5-1-1940

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Recommended Citation

Eck, Laurence T., "The Testing and Concentration of a Low Grade Copper-Nickel Ore." (1940). *Bachelors Theses and Reports, 1928 - 1970.* Paper 108.

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THE TESTING AND CONCENTRATION OF A LOW GRADE COPPER - NICKEL ORE

By

Laurence T. Eck

A Thesis
Submitted to the Department of Mineral Dressing
in Partial Fulfillment of the
Requirements for the Degree of
Bachelor of Science in Metallurgical Engineering

MONTANA SCHOOL OF MINES
BUTTE, MONTANA
MAY, 1940

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THE TESTING AND CONCENTRATION OF A LOW GRADE COPPER - NICKEL ORE

The purpose of this thesis is to investigate the possibility of concentrating a low grade copper-nickel ore, to determine the most effective method of concentration, and to attempt to draw some conclusions to determine whether or not the concentrates produced would be of sufficient high grade to make the recovery of nickel and copper a profitable enterprise.

A serious handicap to the solution of the economic problem is that the location and size of the deposit from which the sample was taken are unknown as no record was available with the sample.

PROCEDURE IN TESTING

Standard methods for testing were used, [1] and follow in the order performed:

1. MACROSCOPIC EXAMINATION

On the examination of the sample with the aid of a hand lens, it was noted that the valuable minerals were associated with dark, basic or ultra-basic minerals. This tends to lead to the speculation that the deposit is of complex igneous origin, possibly similar to that of the Sudbury district in Ontario. The only mineral that was definitely identified by this visual inspection was chalcopyrite, the identification of the remainder of the minerals being postponed until the microscopic examination.

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[1] W. F. Dietrich, A. L. Engel, and Morris Guggenheim, "Ore Dressing Tests and Their Significance," U.S. Bur. of Mines R. I. 3328, pp. 5-29

2. CRUSHING AND SAMPLING

The total amount of ore available (about three sacks) was coned and quartered to a sample of about 50 pounds. This sample was then crushed in stages through rolls from one-half inch size to 10 mesh. This entailed crushing once followed by screening to remove the -10 mesh material, the process being repeated until the whole sample was screened through a 10-mesh screen.

3. ASSAY OF HEAD SAMPLE

About 100 grams of the -10 mesh sample was split out and ground to minus 100 mesh for quantitative work. The analysis of the ore is given in Table I.

TABLE I

CHEMICAL ANALYSIS OF HEAD SAMPLE

Cu	2.33%
Ni	0.70%
Fe	26.6 %
Insol.	46.04%

Because of insufficient time and inadequate equipment for analysis, no attempt was made to determine gold, and metals of the platinum group. However, in many of the nickel ores tested by the Canadian Department of Mines, the above mentioned metals occur, [2] and are probably present in this ore. But the prime interest of this investigation is to recover the copper and nickel, even though the gold and platinum group metals may be present to add to the value of the ore.

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[2] W. B. Timm, "The Selective Flotation of the Lower Grade Nickeliferous Pyrrhotite Ores of Ontario," Can. Dept. of Mines Bull. No. 617, 1923, p. 137.

4. SIZING ANALYSIS

A sizing analysis was conducted on a 500-gram sample of the ore, wet screening being used to clean the particles of the slimes so that the heavy liquid in the float-and-sink determinations would not become too contaminated with this fine material. The ore was then dried, and the sample (minus the slimes) was screened in a nest of sieves on the Ro-Tap shaker. Table II presents the results of the screen analysis.

TABLE II

WET SCREEN ANALYSIS OF ORE

SIZE	WEIGHT	WEIGHT %	CUM. WT.%
-10 + 14	45.5	9.15	9.15
-14 + 20	111.0	22.25	31.40
-20 + 28	85.7	17.20	48.60
-28 + 35	61.5	12.34	60.94
-35 + 48	46.0	9.24	70.18
-48 + 65	33.0	6.63	76.81
-65 +100	30.0	6.02	82.83
-100+150	20.0	4.02	86.85
-150+200	18.0	3.61	90.46
-200	47.5	9.54	100.00
	498.2	100.00	

No assays were made on the above screen sizes for copper and nickel before float-and-sink tests were performed.

5. FLOAT-AND-SINK ANALYSIS

Each size from the wet screening was treated to separate the gangue from the valuable minerals by heavy liquid separation. The heavy liquid used in the tests was acetylene tetrabromide (sp. gr. 2.95). For sizes larger than 48-mesh, the separation was made in 250 cc.

beakers; for finer particles a 500 cc. separatory funnel was used. In the case of these larger sizes, the sample was split down by means of a micro-splitter to about 40 or 50 grams. This sample was then placed in the beaker nearly filled with the heavy liquid. The minerals with a density greater than the fractionating medium sank, while those with a lower density floated. The float fraction was skimmed off with a small wire-screen scoop, drained, and thoroughly washed to remove the last traces of the heavy liquid. Next, the heavy liquid was drained from the sink fraction which remained in the beaker. This product was then washed, dried, and weighed.

The finer sizes were separated by means of the separatory funnel as previously stated. The sink product was drawn off first and collected on a filter paper. The float product was next tapped onto a filter paper and both products drained, washed, and dried as previously noted. Careful washing was necessary to remove all heavy liquid in order to prevent the products becoming charred when dried. A tabulation of the float-and sink results is found in Table III.

TABLE III

FLOAT AND SINK ANALYSIS

MESH	WT. SINK	WT. FLOAT	% SINK	% FLOAT
-10 + 14	39.34	5.08	88.58	11.42
-14 + 20	47.53	7.31	86.67	13.33
-20 + 28	43.50	8.51	83.65	16.35
-28 + 35	37.82	7.93	82.69	17.31
-35 + 48	38.09	7.66	83.26	16.74
-48 + 65	26.77	6.39	80.72	19.28
-65 +100	23.76	6.23	79.20	20.80
-100+150	15.66	4.51	77.60	22.40
-150+200	13.65	4.31	76.00	24.00

6. MICROSCOPIC STUDY

Preparation of Briquettes

In preparing for a study of the minerals under the microscope, a briquette was made of the -28 plus 35 mesh sink product. This entailed thorough mixing of equal volumes of the ore and bakelite. The mixture was then placed in the bottom of the briquette mold, and a calculated volume of bakelite added to fill the mold. The plunger was placed in position, and the mold heated by an electric coil. When a temperature of 130° C. was reached, the briquette was compressed by means of the plunger and an hydraulic jack to a pressure of approximately 10,000 pounds per square inch. After allowing a short time for the bakelite to set, the briquette was removed from the mold and trimmed on an emery wheel. Small pits were removed by grinding the specimen by hand with 600 carborundum on a glass plate. The specimen was then further ground and polished on a mechanical polishing machine using copper or lead laps with elutriated carborundum or alundum. This treatment formed a plane surface on the briquette with little relief between the hard and soft minerals.

Identification of Minerals

In order to determine the mineralogical composition of the ore, the briquetted specimens were examined under the microscope. However, because of the similarity of the colors of pentlandite, $(Ni,Fe)S$, pyrite (FeS_2), pyrrhotite (Fe_nS_{n+1}), and chalcopyrite $(CuFeS_2)$, as viewed through the microscope, microchemical tests were attempted to distinguish the minerals from one another.

Chalcopyrite was determined by its brassy-yellow color, its tendency to scratch more readily than the other minerals, and by microchemical tests.[3] After taking the mineral particle into solution with dilute nitric acid on a small glass plate, it was noted that on the addition of potassium mercuric thiocyanate, small star-like and sphenoid-like crystals appeared in the solution under the microscope, indicating the presence of copper.

A particle of mineral tentatively identified as pentlandite was treated similarly, giving small spheroids of nickel mercuric thiocyanate to indicate the presence of nickel.

Microchemical tests for pyrrhotite were unsuccessful because pentlandite and pyrrhotite both gave tests for iron as well as nickel. The reason for this is made evident on page 7.

Magnetite was identified by its shiny metallic luster as well as by its hardness. Various parts of the specimen were tested for cobalt, but positive results were not obtained.

In order to identify the minerals positively, the

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[3] M. N. Short, "Microscopic Determination of the Ore Minerals", U.S. Geol. Surv. Bull. 825, plates III and IV, p. 136.

method suggested by Professor A. M. Gaudin[4] was attempted. It was found that an excellent differentiation was obtained by staining the briquetted and polished minerals for six minutes at 62° F. by using a solution of chromic oxide, hydrochloric acid, and water. The polished pentlandite was stained a brilliant blue color; pyrrhotite assumed a dark bronze hue whereas the chalcopyrite remained unchanged. This staining also gave a clue to the mineralogical relationships of the minerals.

The inferences that were drawn from the examination are as follows:

1. The pentlandite, chalcopyrite, and pyrrhotite are very intimately associated with each other. It seemed that the pentlandite and pyrrhotite, however, more often occur together, with the pentlandite seemingly replacing the pyrrhotite in places. Small blebs of pentlandite a few microns in diameter were observed to be scattered throughout the pyrrhotite, but there was no such occurrence of the pentlandite with chalcopyrite. Figures I and II demonstrate this relationship.

2. A dark gangue mineral was found but unidentified. It seemed, however, that small blebs of the pentlandite were also present in this mineral. Because of the scarcity of its occurrence, the amount of pentlandite carried by this mineral is practically negligible.

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[4] A. M. Gaudin, "Staining Minerals for Easier Identification in Quantitative Mineragraphic Problems," Economic Geology, Vol. 30, (1935) pp. 558-560.

Fig. I

Pentlandite very finely-disseminated through pyrrhotite.
(X 205)

Fig. II

Pentlandite disseminated through pyrrhotite

(X 205)

LEGEND:	pr	pyrrhotite
	pn	pentlandite

This relationship is shown in Figure III.

Figure III
X 205

LEGEND:

mg	titaniferous magnetite
si	dark silicate
pn	pentlandite

3. The acid stain etched the magnetite somewhat so than an intergrowth structure appeared. This would lead to the conclusion that the mineral is titaniferous. This conclusion has no definite bearing on the problem of separating the copper minerals from the nickel bearing minerals, but demonstrates the effectiveness of staining mineral specimens to facilitate identification of the various minerals present. An example of this is shown also in Figure III.

8. SIZE OF LIBERATION

The float and sink products of each size were examined under the Ultro-Pak microscope to determine the approximate size of liberation. After inspection, it was concluded that an excellent liberation of chalcopyrite from pyrrhotite occurred at approximately 100 mesh. Taking this conclusion as a starting point, chemical analyses for copper and iron were run on the float products of the minus 35 plus 48-, minus 48 plus 65-, and the minus 65 plus 100-mesh products. The analytical results are given in Table IV.

TABLE IV

ANALYSES OF FLOAT PRODUCTS.

MESH	%Cu	%Fe
-35 plus 48	0.10	10.19
-48 plus 65	0.075	4.17
-65 plus 100	0.075	2.55

It can be seen that there is very little copper tied up with the gangue in the case of the minus 65 plus 100 mesh product, so that if all of the ore were ground through 100 mesh, this percentage will probably be reduced more.

Thus, with the size of liberation in the 100-mesh range, the type of treatment is limited to flotation or magnetic concentration. The International Nickel Company of Canada for a few years used magnetic concentration, but selective flotation has superseded this type of concentration in their more recent installations.[5] It is evident that selective flotation offered improved concentration, hence the change. Also, magnetic concentration of weakly paramagnetic minerals is usually applied not to the ore, but to gravity concentrates.[6] Furthermore, the specific gravities of the three

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[5] W.T. MacDonald, "Selective Flotation Mill at Copper Cliff", Engineering and Mining Journal, Vol. 30, (1930), p. 465.

[6] A. M. Gaudin, Principles of Mineral Dressing (New York, 1939), p.457.

minerals are all close to 5.0, so that investigations along this line would not seem practicable.

Although the chalcopyrite was liberated from the pyrrhotite near 100 mesh, grinding to this size by no means liberated the small blebs of pentlandite from the pyrrhotite; these small blebs were only a few microns in width or diameter. However, since the pentlandite is either free or included in the pyrrhotite, it seemed likely that selective flotation and separation of the copper-bearing mineral from the nickel-bearing mineral could be effected without grinding to any finer size.

9. FLOTATION TESTS

With the concentration of the low grade nickel ore limited to flotation methods, there were three methods of approach to the problem.

One method was to assume a copper smelter in the vicinity of the mill and to make a high grade copper concentrate, discarding the nickel by the rejection of the pyrrhotite. The nickel present would be relatively low, but would very likely be subjected to deductions because of contamination of the electrolyte in the electrolytic refining of the copper. This method would probably result in a high ratio of concentration, but would probably eliminate the recovery of gold and metals of the platinum group.

The second method was to make a bulk concentrate of all the sulphides. This would recover all of the copper, nickel, and rare metals mentioned above. However, with a copper smelter, the deductions would again be high because of the nickel. The ratio of concentration would be relatively low.

The third method is to assume that both copper and nickel treatment plants are in the vicinity of the concentrator, and to make a concentration similar to the process used by the International Nickel Company in the Sudbury district.[7] From tests performed on the Sudbury ores,[8] it has been found that by making a concentrate high in copper and low in nickel, the concentrate could be treated in a copper smelter as a copper concentrate with the small amount of nickel slagged off in basic-lined converters. This product could then be treated for recovery of the nickel in nickel converters, or the nickel in the blister copper could be recovered as a by-product of the copper electrolytic refining.

The copper-nickel cells could then float the remaining copper and nickel from the reject of the copper cells to give a concentrate containing as much of the nickel as possible. This concentrate could then be refined in the nickel smelter.

Thus, about half of the copper could be removed and smelted in a copper smelter at a comparatively lower cost than if all the ore were treated as a nickel ore in the nickel treatment plant.

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[7] W. T. MacDonald, op. cit., p. 465.

[8] Ibid., p. 465

This latter method was selected as a basis to the solution of the problem because of the unknown location of the deposits of the ore being tested, and also because results could be compared with that of present selective flotation practice.

Grinding

In the grinding, the required amount of minus 10 mesh ore for the flotation cell used (500 or 600 grams) was placed in a porcelain pebble mill. Flint pebbles were added to approximately 35% mill volume. Water was added until the level was about one inch below the top of the pebbles. In order to prevent over-grinding, stage grinding was practiced so that the ore was ground for a short time, the minus 65 mesh material being screened through a 65 mesh screen, and the plus 65 returned to the mill and ground for another short time. This process was repeated until 95% or more of the material passed through the given screen. The number of stages and length of each grind is indicated in Tables VI, VII, VIII, and ~~VIII~~ IX, which present data from the flotation tests. The undersize was filtered on a suction filter, the cake removed to a pan, covered with water, and placed in the flotation cell for testing.

Reagents

After a study of numerous flotation reagents used in the flotation of copper-nickel ores, a tentative list of reagents was compiled. These are given in Table V.

TABLE V

REAGENTS SUGGESTED FOR THE FLOTATION OF Cu-Ni ORES

FUNCTION	REAGENT	APPROX. CONC. (lb. per ton)
Conditioner	Soda Ash	5
Frothers	Cresylic acid	0.03
	Pine Oil	0.03
Collectors	Aerofloat 15	0.03
	Aerofloat 25	0.03
	Potassium Amyl Xanthate	0.10

A preliminary test to determine the recovery with these reagents resulted in what seemed to be an excellent selective separation. Therefore these same reagents were used throughout the tests.

The general procedure for the addition of these reagents is as follows:

1. After the ore had been placed in the cell and the pulp thoroughly agitated with the correct amount of water to give a pulp dilution of 3:1 to 4:1, the soda ash was added and allowed to condition in the cell for 5 to 10 minutes. The chalcopyrite showed a marked tendency to film with just the conditioner present, although there was no stable froth.
2. A sample was then removed for the determination of the pH of the pulp.
3. Cresylic acid was then added and the pulp thoroughly agitated with the air shut off for a few minutes. The bubbles formed carried a high percentage of chalcopyrite.
4. With the air still turned off, Aerofloat 15 was next added and allowed to mix with the pulp for a few minutes.

When the air was turned on, the bubbles formed floated the chalcopyrite along with small amounts of pyrrhotite. This froth was somewhat brittle, but was stable enough to float the chalcopyrite.

5. The addition of Aerofloat 25 and potassium amyl xanthate as collectors along with pine oil as a frother floated the remaining copper and nickel minerals in a thick, heavy froth.
6. When the froth became barren, the test was concluded, a sample taken for pH of the final pulp, the products dried, weighed, and briquetted for microscopic examination.

In two of the tests performed, the copper rougher concentrate was placed in a 250 gram cell and cleaned up to remove the major portion of the small amount of pyrrhotite which was floated with the chalcopyrite. A flow sheet showing the steps taken in testing the ore is given on page 16.

FLOTATION TESTS NO. 1 and 2

(both samples ground to -65 mesh)

OBJECT: To compare the recoveries obtained by cleaning up the rougher concentrate in one case (test No. 2) with that obtained as a result of no cleaning operation (test No. 1). Both samples of ore were ground to an average of 90 to 95% minus 65 mesh, and the reagents used were identical. Tables VI and VII present the data obtained.

FLOW SHEET OF LABORATORY FLOTATION CONCENTRATION

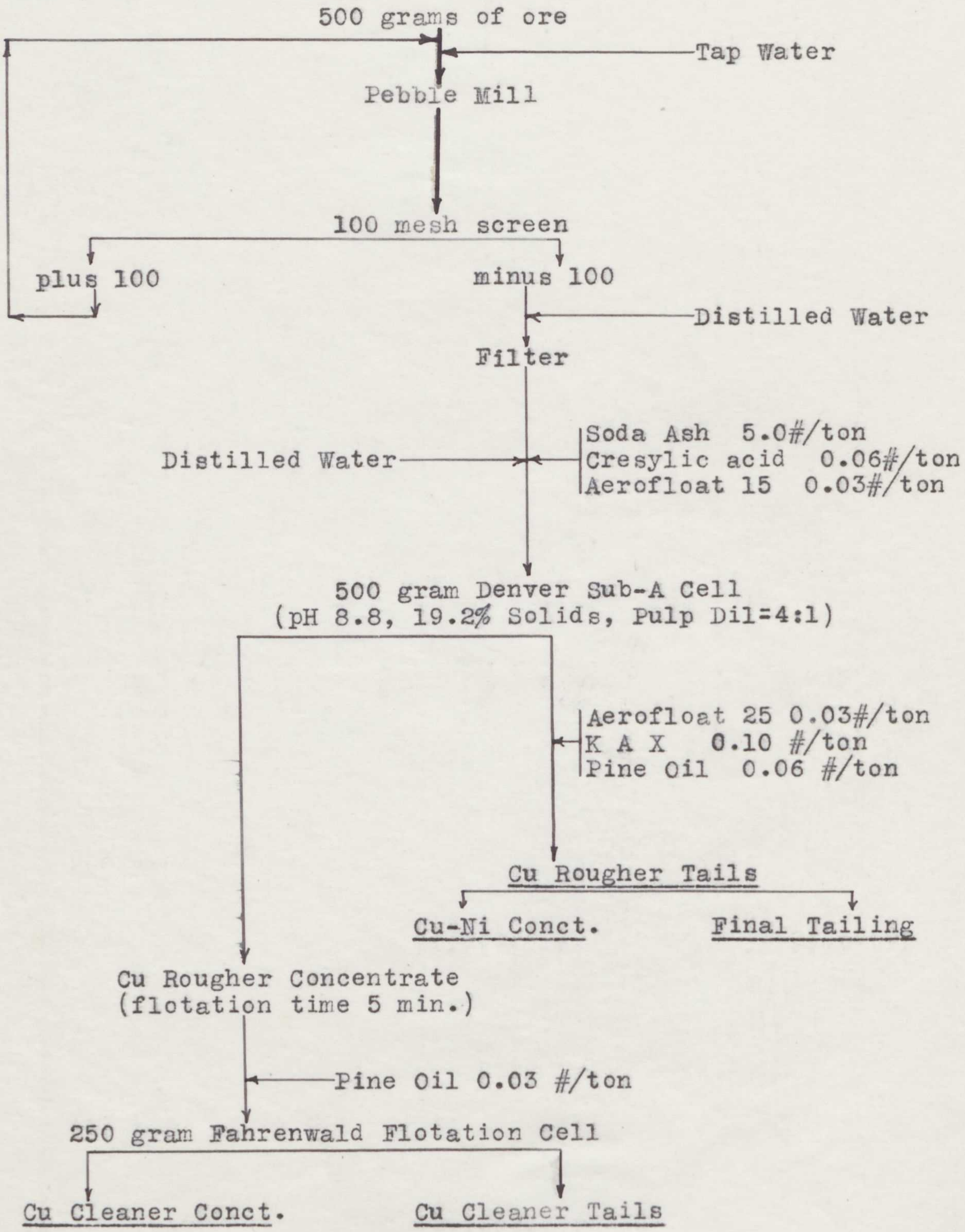

TABLE VI

FLOTATION TEST NO. 1

Grind:

Primary: minus 10 mesh

Final: Abbe mill; 90% -65 mesh; time 5 min. + 5 min.

Water:

Grind: tap water

Flotation: distilled water

Pulp consistency: 12/1

Pulp dilution: 4/1

Percent Solids 19.2%

500 gram Denver cell used.

PRODUCT	WT. (gms)	WT.%	ASSAY (%) Fe	Cu	Ni	RECOVERY (%) Fe	Cu	Ni
Cu Rougher	50.6	12.0	30.48	12.17	1.34	16.15	56.70	29.10
Ni Rougher	150.2	35.6	37.44	2.25	0.923	58.60	31.10	59.32
Tails	220.9	52.4	10.98	0.60	0.122	25.25	12.20	11.58
Composite	421.7	100.0	22.74	2.57	0.55	100.00	100.00	100.00

REAGENTS	COPPER ROUGHER (pounds per ton)	NICKEL ROUGHER
Soda Ash (pH 8.8)	5	
Cresylic acid	0.06	
Aerofloat 15	0.03	
Aerofloat 25		0.03
KAX		0.10
Pine Oil (final pH 8.0 to 8.2)		0.10

TABLE VII

FLOTATION TEST NO. 2

Grind:

Primary: minus 10 mesh
Final: Abbe mill; 95% minus 65 mesh; time 1 min.+ 5 min. +5 min.+ 6 min. + 10 min.

Water:

Grind: Tap water
Flotation: Distilled water

Pulp consistency: 8/1
Pulp dilution: 2.75/1
Percent solids: 27%

600 gram Fagergren cell used for roughing.
250 gram Fahrenwald-type for cleaning.

PRODUCT	WT. (gms)	WT.%	ASSAY (%) Fe	Cu	Ni	RECOVERY (%) Fe	Cu	Ni
Cu Cleaner	45.6	8.08	29.99	14.00	1.065	9.55	45.40	16.25
Cl. Tails	17.2	3.04	20.48	4.10	0.67	3.65	5.00	3.85
Cu Rougher	62.8	11.12	30.10	11.22	0.95	13.20	50.40	20.10
Ni Rougher	164.5	29.05	46.55	3.65	1.09	53.50	42.45	59.90
Tails	338.5	59.83	14.11	0.30	0.177	33.30	7.15	20.00
Composite	565.8	100.00	25.35	2.50	0.53	100.00	100.00	100.00

REAGENTS	COPPER ROUGHER (pounds per ton)	NICKEL ROUGHER	CLEANER
Soda ash (pH 8.8)	5		
Cresylic acid	0.06		
Aerofloat 15	0.03		
Aerofloat 25		0.03	
KAX		0.10	
Pine Oil		0.10	0.03
TIME	22 min.	15 min.	20 min.

REMARKS: 1 drop of pine oil added to cleaner cell to increase froth.

DISCUSSION OF RESULTS

The results of flotation test No. 1 show a copper concentrate containing 12% copper and 1.34% nickel with 57% of the original amount of copper in the ore being recovered. The nickel concentrate runs much lower in nickel than the copper concentrate, but the large amount of pyrrhotite floated still contains nearly 60% of the nickel in the ore. The total recovery of copper in both concentrates is about 88% while that of the nickel is about 89%. The over-all ratio of concentration is 2.1:1. The ratio of copper to nickel in the copper concentrate is about 9:1.

The results of test No. 2 where the copper rougher concentrates were cleaned indicate a concentrate containing 14% copper and 1% nickel with only 45% of the original amount of copper in the ore in this concentrate. In this case, the nickel concentrate has approximately the same percentage of nickel as that of the copper concentrate, but 60% of the nickel is recovered in the nickel concentrate. The total recovery of copper is about 93% while the maximum nickel recovery is only 80%.

The cleaning raised the grade of the copper concentrate and seemingly raised the recovery over that of test No. 1. The recovery of nickel in the copper concentrate was reduced 13%, thereby increasing the ratio of copper to nickel to 14:1. The over-all ratio of concentration was raised in this case to 2.5:1, but this advantage is overshadowed by the high nickel loss in the tailings.

FLOTATION TESTS NO. 3 and 4

OBJECT: To compare the recoveries obtained by cleaning the rougher concentrate in one case (test No. 4) with that obtained as a result of no cleaning operation (test No. 3). Both samples of ore were ground to an average of 90% minus 100 mesh, and the flotation tests conducted with identical flotation reagents. One drop of pine oil was the only reagent added to the cleaner cell to produce a stable froth. Tables VIII and IX present the data obtained.

TABLE VIII

FLOTATION TEST NO. 3

Grind:
Primary: minus 10 mesh
Final: Abbe mill; 90% minus 100 mesh; time+1 min. + 5 min.
+5 min. +10 min.
+5 min.

Water:	**Grind: tap water**	**Pulp consistency:**	12/1
	Flotation: distilled water	**Pulp** dilution:	4/1
		Percent solids:	19.2%

500 gram Denver cell used.

PRODUCT	WT. (gms)	WT.%	ASSAY (%) Fe	Cu	Ni	RECOVERY Fe	Cu	Ni
Cu Rougher	50.6	10.35	25.58	12.90	0.73	10.71	58.80	12.22
Ni Rougher	192.0	39.75	40.96	1.80	1.17	65.90	31.40	74.70
Tails	241.0	49.90	11.56	0.45	0.16	23.40	9.80	13.08
Composite	483.0	100.00	24.71	2.28	0.623	100.00	100.00	100.00

REAGENTS	COPPER ROUGHER (Pounds per ton)	NICKEL ROUGHER
Soda Ash (pH 8.8 - 9.0)	5	
Cresylic	0.06	
Aerofloat 15	0.03	
Aerofloat 25		0.03
KAX		0.10
Pine Oil		0.10
TIME	25 min.	16 min.

TABLE IX

FLOTATION TEST NO. 4

Grind:
Primary: minus 10 mesh
Final: Abbe mill; 90% minus 100 mesh; time+1 min. + 5 min.
+5 min. +10 min.
+5 min.

Water:
Grind: tap water
Flotation: distilled water

Pulp consistency: 8/1
Pulp dilution: 2.75/1
Percent solids: 27%

600 gram Fagergren cell used for roughing.
250 gram Fahrenwald-type for cleaning.

PRODUCT	WT. (gms)	WT.%	ASSAY (%) Fe	Cu	Ni	RECOVERY (%) Fe	Cu	Ni
Cu cleaner	64.0	10.98	32.83	16.15	1.097	13.93	71.20	19.30
Cl. Tails	16.0	2.74	32.54	3.1	0.98	3.52	3.40	4.32
Cu Rougher	80.0	13.72	32.00	13.53	1.07	17.45	74.60	23.62
Ni Rougher	202	34.70	45.77	1.45	1.28	63.10	20.22	71.50
Tails	300	51.58	9.51	0.25	0.059	19.45	5.18	4.88
Composite	582	100.00	25.21	2.49	0.63	100.00	100.00	100.00

REAGENTS	COPPER ROUGHER	NICKEL ROUGHER	CLEANER
	(pounds per ton)		
Soda ash (pH 8.4)	5		
Cresylic acid	0.06		
Aerofloat 15	0.03		
Aerofloat 25		0.03	
KAX		0.10	
Pine Oil		0.10	0.03
TIME	27 min.	15 min.	15-20 min.

DISCUSSION OF RESULTS

The results of test No. 3 show a copper concentrate containing 13% copper and o.74% nickel with 59% of the original amount of copper in the ore recovered in this concentrate. The nickel concentrate assays 1.17% nickel with 75% of the nickel in the ore in this concentrate. The total recovery of copper in both concentrates is 90% while the maximum nickel recovery is 87%. The over-all ratio of concentration is 2:1 while the ratio of copper to nickel in the copper concentrate is about 17.5:1.

The data obtained from test No. 4, where the copper rougher concentrates were cleaned show a concentrate containing 16% copper and 1% nickel which represents 71% of the total copper in the ore. The nickel concentrate averages 0.2% higher in nickel than the copper concentrate with 71% of the nickel being recovered in this operation. The total recovery of copper in the concentrates is 95% with a correspondingly high recovery of nickel. The over-all ratio of concentration is 2.1:1 and the copper to nickel ratio in the copper cleaner concentrate is 14.7:1.

The cleaning here likewise raised the grade of the copper concentrate and also gave the maximum recovery of copper in all of the tests run. The recovery in the nickel concentrate was somewhat lower than in test No. 3, but the total recovery was also the maximum in the tests run.

EXAMINATION OF BRIQUETTED CONCENTRATES

In the examination of the copper concentrate, it was observed that nearly all of the pentlandite present was there not as blebs in pyrrhotite, but as relatively large, free particles as compared to these blebs. This indicates that the nickel-bearing mineral, if free from pyrrhotite, will float easier than the pyrrhotite with small blebs of pentlandite in it. This would lead to the general conclusion that it would be difficult to lower the nickel content in the concentrate unless some depressant for free pentlandite were used because of the free state of some of this mineral. The fact that pentlandite floats more easily than pyrrhotite is confirmed by MacDonald.[9]

A comparatively small amount of pyrrhotite was floated with the chalcopyrite, but these grains are probably mechanically held particles with the chalcopyrite. Several good cleaning operations would probably remove most of the small amount of pyrrhotite present. There were no locked particles of pyrrhotite and chalcopyrite in this concentrate.

Upon examining a briquette of the copper-nickel concentrate, it was found that pentlandite was present both as relatively-large free particles and as blebs in the pyrrhotite. The chalcopyrite present in the examination of the -65 mesh briquette showed some locking with the pyrrhotite indicating that the size of liberation had not quite been reached. This fact will probably account for the copper present in the nickel concentrate in flotation tests 1 and 2 due to the relative hardness of chalcopyrite and pyrrhotite. The major por-

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[9] W. T. MacDonald, op. cit., p. 470

tion of the pyrrhotite present after stage grinding is in the range of -65 plus 100 mesh while the range of most of the chalcopyrite minerals is approximately -150 plus 200 mesh. However, if the ore is ground to pass 100 mesh, this locking of particles of pyrrhotite and chalcopyrite will practically be eliminated.

CONCLUSIONS

From the results of the flotation tests performed, it may be seen that a copper concentrate containing 16% copper and 1% nickel can be made. Although this concentrate is not as high in copper as that of the copper concentrates of the International Nickel company, the ratio of copper to nickel (16:1) is very nearly the same as that obtained in actual practice by that company.[10] Then, too, approximately 70% of the copper in the ore can be recovered in this concentrate whereas the recovery in the commercial concentrator in the Sudbury district is possibly slightly over 50%.[11] Since this concentrate is treated in the copper smelter near Sudbury,[12] it may be reasonable to expect that if such a concentrate can be treated, any other copper smelter could do likewise. Transportation charges, of course, would enter into the considerations of profit and treatment.

The nickel concentrate produced contains about 1.4% copper and 1.3% nickel. This concentrate, too, is probably much lower in copper and nickel than that at International

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[10] W. T. MacDonald, op. cit., p. 465

[11] Ibid., p. 465

[12] Ibid., p. 465

Nickel due to the much lower grade of the ore tested as compared to the grade of the mill feed at Sudbury.[13] However, 70% of the nickel is recovered in this concentrate, and can be treated directly in the nickel smelters. The remainder of the nickel in the copper concentrate can be recovered either from the copper electrolyte or by treating the copper converter slag in nickel converters.[14]

The total ratio of concentration is only 2:1. This ratio compares favorably with commercial practice at Copper Cliff, (2:1) since tests on the Sudbury ore indicated that in order to recover all of the rare metals plus the pentlandite carrying the nickel, it was necessary to float nearly all of the sulphides.[15] A method for rejecting pyrrhotite would give a much larger ratio of concentration, but there would be very little recovery of nickel because of the previously stated mineralogical relationship of pyrrhotite to pentlandite.

The total recovery of copper as well as nickel were respectively 94% and 95% as compared to recoveries of 97% and 93% obtained at the concentrator in the Sudbury district.[16] This indicates that the recoveries obtained in the tests compare favorably with commercial practice. With careful regulation of variables in a properly-designed and equipped plant, the recoveries obtained can probably be raised somewhat from that obtained in the testing.

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[13] W. T. MacDonald, op. cit., p. 472

[14] Ibid., p. 465

[15] Ibid., p. 467

[16] Ibid., p. 472

However, the present concentrator at International Nickel treats about 8000 tons of ore per day that averages 3% nickel and 3.4% copper.[17] It seems logical that in order to treat an ore containing a 2.3% copper with only 0.70% nickel, a concentrator equally as large as that at Copper Cliff would be required to make the operation a profitable one. The ore would have to be mined on a large scale similar to the open pit methods used for the porphyry coppers because of the low grade of ore.

The aforementioned possibilities, however, are hinged upon the size of the ore body. It would be an expensive mistake to construct such a concentrator with less than five or ten years supply of ore.

And were all previously mentioned factors favorable including distance from railroad as well as distance from a copper smelter and a nickel smelter, a concentrator with at least 8000 tons per day capacity would be required in order to make the operation profitable because of the low grade of the ore. Any conclusions more definite than the above would be difficult to draw because nickel smelter charges, transportation costs, and all related costs are not easily obtained.

TABLE X

COMPARISON OF TEST RESULTS WITH COMMERCIAL RESULTS

	ORE TESTED	INCO
Grade of mill feed:		
nickel:	0.70%	3.0%
copper:	2.30	3.40
Ratio of Cu to Ni in Cu conct:	15:1	15 to 20:1
% Solids in Cu rougher feed:	19-27%	27-35%
pH of Cu rougher feed	8.4 -8.8	8.0-8.5
% Cu in Cu rougher conct:	16%	25%
% Ni in Cu rougher conct:	1%	1.25%
Total ratio of concentration:	2:1	2:1

A possible flow sheet for the treatment of the ore is found on page 27.

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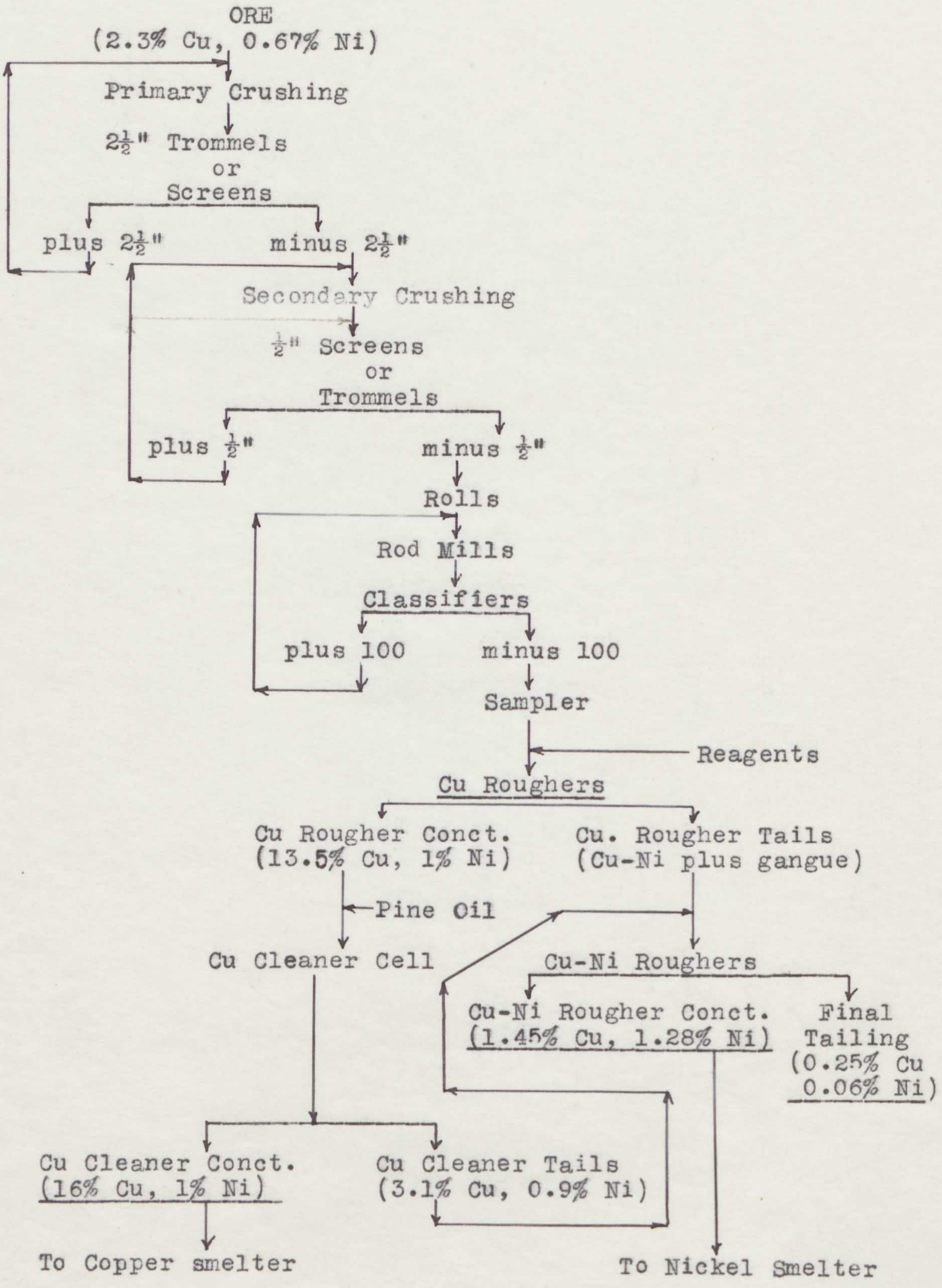
[17] W. T. MacDonald, op. cit., p. 472.

PROPOSED FLOW SHEET (QUALITATIVE)



SUGGESTIONS FOR FURTHER STUDY

1. Throughout these tests, the pH of the pulp was not changed during each test run. This resulted on the addition of the soda ash, in a pH between 8.5 and 9.0 in all four tests. From inspection of the pH values of various products in the flotation section of the concentrator at Copper Cliff, it seems that the pH of the feed to the copper-nickel cells is about 7.6 while that of the feed to the copper cells is 8.2.[18] From this data, along with results obtained by McAllister in work done with a copper-nickel ore,[19] it appears that careful control of the pH around 8.5 in the copper cells and the reduction of the pH in the nickel cells to about 7.0 would probably reduce the nickel loss in the tailings to bring about a higher total recovery.

2. The effect of varying amounts of reagents should also be studied in order to determine the least amount required to obtain a maximum recovery.

3. The effect of other combinations of reagents should be investigated. Reagents in general use other than those applied in the tests are tabulated in Table XI.

TABLE XI

LIST OF FLOTATION REAGENTS IN GENERAL USE
FOR SELECTIVE COPPER - NICKEL FLOTATION

		(lb. per ton)
COLLECTORS:	Ketone residue	-
	Flotogen (mercaptobenzothiazole)	-
	Coal tar creosote	0.20
	Acid creosote	0.16

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[18] W. T. MacDonald, <u>op. cit</u>., p. 467

[19] J. A. McAllister, "A Preliminary Study of Montana Copper-Nickel Ore", <u>B. S. Thesis</u>, Montana School of Mines, p.7

TABLE XI (CONT'D.)

COLLECTORS: (cont'd)	T-T Mixture	0.15
	X-Y Mixture	-
	Thiofizzan	-
DEPRESSANTS: (for nickel)	Sodium cyanide	0.20
	Potassium dichromate	0.20
CONDITIONERS:	Lime	1-2
	Sulphuric acid	0.5
	Sodium pyrophosphate	-

Cyanide has the effect of reducing the nickel content in the copper concentrate, but due to its depressing action on all sulphides, lower recoveries of both metals result.[20] However, it seems that potassium dichromate will depress nickel without affecting the recovery of copper.[21]

According to a report of the Canadian Department of Mines concerning the addition of lime, it seems that "The essential point is to carry an alkaline pulp, obtained by the addition of lime, soda ash, or caustic soda. Lime has the tendency to prevent some of the nickel from floating, but gives the higher grade product. Soda ash gives the best recovery, but a lower grade product. A combination of the two seems to give the best results."[22]

The above references seem to furnish a method of attack for improving the recovery as well as for making a better

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[20] C. S. Parsons, G. B. Walker, and A. K. Anderson, "The Concentration of the Copper-Nickel Ore of the Falconbridge Nickel Mines, Ltd., Garson, Ontario", Can. Dept. of Mines No. 724, p.43.

[21] Ibid., p. 42

[22] C. S. Parsons, "The Concentration of the Lower Grade Copper-Nickel Ores," Can. Dept. of Mines No. 586 (1921), p. 156

separation of the various products.

4. The control of temperature, effects of using mill water to find the effect of soluble salts in the ore, and the results of using iron balls with an iron mill should be investigated.

5. The effect of regrinding the tailings would be interesting to investigate.

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ACKNOWLEDGEMENTS

The writer expresses his appreciation and gratitude to Doctor S. R. B. Cooke, head of the Department of Ore Dressing at the Montana School of Mines, for his aid and assistance in the work represented by this thesis.

The writer is also indebted to Professor L. J. Hartzell and Mr. H. Rush Spedden for suggestions and loan of equipment.