Conglomerate and Amygdaloid flotation tailings

Curve showing early recovery of copper

362

CALUMET AND HECLA CONSOLIDATED COPPER COMPANY

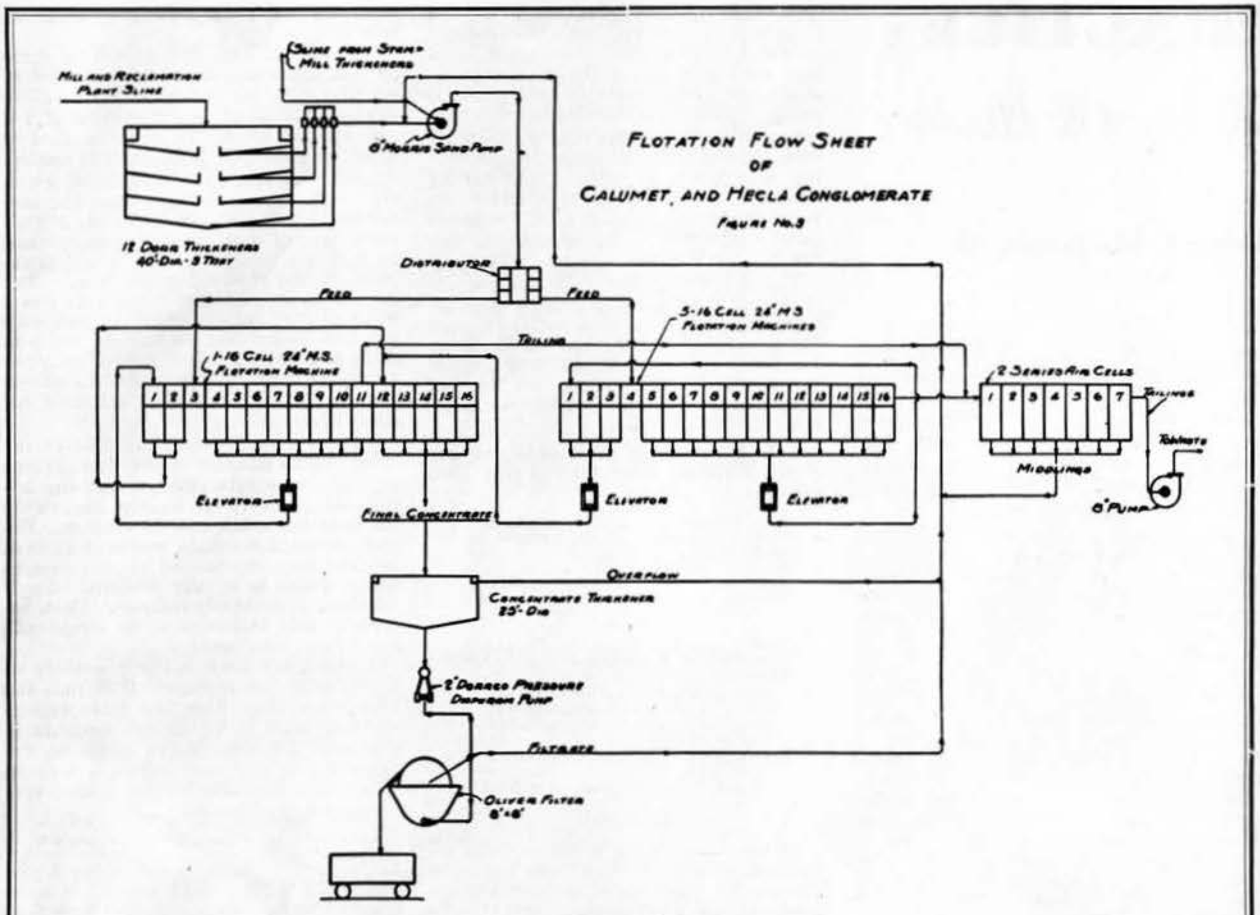


Figure 3

reagents are added to the ball-mill feed. The pulp density overflowing ball mill classifier is 40 percent solids, but this is lowered to 30 to 33 percent solids for flotation.

Of the 800 tons of rock stamped per head about 620 tons goes through the Fahrenwald machine (*Figure 4*) and most of the balance (primary slime overflow) goes to six 25-ft. diameter 3-tray thickeners in the mill basement, the thickened slime being elevated by means of six 4 in. diaphragm pumps to a 40-ft. Forrester flotation machine, which does very good work on this class of material.

The flotation circuit is simple. The feed enters the first cell of the Fahrenwald machine and a final concentrate is taken from the first two cells. From the remaining cells a rougher concentrate is made which flows by gravity into a side passage in the first cell directly over the impeller, thus avoiding the use of an elevator. This coarse concentrate averages nearly 60 percent copper, so retreatment is not necessary. Increasing the grade of concentrate much above this point increases the tailing loss materially. As regards operating conditions on the flotation machines it is advisable to carry a very high pulp level and comparatively small amount of froth for best results in saving the coarser particles of copper. Flotation of most of the copper is rapid, as the following curve shows:

Some of the heaviest particles lag behind and are recovered only in the final cells.

The flotation tailing flows to a 4-ft. 6-in. Dorr duplex classifier which makes an overflow averaging about .04 percent copper that amounts to 80 percent by weight of the tailing and is discarded. The sand product from the classifier is split between four Wilfley tables upon which a small amount of copper is recovered that has proven to be too heavy to float. The tables make a middling which is cleaned up on a finisher table, and the table tailings are combined with the classifier overflow as a final tailing to be pumped to the lake. The tables save about a pound of copper per ton treated, which lowers the flotation tailing approximately .01 percent copper. The table concentrate is high grade. As only a small amount of coarse sand is treated any change in the flotation circuit is magnified so that these tables make exceptionally good pilots.

The flotation concentrate is pumped to a drag classifier for removal of the coarse copper. The drag overflow flows to a 30-ft. traction thickener and the thickened product is elevated by means of a 2-in. pressure diaphragm-pump to the classifier discharge in order to wash the heavy copper into the filter. This latter is a 6 ft. by 4 ft. Dorrco fitted with a short belt conveyor for discharging the filter cake into a concentrate bin which can be emptied into 50-ton hopper-bottom cars for shipment to the smelter. The filter cake carries about 11 percent moisture and the dry assay is 55 to 60 percent copper.

The filter is run with an overflow which was at first returned to the concentrate thickener. This caused the slime to build up and make a dirty overflow. Later the filter overflow was returned to the Forrester flotation machine, resulting in a better grade of concentrate without raising the flotation tailing. The overflow of the concentrate thickener under these conditions is clear and remains so when operating conditions are normal.

Typical screen analyses of flotation feed and tailing and final tailing after tabling are as follows:

Mesh	Flot. Feed		Flot. Tailing		Final Tailing	
	Percent Wt.	Assay	Percent Wt.	Assay	Percent Wt.	Assay
+35	7.9	.50	8.4	.45	7.7	.23
+48	9.0	.50	10.0	.20	9.0	.12
+65	14.0	.52	14.7	.08	15.1	.06
+100	17.1	.46	17.1	.05	17.3	.06
+150	6.9	.42	5.6	.05	5.7	.05
+200	4.6	.44	3.1	.05	6.6	.04
—200	40.5	.19	41.1	.03	38.6	.03
Total	100.0	.37	100.0	.09	100.0	.07

CALUMET AND HECLA CONSOLIDATED COPPER COMPANY

MS-002
Box 40
Folder 8

Tamarack Reclamation – Steam Main

STEAM LINE FROM AHMEEK MILL TO TAMARACK RECLAMATION

Length of Post Supports.
Number of Supports required.

Length given is from ground to center line of 8" steam pipe.

Post Number	Length	Post Number	Length	Post Number	Length	Number of Posts	Length
1	10½'	40	12½'	79	8'	16	3'
2	10½	41	12	80	8	20	4
3	10½	42	12	81	8	19	5
4	3	43	11½	82	7½	17	6
5	3	44	11	83	7½	17	7
6	3	45	10½	84	7	9	8
7	3	46	10½	85	7	1	10
8	3	47	10	86	6½	6	11
9	3	48	3	87	6	3	12
10	3	49	3½	88	6	4	13
11	3½	50	3½	89	6	1	14
12	3½	51	4	90	6	1	15
13	4	52	4	91	6	1	24
14	4½	53	4	92	5½	1	25
15	4½	54	5	93	5½	116	
16	4½	55	5	94	5½		
17	5	56	5½	95	5½		
18	4½	57	5½	96	5		
19	4½	58	5½	97	5		
20	4½	59	5	98	5		
21	4	60	4½	99	5		
22	3½	61	4½	100	5		
23	3½	62	4	101	4½		
24	3	63	4	102	4		
25	3	64	4	103	4		
26	2½	65	3½	104	7		
27	2½	66	3½	105	7		
28	2½	67	3½	106	7		
29	2½	68	4½	107	7		
30	2½	69	5½	108	7		
31	3	70	6	109	7		
32	3½	71	6	110	7		
33	23½	72	6	111	7		
34	24½	73	6	112	7		
35	14½	74	6½	113	7		
36	13½	75	6½	114	7½		
37	13	76	7	115	7½		
38	13	77	7	116	8		
39	13	78	8				

No 1 nearest to Ahmeek Mill Boiler House

Req. #11744
Job #C-1297

Box 200 - Folder 25
L "Milling & Concentrating"

C
O
P
Y

Lake Linden, Michigan

September 6, 1950

Mr. H. L. Williams, Production Manager
Calumet and Hecla Consolidated Copper Company
Calumet, Michigan

Dear Sir:

In addition to the conglomerate slime being treated at the Lake Linden Reclamation Plant, the Company has four amygdoloid sand banks. 1) The remaining Osceola sands near the coal dock. 2) The Osceola Tamarack sands being reworked by the Tamarack Reclamation Plant. 3) The Ahmeek Mill tailing bank. 4) The Point Mills tailing banks.

Osceola Sand Bank

The Hecla and Calumet Mills treated 14,371,000 tons of amygdoloid rock between 1900 and 1925 or 1926, after which all of this rock was stamped at the Ahmeek Mill. During 1947 and a year or so previously the Lake Linden Reclamation Plant treated a total of 3,736,000 tons of amygdoloid sand, and for some years previous to this while the dredge was handling material close to the border line of the conglomerate and amygdoloid sands, a large amount of amygdoloid was treated along with the conglomerate, probably at least 1,500,000 tons. So there is only 8 or 9 million tons remaining and a considerable portion of this settled far out in the lake beyond reach of the dredge. Also the lake gradually gets deeper to the south and east attaining a depth of 130 feet or more, while the dredges have a maximum digging depth of only 110 feet. The engineering department estimated the island of sand near the coal dock which is within reasonable dredging distance at 1,250,000 tons so the total amount of sand that is possible to recover might be 4,000,000 to 5,000,000 tons.

To treat this material it will be necessary to revamp the Regrinding Plant and replace the conveyor, winch, and 12" pump taken from the shore plant. Due to the increase in length of discharge line, it will be necessary to put a 20" booster pump on a scow 1/3 of the distance from the dredge to the shore plant. In addition, it will be necessary to purchase 3000 feet of 22" dredge pipe at about $10 per foot and probably a dozen more pontoons. Last year we sold six to Quincy.

During 1947 which was the last year the Lake Linden Plant handled amygdoloid it treated 997,000 tons less 18,000 of Calumet dam conglomerate, or 979,000 tons and produced 3,846,636 pounds of copper less 400,000 from the dam, or 3,446,636 pounds equal to 3.52 pounds per ton, at a cost of twenty cents per pound. The feed assayed .244 and tailing .068. The original Hecla Mill tailing averaged only 3½ pounds per ton so the remaining sands will probably decrease as we work south and will likely run only 4 to 4½ pounds. This would not, be commercial under our present flow sheet, but if #2 Regrinding Plant were changed over like Tamarack it might be profitable, as the Osceola rock is soft and the copper coarsely disseminated so coarser grinding than is practiced at Tamarack would be practical.

If sink and float is perfected so as to treat material down to 10 mesh satisfactorily, it would change the picture entirely. By putting a sink and float plant on the dredge and discarding fifty to sixty-six per cent of the sand at that point, we could double the grade of sand treated and have a really profitable operation. This would apply to the other sand banks as well.

Tamarack Osceola Sand Bank

There were 29,342,000 tons of amygdoloid sand assaying .231% copper deposited in this bank. At least 1,000,000 tons of amygdoloid was leached along with the conglomerate and since the new plant started we have treated over 4,600,000 tons more. This leaves approximately 23,700,000 tons remaining, but a considerable part of the bank to the south and east is covered with new tailings which it would not be profitable to treat and in addition, there is a considerable tonnage which lies too deep for the dredge to reach. Also, there is a large tonnage that is too refractory and low grade to treat except on a very high metal market. However, this bank should have a life of 12 years remaining.

Ahmeek Sand Bank

Up to the year 1929 the Ahmeek Mill had treated 17,451,000 ton of rock with an average tailing of .222% copper. Since the introduction of flotation in 1929, the tailings have been too low grade to retreat. The Ahmeek rock gives a low tailing by flotation and as the lake is deep here, the percentage of sand above water should be small with not much loss due to oxidation. A recovery of 3 to 3½ pounds should be obtained, and by treating only the older high grade part of the bank this could be increased substantially. After the Tamarack bank is exhausted, the dredge could be moved to the Ahmeek bank and the sand treated in the Tamarack Plant. While this is being done, it will be necessary to carry the Ahmeek Mill tailing across the dredged area with pipe and pontoons.

This bank may have a life of 10 to 12 years as some of the better material in the outer part of the bank is covered by low grade flotation tailing.

<u>Point Mills Sand Bank</u>

This bank is nearly all Allouez and Centennial tailings and amounted to a total of 7,061,000 tons. Included in this tonnage is a small amount of Allouez conglomerate—probably well under 10% of the total.

Some years ago grab samples were taken over the bank. The conglomerate averaged .351% copper and the amygdoloid ran from .188 to .289, so should average slightly over 4½ pounds. The lake in this vicinity is quite shallow so the proportion of sand above water level will be high—probably 1/3 of the total. As a consequence the flotation tailing will be high.

To treat this sand bank it would be necessary to build a Reclamation Plant. It cost $1,000,000 to build the Quincy Reclamation Plant seven years ago. As we already have the necessary dredge and pontoons, and could take part of #2 Regrinding Plant for the building, it should not cost more to erect than the Quincy Plant did, as we have considerable second hand equipment including pumps, tables, piping, etc., although we would have to buy the ball mills needed and motors to drive them.

Yours truly,

/s/ R. M. Haskell

R. M. Haskell, Superintendent

vs

MS-002
Box 203

Research Dept. DATA BOOKS 1951 - 1968

1-A Cuprous Oxide (Standard Ceric Sulfate)
1-B Cuprous Oxide
2. Chloride, Carbonate & Cyanide
3. Greenhouse Experiments
4. Carbonate (Process for Cu Carbonate)
5. Agricultural Chemicals (Analytical) Particle Size
6. Recovery of Cu From Flue Dust
7. Crushing & Milling
8. Zinc Chemicals
9. Pyrometallurgy (Segregation Tests on Slags, Te in Cu, Se in Refin Cu.
10. Special Leaching Tests
11. Congl. Tailings--Lt. Wt. Bldg. Aggregate, Ti in Tailings
12. Sulfide Leaching (Analysis)
13. " " "

14. Sulfide Ore leaching, Sulfur Compound, Sulfite, Thiosulfate, & Sulfate
15. Flue Dust
16. Sulfide Leaching (Tests 2-47 to 2-78)
17. " " (Tests 3-1 to 3-33)
18. " " (Tests 4-79 to 4-103)
19. " " (Tests 5-103 to 5-127)
20. " " (Tests 6-128 to 6-148)
21. " " (Tests 7-34 - 7-41)
22. " " (Tests 8-149 to 8-194)
23. Sulfide Leaching - Manganese Ore (Tests 8-194 to)
24. Cuprous & Metallic - Analytical
25. Cuprous Cyanide & Copper Chlorides
26. Cu Phosphates, CuZn Phosphate & Cu Silicate Fungicides
27. Leaching of Zn Concentrates, & Secondary By-Products

203	204	205	206
1-A to 37	38 to 77	78 to 117	118 to 166

28. Cuprous Cyanide

29. Cuprous & Metallic (Tests 1 - 91)

30. Leaching of Zn Concentrates & Secondary By-Products (Analysi

31. Agric. Chemicals (Analytical) PARTICLE SIZE

32. Leaching Zn Conc. & Sec. Zn Matls., Cu Diammino Carbonate, Copper Powder

33. Distillation of Leach Solns.

34. Agric. Chemicals (Analytical), Storage Stability of Mixed Oxide in Bags, Solubility of Oxides in HCl, TBCS Process from Oxide Process for Cu Carbonate

35. Cuprous & Metallic (Analytical)

36. Pyrite Processing & Sulfuric Acid Mfg.

37. Cuprous & Metallic

38. Metal Ammino Compounds - Copper Powder

39. Aerofall Mill Tests (book could not be found in May 1960)

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40. Aerofall Mill Tests (Analytical)

41. Metal Ammino Compounds, Cu Powder

42. Cu Powder, Cuprous Oxide, Furnace Tests on Cupric, NH_3 Reduction of Minerals

43. Crushing & Grinding (Analytical)

44. Aerofall Mill Tests (Analytical) & Keweenaw Sand Sizings

45. Pelletizing & Briquetting

46. Ion Exchange, Stabilization & Preparation of Cupric Hydrate

47. Combined Leaching Facilities

48. Properties of Leaching Solution

49. Ind. Oxide for Primary Batteries, Competititve Cu Oxides

50. Foundry Sands

51. Aggregate

52. Poor Rock Processing

53. Silver Recovery
54. Allouez Sands
55. Silver Analysis - Sulfide Leaching
56. Ammonia Leaching--White Pine, Silver Recovery, Kenny Paten
57. Agric. Chemicals (Analytical) - Ind. Oxides - Iron Chemicals
58. Selenium & Molybdenum (Analytical)
59. Poor Rock
60. Sink Float - Analytical
61. St. Louis Exploration
62. Fungicidal Cements
63. Cascade Mill Testing
64. Epidote Core Sand - Ti (Concentration) in Tailings
65. Boleo Ore - Ammonia Leaching (inc. El Arco) Santa Rosalia Pr
66. Agricultural Chemicals (Analytical) - Cupric Hydrate Assays ,
MICRON SIZE CONSTANTS, PARTICLE SIZE, CUPRIC ACETATE

67. Cupric Acetate - Cupric Formate
68. Algicides
69. Boleo - Segregation Process - Santa Rosalia Project
70. Boleo - Acid Leaching - Santa Rosalia Project
71. Boleo - Ammonia Leaching - St. Rosalia Project (See book 65)
72. Preconcentration
73. Boleo - Acid Leaching - St. Rosalia Project
74. Countercurrent Decantation - Boleo - St. Rosalia Project
75. Leaching Tests on Rich & Poor - Arsenic
76. Boleo - Acid Leaching (Santa Rosalia Project)
77. Cupric Hydrate - Reclamation of Hydrate & TBCS Wastes
78. Leaching Tests on Rich & Poor - Arsenic (see #75)
79. Leaching Rich & Poor #10 Tank - Arsenic Test
80. " " " " " " " #2

81. Preconcentration - Battelle
82. Cuprous & Metallic
83. Copper Catalysts
84. Ahmeek Mill - Froth Flotation
85. Cu Powder, Leach Rich & Poor Mineral - LL Leach Plt.
86. Selenium & Molybdenum - Laboratory Flotation - see #58
87. L-P-F (Leach-Precipitate-Float) - Santa Rosalia Project
88. " " " " " " "
89. Miscellaneous Analyses - Geological Dept.
90. Agricultural Chemicals (Analytical), See #66
91. Wemco-Remer Jig
92. Copper Oxychloride
93. Heavy Media - Western Machy. Co. Tests
94 Agric. CHEMICALS - Colored Cu Acetate Fertilizer
95 Baum Jig
96. Misc. Analyses (Depts. Other Than Geological)
97. Misc. Analyses (Depts. Other Than Geological)
98. Homogenizer Tests
99. Misc. Analyses (Depts. Other Than Geological)
100. Agricultural Chemicals
101. Copper Dimethyldithio Carbamate
102. Copper Carbonate - Book No. 2
103. Liquid Copper
104. Cascade Mill - Crushing & Grinding
105. Lab. Flotation - Selenium & Molybdenum New Mexico
106. Poor Rock - Field Notes
107. Laboratory Flotation
108A. Kingston
108B. Miscellaneous
109. Copper Alloys

110. 1834 - Algicides - Fluid Copper

111. Miscellaneous

112. Zirconium — o.n.+

113. Miscellaneous

114. Boron

115. Miscellaneous (8/13/65 to 11/19/65)

116. Fred Baldwin - Miscellaneous

117. Miscellaneous 11/19/65 to 5/10/66

118. Miscellaneous 5/13/66 to 9/5/66

119. Gary Corbein - Miscellaneous

120. Gary Corbeil - Miscellaneous

121. Miscellaneous 4/5/66 - 4/27/67

122. Gary Corbeil - Miscellaneous

123. L. G. Stevens - A. A. Data

124. C. E. Lugviel - Spectrophotometer

125. C. E. Lugviel - Miscellaneous Experiments

126. C. E. Lugviel - Chemical Analysis & Other Chem Work for LGS

127. C. E. Lugviel - Chemical Analysis work for LGS

128. C. E. Lugviel - Chemical Analysis etc. for LGS

129. L. G. Stevens - Copper Organics

130. Kingston Drill Core

131. Miscellaneous - C. J. Bastian 5/1/67 - 1/2/68

132. Anne Stafford A. A. 5/67 - 9/67

133. Anne Stafford Misc. 3/1/68 - 10/15/68

134. Anne Stafford 9/28/67 - 3/1/68

135. C. J. Bastian Miscellaneous

136. C. J. Bastina Miscellaneous 6/21/68 - 7/29/68

137. Ion Exchange - Slags, Tails & Ores - Omit

138. C. J. Bastian Misc. 8/14/68 - 1/27/69

139. C. J. Bastian 1/27/69--- omit

140. Anne Stafford 9/16/68 - 12/13/69

141. Gary Binoniemi 6/10/68

142. Fred Baldwin #2 8/9/65 - 6/8/66

143. L. G. Stevens - Inorganics, Gen.

144. L. G. Stevens, Organics, Gen.

145. L. G. Stevens Ethylene Bis- Organics 20 M. etc.

146. L. G. Stevens - B. Nelson (MTU) Dithiocarbamates, Anal. etc.

147. Kingston A. A.

148. Techtron Atomic Absorption (Smelter Lab)

149. Medusa - Wampum - W. M. Lohela = 11-17-66 - 2-21-67

150. Book I - Calumet Foundry - Lohela - 2-13-67 - 6-1-67

151. Book I - Chemical Analysis - J. D. Johnson 3-16-65 - 11-16-66

152. Ripley Foundry - W. M. Lohela - 11-5-66 - 3-29-67

153. Book I. - Photomicrographs of Regular Ni Hard and Boron Modified Ni Hard Grinding Balls - J. D. Johnson 5-12-65

154. Miscellaneous - W. C. Yeh - 12-15-66 - 2-3-67

155. Robert Reilly - Cu Hydrate - 4-467 - 8-21-67

156 - J. Cone's Cu Work - J/ Johnson - 4-1-65 - 1-5-68

157. Chemical Analyses - W. C. Yeh = 1-3-67 - 4-13-67

158. W. C. Yeh (Summary copper book-not a date to date Lab. book)

159. W. C. Yeh (Microstructures - not date to date lab. book)

160. W. C. Yeh Zr Cu Alloy - 4-18-67 - 6-27-67

161. W. C. Yeh - ZAK, Inc.- 12-19-66 - 3-13-68

162. W. C. Yeh - Stress Rupture - 6-27-67 - 6-25-68

163. W. C. Yeh - ATLAS - 1-31-67 - 9-14-67

164. R. J. Marcotte - July 1960 - July 18, 1961

165. W. C. Yeh - Copper Alloys 6-9-67 - 7-18-68

166. W.C. Yeh (Cu) 1836 - 4=20-67 - 11-14-68

MS-002
Box 38
Folder 22

COPY

VERN E. ALDEN COMPANY
ENGINEERS
33 NORTH LA SALLE STREET
CHICAGO 2

J.O. 342

November 15, 1954

Mr. P. H. Ostlender
Project Engineer
Calumet Division
Calumet & Hecla, Inc.
Calumet, Michigan

Report on Lightning Protection
For the Electrical Transmission System of
Calumet & Hecla, Inc.

Dear Mr. Ostlender:

Following your inquiry letter to us dated April 6, 1953, Purchase Order No. 14418 was issued to cover the subject report.

We are sending you herewith five copies of this report. The completion of this report was delayed until the Osceola Unwatering Project was fairly well complete as was suggested in your letter.

A minute survey of your lines and substations was made and this report embodies our findings and our recommendations for certain improvements.

We will be pleased to review this report with you after you have had an opportunity to study it.

Yours very truly,

F. D. Troxel
F. D. Troxel
Project Engineer

Encl

August 25, 1954

Mr. Carl J. Marold
Herrick, Smith, Donald, Farley & Ketchum
294 Washington Street
Boston 8, Massachusetts

Dear Mr. Marold:

This letter is written to try to clarify the factual situation respecting the tailings banks in Torch Lake from which copper has been reclaimed by Calumet & Hecla. I enclose a sketch map showing Torch Lake, the location of the various mills and reclamation plants which are located on Torch Lake or located there in the past, and the site of the original tailings banks from which copper has been or is being reclaimed.

I believe the greatest source of confusion in this matter lies in the fact that company names have been used as names for properties, and that these names have continued with the properties after the companies have ceased to exist.

This discussion has to do with veins, mines, companies, mills, and reclamation plants. In many instances the same name was applied to each of these. This circumstance leads to most of the confusion.

Calumet and Hecla Consolidated Copper Company (whose name was changed to Calumet & Hecla, Inc. in 1952) was formed in 1923 by a consolidation of the following five companies: Calumet and Hecla Mining Company, Osceola Consolidated Mining Company, Ahmeek Mining Company, Allouez Mining Company, and Centennial Copper Company. This consolidation has been ruled by the Treasury Department to have been a tax-free reorganization. All of the companies were Michigan corporations, and the resulting company was also a Michigan corporation. Under the Michigan corporation law, it is expressly stated that in the event of a consolidation of this type, the entity of the constituent companies is continued and merged in the resulting company.

Calumet and Hecla Mining Company conducted mining operations on two veins: the Calumet conglomerate vein, and the Osceola amygdaloid vein. Osceola Mining Company conducted operations on the Osceola amygdaloid vein and on the Kearsarge vein.

About 1900 it purchased the north Kearsarge mine and the south Kearsarge mine.

Ahmeek, Allouez and Centennial all operated exclusively on the Kearsarge vein.

The Tamarack Mining Company was organized about 1880 to mine the lower part of the Calumet conglomerate vein. It did so until 1917, at which time Calumet and Hecla Mining Company purchased all of its assets.

The mills of all of these companies were located on Torch Lake as shown by the enclosed sketch. The most southerly mill was built in 1885 by the Osceola Consolidated Mining Company. At a later date they built another larger mill adjoining the first. The tailings from both these mills were deposited in the same general area in Torch Lake. These mills operated from 1885 to 1920, treating ore from mines operated by the Osceola Company, and, at times, ore from other sources.

The Tamarack Mining Company built its No. 1 mill north of the Osceola mills in 1887. It subsequently built its No. 2 mill between its No. 1 mill and the Osceola mill.

The Tamarack No. 1 mill treated ores from the mines of the Tamarack Mining Company on the Calumet conglomerate vein until Calumet and Hecla Mining Company bought all of its assets. Calumet and Hecla continued stamping rock produced by it from the Calumet conglomerate vein at the Tamarack mill until May 29, 1920, at which time the mill ceased operation.

The Tamarack Mining Company sold its No. 2 mill to the Lake Milling, Smelting and Refining Company in 1914. Lake Milling had been organized as a service facility by some of the smaller mining companies in the area. Its No. 1 mill was located at Point Mills on Portage Lake, and that mill and its tailings are not in issue here. The Tamarack No. 2 mill became known as the Lake No. 2 mill. It operated with several periods of shutdown from 1914 through 1930.

Lake Milling, Smelting and Refining Company performed milling for its shareholders only. It operated on a fee basis intended simply to return its cost. The ore of each of the producers was kept segregated, passing through separate stamps, etc. The concentrates were delivered to the companies furnishing the ore. Tailings were deposited in Torch Lake in in an area between the Osceola sands and the Tamarack sands. The riparian rights to this area had originally belonged to the Osceola, which had transferred them to the Tamarack Company when the mill was originally built. Tamarack transferred these rights to Lake, and they were confirmed by a supplementary grant from Osceola.

After July, 1919, the tailings from all of the foregoing mills were commingled and deposited in a separate tailings bank. None of these have been reworked because of their low grade. The reason for this commingling was that the tailings banks had become so extensive that tailings would no longer flow by gravity and it was necessary to install a pump to elevate them and distribute them to a greater distance.

Enclosed you will find a compilation of the stamping records of the Osceola mills and the Lake mill. You will note that the Osceola stamped principally its own production from the Osceola vein and its north Kearsarge and south Kearsarge mines. From 1905 until 1910 it stamped rock for the Ahmeek Mining Company. The Ahmeek Mining Company at that time built a mill for itself at a location shown on the sketch map.

The Lake mill was shut down in 1919, and during the balance of that year ore from the Allouez, Isle Royale and Centennial mines was stamped in the Osceola mill.

In 1910 and 1911, an arrangement was made by which the amygdaloid ore produced by Osceola was stamped at the Calumet and Hecla mill, and the Calumet and Hecla ore was stamped at the Osceola mill. The Isle Royale mine did not become part of Calumet and Hecla, although Calumet and Hecla owned 20% of its stock and managed its operations. In 1915 the Isle Royale mill burned down, and while it was being rebuilt its ore was stamped at the Osceola mill.

At the northerly end of Torch Lake, Calumet and Hecla Mining Company had operated two mills: the Calumet mill and the Hecla mill. Except for the Osceola rock which was stamped in 1910 and 1911, all of the production going through these mills was from the mines of Calumet and Hecla Mining Company.

In 1915, Calumet and Hecla built a plant at Lake Linden to reclaim its sands. This is called the Calumet Reclamation and is shown on the map. This plant was intended only to handle tailings from the Calumet conglomerate vein. It did so from 1915 until about 1943, at which time operations were suspended on the conglomerate sands and transferred to the amygdaloid sands. After several years, operations were re-transferred to the conglomerate sands and continued until 1953, at which time operations of the reclamation plant stopped entirely.

In 1925 a similar reclamation plant was built for the purpose of reworking the conglomerate sands produced from the Calumet conglomerate ore by the Tamarack Mining Company. This was called the Tamarack reclamation plant, even though the Tamarack Mining Company had been dissolved in 1917. This plant continued to operate on these sands until they were exhausted, at which time the plant was modernized and transferred its activities to the amygdaloid sands produced by the Osceola and Lake mills. It is these activities that are mostly in question.

I am not sure that this description will do much to clarify the fact situation for you. I believe that a trip to Calumet for a day or two will be necessary. If you will let me know what date would be convenient for you, I will make arrangements to meet you in Chicago and go to Calumet with you.

Sincerely,

A. E. P.

AEP:hp

J.O. 342 November 12, 1954

REPORT ON LIGHTNING PROTECTION FOR THE ELECTRICAL TRANSMISSION SYSTEM OF CALUMET & HECLA, INC.

1. During past years there have been a considerable number of outages of the electrical transmission system which were traceable to lightning disturbances. This study was made in accordance with the instructions in your purchase order No. 14418 and was made in an effort to find the points in your electrical transmission system which are susceptible to lightning troubles and to determine what could be done, at not too great a cost, to correct these conditions, and thus provide more reliable service from the electrical transmission system.

2. In an electrical transmission system such as exists here, the principal forms of lightning protection which might be considered are:

    A. The use of modern lightning arresters correctly applied.

    B. The use of overhead ground wires and lightning rods properly installed.

    C. The maintaining of low resistance ground connections for the arresters, the overhead ground wires and the lightning rods.

3. We have looked at each of your substations individually and find that in general there are many old lightning arresters

now in service on your system. Many of these arresters are undoubtedly useless as lightning arresters and, perhaps, even worse than no lightning arrester at all. In general this report recommends the replacement of all these old arresters with new modern arresters. The new arresters should be as follows:

15 kv station type grounded neutral service
Westinghouse Type "SV" outdoor Style No. 1533116 or equal.

2300 V Line type ungrounded neutral service
Westinghouse Type "LV" Style No. 1535071 or equal.

At certain points on your system some of the old arresters have already been replaced and these, of course, do not have to be replaced again. This report discusses in detail the conditions which exist at each substation on your system. At a few points where overhead lightning rods or ground wires are not now installed, it is suggested that they be added. In general in protecting equipment from lightning disturbances, it is well to place the lightning arrester as close to the equipment being protected as is possible. On your system, for the most part, the major equipment which is being protected is transformers and for that reason the arresters in each case should be placed as close to the transformers as is possible.

4. This report is a part of the general effort to raise the level of the reliability of your entire electrical system and other items, germane to this program, have been carried out in the past year or so. These items included the reconditioning of the 13.2 kv breakers and relays in the Lake Linden power house, revision of relay settings, automatic transfer of auxiliary supply, replacing poles on lines, etc. This work has progressed along with the work on the Osceola project. In our report to you dated February 8, 1949 we made an engineering study of each of your substations. This report covered principally the possible short circuit conditions on your system at the various points and the interrupting abilities of the breakers at the various points. In this report we made a series of recommendations. While reviewing your system in connection with this lightning study, we checked to see if the recommendations contained in the report dated February 8, 1949 had been carried out. In many cases these recommendations had not been carried out. In the earlier report certain hazards were pointed out and recommendations made to remove these hazards. As long as these conditions exist, they are a hazard to the reliable operation of this system just as are the hazards from lightning disturbances. We suggest that the report dated February 8, 1949 be reviewed and the things recommended therein, which have not been done, be carried out as soon as possible.

5. Any overhead 2300 volt or lower voltage lines which are exposed to lightning surges should be equipped with modern lightning arresters at both ends of each line. All electrical equipment and lighting circuits should be solidly and permanently connected to ground.

6. Below are listed in detail our recommendations in regard to each of the substations. You will note that in certain cases we are referring to substations which no longer carry much load. However, as long as these substations are connected to your system they constitute just as much a hazard to the reliable operation of your system as if they were carrying a heavier load. In cases where certain substations are no longer carrying any load, we would suggest that that substation and as much of the line as possible, that originally supplied such a substation, be disconnected from your system.

   A. <u>Quincy Substation</u> - This substation is relatively new. The equipment for the most part is relatively modern. Modern G.E. type lightning arresters are installed. The substation is located in a low spot which should be relatively free from lightning occurrence. There is no record of a lightning stroke at this location. The ground resistance is quite low. Therefore, we would suggest no changes insofar as the lightning protection for this particular yard is concerned even though there are no ground wires above the substation proper. In our report dated February 1, 1949, we stated that the breaker which is installed on the high tension side of this transformer had an interrupting capacity of 50,000 kva and that the short circuit current that might flow in case of a short at the terminals of the transformer, would be about 91,000 kva. We suggested that this breaker be removed from service since it constitutes a serious hazard to the reliable operation of the system and is a source of fire hazard as well. This breaker has not been removed and we again recommend that it be removed just as soon as practical. When the breaker is removed, it will be necessary to change the settings of the relays for the Quincy Line at the Ahmeek Power House since the clearing of a fault will depend upon the operation of these relays.

B. <u>Tamarack Reclamation Substation</u> - In this substation there are two banks of transformers. These transformers are each 1,000 kva in capacity, arranged in two banks of three each, one stepping down to 440 volt and the other to 2300 volt. All of this equipment is located indoors. Both transformer banks are supplied by one feeder from the Ahmeek Power Station. There are located here, three old style G.E.Co.'s oxide film type of lightning arresters. These arresters are quite old and are obsolete. Most arresters of this type have been removed from power systems many years ago since it was found that the discs in the arresters deteriorated after a period of time and that it was impossible to determine their condition in any satisfactory manner. Furthermore, the porous block type of arrester has been developed since and it is a much better arrester in every respect. We would recommend that these arresters be removed and be replaced with modern arresters. In our report dated February 1, 1949 we stated that the three oil circuit breakers located in the circuit on the high tension side of these transformers each had an interrupting rating of 25,000 kva and that the possible short circuit current that might flow with a short circuit at the high tension terminals of these transformers could be about 136,000 kva. We, therefore, recommended that these breakers be all removed from service just as soon as possible. This has not been done. We recommend again that these breakers be removed just as soon as practical as they constitute a serious fire and system reliability hazard. Furthermore this substation is located in a room which has a door which leads to the Reclamation Plant. If the oil in these transformers or the breakers should get out of the tanks and catch fire, as it does when there is a fault in the equipment, this flaming oil could flow out through this door onto a wooden platform, down a wooden stairs, thereby starting what could be a serious and costly fire. We would recommend that a curb be placed immediately at this door entrance so that this flaming oil could not flow out onto the wooden platform and stairs. Furthermore certain of this equipment is located on an elevated platform which has a stairway at one end. If a fault occurs when a man was at the opposite end of this platform he would be trapped. We would suggest that at least an escape ladder of some sort be provided at the opposite end of this elevated platform.

C. <u>No. 2 Regrinding Plant Substation</u> - This substation was originally supplied by two lines from the Lake Linden power house bus. Two transformer banks were installed, one 3750 kva in capacity and another 6000 kva. The load at this time, however, has been very much reduced and at present only one line is connected from Lake Linden to the substation. All of the transformer capacity is, however, yet in service. Apparently there is a good chance that all of this equipment will be taken out of service before long due to the regrinding plant operation being discontinued. If this is true, of course, it is not desirable to spend any money here. However, if this equipment is to continue to be energized, even though it is not carrying any load for any length of time, we would suggest that the old lightning arresters on the one line that is in service be replaced with modern type of arresters. In our report dated February 1, 1949 we stated that the breakers which are in the high tension side of the feeds to these two transformer banks each had an interrupting rating of about 25,000 kva. The short circuit current that might flow is something in the order of 232,000 kva. We, therefore, recommended that these breakers be removed from service immediately. This has not been done. If this equipment is to remain energized we would recommend again that these breakers be removed just as soon as possible. All of this equipment is quite old and it is a source of hazard to the whole system. The equipment, however, is located in a separate building and if it caught on fire it probably would not do much damage to other buildings. When the high tension breakers are removed, the relay settings at the Lake Linden bus should be changed.

D. <u>The Smelter Substation</u> - The smelter substation has three 1000 kva single phase transformers. These transformers were formerly fed through breakers on the high tension sides from a line from the Lake Linden Power House. These breakers have been removed in accordance with the recommendations of our report dated February 1, 1949. The incoming line to this substation is provided with old G.E. lightning arresters which should be replaced with the new modern type. The ground resistance at this station is low and with the new lightning arresters, a minimum amount of lightning disturbance would be anticipated. We, however, would recommend that lightning rods be placed above the switchyard structure in the usual manner that has been done on many of the substations on the C&H system.

E. Coal Dock Substation - The coal dock substation is an outdoor substation. The transformer bank formerly had an oil circuit breaker on the high tension side but this breaker has been removed in accordance with the recommendation of our report of February 1, 1949. The incoming line is provided with old style Westinghouse lightning arresters and these should be replaced with new style arresters. We would also suggest that lightning rods be added above this switchyard. This substation is fed from the same breaker and line that feeds the smelter. The relay settings for this breaker in the Lake Linden Power House are satisfactory as indicated on the last relay settings list which we gave Calumet & Hecla.

F. Ahmeek Power Plant - All the lines going out of the Ahmeek Power Plant are equipped with modern lightning arresters. These arresters are mounted on a modern steel pull-off structure. The pull-off structure does not have lightning rod protection above it but it is set adjacent to the rather tall power house smokestack which is equipped with lightning rods and this affords good protection from direct lightning strokes. We would, therefore, suggest no changes at this point. The ground resistance at this point is very low which also will lead to good operation from these lightning arresters.

G. Lake Linden Power Plant - All of the lines going out from the Lake Linden Power Plant are equipped with modern arresters. These have been replaced in recent years. The switching equipment in the Lake Linden Power Plant is all relatively modern and has recently been reconditioned and tested. The relays have been carefully cleaned and adjusted. The relay settings have been checked in accordance with the system as it is at present. Therefore, no further work would need to be done at the Lake Linden Power Plant and a minimum of trouble should be expected from lightning at this point.

H. All of the transmission lines at Substation "B", both the 60 cycle and the 25 cycle, have been provided with modern lightning arresters. Ground resistance at this point is low. Overhead lightning rods have been provided. Therefore it would appear that no work need to be done at this point in order to have a good level of lightning protection.

I. <u>Lines to Osceola No. 13 and No. 6 Shafts and to Tamarack No. 5</u> - These lines and the associated substations have recently been built and they are equipped with modern lightning arresters and other lightning protection features. All of this should give good lightning protection for these lines and the substations.

J. <u>Calumet Waterworks Substation</u> - All of the lightning arresters at Calumet Waterworks are old arresters. These should be removed and replaced by modern arresters. There are two sets, one for each of the two transformer banks. Lightning rods should be added over the switchyard. The ground resistance at this point is rather low and with the addition of the two items mentioned above a good level of protection would be assured. The arresters on both ends of the 2300 V line to the Tamarack Water Works should also be replaced.

K. Centennial Substation and Adjacent Lines - The area around the Centennial Substation has the record of having more lightning difficulties than any other spot on the C&H system. This is probably due to exceedingly high ground resistance and to, perhaps, a rather exposed natural position. We would recommend that the two old lightning arresters on the line to the transformer bank be removed and that the one modern arrester be retained. We would suggest the addition of a modern arrester at the point where this connection to the transformer bank cuts into the main line. We would suggest that another arrester be placed about one thousand feet away on the line to Substation "B" and that another be placed about one thousand foot distance in the opposite direction from this tap point. Anything that can be done to lower the ground resistance of all of these arresters should be done. Modern lightning arresters should be provided on the 2300 V. distribution system. With the addition of these arresters and a low ground resistance we would think that the protection afforded was about as good as possible.

L. Alloway No. 3 Substation - There are modern arresters at this substation and we would suggest no change here.

M. Ahmeek No. 2 Substation - There are choke coils in the connection to the transformer bank to Ahmeek No. 2. These choke coils serve no useful purpose and in fact are a hindrance insofar as lightning protection is concerned. They should be removed. The lightning arresters at this point are very old and should be replaced with modern arresters.

N. Ahmeek No. 3 and No. 4 Substation - The lightning arresters at this point on both the incoming lines and outgoing lines are all very old and should be replaced with modern arresters. All of these new arresters should be located out of doors.

O. Seneca No. 2 Substation - The lightning arresters at this point are very old and should be replaced with modern lightning arresters. Lightning rods should be placed above the substation.

P. Iroquois Substation - The lightning arresters at this substation are very old and should be replaced with modern arresters. Lightning rods should also be placed over the substation.

Q. Trap Rock Valley Line and Substation - This line and substation have been built recently and are provided with modern arresters, etc. and should be relatively free from lightning troubles.

The estimated cost of carrying out the foregoing recommendations is as follows:

42 -	15 kv lightning arresters @ $200. ea. installed =	$8,400.
12 -	2300 V. lightning arresters @ $10.ea. installed =	120.
5 -	Sets of lightning rods above substations =	500.
	Total =	$9,020.

We feel that if the recommendations which are made in this report are carried out that the reliability of your electrical transmission system, insofar as lightning disturbances are concerned, will be much improved. We will be happy to discuss this report with you, after you have had an opportunity to review it, if you so desire.

Signed: F. D. Troxel

F. D. Troxel
Senior Electrical Engineer

MS-002
Box 38
Folder 22

R E P O R T   O N   S T A T U S   O F   R E C O M M E N D A T I O N S

OF Vern E. Alden Co.'s Engineering Studies

of Feb. 1, 1949 and Nov. 17, 1954

CONTENTS

This report lists by substations, the status of Vern E. Alden's recommendations contained in their two reports:

Engineering Study of Distribution Substations
February 1, 1949

Lightning Protection for Elec. Transmission System
November 17, 1954

Submitted to

F. H. OSTLENDER

by

A. J. KLEVEN

January 6, 1955

Note: This status report does not take into consideration the advisability of installing some of the recommended equipment in the light of the life of some of the properties involved nor does it reflect some changes that have been made in the distribution system since the 1949 report. This report, therefore, can only serve as information for further studies. A.J.K.

The following is a list of recommendations taken from Vern E. Alden Company reports of February 8, 1949, and November 12, 1954, and what has been done toward carrying them out.

Quincy Substation

Recommendations:

Remove 13.2 KV Breaker.

Set relays at Ahmeek to clear fault when breaker is removed.

Work done on recommendations:

There seems to be some question as to the interrupting capacity of this breaker. V. E. Alden's report says 50,000 KVA and a letter attached to the report from C. K. Nauman of General Electric addressed to J. E. Breth claims 250,000 KVA.

New bushings have been installed in the 13.2 KV breaker and it is still in service.

Tamarack Reclamation Substation

Recommendations:

Remove three 13.2 KV Breakers from service, one on line from Ahmeek and two on high side of transformers.

Set relays at Ahmeek to trip on fault on either low voltage bus at Tamarack.

Overhaul 2.3 KV Breakers.

Install reactor on high side of transformer banks.

On 2300 Volt bank, reactor specifications - 4.1% reactance, 132 amps, 13.8 KV on 3000 KVA base.

On 460 Volt bank, reactor specifications - 30% reactance, 132 amps, 13.8 KV on 3000 KVA base.

Build curb to prevent oil from transformers from flowing down wooden stairs.

Provide escape ladder from elevated platform on opposite side from stairs.

Replace old G.E. lightning arresters.

Work done on recommendations:

The trip circuit has been taken out of the 13.2 KV breaker at Tamarack.

No other changes have been made.

No.2 Regrinding Plant Substation

Recommendations:

Remove 13.2 KV breakers.

Put both transformer banks on one circuit.

Change relay settings at Lake Linden plant when breakers are removed.

Replace old lightning arresters if station is to continue energized.

Work done on recommendations:

Breakers have been taken out of service.

Two transformer banks have been put on one circuit.

The Smelter Substation

Recommendations:

Separate breakers at Lake Linden for Coal Dock and Smelter.
Remove 13.2 KV breaker at Smelter.
Set relays at Lake Linden to trip on fault on low voltage bus at Smelter.
Install a reactor on the high side of transformers to limit short circuit current as the 2.3 KV breakers have too low interrupting capacity.
Reactor specifications - 9% reactance, 132 amps, 13.8 KV on a 3000 KVA base.
Clean and overhaul low voltage equipment.
Replace old G.E. lightning arresters.
Install lightning rods above station.

Work done on recommendations:

13.2 KV breaker removed from service.

Coal Dock Substation

Recommendations:

Remove remaining 13.2 KV Breaker.
Feed substation from separate breaker at Lake Linden.
Set relays at Lake Linden to trip with a fault on low voltage bus at Coal Dock.
Clean and overhaul 2.3 KV breakers.
Replace old Westinghouse lightning arresters.
Install lightning rods above station.

Work done on recommendations:

13.2 KV Breaker has been removed.
Indoor transformer banks have been replaced with a new outdoor bank.
S & C fuses with 209,000 KVA interrupting capacity have been installed on new transformer bank.
Cleaning and overhauling 2.3 KV breakers is being done.
Sub is fed from same breaker and line which feeds Smelter, and the relay settings at Lake Linden are satisfactory.

Calumet Substation

Recommendations:

New 13.2 KV breakers.
Operate on 600 KVA bank with 1500 KVA bank as spare.
Overhaul 2.3 KV breakers.

Work done on recommendations:

13.2 KV in Calumet Substation used only in emergency.
600 KVA bank removed.
1500 KVA bank used only in emergency.

Calumet and Tamarack Water Works Substation

Recommendations:

Check equipment.
Replace lightning arresters on both transformer banks.
Install lightning rods over the switchyard.
Replace lightning arresters on both ends of 2.3 KV line to Tam. Water Works.

Work done on recommendations:

None of the recommendations carried out so far.

Centennial Substation

Recommendations:
Check 13.2 KV Breaker and relays.
Remove old lightning arresters on line to transformer and retain modern station arresters.
Install modern arresters at point where connection to transformer bank cuts into main line.
Install lightning arresters 1000' along line to "B" Station from tap point and 1000' along line in opposite direction from tap point.
Drive additional ground rods to lower ground resistance.
Put lightning arresters on 2.3 KV distribution system.
Work done on recommendations:
Lightning arresters installed on line to compressors.

Allouez No.3 Substation

Recommendations:
Install modern lightning arresters.
Work done on recommendations:
No changes have been made.

Ahmeek No.2 Substation

Recommendations:
Remove choke coil in connection to transformer bank.
Replace lightning arresters.
The relays at this station should be changed.
Work done on recommendations:
No changes have been made.

Ahmeek No.3 & 4 Substation

Recommendations:
Replace lightning arresters on all incoming and outgoing lines.
Locate all arresters outside.
Work done on recommendations:
No changes have been made.

Seneca No.2 Substation

Recommendations:
13.2 KV breaker at No. 3 & 4 Ahmeek on Seneca line should be removed or replaced.
Replace lightning arresters.
Install lightning rods over substation.
Work done on recommendations:
No changes have been made.

Iroquois Mine Substation

Recommendations:
Put 13.2 KV breaker in service if it is in good condition.
Replace lightning arresters.
Install lightning rods over substation.
Work done on recommendations:
No changes have been made.
The 13.2 KW breaker is not in operating condition.

The substations listed below have modern lightning protection and modern equipment and no changes are suggested.

Lake Linden Power Plant

Ahmeek Power Plant

"B" Station

Trap Rock Valley Substation

No.13 Osceola Substation

No.6 Osceola Substation

No.5 Tamarack Substation

TABLE II

Summary - Average of Various Tests
Assays - Per Cent Copper

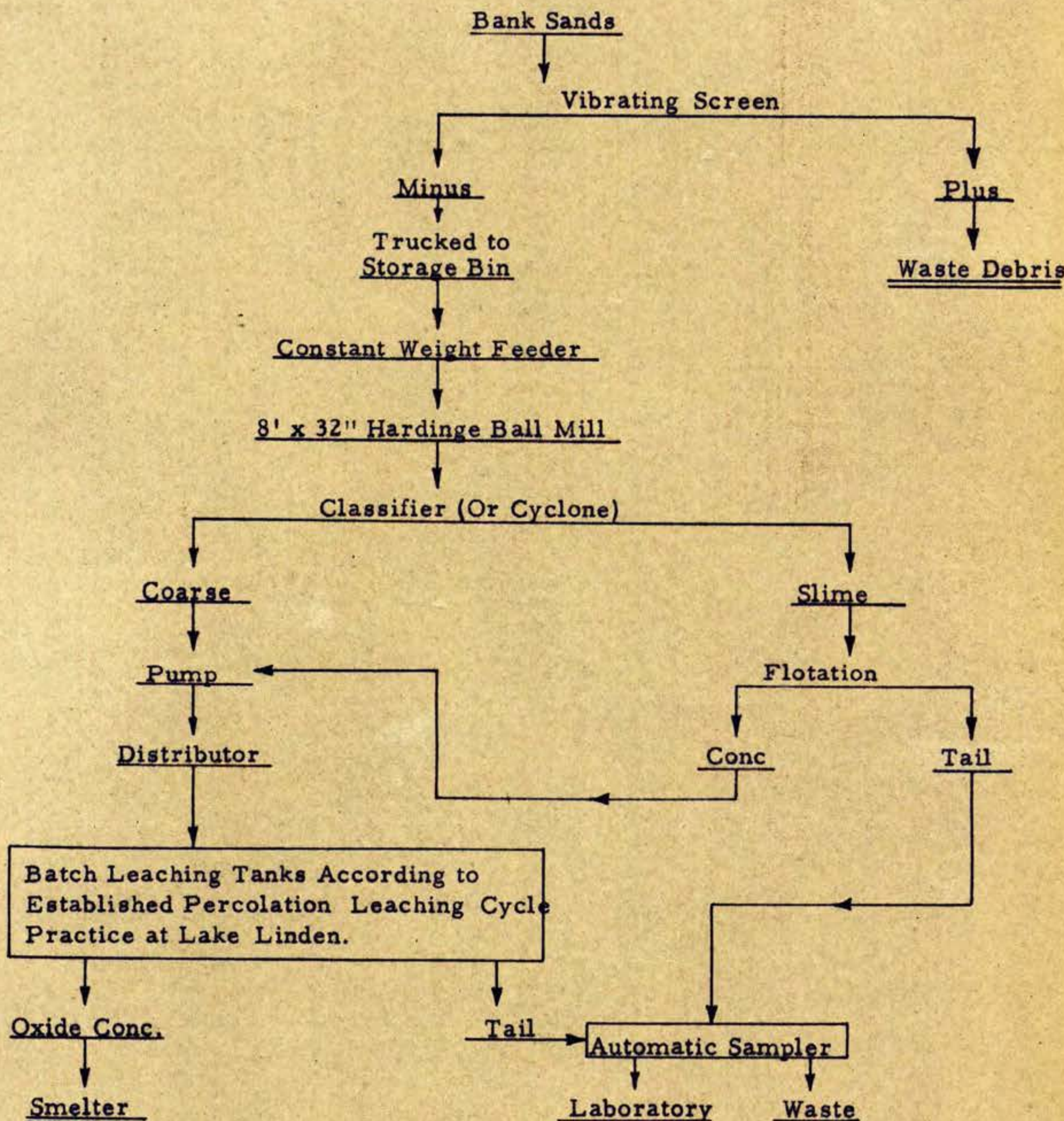
Test	Belt Feed	Belt Tons	Plt. Fines	Gen. Feed	Gen. Tail	Shifts Day	Shifts Aft.	Shifts Nite	Test No. 2 Flot.	Remarks
A	.288	2,525	.260	.258	.148	.145	.150	.149	.157	Regular practice Z-11, #5 P.O., #11 P.O.
B	.264	2,295	.260	.249	.142	.140	.146	.140	.150	Z-11, ≁ #250 Dowfroth ≁ No. 2 Fuel Oil
C	.282	2,351	.261	.240	.149	.153	.144	.149	.148	Z-11 ≁ #5 P.O.
D	.272	2,278	.258	.237	.146	.147	.144	.146	.146	Z-11 ≁ #5 P.O. ≁ #11 P.O.
E	.267	2,721	.273	.287	.162	.173	.154	.160	.168	Z-11 Mix. @ Ahm.Mill, #5 P.O., #11 P.O.
F	.280	2,400	.272	.252	.154	.158	.146	.159	.154	Z-11, #250 Dowfroth, #2 F.O.

Condensed Operating Summary Showing Relative Increase of "Tonnage" with Feed and Tail Assays - Various Tests
TABLE III

Test	Plant Tons/24 Hrs.	Slimes Plant Fines	Gen. Feed #2 Class. O'flows	No. 2 Mach. Flot Tail	Reagents Used
D	2,278	.258	.237	.146	Z-11, #5 P.O., #11 P.O.
B	2,295	.260	.249	.150	Z-11, No. 250 D.F., ≁ #2 F.O.
C	2,351	.261	.240	.148	Z-11, #5 P.O.
F	2,400	.272	.254	.154	Z-11, #250 D.F., #2 F.O.
A	2,525	.260	.258	.157	Z-11, #5 P.O., #11 P.O.
E	2,721	.273	.287	.168	Z-11 Ahm.Mix., #5 P.O., #11 P.O.

PROPOSED RECLAMATION FLOWSHEET
GRINDING-FLOTATION-LEACHING

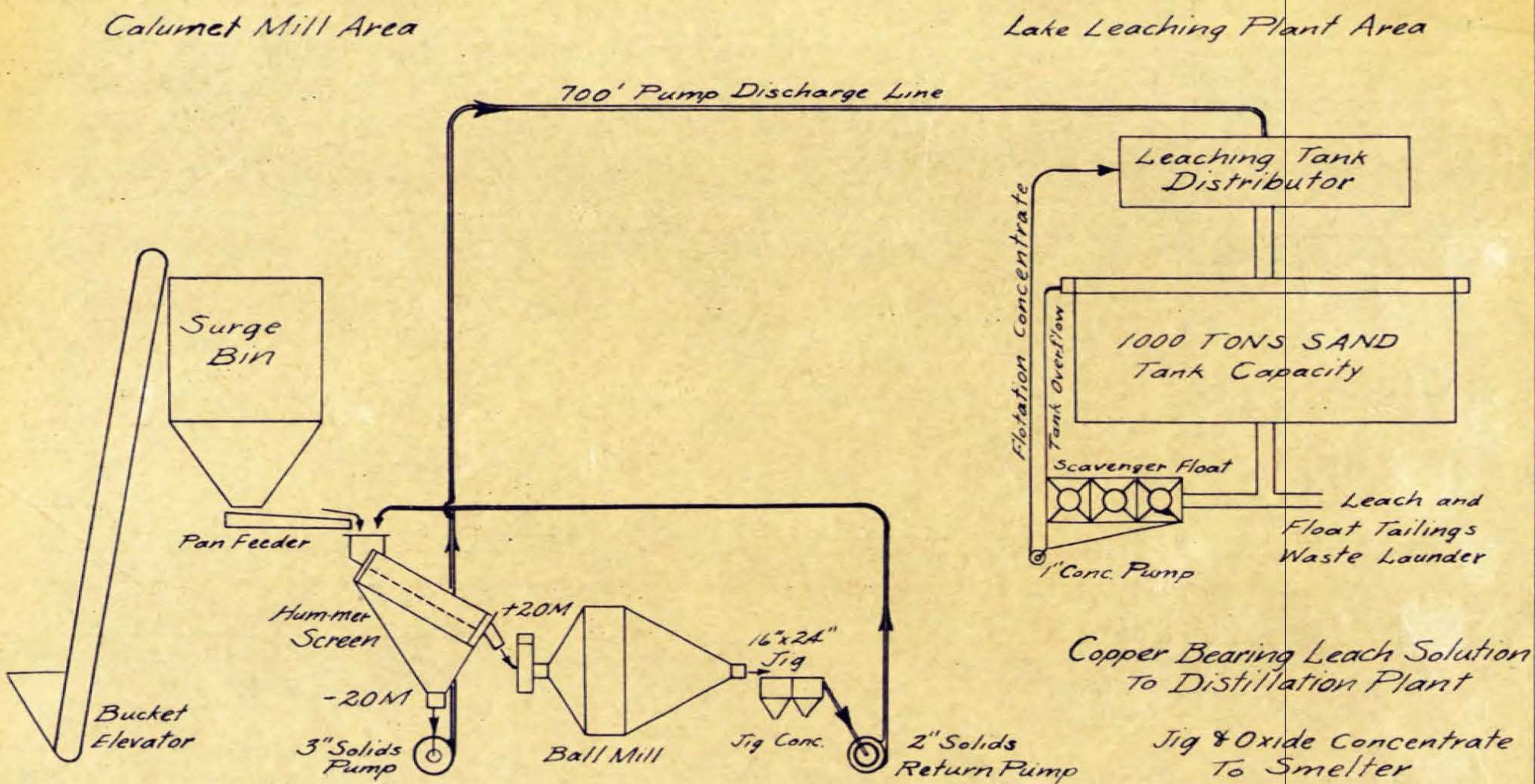
Conglomerate Tailings
(Capacity 400 T.P.D.)



Oxide Conc. Upwards of 90% Cu.
Flotation Conc. About 25 to 30% Cu.
Anticipated Overall Recovery 85%.

Research Dept.
A. B. Landstrom
12/14/55

RECLAMATION FLOWSHEET - ALLOUEZ SANDS



Calumet Division - Calumet & Hecla, Inc.
Research & Development Department
A. B. Landstrom 2/24/56

SCREEN ANALYSIS
TABLE VIII

C O P Y

MATERIAL Underflow (To Flotation) "V" Tanks, Tamarack, December 13, 1957

Name of Company MR. J. J. VITTON (24-hourly samples)

Location

Size of Crusher

Feed to Crusher

Crusher Setting

Moisture

Specific Gravity

Hardness

Remarks

Form 71

Screen Analysis made with Tyler Test Sieves

Through	On	Weight on Screen	% on Screen	% Cumulative	% Copper	% Copper
	6	.02	.01	.01		
	10	.10	.05	.06		
	14	.12	.06	.12	.384	.0004608
	20	.80	.40	.52	.370	.0014800
	28	1.90	.95	1.47	.579	.0055005
	35	3.70	1.85	3.32	.401	.0074185
	48	15.00	7.50	10.82	.202	.0151500
	65	24.10	12.05	22.87	.243	.0292815
	100	49.50	24.75	47.62	.239	.0591525
	150	48.40	24.20	71.82	.269	.0650980
	200	30.50	15.25	87.07	.274	.0417850
200		25.86	12.93	100.00	.471	.0609003
Totals		200.00	100.00			.2862271
		Head Assay		.296% Copper		
		By Screen Analysis		.286% Copper		

Copies to:
A. Edwards
W. G. Gagnon
L. C. Klein
R. K. Poull
J. J. Vitton

/s/ W. G. Gagnon
W. G. Gagnon 12-17-57
wgg/gws

SCREEN ANALYSIS
TABLE VIII

COPY

MATERIAL Underflow (To Flotation) "V" Tanks, Tamarack, December 13, 1957

Name of Company MR. J. J. VITTON (24-hourly samples)

Location

Size of Crusher

Feed to Crusher

Crusher Setting

Moisture

Specific Gravity

Hardness

Remarks

orm 71

Screen Analysis made with Tyler Test Sieves

Through	On	Weight on Screen	% on Screen	% Cumulative	% Copper	% Copper
	6	.02	.01	.01		
	10	.10	.05	.06		
	14	.12	.06	.12	.384	.0004608
	20	.80	.40	.52	.370	.0014800
	28	1.90	.95	1.47	.579	.0055005
	35	3.70	1.85	3.32	.401	.0074185
	48	15.00	7.50	10.82	.202	.0151500
	65	24.10	12.05	22.87	.243	.0292815
	100	49.50	24.75	47.62	.239	.0591525
	150	48.40	24.20	71.82	.269	.0650980
	200	30.50	15.25	87.07	.274	.0417850
200		25.86	12.93	100.00	.471	.0609003
otals		200.00	100.00			.2862271
		Head Assay		.296% Copper		
		By Screen Analysis		.286% Copper		

ies to:
Edwards
G. Gagnon
C. Klein
K. Poull
J. Vitton

/s/ W. G. Gagnon
W. G. Gagnon 12-17-57
wgg/gws

COPY

SCREEN ANALYSIS

MATERIAL Tamarack Reclamation January 1958

TABLE IV

Name of Company Mr. L. C. Klein

Location

Size of Crusher

Feed to Crusher

Crusher Setting

Moisture

Specific Gravity

Hardness

Remarks

Form 71

Screen Analysis made with Tyler Test Sieves

Through	On	Weight on Screen	% on Screen	% Cumulative	% Copper	% Copper
		General Feed				
	28	1.3	.65	.65		
	35	1.2	.60	1.25	.250	.0031250
	48	30.5	15.25	16.50	.185	.0282125
	65	31.5	15.75	32.25	.231	.0363825
	100	35.3	17.65	49.90	.249	.0439485
	150	21.8	10.90	60.80	.334	.0364060
	200	13.5	6.75	67.55	.334	.0225450
200		64.9	32.45	100.00	.287	.0931315
Totals		200.0	100.00			.2637510

Head Assay .253% Copper

By Screen Analysis .264% Copper