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 ⅜-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

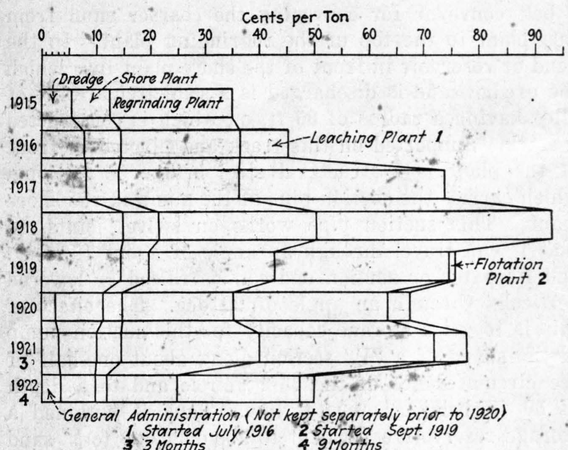


Fig. 4—Reclamation plant costs, 1915 to date

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 2½ 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:

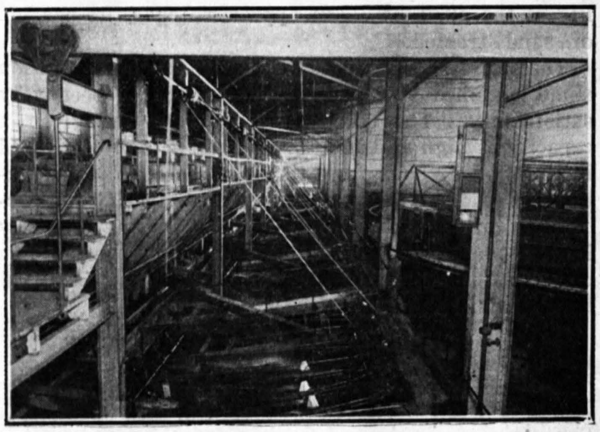
Tons treated	866,524
Assay feed, per cent copper	0.723
Assay tailing, per cent copper	0.468
Pounds refined copper	4,458,100
Pounds copper per ton sand	5.14
Cost per pound copper, excluding smelting and selling, cents	6.40

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

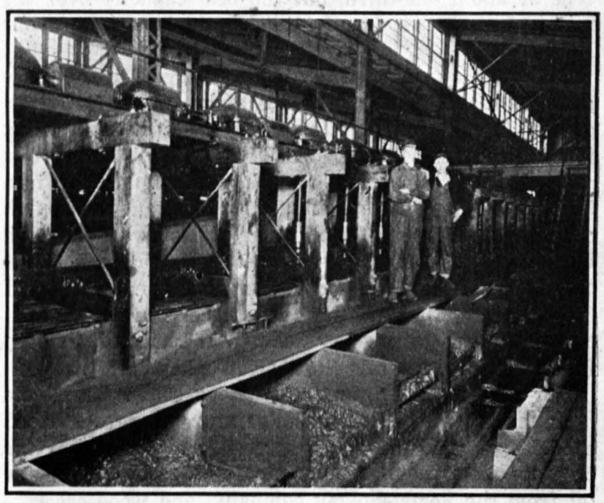


Leaching plant, showing "V" settling tanks, Dorr quadruplex classifier, and leaching tanks with covers in place. Colored lights against column in foreground are signals flashed from plant to plant to show load conditions.

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.

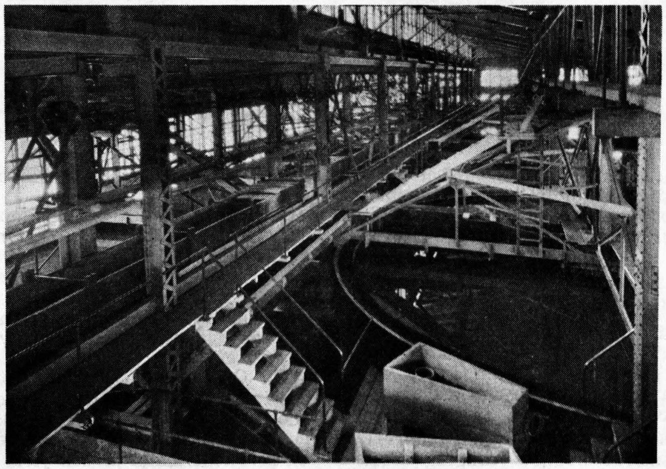


Figure 1. 40 ft. thickeners in north flotation plant, Lake Linden.

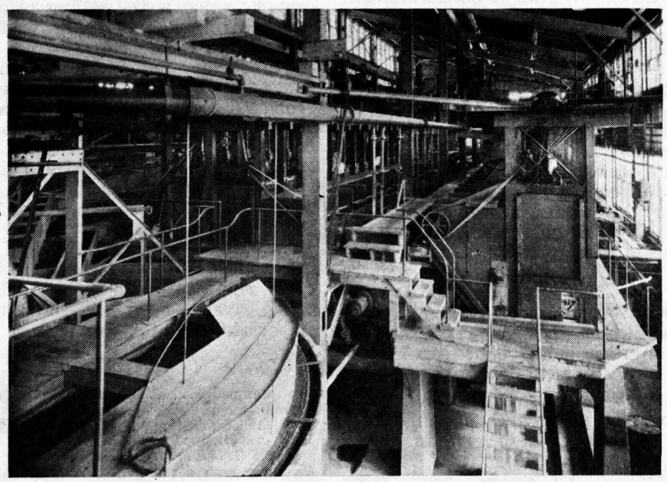


Figure 2. Standard minerals separation flotation machines, north flotation plant, Lake Linden.

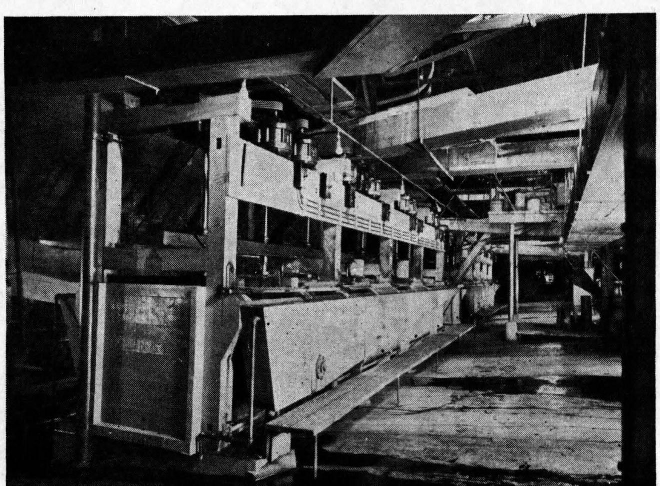


Figure 4. Fahrenwald flotation machines, Ahmeek mill.

FLOTATION

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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

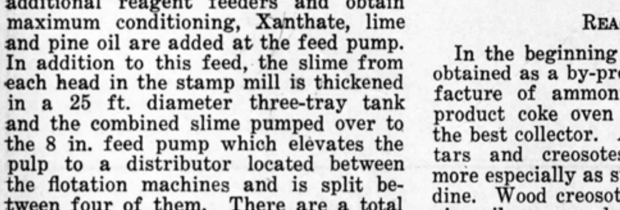
G. N. S. No. 5 pine oil containing 3 percent A. T. & T. No. 11 pine oil is used. The oil consumption is higher here than in western practice, as it is impractical under local conditions to settle the mill tailings and reuse the water.

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.

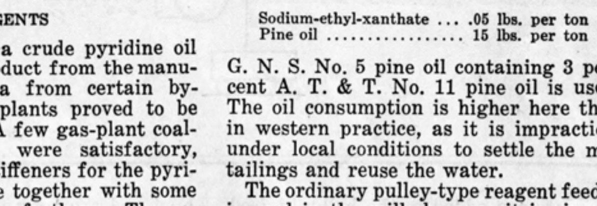
FLOTATION ON AMYGDALOID

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

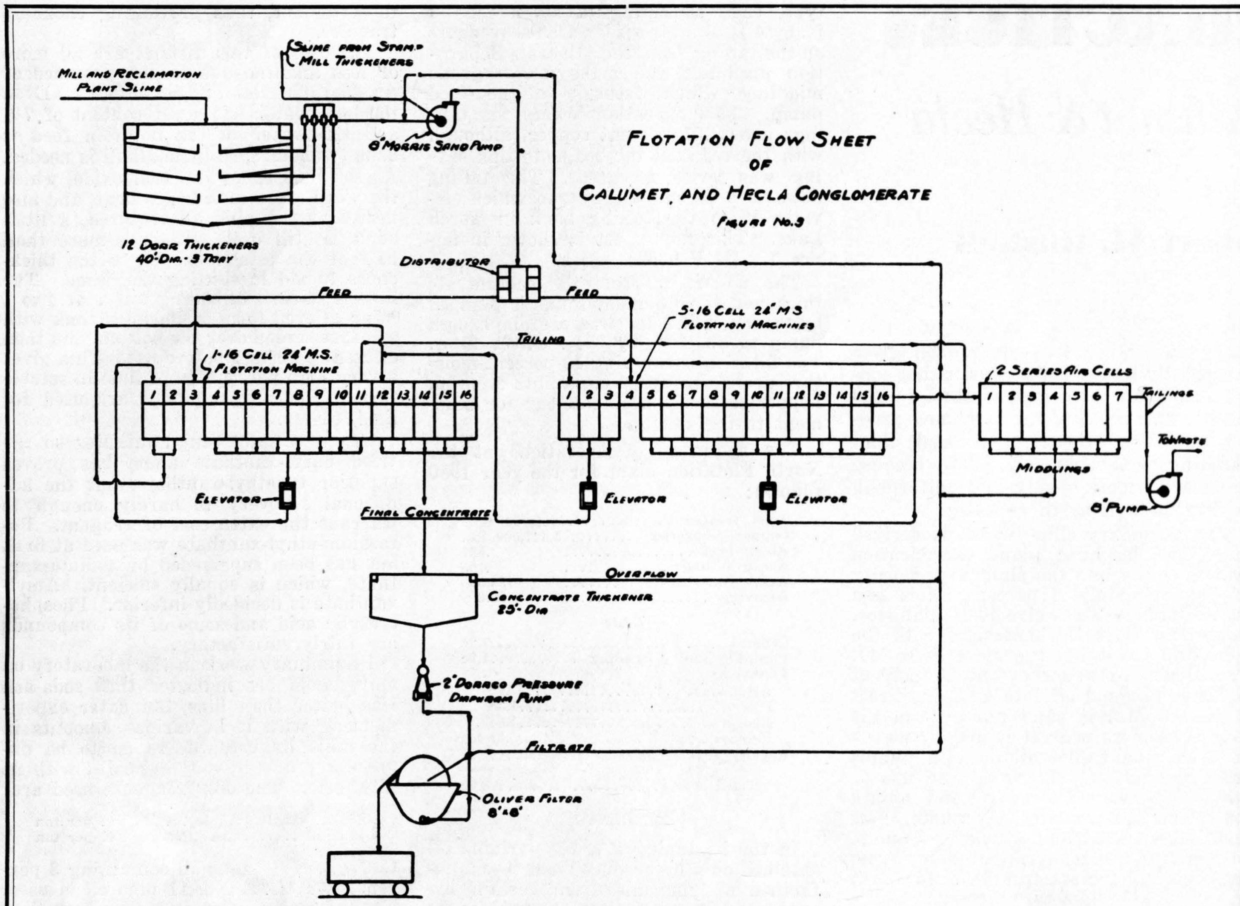


Figure 3

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 Percent Wt.	Flot. Feed Assay	Flot. Tailing Percent Wt.	Flot. Tailing Assay	Final Tailing Percent Wt.	Final Tailing 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

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"

COPY

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.

Point Mills Sand Bank

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.m.t

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--- omic

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