The Non-Metallic Laboratory—for investigations into the treatment and utilization of non-metallic ores and minerals. The laboratory is 60 feet by 58 feet, has two floors and is equipped with both small- and large-scale machinery and apparatus.

CANADA

DEPARTMENT OF MINES

HON. CHARLES STEWART, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER

MINES BRANCH

JOHN McLEISH, DIRECTOR

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(*Testing and Research Laboratories*)

1925

OTTAWA
F. A. ACLAND
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
1926

No. 670

Annual reports on Mines Branch investigations are now issued in four parts, as follows:—

Investigations of Mineral Resources and the Mining Industry.

Investigations in Ore Dressing and Metallurgy (Testing and Research Laboratories).

Investigations of Fuels and Fuel Testing (Testing and Research Laboratories).

Investigations in Ceramics and Road Materials (Testing and Research Laboratories).

Other reports on Special Investigations are issued as completed.

MINES BRANCH INVESTIGATIONS IN
ORE DRESSING AND METALLURGY, 1925

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm
Chief of Division

During the year 1925 the laboratories of the Division were devoted entirely to experimental test and research work in connexion with the treatment of Canadian ores and metallurgical products. In order that the laboratories might be equipped with the latest apparatus for test operations new equipment was purchased, such as a Fahrenwald classifier, three new quarter-size concentrating tables, a rod mill, and a six-hearth, mechanically rabbled, roasting furnace. A new non-metallic laboratory is being equipped to study the preparation and utilization of Canadian non-metallic minerals, on completion of which more extensive experimental test work will be carried on. A pamphlet descriptive of the non-metallic laboratory will shortly be issued. On request of the National Research Council their investigatory engineers were accommodated with the use of the laboratories for their investigation of Canadian magnesites.

The investigations have been carried out under the direction of W. B. Timm, Chief of Division of Ore Dressing and Metallurgy; those in Section II on metallic ores, under the immediate supervision of C. S. Parsons, assisted by J. S. Godard, and those on non-metallic ores under R. K. Carnochan; those in Section III under the immediate supervision of R. J. Traill, assisted by W. R. McClelland; and those in Section IV under H. C. Mabee, Chief Chemist of the Division, assisted by A. E. Smaill. A brief review and summary of the investigations is as follows:—

Section II (pages 11-71, inclusive)

The reports under this section include the results of the investigations conducted in the Ore Dressing Laboratories by C. S. Parsons, R. K. Carnochan, and J. S. Godard.

Report No. 221 (pages 11-13, inclusive)

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE BAIN MINE, INDIAN LAKE, MASHAM TOWNSHIP, QUE., BY C. S. PARSONS

The ore from the Bain mine represents a type of molybdenite ore found in eastern Canada, in which the molybdenite is of the medium flake variety, associated with massive iron pyrite and pyrrhotite. It varies from the ore of the Moss mine, Quyon, Que., in which the molybdenite with an appreciable amount of iron sulphides, is disseminated throughout a gangue of quartz, feldspar, and fluorite (altered syenite gneiss) and from the ore of the Benjamin mine, south of Amos, Que., in which the molybdenite occurs in feldspathic quartz veins with sericite. The report shows that due to the presence of so much iron sulphides, it was a more difficult ore to concentrate by flotation than the other two types mentioned. A lower grade concentrate was obtained on account of not obtaining the same differential action between the molybdenite and the gangue minerals. It was shown, however, that on an ore containing 1 per cent MoS_2, a concentrate averaging 80 per cent MoS_2 could be obtained with a recovery of 88 per cent of the MoS_2 in the ore. Although 80 per cent MoS_2 concentrate may not be so satisfactory a product as the higher grade of 90 per cent and over for certain purposes, it is quite satisfactory for the production of ferro-molybdenum, for which purpose the greater part of the production of molybdenite concentrates is used. Included in the report is a flow-sheet followed in conducting the test. This flow-sheet will be of interest to those intending to develop this type, or any type of molybdenite deposit, as it gives a good idea of the equipment required for concentration.

Report No. 222 (pages 13-22, inclusive)

THE CONCENTRATION OF THE ZINC-IRON MIDDLING DUMP AT NOTRE DAME DES ANGES STATION, QUE., BY C. S. PARSONS

When the zinc-iron middling dump at Notre Dame des Anges station was being concentrated by the British Metal Corporation (Canada), Ltd., a part of the dump was encountered on which poor recoveries were being made. A shipment consisting of 15 tons of the material and 15 tons of the tailings from this material was submitted to determine where the trouble lay and whether it could be overcome. The report gives the results of small-scale preliminary tests and of tonnage check tests on both samples, and shows that by using the proper reagents and control of pulp densities, etc., high-grade products could be obtained with good recoveries and that the difficulties experienced could be overcome. The report is illustrated by flow-sheets used in conducting the tests.

Report No. 223 (pages 23-28, inclusive)

THE CONCENTRATION OF SILVER-LEAD-ZINC ORE FROM THE ENTERPRISE MINE, SLOCAN DISTRICT, B.C., BY C. S. PARSONS

The report on the concentration of this ore shows that it can be concentrated by selective flotation which gives a lead concentrate and a zinc concentrate, both marketable products, with good recoveries of the silver, lead, and zinc values in the ore. The report deals in a general way with the concentration of the Slocan ores containing lead, zinc, and silver values, the author claiming that straight selective flotation would give a simpler flow-sheet and thus lower the costs of plant installation and plant operation, and also would give higher recoveries of the values, with a greater monetary return from the sale of the products, than a combination of gravity concentration and flotation, although the latter process gives a higher grade lead concentrate containing less zinc.

Report No. 224 (pages 28-36, inclusive)

THE TREATMENT OF THE GOLD-COPPER ORE OF THE ARGONAUT MINE, LARDER LAKE, ONT., BY J. S. GODARD

The presence of copper in the gold ore of the Argonaut mine has been detrimental to good results by the cyanide process. The ground ore does not settle or filter freely, making more difficult the replacement of the dissolved values after leaching. The consumption of cyanide is high and poor recoveries are obtained. A series of tests were made to obtain a comparison of results by straight cyanidation, amalgamation, and combinations of amalgamation, concentration, and cyanidation. The tests show that the ore is amenable to amalgamation and concentration by flotation with good results and that the tailing can be cyanided with a moderate consumption of cyanide after the large bulk of the copper is eliminated by concentration. Whether cyanidation will pay for the extra recovery of the gold values after concentration is problematical. The installation of a customs smelter in the Rouyn district, Que., will assist materially the operation of this property with respect to disposal of the copper-gold concentrates.

Report No. 225 (pages 36-39, inclusive)

THE RECOVERY OF FOUNDRY METAL FROM SWEEPINGS, BY J. S. GODARD

The tests show a simple method for the recovery of metal from foundry sweepings. A simple installation of a small ball mill, screening device, and concentrating table can be used to advantage in such foundries having an accumulation of sweepings, which are being wasted or shipped out of the country for the recovery of the metal. By such a method, the metal in the sweepings can be recovered.

Report No. 226 (pages 39-42, inclusive)

AMALGAMATION AND CONCENTRATION OF A MATACHEWAN GOLD ORE, BY C. S. PARSONS AND J. S. GODARD

The report covers the treatment of the gold ore from the Young-Davidson mine at Matachewan, Ont. A large body of low-grade ore has been developed on this property. The values are too low to permit of any extensive process such as grinding to 200 mesh for cyanidation. The gold values being associated with the sulphides permits of grinding to about 30 mesh, amalgamation, and concentration of the sulphides with the recovery of the gold in the sulphides by regrinding and cyanidation. By such a process, 85 per cent of the gold values in an ore averaging $2.80 per ton can be recovered with greatly reduced cost of treatment over an all-sliming cyanidation process. Such a process would seem applicable to other ore-bodies of a similar character.

Report No. 227 (pages 43-45, inclusive)

THE PREPARATION OF VOLCANIC ASH FROM WALDECK, SASK., FOR INDUSTRIAL PURPOSES, BY R. K. CARNOCHAN

The test work on this material included a method of grinding and separation of the ground material into various products for such uses as oil filtering, hand cleanser, household cleanser, and metal polish. A flow-sheet of a simple plant is given in the report. Such a treatment as described is now being used for the preparation of the various products given above.

Report No. 228 (pages 45-47, inclusive)

THE CONCENTRATION OF A DRY SILVER ORE FROM THE SLOCAN SILVER MINES, LTD., ALAMO, B.C., BY C. S. PARSONS

The report gives the results obtained from a dry silver ore from the Slocan district. On No. 1 sample representing the bulk of the ore a high-grade silver concentrate can readily be obtained by flotation with recoveries of over 90 per cent of the silver values. Samples Nos. 2 and 3 being oxidized, good recoveries were not obtained. The ore represented by the latter samples was of sufficient grade to be shipped without concentration.

Report No. 229 (pages 48-50, inclusive)

THE CONCENTRATION OF A COPPER ORE FROM PITT LAKE, B.C., BY J. S. GODARD

The report gives the results obtained on a copper ore carrying silver and gold values from Pitt Lake, B.C. The ore is very amenable to concentration by flotation grinding to 65 mesh. A high-grade copper concentrate is produced carrying the silver and gold values with recoveries between 90 and 95 per cent of the copper values.

Report No. 230 (pages 50-51, inclusive)

THE CONCENTRATION OF A COPPER-LEAD-ZINC ORE FROM ALBERT COUNTY, N.B., BY C. S. PARSONS

The ore on which the experimental test was conducted was a mixture of fine sulphides of iron, copper, lead, and zinc, carrying some gold and silver values. The results of the test show that the ore can be concentrated into a copper-lead product carrying the most of the precious metals and over 90 per cent of the copper and lead values, and into a 50 per cent zinc product, with a recovery of over 80 per cent of the zinc values. The results were obtained by selective flotation.

Report No. 231 (pages 52-56, inclusive)

THE CONCENTRATION OF THE COPPER ORE ("C" ORE-BODY) OF THE HORNE MINE, NORANDA MINES LTD., ROUYN, QUE., BY J. S. GODARD

The report on the concentration of the ore of "C" ore-body of the Horne mine shows that it can be concentrated with good recoveries of the copper values and fair recoveries of the gold values. In the concentration of this class of ore, selective action of the copper cannot be carried too far, as it is done at the expense of the gold values. The gold seems to be associated to some extent with the pyrite in the ore and the aim should be to make a concentrate of 12 to 15 per cent copper containing as well the pyrite in the ore, eliminating in the tailing only pyrrhotite and gangue. When such a concentrate is made it contains 95 per cent of the copper values and 80 per cent of the gold values. The economic point is one of balancing smelter costs of treating the lower grade concentrate against the loss of gold values in the higher grade concentrate and also in determining whether it is cheaper to smelt the ore direct as against concentration to this grade with the loss of 5 per cent of the copper and 20 per cent of the gold. The 20 per cent loss of gold values is less than 20 cents per ton of ore of the grade submitted.

Report No. 232 (pages 56-58, inclusive)

THE CONCENTRATION OF A LEAD-ZINC ORE FROM THE KICKING HORSE MINE, FIELD, B.C., BY C. S. PARSONS

The ore consisted of galena and sphalerite carrying silver values. Grinding to about 60 mesh was necessary to free the sulphides from each other and from the gangue. Selective flotation at this mesh gave good results. A recovery of 80 per cent of the lead and of 85 per cent of the zinc in a zinc concentrate assaying 50 per cent zinc could be expected. Ninety per cent of the silver values report in the two concentrates, about half in each. A higher grade lead concentrate containing less zinc could very probably be obtained in practice by recleaning the lead concentrate.

Report No. 233 (pages 58-63, inclusive)

EXPERIMENTAL TESTS ON A GOLD ORE FROM THE CONTACT MINES, PAULSON, B.C., BY J. S. GODARD

The experimental tests conducted comprised amalgamation, table concentration, flotation and cyanide tests, and combinations of these, to determine which would be more suitable for the treatment of the ore. Cyanidation gave good results on the sample submitted, but a change in the character of the ore, such as the presence of more copper and zinc might readily affect its treatment by this process. Over 50 per cent of the gold values is recoverable by amalgamation so that a combination of amalgamation and flotation would seem to be more applicable, especially as the property is fairly favourably situated for the disposal of concentrate to a smelter. Flotation alone gave good results, over 90 per cent of the values being recoverable in a concentrate assaying 1·6 ounces gold per ton and 2·7 ounces silver per ton, a concentration of the values of over three times that in the ore.

Report No. 234 (pages 63-64, inclusive)

THE CONCENTRATION OF A ZINC ORE FROM RENFREW, ONT., BY J. S. GODARD

The report shows that the ore is readily concentrated to a product containing over 50 per cent zinc with recoveries of about 95 per cent of the zinc values. In the samples submitted for test purposes only traces of lead were present. Other samples from the deposit showed lead to be present, in which case, selective flotation of the lead from the zinc would have to be applied in order to recover a high percentage of the lead values.

Report No. 235 (pages 65-66, inclusive)

THE CONCENTRATION OF A LEAD ORE FROM THE FRONTENAC MINE, PERTH ROAD, ONT., BY J. S. GODARD

The test work consisted of some small table tests to determine the grade of concentrate and recoveries that could be expected. It showed that the ore would be amenable to concentration by jigging and table concentration with good recoveries of the lead in a high-grade product.

Report No. 236 (pages 66-68, inclusive)

THE CONCENTRATION OF A COPPER-ZINC ORE FROM THE AMULET MINE, ROUYN, QUE., BY C. S. PARSONS

The report is interesting in that it shows that the copper-zinc ores of western Quebec can be concentrated by selective flotation into copper and zinc products. High-grade copper concentrates were obtained from this ore containing 90 per cent of the copper values. High-grade zinc concentrates were obtained indicating a possible recovery of 75 per cent of the zinc values. A great deal more research should be done on this class of ore to obtain a better selective action between the copper and the zinc with the lowering of the zinc content in the copper concentrate or in the re-treatment of the copper concentrate for the same purpose. Preliminary work along these lines should be followed by large-scale tests on a tonnage scale. The recovery of the gold and silver values is also of importance.

Report No. 237 (pages 68-71, inclusive)

THE CONCENTRATION OF A ZINC-SILVER ORE FROM THE WONDERFUL MINE, SANDON, B.C., BY J. S. GODARD

The ore from the Wonderful mine was being concentrated in the Alamo mill, Alamo, B.C., by gravity concentration on tables to recover the lead values, flotation of the zinc to recover the zinc values, and flotation of the slimes from gravity concentration to recover the silver values in them. This made a fairly complicated flow-sheet and the recovery of the values was poor. The results of the tests show that selective flotation without gravity concentration would give good results if the ore contained sufficient lead values to warrant such a process. The report shows that the lead content in the sample submitted is too low to produce a high-grade lead concentrate and the ore must be regarded as a zinc-silver ore. Straight flotation of the ore gives a high-grade zinc-silver concentrate containing 50 per cent zinc, 35 ounces silver per ton, with recoveries of 95 per cent of the zinc and 90 per cent of the silver values in the ore. The adoption of a simple process of straight flotation, disregarding the small amount of lead in the ore would give good results.

Section III (pages 72-88, inclusive)

The reports under this section include the results of investigations conducted in the hydrometallurgical or electrochemical laboratory by R. J. Traill and W. R. McClelland. These officers give a summary of the work accomplished during the year on "the treatment of iron sulphide ores (pyrrhotite and pyrite) for the production of electrolytic iron, with the recovery of sulphur and other metals as by-products." This report is a continuation of the work of 1924, and shows the progress made with this investigation. Progress was made on the conversion of pyrite to artificial pyrrhotite, on the purification of the electrolytes and on the electrodeposition of the iron from solution. The work on this investigation was somewhat curtailed by the taking up of a similar investigation on the treatment of Canadian ilmenites.

Report No. 238 (pages 78-81, inclusive)

A HYDROMETALLURGICAL TREATMENT FOR PYRRHOTITE OF LOW GOLD AND COPPER CONTENT FROM THE NORANDA MINES, LTD., ROUYN TOWNSHIP, QUE., BY R. J. TRAILL AND W. R. MCCLELLAND

The report gives a method of treatment for the massive pyrrhotite ore-bodies of the Horne mine, too low in copper and gold to be economically workable for the recovery of the copper and gold values alone, but which might possibly be worked for the recovery of the iron and sulphur content with the copper and gold as by-products. The report shows that on an ore containing 50·3 per cent iron, 1·2 per cent copper, 0·37 per cent zinc, and 0·10 ounce gold per ton, that 89·5 per cent of the iron can be recovered as pure electrolytic iron (99·96 per cent Fe), 88·6 per cent of the copper can be recovered in the process, 60·4 per cent of the sulphur can be recovered as elemental sulphur, and that the gold values have been concentrated in the residue in the ratio of 1 : 8, making this residue an ore from which the gold values and remaining copper values can be recovered in smelting operations. These results were obtained on two preliminary tests; better results as to sulphur and iron recovery could no doubt be obtained by further test and research. The report shows a possible method of treatment for this class of ore. Whether such a process would be economically possible will depend on the cost of power, plant installation, and operating costs, the latter two items determined by pilot-plant scale operations.

Report No. 239 (pages 81-88, inclusive)

A NEW PROCESS FOR THE TREATMENT OF ILMENITE ORES FOR THE PRODUCTION OF ELECTROLYTIC IRON AND TITANIUM OXIDE CONCENTRATE FOR PIGMENT AND OTHER PURPOSES, BY R. J. TRAILL AND W. R. MCCLELLAND

The report outlines a new process for the treatment of ilmenite ores and titaniferous iron ores, whereby the iron content in the ores is metallized and leached with ferric chloride, the iron deposited as pure electrolytic iron (99·96 per cent Fe), leaving a high-grade titanium oxide residue to be used for the manufacture of titanium pigments, titanium salts, and ferro-titanium. Results are given of the metallization of the iron to the sponge state; the leaching of the sponge; the purification of the electrolyte, and the electro-deposition of the iron. The detailed results of a test example on ilmenite ore from Ivry, Que., are given, showing that the process is applicable from a metallurgical standpoint. Sufficient laboratory research has been conducted to warrant the installation of a small pilot plant to determine cost data.

Section IV (pages 89-94, inclusive)

In this section, the progress made on the investigation into a method of treatment of heavy sulphide ores for the recovery of the iron content, as well as the other contained metals, is given. When heavy sulphide ores of copper and nickel are reduced in a furnace, under non-oxidizing conditions using an alkali flux to slag off the gangue minerals, there results a low-grade iron sulphide matte containing the copper, nickel, and precious metal values. This matte disintegrates to a fine powder in a humid atmosphere. When roasted and chloridized, the copper and a large percentage of the nickel is rendered soluble leaving an iron oxide residue suitable for reduction to pig iron or nickel-iron pig. The soluble metals are treated for their recovery. The report contains the results obtained during the year from experimental tests on lots up to 500 pounds of ore or concentrates. When consideration is given to the rate at which iron ore reserves are being used up by industrial expansion, the time is fast approaching when the iron content in sulphide ores will have to be utilized. Canadian sulphide ore-bodies being worked for their other metal contents such as copper, nickel, zinc, and the precious metals, contain large quantities of iron, which are going out into the rock, tailing, and slag dumps. An economical process which will recover this iron in the process of treatment for the other metals would be of great national importance.

Section V (page 95)

A brief summary of the work of the chemical laboratories of the Division is given. The report shows the diversified character of the work, which was mainly on the determination of the metal values in the ores and products from test operations. B. P. Coyne, R. A. Rogers, and L. Lutes were continuously employed on this work, the first two performing the analytical and the latter the fire assay work. Considerable analytical work was also performed in the electrochemical laboratory in connexion with the electrolytic iron investigations. This included analysis of the ores and products of test operations with numerous control samples of the operations. The laboratories were also used by the National Research Council who had a technical officer working in them for a part of the year.

Section VI (pages 96-123, inclusive)

This section includes two articles of the Mines Branch Memorandum Series, Nos. 22 and 25, on Molybdenite and Graphite, and one on the ores of western Quebec. They are inserted to preserve in printed form the information contained therein. The articles on molybdenite and graphite deal with the concentration of the ores from the viewpoint of the operator and millman. The practical side of milling operations is stressed. The article on the ores of western Quebec is a continuation of Section III of the Investigations of Mineral Resources and the Mining Industry, 1924, and deals with the character of the ores and their metallurgical treatment. The articles are as follows:—

THE CONCENTRATION OF CANADIAN MOLYBDENITE ORES, BY W. B. TIMM AND C. S. PARSONS (pages 96-108, inclusive)

The report sub-divides Canadian occurrences of molybdenite ores into their several types and gives concrete examples of the results obtained from the concentration of each type of ore, the procedure followed, the flow-sheet used, grinding, control of pulp densities, reagents, and manipulation of plant machinery to obtain the best results. In a number of cases the tests were performed on carload lots of ore, under actual milling conditions, so that the results obtained are those that could be expected in practice.

THE CONCENTRATION OF CANADIAN FLAKE GRAPHITE ORES, BY C. S. PARSONS

(pages 109-118, inclusive)

The report deals with the general principles to be considered in the design of a concentrator for flake graphite ores. It gives two flow-sheets of modern plants, and a theoretical one embracing the author's opinion of the most suitable one for graphite ores in general. The author discusses the important points of each and gives details of operation of the grinding mills, flotation units, tables, filters, and dewaterers, driers, and finishing plant. The author's experience in the concentration of graphite ores, together with the experimental work conducted on a large number of graphite ores, has enabled him to stress the practical side of the operations, so that the report should be found of interest to graphite operators and those contemplating entering the field.

THE ORES OF WESTERN QUEBEC—THEIR CHARACTER AND METALLURGICAL TREATMENT, BY W. B. TIMM

(pages 119-123, inclusive)

The report classifies the ores into three main types, gives examples of the three types on which exploration and development work has been done, and discusses the metallurgical treatment of each. The third type, comprising the large pyrrhotite ore-bodies of the district containing commercial values in gold, copper, and zinc, is of outstanding importance at present. The treatment of the ores of this latter type will require the establishment of a large smelting industry in the district.

List of Ores and Metallurgical Products on which Experimental Tests and Research Work was Conducted

In the following table is given a list of the ores and metallurgical products received at the Ore Testing Laboratories for experimental test and research. The tabulated statement includes the report numbers, class of ore or product, source of shipment, name of shipper, and weight of shipment.

Report number	Page	Ore or product	Source of shipment	Shipper and address	Weight
					lbs.
221	11	Molybdenite	Bain mine	H. H. Claudet, Rockcliffe, Ont.	61,500
222	13	Lead-zinc	Notre Dame mine, Que.	British Metal Corporation (Canada), Ltd., Montreal.	60,000
223	23	Silver-lead-zinc	Enterprise mine, Silverton, B.C.	E. C. Wragge, Nelson, B.C.	100
224	28	Gold-copper	Argonaut mine	Argonaut Mines Ltd., Larder Lake, Ont.	80
225	36	Foundry sweepings.	Canada Bronze foundry, Montreal	Canada Bronze Company, Montreal.	400
226	39	Gold	Young-Davidson mine, Matachewan, Ont.	Porcupine Goldfields Development and Finance Co., Montreal.	16,312
227	43	Volcanic ash	Waldeck, Sask.	W. C. Vance, Val-Kel Cleaners, Ltd., Swift Current, Sask.	300
228	45	Silver	North fork Carpenter creek, Slocan district, B.C.	Slocan Silver Mines, Ltd., Alamo, B.C.	200
229	48	Copper-silver	Pitt mine, Pitt Lake, B.C.	Pitt Mining Company, Vancouver.	218
230	50	Copper-lead-zinc.	Albert county, N.B.	John E. Teahan, Sr., Kerry P.O., N.B.	119
231	52	Copper-gold	"C" ore-body, Horne mine.	Noranda Mines, Ltd., Rouyn, Que.	115
232	56	Lead-zinc	Kicking Horse mine, Field, B.C.	Pacific Mines, Ltd., Vancouver.	424
233	58	Gold	Contact mine, Paulson, B.C.	K.V. Mines Ltd., Vancouver	117
234	63	Zinc	Pucker Street mine, Renfrew, Ont.	Alderson and MacKay, Montreal	192
235	65	Lead	Frontenac mine, Perth Road, Ont.	J. M. Forbes, Ottawa	56
236	66	Copper-zinc	Amulet mine, Rouyn, Que.	Amulet Mines, Ltd., Rouyn, Que.	80
237	68	Zinc-silver	Wonderful mine, Sandon, B.C.	Cunningham Mines, Ltd., Alamo, B.C.	235
238	78	Copper-gold	Noranda mine, Rouyn, Que.	Noranda Mines, Ltd., Rouyn, Que.	60
239	81	Iron-titanium	Ivry mine, Ivry, Que.	McArthur-Irwin, Ltd., Montreal	200
240	89	Copper-nickel	Murray mine, Nickelton, Ont.	Toronto General Trust Co., Toronto.	4,000

II

REPORTS OF INVESTIGATIONS: ORE DRESSING AND METALLURGICAL LABORATORY

Report No. 221

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE BAIN MINE, INDIAN LAKE, QUEBEC

C. S. Parsons

Shipment.—A carload shipment of 30·75 tons of molybdenite ore was received March 17, 1925, from the Bain mine, at Indian lake, Masham township, Hull district, Que. The shipment was submitted by H. H. Claudet, Esq., Rockcliffe, Ottawa, Ont.

Purpose of Experimental Test.—A large-scale tonnage check test was desired to determine the amenability of this type of molybdenite ore to concentration.

Characteristics of the Ore.—The molybdenite is of the medium-large flake variety associated with a large amount of massive iron pyrite. The gangue is pyroxenite and other lime-silicate minerals. The richer parts of the ore consist almost entirely of iron sulphides. This type of ore, occurring in many parts of eastern Canada, is distinguished by the above characteristics from the more disseminated and highly siliceous types, such as the Moss mine ore, north of Quyon, Que., and that of occurrences to the south of Amos, Que. The ore showed evidence of considerable oxidization, and had apparently been taken from near the surface of the deposit.

Sampling and Analysis.—Due to the large flake and spotty nature of the ore it is very difficult to obtain an accurate sample of this type of ore. A sample of one-twentieth of the feed to the ball mill, which had been crushed to 1¼ inch by a jaw crusher, was cut by a Vezin sampler. This was reduced by graded crushing and sampling. A sample was also taken of the ground feed to the flotation cells. This wet sample showed a higher assay for the ore than the dry sample, and both samples were slightly lower than the calculated assay of the ore from the contents of the concentration products. The calculated assay of the ore showed it to contain 1·0 per cent molybdenite.

Experimental Tests

The flow-sheet, Figure 5, was followed in conducting the test. This flow-sheet has proven applicable to the concentration of most types of molybdenite ores:—

Reagents Used—

Kerosene (British American Oil Co's. "Lampolene" brand).......	0·8 pound/ton
Pine oil (steam distilled)..	0·3 " "

Screen Test of Feed to Flotation Cells—

Mesh	Weight grms.	Weight %	Cumulative %
+35	2·7	0·5	0·5
− 35+ 48	14·4	2·9	3·4
− 48+ 65	45·7	9·1	12·5
− 65+100	107·2	21·5	34·0
−100+150	74·0	14·8	48·8
−150+200	68·3	13·7	62·5
−200	187·7	37·5	100·0

Daily Record of Run—

Date	Description of sample	Weight of product	Analysis MoS$_2$
		pounds	per cent
March 23	Feed to flotation cells (classifier overflow)		1·17
	Concentrates—first two hours' operation	26·5	76·07
	Concentrates—remainder of day	31·0	79·09
	Tailings		0·11
March 25	Feed to flotation cells (classifier overflow)		0·98
	Concentrates	122·0	81·88
	Tailings		0·17
March 26	Feed to flotation cells (classifier overflow)		1·04
	Concentrates	70·0	80·70
	Tailings		0·09
March 30	Feed to flotation cells (classifier overflow)		0·85
	Concentrates	89·5	79·84
	Tailings		0·11
March 31	Feed to flotation cells (classifier overflow)		0·74
	Concentrates	74·0	78·12
	Tailings		0·13
April 1	Feed to flotation cells (classifier overflow)		0·97
	Concentrates	66·5	81·56
	Tailings		0·08
April 2	Feed to flotation cells (classifier overflow)		0·78
	Concentrates	66·0	73·38
	Tailings		0·12
April 3	Feed to flotation cells (classifier overflow)		0·65
	Concentrates	123·0	74·46
	Tailings		0·18
	Clean-up of ball mill, classifier, etc	669·0	2·98

NOTE.—Last two days' run after ore all in ball mill and circuit being run out to obtain as much concentrate as possible; concentrates of lower grade, and tailings higher than average. First two hours' run on first day, some zinc picked up from previous operations, concentrates lower grade than average, contaminated with zinc.

Screen Analysis of Concentrates—Second Day's Run—

Mesh	Weight	Molybdenite
	per cent	per cent
+ 65	22·5	86·08
− 65+100	14·0	78·11
−100+150	13·0	82·80
−150	50·5	78·32

Analysis of Picked Flakes—

Hand-picked large flakes	98·50 per cent MoS_2
Hand-picked flakes, +28 mesh concentrates	95·30 "

These analyses show that the flake contains a small amount of iron sulphides between the laminae, and it would be impossible to obtain a concentrate, by mechanical means, much over 90 per cent MoS_2.

Summary of Concentration Results—

One car of ore, net weight	61,500·0 pounds
Analysis of sample cut from dry ore by Vezin sampler	0·83 per cent MoS_2
Average analysis of daily sample wet feed to cells	0·92 "
Analysis of feed obtained by actual weight of MoS_2 recovered in concentrate and by calculation of tailing and clean-up	1·00+ "
Content in MoS_2, using 1·00+ per cent MoS_2 assay	616·72 pounds
Amount of concentrate obtained	668·5 "
Analysis of total concentrate by calculation	78·5 per cent MoS_2
Content MoS_2 in concentrate	524·58 pounds
Clean-up from run (669 pounds at 2·98 per cent MoS_2)	19·94 "
Average analysis of daily tailing samples	0·12 per cent MoS_2
Content MoS_2 in tailing	72·20 pounds
Recovery of MoS_2 by actual weights of products made	87·9 per cent
Recovery figured from formula $R = \frac{100.(H-T)\,c}{H\,(C-T)}$	88·2 per cent

CONCLUSIONS

The above results show that on an ore of this type averaging 1 per cent MoS_2, a concentrate containing 80 per cent MoS_2 can be produced with a recovery of better than 88 per cent of the molybdenite values in the ore.

The ore submitted was taken from near the surface and was more or less oxidized. This state of the ore did not have any appreciable effect on the recovery, as tailings as low as 0·08 per cent MoS_2 were produced during the run. It may, however, have had some effect on the grade of the concentrate. It is possible that on fresh ore of this grade, a concentrate of 85 per cent MoS_2 could be obtained.

The production of a high-grade concentrate from this type of molybdenite ore in which the iron sulphides predominate over the siliceous gangue minerals, is more difficult than from the more highly siliceous ores. More careful control of reagents, pulp densities, deflocculation devices, etc., is required. With proper control, the results given above should be obtained, and on freshly mined ore, it would be possible to produce higher grade concentrates.

Report No. 222

THE CONCENTRATION OF THE ZINC-IRON MIDDLING DUMP AT NOTRE DAME DES ANGES, QUEBEC

C. S. Parsons

Shipment.—A carload shipment of zinc-iron middlings was received from the British Metals Corporation (Canada), Ltd. (head office, Bank of Nova Scotia Building, Montreal). The shipment contained two kinds of material, designated as Lots Nos. 1 and 2. Lot No. 1 consisted of tailings from the flotation plant which was in operation during the summer of 1924. Lot No. 2 consisted of a part of the dump encountered just before the mill closed for the winter, and from which the tailings represented by Lot No. 1 were produced.

Location of Dump.—The dump from which these samples were taken is situated at Notre Dame des Anges, Portneuf district, Que. It was produced by the Weedon Mining Co. during the operation of the Notre Dame mine, which the company had under lease during 1915-1917. It was purchased by the British Metals Corporation in 1924, and a 100-ton flotation plant was built for the recovery of the lead and zinc. The mill was operated successfully during the summer of 1924 and produced lead concentrate and zinc concentrate, both of which were marketable.

Purpose of Experimental Tests.—Just before the concentrator was closed for the winter season, a part of the dump was encountered which had become badly oxidized. Trouble was experienced in floating this material, and in order to determine whether the results obtained could be improved upon, a sample of about 15 tons, designated Lot No. 2, representing the badly oxidized material was sent to the experimental plant of the Ore Dressing and Metallurgical Division, so that the data obtained there would be available for the resumption of operations in the spring. Lot No. 1, of about 15 tons representing the tailings obtained from this badly oxidized material, was sent to determine whether a further recovery of the zinc could be obtained by re-treatment.

Character and Analysis of Material.—Lot No. 1 contained approximately 11·5 per cent zinc as zinc blende, the balance being chiefly iron sulphides. Lot No. 2 contained approximately 24·79 per cent zinc and 2·45 per cent lead, the balance being chiefly the iron sulphides, pyrite, and pyrrhotite.

EXPERIMENTAL TESTS

Small-scale Tests on Lot No. 1

A series of small-scale tests were run before any attempt was made to operate on a tonnage scale. Different reagents and combinations were tried, the results and conclusions drawn from these tests are given below:—

Test No.	Product	Weight		Analysis Zn per cent	Distribution of Zn values per cent
		Grms.	Per cent		
1	Zinc concentrate	225·0	20·0	33·88	57·9
	Zinc middling	115·5	9·9	18·35	15·4
	Tailing	789·0	70·1	4·44	26·6
2	Zinc concentrate	165·5	16·9	51·00	68·7
	Zinc middling	127·0	13·9	21·64	22·4
	Tailing	689·4	70·2	1·59	8·9
3	Zinc concentrate	159·5	16·2	43·56	66·6
	Zinc middling	52·5	5·3	17·88	8·1
	Tailing	770·0	78·5	3·81	25·3
4	Zinc concentrate	191·0	19·2	51·40	81·6
	Zinc middling	60·0	6·0	16·39	8·2
	Tailing	745·0	74·8	1·65	10·2

Reagents Used—

Test No. 1: 10·0 lb./ton soda ash.
0·7 " neutral coal-tar creosote No. 2 (Dominion Tar and Chemical Co.).
0·25 " TT mixture.
0·05 " steam distilled pine oil.

Test No. 2: 10·0 lb./ton soda ash.
0·7 " neutral coal-tar creosote oil No. 2.
0·05 " potassium xanthate.
2·0 " copper sulphate.
0·15 " TT mixture.
0·05 " pine oil.

Test No. 3: 10·0 lb./ton soda ash.
0·7 " neutral coal-tar creosote oil No. 2.
2·0 " copper sulphate.
0·25 " TT mixture.
0·05 " pine oil.

Test No. 4: 10·0 lb./ton soda ash.
0·05 " potassium xanthate.
2·0 " copper sulphate.
0·05 " pine oil.

Summary and Conclusions

The first test was run to determine whether it was necessary to add copper sulphate. It was thought that copper sulphate having been used in the previous treatment it might not be necessary to use it again. The results of tests Nos. 1 and 3 show conclusively that copper sulphate must be added. Tests Nos. 2 and 3 were run to determine the effect of the addition of potassium xanthate. The comparison of the results of these two tests suggested that potassium xanthate was producing a lower tailing. Test No. 4 was therefore made using potassium xanthate alone. The results obtained from test No. 4 were far better than any obtained in the previous experiments, and proved the value of xanthate as a reagent for the flotation of zinc.

Small-scale Tests on Lot No. 2

As on Lot No. 1, a series of small-scale tests were run, trying out different combinations of reagents. Particular attention was given to the possibility of replacing soda ash with lime, either wholly or in part, which would reduce the cost of reagents considerably owing to the large amount of alkaline necessary to neutralize the acidity of this material. The amount required varied between 15 and 18 pounds per ton. The attempt to replace the soda ash with lime was not successful. The grade of the lead concentrate was materially lowered as was also the recovery of the zinc. The results of these tests will not be given in detail with the exception of the final test No. 6, in which potassium xanthate was used. This test is given for the sake of comparison with the results of the large-scale tests:—

Test No.	Product	Weight		Analysis		Distribution of values	
		Grms	%	Pb %	Zn %	Pb %	Zn %
6	Lead concentrate........	35·1	3·4	24·25	11·86	34·8	1·6
	Lead middling..........	56·5	5·5	10·38	22·50	24·0	4·9
	Zinc concentrate........	365·3	35·1	1·07	53·65	15·9	76·1
	Zinc middling..........	104·5	10·2	2·20	24·13	9·4	9·8
	Tailings................	474·5	45·8	0·82	4·17	15·9	7·6

Reagents Used—

Test No. 6: Lead—18·0 lb./ton soda ash.
0·2 " thiocarbanilide.
0·25 " sodium cyanide.
0·20 " cresylic acid.
Zinc— 2·0 " copper sulphate.
0·2 " potassium xanthate.
0·1 " pine oil.

Large-scale Tonnage Check Tests on Lot No. 1 (Tailings)

The flow-sheet, Figure 1, was followed in conducting the tests on the tailings. Each test was for a period of approximately six hours The information as to reagents used, pulp densities, samples taken, etc., and results obtained, are given in condensed form.

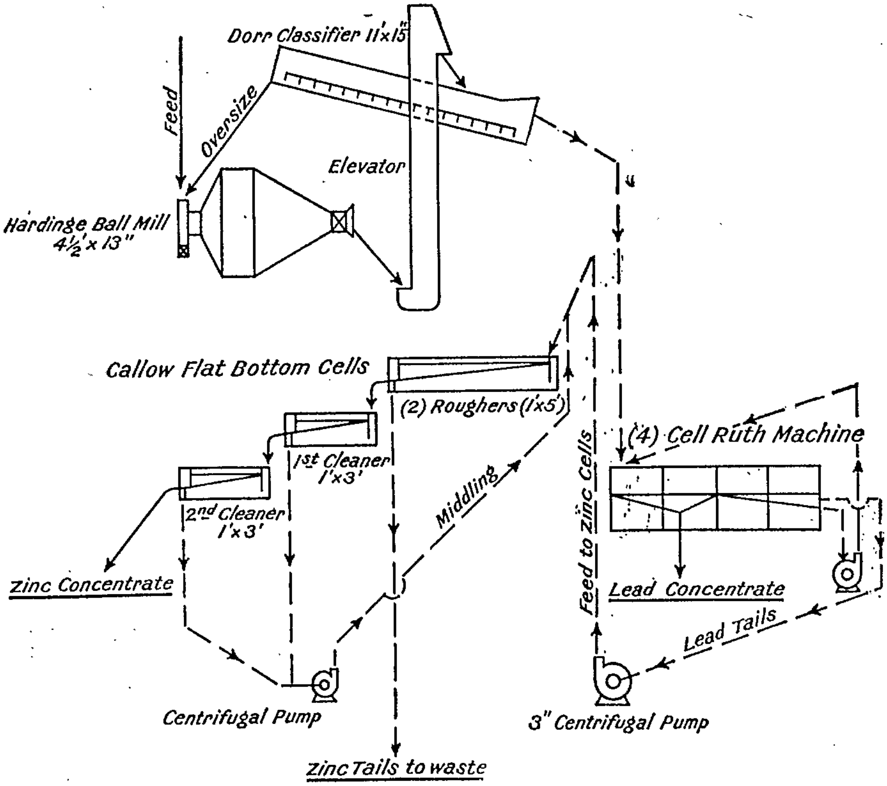


Figure 1. Flow-sheet, lot No. 1—tailings, report No. 222.

LOT NO. 1

Test No. 1

Rate of feed per hour	750 pounds.

Reagents—

Soda ash	10.00 a.m. to 12.00	8·3 lb./ton
	12.00 to 4.00 p.m.	11·0 "
Copper sulphate		2·5 "
Potassium xanthate	12.00 to 2·50 p.m.	0·07 "
	2.50 p.m. to 4.00 p.m.	0·14 "

Pulp densities—
Feed to cells, approximately 1 : 3.

Remarks—
A charge of 2,000 pounds of 1½-inch balls was used in the ball mill. The conclusion reached was that the grinding was too fine.

Key to sampling periods—

No. 1 sample	10.00 a.m. to 12.00
No. 2 sample	12.00 to 2.50 p.m.
No. 3 sample	2·50 p.m. to 4.00 p.m.

Analysis of samples—

Head sample, or feed	11·71 per cent Zn
No. 1 zinc concentrate	30·44 "
No. 1 zinc tailing	6·21 "
No. 2 zinc concentrate	27·59 "
No. 2 zinc tailing	8·59 "
No. 3 zinc concentrate	25·04 "
No. 3 zinc tailing	2·64 "

Test No. 2

Rate of feed per hour	865 pounds.

Reagents—
Soda ash-lime mixture (lime 3½ to soda 4)—

	10.50 a.m. to 11.30 a.m.	5·20 lb./ton
	11.30 a.m. to 12.35 p.m.	3·50 "
	12.35 p.m. to 2.10 p.m.	4·65 "
	2.10 p.m. to 2.45 p.m.	9·50 "
Soda ash	2.45 p.m. to 3.40 p.m.	11·00 "
	3.40 p.m to 4.10 p.m.	Nil
Pine oil		0·03 "
Copper sulphate		2·0 "
Potassium xanthate		0·3 "
TT reagent		0·1 "

Pulp densities—
Feed to cells averaged 1 : 2·22.

Remarks.—1,500 pounds of balls was removed from the mill leaving only 500 pounds.

Key to sampling periods—

No. 1 samples	11.30 a.m. to 12.30 p.m.
No. 2 samples	12.30 p.m. to 2.20 p.m.
No. 3 samples	2.20 p.m. to 3.45 p.m.
No. 4 samples	3.45 p.m. to 4.10 p.m.

Analysis of samples—

Feed or head sample	10·89 per cent Zn
No. 1 zinc concentrate	46·62 "
No. 1 zinc tailing	11·45 "
No. 2 zinc concentrate	45·00 "
No. 2 zinc tailing	4·21 "
No. 3 zinc concentrate	40·50 "
No. 3 zinc tailing	4·48 "
No. 4 zinc concentrate	35·58 "
No. 4 zinc tailing	3·97 "
Analysis of total concentrate	42·81 "

Test No. 3

Rate of feed per hour.. 300 pounds

Reagents—

Reagent	Period	Amount	
Soda ash-lime mixture (lime 2 to soda 4)—			
	10.20 a.m. to 12.20 p.m.	8·3	lb./ton
	12.20 p.m. to 1.45 p.m.	5·0	"
	1.45 p.m. to 4.45 p.m.	10·0	"
Copper sulphate		2·5	"
Potassium xanthate	10.25 a.m. to 1.45 p.m.	0·3	"
	1.45 p.m. to 4.00 p.m.	0·2	"
Pine oil		0·03	"
Neutral creosote oil No. 2	4.00 p.m. to 4.45 p.m.	0·3	"

Pulp densities—
Feed to cells averaged 1:1·25.

Key to sampling periods—

Samples	Period
No. 1 samples	12.45 p.m. to 1.45 p.m.
No. 2 samples	1.45 p.m. to 3.25 p.m.
No. 3 samples	3.25 p.m. to 4.05 p.m.
No. 4 samples	4.10 p.m. to 4.45 p.m.

Analysis of samples—

Sample	Zn
Feed or head sample	11·17 per cent Zn
No. 1 zinc concentrate	41·03 "
No. 1 zinc tailing	1·60 "
No. 2 zinc concentrate	42·76 "
No. 2 zinc tailing	1·62 "
No. 3 zinc concentrate	38·38 "
No. 3 zinc tailing	1·29 "
No. 4 zinc concentrate	46·73 "
No. 4 zinc tailing	6·82 "
Total concentrates	41·45 "

Summary and Conclusions, Lot No. 1.—These tests show conclusively that the tailings can be re-treated and a fair grade of zinc concentrate produced with a low tailing. There seems to be no advantage in substituting lime for part of the soda ash required. Any saving in the cost of reagent is offset by a lower recovery. Potassium xanthate stands forth as a reagent of great merit; a saving in cost of reagents as compared to TT mixture is effected as well as an increase in the grade of the concentrate and in the recovery. In actual operation, judging from the results of the small-scale tests, it is safe to assume that a 50 per cent zinc concentrate can be produced, with a tailing containing less than 2·0 per cent zinc, using soda ash pulp.

Large-scale Tonnage Check Tests on Lot No. 2 (Middlings)

The flow-sheet, Figure 2, was followed in conducting the tests on the zinc-iron middlings, with the exception of test No. 6. Each test was for a period of approximately 6 hours. The information as to reagents used, pulp densities, samples taken, etc., and results obtained, is given in condensed form.

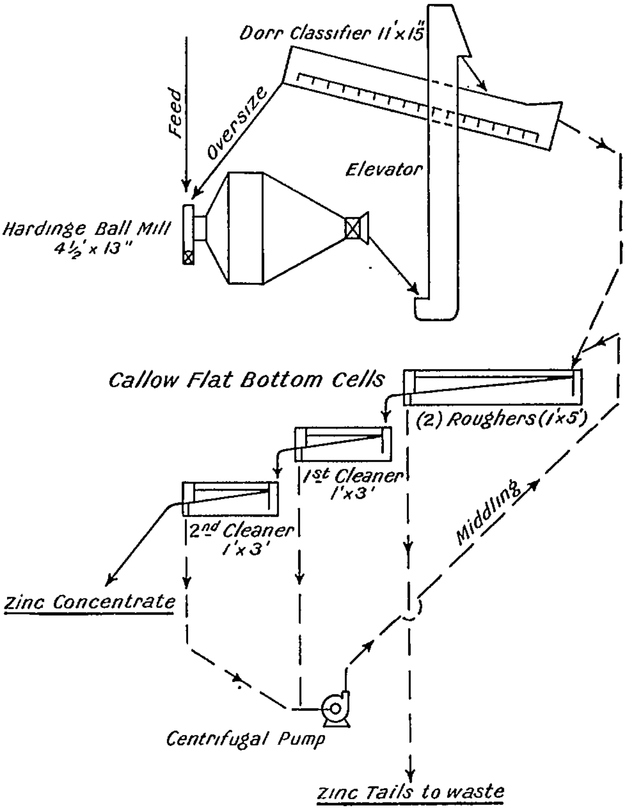


Figure 2. Flow-sheet, lot No. 2—middlings, report No. 222.

LOT NO. 2

Test No. 1

Rate of feed per hour		900 pounds
Reagents—		
Lead—		
Soda ash to ball mill	9.30 a.m. to 12.20 p.m.	18·0 lb./ton
	12.20 p.m. to 4.00 p.m.	16·0 "
Sodium cyanide		0·266 "
Oil mixture to ball mill (Barrett's water-gas tar, 50 per cent; D. T. & C. Co. acid creosote, 50 per cent)		0·42 "
Zinc—		
Copper sulphate		2 to 2·5 "
Potassium xanthate	11.30 a.m. to 12.25 p.m.	0·124 "
	12.25 p.m. to 12.45 p.m.	0·37 "
	12.45 p.m. to 3.00 p.m.	0·4 "
	3.00 p.m. to 4.10 p.m.	0·12 "
Neutral creosote No. 2 (D. T. & C. Co.)	3.00 p.m. to 4.10 p.m.	0·8 "
Pine oil		very little
YZ mixture—tried a little for a few minutes but did not like character of froth produced.		
Density samples—		
Average to lead cells		1 : 1·6
Average to zinc cells		1 : 2·8
Key to sampling periods—		
Lead concentrate		whole period
Lead tailing		"
Zinc sample No. 1		12.35 p.m. to 3.00 p.m.
Zinc sample No. 2		3.00 p.m. to 4.00 p.m.

NOTE.—Sample No. 2 shows the effect of replacing part of the xanthate by neutral creosote.

Analysis of samples—	Pb%	Zn%
Feed, or head sample	2·45	24·79
Lead concentrate	16·47	12·42
Lead tailing	0·40	22·50
No. 1 zinc concentrates	0·30	51·62
No. 1 zinc tailings	0·45	0·85
No. 2 zinc concentrates	0·55	48·86
No. 2 zinc tailings	0·55	3·00
Zinc concentrates, whole period	0·55	47·13

Test No. 2

Rate of feed per hour	870 pounds	
Reagents—		
Lead—		
Soda ash-lime mixture (soda 3½ to lime 4) whole run	15·0	lb./ton
Sodium cyanide	0·26	“
Thiocarbanilide	0·20	“
Cresylic acid	0·40	“
Zinc—		
Copper sulphate	2·50	“
Potassium xanthate	0·33	“
Pulp densities—		
Feed to lead cells averaged	1 : 1·2	
Feed to zinc cells averaged	1 : 3·0	
Discharge to zinc cells averaged	1 : 15	

Key to sampling periods—	
Lead concentrates	Whole run
Lead tailings	“
Zinc sample No. 1	11.00 a.m. to 3.00 p.m.
Zinc sample No. 2	3.00 p.m. to 4.00 p.m.

Analysis of samples—	Pb%	Zn%
Feed, or head sample	2·20	23·00
Lead concentrate	11·78	13·31
Lead tailing	0·80	24·58
No. 1 zinc concentrate	0·75	49·37
No. 1 zinc tailing	0·58	5·36
No. 2 zinc concentrate	1·50	52·12
No. 2 zinc tailing	0·67	14·40

Test No. 3

Rate of feed per hour	870 pounds	
Reagents—		
Lead—		
Soda ash	18·0	lb./ton
Oil mixture (Barrett's water-gas tar, 50 per cent; D. T. & C. Co. acid creosote, 50 per cent)	0·6	“
Sodium cyanide	0·26	“
Cresylic acid	0·05	“
Zinc—		
Copper sulphate	2·5	“
Potassium xanthate	0·5	“
Pulp densities—		
Feed to lead cells	1 : 1·5	
Lead to zinc cells	1 : 3·0	

Analysis of samples—	Pb%	Zn%	Au. oz.	Ag. oz.
Feed, or head sample	2·35	24·60	0·05	7·16
No. 1 lead concentrate	22·36	9·48	0·20	49·30
No. 2 lead concentrate	19·46	11·82	0·15	43·95
No. 3 lead concentrate	20·66	10·87	0·20	46·20
Average, Cu. 4·33 per cent	20·81	9·39	0·25	46·54
No. 1 lead tailings	0·45	21·03	0·05	3·52
No. 2 lead tailings	0·35	24·68	0·04	3·14
No. 1 zinc concentrate	0·30	51·97	tr	1·20
No. 1 zinc tailings	0·52	7·41	0·04	3·42
No. 2 zinc concentrate	0·40	50·19	tr	2·20
No. 2 zinc tailings	0·30	3·80	0·03	2·32
No. 3 zinc concentrate	0·35	50·78	tr	1·82
No. 3 zinc tailings	0·27	0·84	0·03	2·30
Average zinc concentrate	0·27	52·30	tr	2·50

Test No. 4

Rate of feed per hour	870 pounds

Reagents—

Lead—	
Soda ash, whole run	18·0 lb./ton
Oil mixture (Barrett's water-gas tar, 50 per cent; D. T. & C. Co. acid creosote, 50 per cent)	0·05 "
Sodium cyanide	0·18 "
Cresylic acid	0·05 "
Zinc—	
Copper sulphate	2·5 "
Potassium xanthate	0·5 "

Pulp densities—

Feed to lead cells	1 : 1·5
Feed to zinc cells	1 : 3·0

Analysis of samples—	Pb %	Zn %	Au. oz.	Ag. oz.
Feed, or head sample	2·20	24·54	0·05	6·08
No. 1 lead concentrate	20·26	10·07	0·20	44·80
No. 2 lead concentrate	19·86	10·87	0·20	44·90
No. 3 lead concentrate	20·01	11·26	0·20	43·40
Average, Cu. 4·64	21·06	10·67	0·25	45·15
Lead tailing	0·27	26·07	tr.	1·76
No. 1 zinc concentrate	0·15	52·17	tr.	1·40
No. 1 zinc tailing	0·35	4·94	tr.	1·56
No. 2 zinc concentrate	0·30	50·98	tr.	0·50
No. 2 zinc tailing	0·47	1·53	tr.	2·48
No. 3 zinc concentrate	0·25	51·17	tr.	1·70
No. 3 zinc tailing	0·45	1·43	tr.	1·82
Average zinc concentrate	0·27	52·30	tr.	2·50

Test No. 5

Rate of feed per hour	950 pounds

Reagents—

Lead—	
Soda ash	12·0 lb./ton
Thiocarbanilide	0·2 "
Cyanide	0·25 "
Cresylic acid	0·05 "
Zinc—	
Copper sulphate	2·5 "
Potassium xanthate	0·5 "
Pine oil	0·05 "

Pulp densities—

Feed to lead cells	1 : 1·5
Feed to zinc cells	1 : 3·5

Key to samples—

Sample No. 1	11.50 a.m. to 1.45 p.m.
Sample No. 2	1.45 p.m. to 2.15 p.m.
Sample No. 3	2.15 p.m. to 4.20 p.m.

Analysis of samples—	Pb %	Zn %	Au. oz.	Ag. oz.
Feed, or head sample	2·50	24·30		
No. 1 lead concentrate	18·86	11·95	0·20	42·00
No. 2 lead concentrate	17·96	10·27	0·15	43·45
No. 3 lead concentrate	17·96	10·97	0·25	40·95
Average	17·71	12·80		
Lead tailing	0·35	25·19	tr.	2·10
No. 1 zinc concentrate	0·45	51·37	0·01	3·10
No. 1 zinc tailing	0·67	1·85		
No. 2 zinc concentrate	0·45	51·77	tr.	2·90
No. 2 zinc tailing	0·47	1·70		
No. 3 zinc concentrate	0·20	50·88	tr.	2·46
No. 3 zinc tailing	0·45	0·85		
Average zinc concentrate	0·37	51·80		

Test No. 6

This test was run to determine whether the same results could be obtained in the Callow cells as in the Ruth cells, on the flotation of the lead.

Rate of feed per hour .... 950 pounds

Reagents—

Lead—

Soda ash	11.45 a.m. to 12.20 p.m.	14·0 lb./ton
	12.20 p.m. to 12.30 p.m.	1·4 “
	12.30 p.m. to 1.00 p.m.	none
	1.00 p.m. to 3.20 p.m.	7·0 “
Thiocarbanilide	11.45 a.m. to 12.20 p.m.	0·20 “
	12.20 a.m. to 12.30 p.m.	none
	12.30 p.m. to 1.00 p.m.	0·20 “
	1.00 p.m. to 3.20 p.m.	0·20 “
Oil mixture (Barrett's water-gas tar, 50 per cent; D. T. & C. Co. acid creosote, 50 per cent)		0·30 “
Cyanide		0·24 “
Cresylic acid		0·05 “

Pulp densities—

Feed to lead cells .... 1 : 2·5

Key to samples—

Sample No. 1	11.45 a.m. to 12.20 p.m.
Sample No. 2	12.20 p.m. to 12.30 p.m.
Sample No. 3	12.30 p.m. to 1.00 p.m.
Sample No. 4	1.00 p.m. to 3.20 p.m.

Analysis of samples—	Pb %	Zn %
Head sample or feed	2·52	25·00
Lead concentrate (whole run)	19·06	14·70
No. 1 lead concentrate	16·72	15·51
No. 1 lead tailing	0·50	25·70
No. 2 lead concentrate	13·77	17·50
No. 2 lead tailing	0·60	25·80
No. 3 lead concentrate	13·27	15·40
No. 3 lead tailing	0·70	25·70
No. 4 lead concentrate	12·67	16·50
No. 4 lead tailing	0·65	25·30

Summary and Conclusions Lot No. 2.—The conclusions drawn from the results of these tests are summarized as follows:—

1. That there is no advantage gained in the use of lime as a substitute for a part or the whole of the soda ash. The lime not only lowered the grade of the lead concentrate, but increased the loss of zinc in the zinc tailing.

2. That higher grade lead concentrate was obtained with the oil mixture than with thiocarbanilide.

3. That by the use of xanthate for the flotation of the zinc, a very high-grade concentrate—over 50 per cent zinc—can be obtained with a very low tailing—less than 1·5 per cent zinc.

4. That there is no difficulty in obtaining a high recovery of the zinc when using xanthate under conditions of pulp densities used in these tests.

5. That the whole of the required amount of soda ash should be fed to the ball mill, thereby reducing the amount required.

6. That we find xanthate to be far superior for the flotation of the zinc as compared to TT mixture, YZ mixture, or oil mixtures of neutral coal-tar creosote or water-gas tar.

Report No. 223

THE CONCENTRATION OF SILVER-LEAD-ZINC ORE FROM THE ENTERPRISE MINE, SLOCAN DISTRICT, B.C.

C. S. Parsons

Shipment.—A shipment of 100 pounds of silver-lead-zinc ore was received from the Enterprise mine, Slocan district, B.C., on November 20, 1924. The shipment was submitted by E. C. Wragge, Esq., Nelson, B.C.

Purpose of Experimental Tests.—The shipment was submitted to determine the best method of concentration for the recovery of the silver values, and at the same time produce a lead product which would have a zinc content low enough to avoid excessive penalization by the smelter.

General Practice used on Slocan Ores.—The most serious problem in the concentration of the Slocan ores is the recovery of the silver values, in a smelting product from which the greatest monetary return will be received. The general practice throughout the district has been graded crushing with rather elaborate systems of sizing and classification of the crushed products for feed to jigs and tables on which high-grade lead products are made. The tailings and overflow slime are dewatered, the remaining fine lead and silver values in the slime partly recovered by flotation in lead cells, the tailings from which go to zinc cells for the recovery of the zinc by flotation. By this method the silver losses are appreciable, due to reasons which are given later in this report. In some cases where this general procedure is followed, but on a less elaborate scale, the silver losses amount to 60 per cent of the silver values in the ores.

Character and Analysis of Sample.—The sample received was very high-grade. The analysis shows it to contain:—

Lead	9·92 per cent
Zinc	34·42 "
Silver	24·91 ounces per ton
Gold	0·04 "

The high zinc content makes it extremely difficult to produce a lead concentrate low in zinc by flotation. The sulphides are coarsely crystalline and separate readily from each other and the gangue by crushing to 60 mesh. The gangue is chiefly carbonate of iron (spathic iron) and siliceous material.

Examination of Ore as to Silver Values.—A careful study was first made to determine the association of the silver. Pure pieces of galena, blende, and gangue, approximately $\frac{3}{8}$ inch in size, were picked out and assayed for silver. The results were as follow:—

Galena crystallized in large cubes	100·7	ounces per ton
Galena finely disseminated aggregates of crystals	102·0	"
Zinc blende (0·16 per cent lead, 63·30 per cent zinc)	7·5	"
Gangue (trace 1·66 per cent)	1·18	"

The assays show conclusively that the silver in some form is associated almost exclusively with the galena. Samples of the ore were then crushed to about 20 mesh and a lead concentrate made by tabling. This concentrate contained 70 per cent lead but only 67 ounces silver per ton as against 100 ounces per ton in the coarse unbroken pieces of pure galena. Other tests showed that the finer the galena was crushed the less silver it contained. It is obvious that the silver was freed by crushing. This observation was confirmed by the results of the flotation tests, where the largest proportion of the silver was generally found in the zinc concentrate. The silver which is freed from the galena by crushing is evidently in an extremely fine form. This opinion is further borne out by some work on the Silversmith ores, which proved that 75 per cent of the silver in the zinc flotation concentrates was in the –200-mesh material, and that the silver losses in the tailing were due to very finely divided silver mineral; therefore, owing to this fine condition of the silver, a high recovery would not be expected with the galena concentrate on tables. This is proven by practice. Large losses would also occur in thickening the table tailing for flotation of the zinc, due to the tendency of the fine particles of silver mineral to remain in suspension in the overflow water.

It has been shown by experience that a mineral in such a finely divided state as this silver is believed to be in, is very difficult to recover. This applies to flotation as well as to gravity concentration. The writer has had occasion in a number of cases to observe that very fine particles of mineral in a flotation pulp repel the oil charges, and even if oiled, do not exert enough force to enter the interfaces of the air bubbles of the froth, in other words, they are difficult to oil and collect in the froth. There is, however, a marked tendency for these very fine grains to form aggregates around the large ones to which they seem to cling tenaciously. Experience with other ores has led to the opinion that to collect and recover a mineral which is in this state, a large proportion of large grains must be present in the pulp, so that by the bulk action of the flotation of the coarse grains the fine grains are gathered in and collected.

For these reasons it was decided to confine most of the experimental work to straight selective flotation with the production of a lead and zinc concentrate, and to endeavour by the bulk action, to make a high recovery of the silver, and at the same time produce a lead concentrate sufficiently free from zinc to avoid being penalized to such an extent by the smelter that the economic gain from the increased recovery of the silver would not be offset by penalization charges for the zinc content.

Experimental Tests

A number of selective flotation tests were made, first on the sample as submitted, and second, on the ore diluted by the addition of some barren quartz. It was thought that the sample submitted was higher in grade than would be the case with the milling ore, so that 50 per cent by weight of barren quartz porphyry was added to make up the sample used in the latter tests. It was believed that the lead concentrate produced would contain much less zinc than that obtained from the first series of tests on the high-grade sample submitted. A comparison of the results shows this to be the case. The results from the two series of tests are given in the following table:—

Results of First Series of Selective Flotation Tests on Sample as Submitted

Test No.	Product	Weight		Analysis			Per cent of values			Reagents used, etc.
		Grms.	%	Pb %	Zn %	Ag. oz./ton	Pb	Zn	Ag	
1	Lead concentrate......	91·5	9·0	63·43	15·44	109·7	59·8	4·5	37·8	*Lead:* added to ball mill, 7 lb./ton soda ash, 0·25 lb./ton water-gas tar and coal-tar creosote; added to cells, 0·5 lb./ton cyanide, 0·05 lb./ton cresylic acid. *Zinc:* 2 lb./ton copper sulphate, 0·6 lb./ton neutral creosote, 0·2 lb./ton TT mixture.
	Lead middling........	132·5	13·1	25·05	40·44	50·9	34·1	16·9	25·5	
	Zinc concentrate......	324·5	31·9	0·90	61·58	18·2	3·0	63·1	22·8	
	Zinc middling.........	141·5	13·9	1·00	22·23	16·2	1·4	9·9	8·6	
	Tailing..............	325·7	32·1	0·50	5·37	4·3	1·7	5·5	5·3	
2	Lead concentrate......	82·0	8·2	70·54	9·90	89·1	63·0	3·1	28·8	*Lead:* added to ball mill, 3·5 lb./ton soda ash, 0·2 lb./ton thiocarbanilide; added to cells 0·05 lb./ton cresylic acid, 0·6 lb./ton cyanide. *Zinc:* 1 lb./ton copper sulphate, 0·1 lb./ton potassium xanthate.
	Lead middling........	48·5	4·9	51·05	19·50	38·7	26·9	3·6	7·5	
	Zinc concentrate......	272·0	27·3	1·55	62·07	34·7	4·6	64·9	36·8	
	Zinc middling.........	40·4	4·1	3·01	21·98	36·8	1·3	3·4	5·9	
	Tailing..............	553·5	55·5	0·70	11·71	9·6	4·2	24·9	21·0	
3	Lead concentrate......	110·0	11·7	52·15	24·20	77·80	69·0	8·4	36·0	*Lead:* added to ball mill, 8 lb./ton soda ash, 0·15 lb./ton coal-tar creosote; 1 lb./ton water-gas tar; added to cells, 1 lb./ton cyanide; 0·05 lb./ton cresylic acid. *Zinc:* 0·6 lb./ton neutral creosote; 0·2 lb./ton TT mixture, 0·05 lb./ton No. 5 pine oil.
	Lead middling........	50·0	5·3	28·26	28·17	42·90	16·8	4·4	9·0	
	Zinc concentrate......	315·0	33·3	2·45	59·72	31·3	9·2	59·0	41·0	
	Zinc middling.........	86·5	9·1	2·15	50·14	17·0	2·2	13·6	6·0	
	Tailing..............	384·5	40·6	0·60	12·10	5·0	2·8	14·6	8·0	
6	Lead concentrate......	132·5	13·3	36·87	23·21	117·1	63·5	8·9	58·5	*Lead:* 5 lb./ton soda ash; 0·2 lb./ton thiocarbanilide, 0·4 lb./ton special reagent, 0·05 lb./ton cresylic acid. *Zinc:* 0·6 lb./ton No. 2 neutral creosote, 1 lb./ton copper sulphate, 0·05 lb./ton pine oil No. 5.
	Lead middling........	113·0	11·3	21·74	40·17	35·5	31·8	13·3	15·1	
	Zinc concentrate......	385·2	38·5	0·55	62·09	14·1	2·7	69·8	20·4	
	Zinc middling.........	49·1	4·9	0·90	44·84	13·8	0·6	6·4	2·5	
	Tailing..............	318·7	31·9	0·33	1·77	2·9	1·4	1·6	3·5	
7	Lead concentrate......	130·0	12·9	61·43	16·60	120·5	78·2	6·3	57·8	*Lead:* added to ball mill, 6 lb./ton soda ash, 0·2 lb./ton thiocarbanilide; added to cells, 0·2 lb./ton special reagent, 0·05 lb./ton cresylic acid. *Zinc:* 0·4 lb./ton potassium xanthate, 1 lb./ton copper sulphate, 0·05 lb./ton pine oil.
	Lead middling........	65·0	6·4	18·71	32·55	29·8	11·9	6·2	7·2	
	Zinc concentrate......	343·5	34·1	1·15	60·56	15·0	3·9	61·1	19·0	
	Zinc middling.........	180·0	17·8	2·99	47·62	18·3	5·3	25·1	12·2	
	Tailing..............	290·0	28·8	0·20	1·57	3·56	0·6	1·3	3·8	
8	Lead concentrate......	105·0	10·5	56·09	20·50	129·0	62·8	6·8	51·3	*Lead:* added to ball mill, 6 lb./ton soda ash, 0·2 lb./ton thiocarbanilide; added to cells, 0·05 lb./ton cresylic acid. *Zinc:* 0·3 lb./ton potassium xanthate, 2 lb./ton copper sulphate, 0·05 lb./ton cresylic acid.
	Lead middling........	126·0	12·6	23·45	18·15	38·8	32·9	7·3	18·5	
	Zinc concentrate......	469·0	47·0	0·72	56·80	14·1	3·6	84·8	25·1	
	Tailing..............	297·0	29·8	0·22	1·15	4·54	0·7	1·1	5·1	
10	Lead concentrate......	No flotation of lead under these conditions..........................								*Lead:* added to mill, 6 lb./ton lime, 0·3 lb./ton thiocarbanilide; added to cells, 2 lb./ton sodium sulphite, 0·05 lb./ton cresylic acid.

Results of Second Series of Selective Flotation Tests on Ore Diluted with 50 per cent Barren Quartz

Test No.	Product	Weight		Analysis			Per cent of values			Reagents used, etc.
		Grms.	%	Pb %	Zn %	Ag. oz./ton	Pb	Zn	Ag	
12	Lead concentrate......	53·0	5·3	61·85	13·85	69·30	75·4	4·4	31·2	*Lead:* added to ball mill, 6 lb./ton soda ash, 0·4 lb./ton thiocarbanilide; added to cells, 0·3 lb./ton cyanide, 0·01 lb./ton cresylic acid. *Zinc:* 2 lb./ton copper sulphate, 0·4 lb./ton potassium xanthate, 0·05 lb./ton pine oil.
	Lead middling........	30·0	3·0	24·94	25·65	46·80	17·2	4·5	12·0	
	Zinc concentrate......	274·0	27·5	0·85	52·00	20·50	5·4	84·5	47·8	
	Tailing..............	639·5	64·2	0·13	1·75	1·66	1·9	6·6	9·0	
13	Lead concentrate......	40·4	4·0	70·07	5·90	69·90	56·9	1·4	20·9	*Lead:* added to ball mill, 6 lb./ton soda ash, 0·2 lb./ton thiocarbanilide; added to cells, 0·3 lb./ton cyanide, 0·01 lb./ton cresylic acid. *Zinc:* 2 lb./ton copper sulphate, 0·5 lb./ton potassium xanthate, 0·05 lb./ton pine oil.
	Lead middling........	35·7	3·5	43·96	19·00	47·20	31·5	3·9	12·5	
	Zinc concentrate......	254·0	25·0	1·35	59·80	29·70	8·4	86·4	55·8	
	Zinc middling.........	43·7	4·3	1·75	16·50	9·60	1·5	4·1	3·1	
	Tailing..............	642·0	63·2	0·13	1·15	1·64	1·7	4·2	7·8	

SUMMARY

It will be observed that in each of these tests there is a lead middling product which carries a large portion of the lead values. We have found that in continuous operation on a tonnage scale the middling takes care of itself by being returned continuously to the feed end of the cells. The following examples are cited to support this:—

Results of Small- and Large-scale Tests on a Lead-Zinc Ore:—

Product	Weight	Analysis		Per cent of values	
		Pb %	Zn %	Pb.	Zn.
Small-scale test—					
Lead concentrate*	35·1	24·25	11·86	34·8	1·6
Lead middling	56·5	10·38	22·50	24·0	4·9
Zinc concentrate	365·3	1·07	53·65	16·0	76·0
Zinc middling	104·5	2·20	24·13	9·4	9·8
Zinc tailing	474·4	0·82	4·17	15·8	7·7
Large-scale test—					
Lead concentrate*	....	22·36	9·39	Approximate recoveries: lead 85 per cent, zinc 90 per cent.	
Zinc concentrate	....	0·27	52·30		
Tailing	....	0·30	1·50		

*It may seem that the lead concentrates in the above tests are low-grade, but in this particular case they carry high values in silver, one-half ounce in gold, and some copper values. The ore contained considerable micaceous material. It was, therefore, important that the precious metal values together with the copper and the flotable micaceous material report with the lead concentrate, and thus permit of a much higher grade zinc product being obtained. The lead product, while low in lead values, was a very desirable smelter product.

Notes on Reagents Used.—Thiocarbanilide is apparently a satisfactory reagent for the flotation of the lead, but care should be taken not to use over 0·2 pound per ton as any excess will raise zinc. The silver content of the lead concentrate seems to be governed by the kind and quantity of reagents used to obtain the selective action between the lead and zinc. Four different modifying reagents were experimented with, namely, cyanide, sodium sulphite, sodium di-phosphite, and a special reagent. Cyanide and the special reagent were the only ones which produced any decided selective action between the lead and zinc, but the special reagent produces a lead concentrate containing more silver, thereby increasing the recovery of the silver in the lead concentrate instead of in the zinc concentrate. Soda ash is necessary, as well as copper sulphate. Xanthate is undoubtedly one of the best reagents for the zinc. We recommend the use of a thick pulp of 1 : 2·5 for the lead, and 1 : 3 for the zinc.

CONCLUSIONS

In suggesting that straight selective flotation would seem to have great possibilities on this ore, the writer realizes that he is unfamiliar with local conditions. The economic side will have to be studied by the operators who know these conditions best. He believes that a calculation of the smelter rates will show that the increased recovery of the silver, which is 92 per cent in the case of the second series of tests, will offset the penalty charges for 15-20 per cent zinc in the lead concentrate, and show a considerable margin. It must be remembered that these results have been obtained by small laboratory tests, but we believe that there is enough merit in them to warrant serious consideration being given to the possibility of treating the ore by selective flotation.

It is suggested, in order to settle the question definitely as to which is the better method of treatment for Slocan ores, i.e., straight selective flotation as against gravity concentration and flotation, that a small pilot plant be built by co-operative arrangement among the mine owners, at one of the large mills of the district, using the latter process. This small unit of 12 to 15 tons daily capacity should be arranged so that it would receive a representative tonnage sample cut from the feed to the main mill, in order that a definite comparison of the two methods could be obtained.

Report No. 224

THE TREATMENT OF THE GOLD-COPPER ORE OF THE ARGONAUT MINE, LARDER LAKE, ONT.

J. S. Godard

Shipments.—A shipment of gold-copper ore, weighing 80 pounds, was received December 15, 1924, from the Argonaut Mine, Larder Lake, Ont.

Purpose of Experimental Tests.—It was desired to compare the merits of various methods of treatment for the extraction of the gold values contained in the ore.

Sampling and Analysis.—The entire lot was crushed to ½-inch, cut in a riffle sampler to 20 pounds, which was crushed to −8 mesh and cut to 4 pounds. This quantity was ground dry in a pebble mill to −150 mesh and then cut to obtain an assay sample:—

Gold	1·06 ounce per ton
Copper	2·21 per cent
Iron	11·23 "

Experimental Tests

Cyanidation

Four single-cycle, cyanidation tests of the bottle type were made. The ore was ground dry in a pebble mill and screened through 150 mesh. The cyanide solution was strengthened to 0·05 per cent cyanide at 8-hour intervals. Following are the results of the tests:—

Test No.	Ore grms.	Solution c.c.	KCN per cent	CaO lb./ton	Assay		Extraction, per cent	Consumption lb./ton		Time agitated, hours
					Heads	Tails		KCN	CaO	
1	200	1000	0·05	3	1·06	0·08	92·4	8·0	1·8	24
2	200	1000	0·05	3	1·06	0·04	96·2	13·0	1·8	48
3	200	600	0·05	3	1·06	0·24	77·4	6·9	2·0	24
4	200	600	0·05	3	1·06	0·05	95·2	11·1	2·1	48

The tailings were washed with 100 c.c. 0·05 per cent cyanide, 100 c.c. 0·025 per cent cyanide and water.

Summary.—The extraction was good using dilute pulp and agitating for 48-hour periods. The cyanide consumption was high; little lime was required.

Amalgamation

Two amalgamation tests, Nos. 5 and 6, were made. The samples were cut at −14 mesh and ground wet in a porcelain ball mill using grey iron balls. Amalgamation was conducted for a period of two hours in pebble jars, using 100 grammes mercury and a pulp density of 1 : 1. The tailings were screened and the products assayed.

Test No.	Size	Weight per cent	Assay gold oz./ton	Content per cent	Per cent of values
5	+ 35	0·6	0·34	0·2	0·5
	− 35+ 48	2·2	0·32	0·6	1·7
	− 48+ 65	12·3	0·52	6·4	15·4
	− 65+100	17·0	0·58	9·9	23·8
	−100+150	18·6	0·56	10·4	25·0
	−150+200	22·2	0·34	7·5	18·0
	−200	27·1	0·24	6·5	15·6
6	+ 65	0·2	0·33	0·1	0·3
	− 65+100	3·8	0·42	1·6	4·8
	−100+150	18·0	0·44	7·9	23·8
	−150+200	43·0	0·32	13·8	41·6
	−200	35·0	0·28	9·8	29·5

	Test No. 5	Test No. 6
Weight of ore, grms	1,475	1,475
Degree of crushing, mesh	−35	−65
Average value of tailing after amalgamation	0·42 oz./ton	0·33 oz./ton
Head sample, assay	1·10 "	1·10 "
Per cent of gold amalgamated	61·8	70·0

Summary.—Good results were obtained on the amalgamation of this ore. The gold remaining was largely found in the intermediate sizes. No flowering of the mercury was noticed.

Test No. 7

Flotation at 48 Mesh and Tabling.—One thousand grammes of ore at −14 mesh was ground wet in a ball mill in a dense pulp, 2 : 1, then floated in a Ruth machine. The flotation tailing was tabled on a Wilfley table. The table tailing was screened and the products assayed:—

Product	Weight per cent	Assays		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate	15·6	6·16	14·74	86·7	94·6
Table concentrate	3·0	1·04	0·42	2·8	0·5
Table tailing	56·9	0·17	0·08	8·7	1·9
Slimes	24·5	0·08	0·30	1·8	3·0

Flotation reagents—

NaOH	2.0 lb./ton added to ball mill.
QED	0·05 lb./ton added to Ruth machine.
Pine oil	2 drops " "

NOTE.—Tailing values were taken from results of screen analysis.

Results of Screen Analysis on Table Tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
+ 48 − 48+ 65	6·1	0·20	0·05	7·0	3·8
− 65+100	18·2	0·18	0·03	19·2	6·3
−100+150	20·6	0·12	0·04	14·5	10·1
−150+200	12·9	0·20	0·06	15·1	10·2
−200	42·2	0·18	0·13	44·2	69·6

Ratio of concentration.. 5·7 : 1
Loss of original feed.. 5·9 per cent

Summary.—The recoveries by this method of treatment were as follows: gold, 89·5 per cent, of which 86·7 per cent reported in the flotation concentrate; copper, 95·1 per cent, of which 94·6 per cent was in the flotation concentrate. Tabling is of little value, and from the results obtained might easily be dispensed with. The screen analysis revealed that the gold remaining is evenly distributed throughout the sizes, although the 200-mesh material carried the bulk of the copper values.

Test No. 8

Amalgamation and Flotation at 100 Mesh.—One thousand grammes of ore at −14 mesh was ground wet to 100 mesh in a ball mill and amalgamated for two hours with 100 grammes of mercury. The tailing was floated in a Ruth machine and the flotation tailing screened on 100 mesh.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate	11·1	1·00	19·10	50·4	94·1
Tailing +100	1·3	0·28	0·22	1·8	0·1
Tailing −100	87·6	0·12	0·15	47·8	5·8

Head sample..............................Gold, 1·10 oz/ton, copper, 2·25 per cent
After amalgamation......................Gold, 0·22 oz./ton
Gold amalgamated, per cent............. 70·9
Copper recovery, per cent................ 94·1

Screen Test on Flotation Tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz./tor.	Cu per cent	Au	Cu
+100	1·5	0·28	0·22	3·3	2·0
−100	98·5	0·12	0·15	96·7	98·0

Average value of tailings.................Gold, 0·12 oz./ton, copper, 0·15 per cent
Flotation reagents—
  NaOH..................................2.0 lb./ton added to Ruth machine
  QED...................................0·05 lb./ton " "
  Pine oil..................................2 drops " "

Summary.—The total gold recovery was 89 per cent, of which 70·9 per cent was due to amalgamation. The copper recovery in the flotation concentrate amounted to 94 per cent. Grinding to 100 mesh previous to amalgamation would hardly be feasible in mill operation. Should coarser crushing be tried the values in the flotation concentrates would be raised at the expense of amalgamation recovery.

Test No. 9

Amalgamation and Flotation.—For this test 1,646 grammes of ore at −20 mesh was ground wet to 35 mesh and amalgamated. The amalgamation tailing was floated, the rougher concentrate cleaned, and the tailing screened and assayed.

Product	Weight per cent	Assays		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate	6·7	3·40	30·16	46·8	85·2
Flotation middling	4·2	1·50	4·54	12·9	8·1
Flotation tailings	89·1	0·22	0·18	40·3	6·7

Reagents—
TT mixture.............................. 0·5 lb./ton added to Ruth machine
No frother was used.
0·1 " TT was added for cleaning concentrate

The values in the tailing were taken from the screen analysis.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
− 35+ 48	2·0	0·32	0·14	2·9	1·5
− 48+ 65	13·5	0·28	0·14	17·0	10·4
− 65+100	20·8	0·26	0·16	24·2	18·4
−100+150	12·6	0·32	0·17	18·1	11·8
−150+200	19·7	0·22	0·18	19·5	19·7
−200	31·4	0·13	0·22	18·3	38·2

Summary.—Amalgamation recovered 55 per cent of the gold. The flotation concentrate contained 46·8 per cent of the gold remaining after amalgamation. Only 85 per cent of the copper was recovered in the concentrate. The middling was too high both in quantity and grade to be returned to the mill circuit as it would be in actual practice. The tailing contained sufficient gold to warrant cyanidation providing the cyanide consumption was moderate.

Test No. 10

Amalgamation, Flotation, and Cyanidation.—One thousand grammes of ore at −14 mesh was ground wet in a ball mill to 48 mesh and amalgamated for 1 hour in a 1 : 1 pulp using 100 grammes of mercury. The amalgamation tailing was dewatered and floated. The flotation tailing was sampled and cyanided for 65 hours in a 1 : 2 pulp with cyanide strength of 0·075 per cent. The cyanide tailing was screened.

Head sample before amalgamation	1·10 oz./ton
After amalgamation	0·32 "
Gold amalgamated	70·8 per cent

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate	9·0	1·58	21·38	43·8	89·4
Flotation tailing	91·0	0·20	0·25	56·2	10·6

Screen analysis of cyanide tailing (605·7 grammes).

Mesh	Weight per cent	Assay Au oz./ton	Per cent of values
− 48+ 65	11·1	0·18	27·8
− 65+100	14·2	0·12	23·6
−100+150	15·0	0·09	19·4
−150+200	26·0	0·03	11·1
−200	33·7	0·04	18·1

Test No. 11

Amalgamation, Flotation, and Cyanidation.—Procedure as in test No. 10, except that the flotation tailing was ground to −200 mesh before cyanidation. The grinding was done in a ball mill, the discharge being screened and the oversize returned to the mill.

Head sample before amalgamation	1·10 oz./ton
After amalgamation	0·33 "
Gold amalgamated	70·0 per cent

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate	7·9	1·15	21·62	27·5	77·4
Flotation tailing	92·1	0·28	0·54	72·5	22·6

Test No. 12

Amalgamation, Flotation, and Cyanidation.—Procedure as in test No. 11, except that amalgamation tailing was ground to −200 mesh for flotation:—

Head sample before amalgamation	1·10 oz./ton
After amalgamation	0·30 "
Gold amalgamated	72·7 per cent

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate	12·4	1·42	14·40	58·8	83·9
Flotation tailing	87·6	0·14	0·39	41·2	16·1

Results of Cyanidation, Tests 10, 11, and 12—

Test No.	Weight grms.	Solution c.c.	KCN per cent	Assay		Extraction per cent	Consumption lb./ton		Time hours
				Heads	Tails		KCN	CaO	
10	605·7	1195	0·075	0·20	0·072	64·0	4·76	2·2	65
11	611·2	1789	0·050	0·28	0·060	78·6	6·50	3·9	47
12	611·7	1788	0·10	0·14	0·040	71·5	7·40	3·9	47

Details of Tests—

Test No.	Time	KCN per cent	CaO per cent	KCN raised to	Remarks
10	17	0·014	0·023	0·075	
	24	0·066	0·023	....	No KCN added.
	41	0·046	0·022	0·075	
	47	0·066	0·021	....	No KCN added.
	65	0·042	0·019	....	Ceased agitation.
11	17	0·006	0·009	0·050	0·5 grm. CaO added.
	24	0·040	0·021	....	
	41	0·016	0·019	0·05	0·1 grm. CaO added.
	47	0·040	0·018	....	Ceased agitation.
12	17	0·042	0·015	0·10	0·2 grm. CaO added.
	24	0·066	0·018	....	0·2 grm. CaO added.
	41	0·038	0·02	0·10	
	47	0·092	0·022	....	Ceased agitation.

In each test the cyanide tailing was washed with 200 c.c. 0·10 per cent cyanide; 200 c.c. 0·05 per cent cyanide; and 400 c.c. water.

Summary Tests 10, 11, and 12

The amalgamation in each test was good. About 70 per cent of the gold was extracted at 48 mesh.

Flotation—Gold.—The best flotation results of gold were obtained in test No. 12, where the operation was conducted at −200 mesh. In this test more bulk was removed in the concentrate with a corresponding lower grade of copper. *Copper.*—The flotation results in all three tests were poor, the best recovery being 89·4 per cent in test No. 10. The flotation reagents used in each test were:—

Sodium carbonate	3·0 lb./ton
QED	0·5 "
Pine oil, G.N.S. No. 5	0·10 "

Cyanidation.—Extraction of the gold from the flotation tailings varied from 64 to 79 per cent. The 64 per cent was obtained on tailings cyanided at 48 mesh in test No. 10. In this case the coarser sizes contained considerable gold. The cyanide consumption varied from 4·8 pounds to 7·4 pounds per ton. The high copper in the flotation tailings is largely responsible for this high consumption.

Test No. 14

Amalgamation, Flotation, and Cyanidation.—Procedure as in test No. 11.

Head sample before amalgamation	1·10 oz./ton
After amalgamation	0·32 "
Gold amalgamated	70·4 per cent

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate	10·6	1·21	17·80	39·4	90·2
Flotation tailing	89·4	0·22	0·23	60·6	9·8

Reagents—	
Water-gas tar, coal-tar creosote, 50 per cent mixture	0·7 lb./ton
Pine oil	0·1 "
Cyanide results—	
Flotation tailings, weight	593·5 grms.
Solution	1200 c.c.
Cyanide	0·05 per cent
Assay—	
Heads	0·22 oz./ton
Tails	0·03 "
Extraction	86·4 per cent
Reagents consumed—	
KCN	4·8 lb./ton
CaO	4·5 "
Time	48 hours
Original head sample	1·10 oz./ton
Final tailing	0·03 "
Total extraction	97·3 per cent

Summary

Amalgamation recovered 70·4 per cent of the total gold.

Flotation.—The copper recovery was greatly improved over previous attempts, but the gold recovery was low, only 39·4 per cent reporting in the concentrate.

Cyanidation.—Gold extraction 86·4 per cent was obtained using less than 5 pounds per ton cyanide.

Test No. 15—Flotation and Cyanidation

One thousand grammes of the ore at –14 mesh was ground wet to 48 mesh and floated in a Ruth machine. The concentrate was cleaned and the tailings were reground to –200 mesh, sampled, and cyanided.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Concentrate	6·6	12·50	30·06	68·3	90·3
Middling	2·0	5·97	2·96	9·8	2·7
Tailing	91·4	0·29	0·17	21·9	7·0

Reagents—

Lime	5·0	lb./ton added to ball mill
Xanthate	0·07	" " cells
Pine oil	0·10	" " "

Cyanidation—

Flotation tailing	599·4	grms.
Solution	1200	c.c.
Cyanide	0·05	per cent
Heads, gold	0·29	oz./ton
Tailing, gold	0·025	"
Extraction	91·4	per cent
Cyanide consumed	4·0	lb./ton
Lime consumed	2·9	"
Time	48	hours

Original head sample	1·21 oz./ton Au
Final tailing	0·025 "
Total extraction	98 per cent

Summary.—The results of the flotation test were the best yet obtained. The concentrate carried 68·3 per cent of the gold and 90·3 per cent of the copper, and the middling is more commensurate with what would be obtained in large-scale practice. The tailing contained 0·29 ounce gold per ton and 7·0 per cent of the total copper. The cyanidation results are good, 91·4 per cent of the gold in the tailing goes into solution, using 4·0 pounds per ton cyanide.

GENERAL SUMMARY

Cyanidation.—In test No. 4, 95 per cent of the gold is dissolved by 0·05 per cent cyanide solution in a pulp density of 1: 3 when agitated for 48 hours. The cyanide consumption is high, 11 pounds per ton, an amount which would almost prohibit this type of treatment. Our experience on a similar ore has shown that on repeating the operation using the fouled solution, the cyanide consumption will decrease with each succeeding cycle, but the extraction also decreased. In addition to the lessened dissolving power of the solution, incomplete precipitation might reasonably be expected. This is due to the presence of copper in the solution. Strengthening the solution just previous to precipitation and the addition of an excess over the usual practice of lead acetate will ameliorate this condition. It might, in mill practice, pay to run to waste say one-fifth of the solution after precipitation, which would not allow the copper to accumulate indefinitely in the mill circuit.

Amalgamation.—Amalgamation was satisfactory. The recovery by this method of treatment varied from 55 per cent at –35 mesh in test No. 9, to 70 per cent at –48 mesh in tests Nos. 10, 11, and 12. Grinding finer to 100 mesh as in test No. 8 produced little better results. Accordingly amalgamation could be inserted with profit in the mill flow-sheet.

Flotation.—Flotation can be applied with good results to this ore. It may be used as a primary step where 90 per cent of the copper and 68 per cent of the gold may be obtained in a high-grade concentrate which could be shipped to a smelter, or it might be inserted after amalgamation, should this be installed. In either case, the tailing from the cells contains sufficient gold to warrant cyanidation. In practice an 0·12 per cent copper tailing could be obtained. On such a tailing less than 4 pounds cyanide per ton would be sufficient. It would also eliminate possible precipitation troubles. The flotation tailing when ground to –200 mesh is more readily cyanided with a lower final tailing, 0·025 ounce gold per ton.

CONCLUSIONS

From the results obtained in the above tests, amalgamation, flotation, and cyanidation of the reground flotation tailing would recover 98 per cent of the gold and 90 per cent of the copper. Possibly the concentrate could be stacked pending better conditions as to freight rates, price of copper, or the erection of a smelter nearby.

Report No. 225

THE RECOVERY OF FOUNDRY METAL FROM SWEEPINGS

J. S. Godard

Shipments.—A shipment of 400 pounds of sweepings was received February 7, 1925, from the Canada Bronze Co., Montreal, Que.

Characteristics of the Sweepings.—The material consisted of sweepings remaining after casting, and was composed of gates, spills, etc., from the castings, as well as pieces of coal, and considerable sand from the moulds, all varying in size from 2 inches to a very fine material, 100 mesh, and analysed approximately, copper 30 per cent, lead 10 per cent, tin 3·5 per cent, zinc 7 per cent.

Sampling and Analysis.—It was almost impossible to obtain a satisfactory head sample owing to the nature of the material. The copper in the head sample will be calculated from the products of the various tests.

Purpose of Experimental Work.—The purpose of these tests was to raise the metallic content of the material, particularly the copper, by grinding and gravity concentration, to such a point as would permit of its being re-melted and re-cast. Formerly the material was shipped to a U.S. smelter, but high freight rates made it advisable to find a simple and cheap method of treatment on the company's premises. The company was desirous of recovering the copper which was the predominating metal and as they already had a Wilfley table they desired, if possible, a process which might employ tabling.

EXPERIMENTAL TESTS

Test No. 2

Grinding, Screening, and Tabling.—A sample of 20 pounds was cut from 300 pounds of the sweepings crushed in rolls and screened on a 4-mesh screen. The +4-mesh product was a good grade of metallics. From the –4 mesh a sample of 3,887 grammes was cut with a Jones sampler. This was crushed once in the rolls and screened on 48 mesh. The +48-mesh product was ground wet in a ball mill and again screened on 48 mesh. The +48-mesh product was a good grade of metallics and was not further treated. The –48 mesh from the rolls was tabled on a small Wilfley table, re-running the middlings to make a concentrate and tailing. The –48 mesh from the ball mill was treated similarly and the slimes from both tabling operations were combined.

Results.—

1. Sampling, grinding, and screening—

Oversize from 20 pounds crushed to 4 mesh	329·5 grms.
Sample cut from −4 mesh	3,887·0 "
+48 mesh after passing once through rolls	1,913·0 "
−48 mesh after passing once through rolls	1,974·0 "

2. Concentration—

Product	Weight		Analysis Cu per cent	Per cent of Cu values
	Grms.	Per cent		
+48 mesh from ball mill	565·0	16·5	74·05	58·6
Table concentrate −48 mesh from ball mill	228·0	6·6	52·02	16·6
Table concentrate −48 mesh from rolls	351·2	10·2	23·04	11·3
Table tailing −48 mesh from ball mill	408·7	11·9	6·14	3·5
Table tailing −48 mesh from rolls	1,242·5	36·2	3·12	5·5
Slimes from both tablings	635·5	18·6	5·01	4·5

Head sample from products, omitting the +4 mesh, (copper)	20·72 per cent
Total feed	3,887 grms.
Products recovered	3,431 "
Loss in slimes	456 "
Loss of original feed	11·7 per cent
Value in the concentrates	86·5 "

Summary.—The loss from sliming indicated the tendency of the material to slime when treated in this way. The concentrate on 48-mesh screen after grinding in ball mill is very good. That produced from tabling the −48 mesh from the rolls was the poorest of the three. The table showing results of concentration is given below.

Product	Weight		Analysis Cu per cent	Per cent of values
	Grms.	Per cent		
+48 mesh from ball mill	565·0	49·4	74·05	67·7
Table concentrate −48 mesh from ball mill	228·0	19·9	52·02	19·2
Table concentrate −48 mesh from rolls	351·2	30·7	23·04	13·1

Average copper content in concentrates	54·0 per cent

Test No. 4

Grinding in Ball Mill, Screening on 35 Mesh, and Tabling the −35 Mesh.—A sample of 15 pounds was cut from 180 pounds of the sweepings. This was screened on 4 mesh. The oversize was passed once through rolls and again screened on 4 mesh. From the −4 mesh a sample of 1,562 grammes was cut and this was ground wet in a ball mill for 15 minutes and screened on 35 mesh. The oversize was reground for a further 15 minutes and again screened on 35 mesh. The oversize from this was a good metallic product. The −35 mesh product was tabled on a Wilfley table, re-running the middlings and making a final concentrate, middling, and tailing. The slimes from the screenings were kept separate from those of the table.

Results.—

1. Sampling, grinding, and screening—

Oversize from 15 pounds crushed to 4 mesh	277·7 grms.
Sample cut from −4 mesh	1,562·0 "

2. Concentration:—

Product	Weight		Analysis Cu per cent	Per cent of Cu values
	Grms.	Per cent		
+35 mesh screen	341·4	22·2	62·5	69·0
Table concentrate −35 mesh	360·7	23·5	22·08	25·8
Table middling −35 mesh	40·0	2·6	2·38	0·3
Table tailing	599·7	39·1	1·14	2·2
Slimes (1) screen	110·7	7·2	4·52	1·6
Slimes (2) table	82·7	5·4	3·09	1·1
Table of concentrates—				
+35 mesh	341·4	48·6	62·50	72·8
Table concentrates	360·7	51·4	22·08	27·2

Average copper content of concentrates	41·73 per cent
Original feed	1,562·0 grms.
Products from operations	1,535·2 "
Loss (slimes from table)	26·8 "
Loss of original feed	1·7 per cent
Values in copper concentrates	94·8 "

Test No. 5

Grinding, Screening, and Tabling.—A sample of 1,869·5 grammes was cut from 160 pounds. This was screened dry on 35 mesh, and the oversize reground 15 minutes in a ball mill and again screened; this operation was twice repeated. The final +35 mesh was a good grade of metallics. The −35 mesh was tabled re-running the middlings to make a concentrate and tailing. The slimes from the screen were kept separate from those from the table.

Results—

Product	Weight		Analysis Cu per cent	Per cent of Cu values
	Grms.	Per cent		
+35 mesh concentrate on screen	437·4	24·1	60·70	67·3
Table concentrate −35 mesh	414·9	22·8	25·00	26·2
Table tailing −35 mesh	778·4	42·8	2·19	4·3
Slimes (1) screen	55·2	3·1	5·06	0·7
Slimes (2) table	131·0	7·2	4·58	1·5
Table of concentrates—				
+35 mesh	437·4	51·3	60·7	71·9
Table concentrates	414·9	48·7	25·0	28·1

Average copper content of concentrates	43·31 per cent
Average head sample from products, (copper)	21·75 per cent
Original feed	1,869·5 grms.
Products, total	1,816·9 "
Loss	52·6 "
Loss of original feed	2·8 per cent
Recovery in concentrate (of the copper)	93·5 "

Test No. 6

Grinding, Screening, and Tabling.—A sample of 1,795 grammes was cut from 150 pounds, and crushed dry in rolls, then ground wet in a ball mill for 20 minutes and screened on 65 mesh. The oversize was again ground in a ball mill and screened. These operations were twice repeated. The oversize from the last screening was a good grade of metallics. The –65-mesh material from the screening was tabled, re-running the middlings and making a final concentrate, middling, tailing, and slimes.

Results.—

Product	Weight		Analysis Cu per cent	Per cent of Cu values
	Grms.	Per cent		
+65 mesh	360·4	21·1	74·20	71·6
Table concentrate	185·7	10·9	37·02	18·5
Table middling	25·7	1·5	12·63	0·9
Table tailing	907·0	53·1	2·53	6·1
Table slimes	228·0	13·4	4·78	2·9
Table of concentrates—				
+65 mesh	360·4	66·0	74·20	79·6
Table concentrates	185·7	34·0	37·02	20·4

Average head sample from products (copper)	21·86	per cent
Original feed	1,795·0	grms.
Products	1,706·8	“
Loss	88·2	“
Loss due to slimes (of original feed)	4·9	per cent
Recovery in concentrates (copper)	90·1	“
Average copper content of concentrate	61·56	“

Summary.—In the above tests, good recoveries were made by grinding wet in a ball mill, screening out the metallics, and tabling the undersize from the screen. The best results were obtained on material ground to –65 mesh.

Recommendations.—An installation consisting of a 3-foot 8-inch cylinder Hardinge ball mill, maximum capacity 1 ton per hour, is recommended. This could be operated in conjunction with the Wilfley table already in possession of the company. Fifteen horsepower will be sufficient to operate both the ball mill and the table.

Grinding to 65 mesh will be sufficient to free the copper, the –65-mesh material being fed to the table. The +65-mesh product can be obtained by emptying the ball mill about once per week.

Report No. 226

AMALGAMATION AND CONCENTRATION OF A MATACHEWAN GOLD ORE

C. S. Parsons and J. S. Godard

Shipment.—A shipment of 16,312 pounds of gold ore was received at the Ore Dressing and Metallurgical Laboratories, March 26, 1925, from the Porcupine Goldfields Development and Finance Corporation, Matachewan. The ore was from the Young-Davidson property, Powell township, Ontario.

Purpose of Experimental Work.—The erratic distribution of the free gold in this ore made underground sampling unreliable. A mill run was desired to check the results obtained by underground sampling, and to obtain data and determine a method for the recovery of the gold.

Characteristics of the Ore.—The ore consisted of a hard, red granitic porphyry, mineralized with iron pyrite and containing fine free gold. The gold values were very spotty and it was difficult to obtain check assays from the samples.

Sampling and Analysis.—The sampling was given careful attention. Six samples in all were cut from the shipment. The ore was crushed in a jaw crusher and rolls to ½-inch, passed through a Vezin sampler which cut out a one-tenth part. This was reduced to 10 mesh and then coned and quartered. Each quarter was carried down by graded crushing and splitting, giving four separate head samples. Two additional samples were taken as a re-check on the four, by reducing the rejected half of the first tenth out of the Vezin and reducing it by graded crushing and splitting in a Jones riffle sampler. The results were as follows:—

Sample No. 1	0·140 oz./ton gold
No. 2	0·141 " "
No. 3	0·16 " "
No. 4	0·132 " "
No. 5	0·145 " "
No. 6	0·130 " "
Average	0·141 " "

Duplicate assays made of each of these samples were either the same or showed a difference of 0·01 and not exceeding 0·02 ounce per ton.

Stamp Battery, Amalgamation and Concentration

The ore was fed to a 5-stamp battery carrying a 40-mesh screen. The pulp passed over amalgamation plates to recover the gold. The tailing from the plates was run direct to a standard deck Wilfley table without classification of any kind. The table under this condition did not do good work, as it was operated above capacity and the fine sulphide lay in the riffles under the coarse sand and escaped into the tailing. The tailing was pumped to a tank for settling to await further treatment. In order to recover the fine sulphides the tailing was passed over a 60-mesh Callow belt screen, the undersize going direct to the table. The fines represented 85 per cent by weight, and the coarse 15 per cent. Had the concentration been made on a classified feed, a cleaner and higher grade concentrate would have been produced. The sample of the tailing from the first tabling was taken between the table and the pump before the pulp was pumped to the settling tank. The content in the sample plus the content in the concentrate checked with the amalgamation tailing, which was the feed to the table, showing that the sampling was accurate. In settling this table tailing, there was a loss of fine slimes and probably some fine concentrate in the overflow, so that the accuracy of the final tailing samples, which showed 0·02 ounce gold per ton and 0·012 ounce per ton, is questionable. It would, therefore, be more accurate to take the content of gold in the concentrate obtained and subtract it from the original amalgamation tailing in order to arrive at the values for the final table tailing. This would give a value of 0·0248 ounce per ton representing a loss of 16·61 per cent of the gold values.

The first concentrate contained considerable gangue and was fairly coarse, whereas the concentrate from the −60-mesh material was fine, but much cleaner. The first concentrate contained 0·56 ounce per ton and the second 0·36 ounce per ton. This difference must be due to the gold values in the coarse pyrite not being amalgamated to the same extent as those from the fine pyrite.

SUMMARY

Crushing—

Stamp battery	Five 1,250-pound stamps
Height of drop	6 inches
Number of drops	95 per minute
Height of chuck block	4½ inches
Screen	40 mesh, diagonal slots
Stamp duty	3·2 tons per stamp per 24 hours

With a greater drop the stamp duty would be increased, probably to 4 tons per stamp per 24 hours.

Amalgamation—

Apron plates	45·0	square feet
Lip plates	2·33	"
Chuck block plate	2·0	"
Recovery in fine bullion 0·63 ounce or	54·8	per cent
Recovery, figuring from assay of amalgamation tailing	61·05	"

The latter recovery will be more correct as it is impossible to scrape all the amalgam from the plates.

Ore to battery, weight	16,312·0	pounds
Assay, gold	0·141	oz./ton
Content, gold	1·15	ozs.
Clean up of battery, weight	67·5	lb.
Assay	0·65	oz./ton
Content	0·022	ozs.
Amalgamation tailing, weight (by calculation)	16,244·5	lb.
Assay	0·0525	oz./ton
Content	0·426	oz.
Fine gold bullion recovered from inside, and from plates	0·63	oz.
Recovered as bullion	54·8	per cent
Recovery as figured from tailing assay	61·06	"
Gold absorbed by battery plates	0·235	oz.
Per cent absorbed by battery plates	6·26	

Concentration—

Concentrate from first tabling of amalgamation tailing	525·5	pounds
Assay	0·68	oz./ton
Content	0·179	oz.
Per cent recovered from tailing	42·0	
Per cent of total gold values in heads	15·5	
Tailing from first tabling, by calculation	15,719·0	pounds
Assay	0·03	oz./ton
Content	0·236	oz.
Per cent of gold	55·4	
Per cent of total gold	20·5	
Concentrate from second tabling, −60 mesh	270·0	pounds
Assay	0·36	oz./ton
Content	0·0486	oz.
Tailing from second tabling	13,091·0	pounds
Assay	0·012	oz./ton
Content	0·0785	oz.
Concentrate, +60 mesh	42·5	pounds
Assay	0·36	oz./ton
Content	0·00765	oz.
Tailing, +60 mesh	2,315·5	pounds
Assay	0·02	oz./ton
Content	0·0232	oz.

Recapitulation of Concentration Results—

Product	Weight lb.	Assay oz./ton	Content oz.	Per cent of Au values
First concentrate	525·5	0·68	0·179	15·54
−60 concentrate	270·0	0·36	0·0486	4·23
+60 concentrate	42·5	0·36	0·0076	0·66
Total concentrate	838·0	0·56	0·235	20·43
−60 tailing	13,091·0	0·012	0·0785	6·83
+60 tailing	2,315·5	0·02	0·0232	2·02
Total tailing	15,406·5	0·013	0·1017	8·85

Summary of Concentration Results—

Table concentrates, weight	838·0	pounds
Percentage by weight in table concentrates	5	
Assay table concentrates	0·56	oz./ton
Content in table concentrates	0·235	oz.
Per cent of gold values in table concentrates	20·43	
Table tailing, weight	15,406·5	pounds
Calculated content in table tailing	0·191	oz.
Calculated assay of table tailing	0·0248	oz./ton
Per cent of gold values in tailing	16·61	

Balance—

Product	Content oz.	Per cent
Amalgamated	0·702	61·05
Clean up battery	0·022	1·91
Concentrates	0·235	20·43
Tailings	0·102	8·85
Totals	1·061	92·24
Feed to battery	1·150	
Discrepancy	0·089	7·76
Gold recovered by amalgamation and in table concentrates		81·47

Cyanidation of Table Concentrates.—A number of cyanidation tests were made on the table concentrates. A summary of the results obtained is given in brief form in the following table.

Remarks.—The concentrate was ground wet and intermittently screened through the required mesh. The grinding was done with pebbles, and lime equivalent to 10 pounds per ton was added to the pebble mill during grinding.

In the first three tests the grinding was through 150 mesh, and in the last two tests, through 200 mesh. Finer grinding and longer contact increased the cyanide consumption, but the increase in extraction was over 10 per cent.

Summary of Cyanide Tests on Pyrite Concentrates from Amalgamation Tailings

Test	Mesh	Weight grms.	Solution c.c.	Time of agitation	KCN per cent	Assay oz./ton		Extraction per cent	Reagents consumed	
						Heads	Tails		KCN	CaO
				hours					lb./ton	lb./ton
A	150	489·0	1511	48	0·15	0·56	0·14	75·0	2·53	3·7
B	150	289·0	1213	48	0·20	0·56	0·12	78·8	3·1	2·5
C	150	241·0	1259	48	0·10	0·56	0·10	82·2	2·4	10·3
D	200	512·0	2500	68	0·075	0·56	0·06	89·2	4·34	
E	200	513·5	2500	68	0·150	0·56	0·06	89·2	4·85	

Report No. 227

THE PREPARATION OF VOLCANIC ASH FROM WALDECK, SASKATCHEWAN, FOR INDUSTRIAL PURPOSES

R. K. Carnochan

Shipments.—Two bags of volcanic ash, net weight 192½ pounds, were received January 8, 1925; and two bags, net weight 95 pounds, January 23, 1925, at the Ore Dressing and Metallurgical Laboratories. Both shipments were submitted by Mr. W. C. Vance, of the Van Kel Cleaners, Limited, Swift Current, Sask., and were taken from the company's volcanic ash deposit near Waldeck.

Characteristics of Volcanic Ash Submitted—

Size	Pieces broken to about 4 inches.
Colour	Light buff.
Weight	Very light.
Texture	Fine-grained; the grains were not bonded together solidly; a powder could be produced by rubbing with the fingers; the ash could readily be cut with a knife.

Purpose of Experimental Tests.—Tests were desired to determine possible methods of crushing, grinding, and classification, to produce various grades of products, which could be used in oil filtering, as well as hand cleaner, household cleanser, and metal polish.

Experimental Tests on Shipment No. 1.—After selection of a few specimens, the remaining 192 pounds of shipment No. 1 was crushed by means of a jaw crusher and rolls to all pass an 8-mesh screen. The −8-mesh material was used in making grinding tests with a Raymond grinder in which an effort was made to grind to almost all pass −48 mesh, and have as much as possible +100 mesh. The ground product from the Raymond was put through a Gayco air separator so as to separate it into −48+100 and −100-mesh products.

SUMMARY

The tests made on shipment No. 1 show:—

1. The ash crushed very easily in the jaw crusher.

2. Rolls are unsuited for the crushing of this material, as the ash shows a tendency to form cakes which do not go through the screen or crush up when re-run through the rolls.

3. The Raymond grinder can be adjusted to take material −8 mesh and give a product:—

+ 48 mesh	0·2 per cent
− 48+100 "	26·8 "
−100 "	73·0 "

4. The Gayco air separator gives a good fine product:—

+100 mesh	5·4 per cent
−100 "	94·6 "

The coarse product is not very good, as it tests:—

+100 mesh	24·8 per cent
−100 "	75·2 "

The trouble in the operation of this separator is that the ash keeps breaking up as it is going through, so that there are always fines in the coarse product. When this product is re-run five times, as was done to clean the fines out, a very small amount of coarse material remains.

Experimental Tests on Shipment No. 2.—Seventy pounds was crushed in a jaw crusher to 1½ inch, and then crushed in a hammer mill to all pass a 6-mesh screen. The hammer mill product was screened and gave:—

− 6+10 mesh	18 pounds
−10+48 "	24 "
−48 "	24 "

The −10+48 was crushed in a Raymond grinder to nearly all pass 48 mesh and have as much as possible +100 mesh. The product from the Raymond mill was mixed with the −48-mesh material from the screen, and the mixture separated in a Gayco air separator into:—

Oversize	0·5 pounds
Fines	46·5 "

A screen test on the oversize shows:—

+ 48 mesh	18·4 per cent
− 48+100 "	41·4 "
−100 "	40·2 "

A screen test on the fines shows:—

+ 48 mesh	2·5 per cent
− 48+100 "	6·4 "
−100 "	91·1 "

SUMMARY

1. The hammer mill is well adapted to the crushing of the ash.
2. The Raymond grinder takes the −10+48-mesh product from the screen and gives a product:—

+ 48 mesh	7·8 per cent
− 48+100 "	7·6 "
−100 "	84·6 "

3. As in the test on shipment No. 1, the Gayco air separator gave a good fine product. The coarse product was better than that obtained from shipment No. 1, as it tested:—

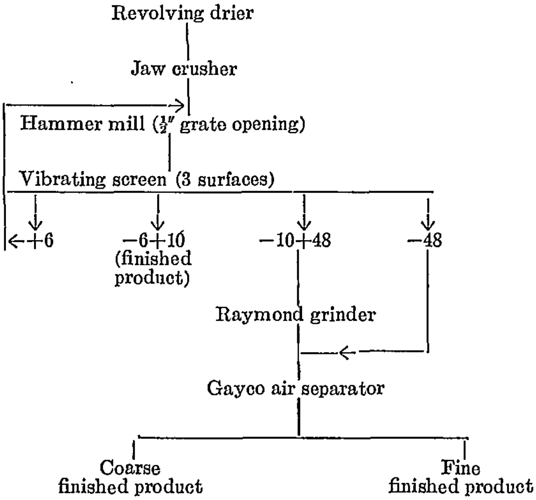
+ 48 mesh	18·4 per cent
− 48+100 "	41·4 "
−100 "	40·2 "

This is 59·8 per cent +100 mesh, whereas the coarse from shipment No. 1 is only 24·8 per cent +100 mesh. This improvement is due to the fact that the proper setting of the machines had been determined when working on the first shipment.

The ash keeps breaking up when put through the Gayco air separator, and this results in a very small amount of coarse being produced. The Gayco air separator used was a small one, and in commercial practice where a larger machine would be used, one pass instead of five as used in these tests, would be sufficient to secure a coarse product free of fines.

CONCLUSIONS

1. The following flow-sheet is recommended for the dry crushing of the volcanic ash:—



2. Small tests made on the volcanic ash show that it is very porous and absorbs moisture, and that it is consequently very difficult to dry, hence the drier installed should be large enough to handle a material of this nature at the required capacity.

3. The above flow-sheet would give a −6+10-mesh product suitable for oil filtering, a coarse product from the air separator suitable for making a hand cleaner, and a fine product for making household cleanser. The tests on shipment No. 2 show that these products would be produced in the following proportions:—

−6+10 mesh	18·0 pounds	25·7 per cent
Air separator, coarse	0·5 "	0·7 "
Air separator, fines	51·5 "	73·6 "
Feed	70·0 "	100·0 "

In the above table it is assumed that the dust loss in the tests would be fines, and that in regular mill work only a small amount would be lost in dust. In regular practice with a large air separator, more coarse material would be produced and less fines, as only one pass would be needed.

4. If a very fine product was desired for metal polish, or any other purpose, a part of the fines from the air separator could be re-run, or put through a second air separator, so as to give a very fine product.

Report No. 228

THE CONCENTRATION OF A DRY SILVER ORE FROM THE SLOCAN SILVER MINES LTD., ALAMO, B.C.

C. S. Parsons

Shipment.—A shipment of 200 pounds of silver-lead ore was received May 7, 1925, from the Slocan Silver Mines, Alamo, B.C. The shipment consisted of two sacks designated as 'Lot No. 1', one sack as 'Lot No. 2,' and one sack as 'Lot No. 3'.

Characteristics and Analysis of Shipment.—The three lots represented distinct types of ores. Lot No. 1 consisted of clean siliceous ore representing 90 per cent of the minable ore. Lot No. 2 consisted of badly oxidized ore, containing a large amount of brown manganese oxide, representing 2 per cent of the minable ore. Lot No. 3 consisted of ore found near the breaks in the vein in which secondary minerals have formed, and represented 8 per cent of the minable ore. The analysis of the samples showed them to contain:—

	Lead per cent	Zinc per cent	Copper per cent	Silver oz./ton
Lot No. 1	1·05	0·80	0·27	37·7
Lot No. 2	2·61	0·75	2·80	349·6
Lot No. 3	0·65	0·75	1·14	150·3

Purpose of Experimental Tests.—The experimental tests were conducted to determine the recovery and grade of concentrate that could be obtained from each type of ore by flotation, and from the ores when mixed in the proportions given above, so that the owners might determine whether it would be more profitable to ship the ore direct to the smelter, or concentrate it to obtain a high-grade shipping product.

EXPERIMENTAL TESTS

Procedure.—A series of tests were made on each of the three lots, and on a composite sample made up in the proportion of the percentages of each in the run-of-mine. The sample cut out from each lot and from the composite lot for test purposes, was assayed for silver content only. The results are given in the following table:—

Results of Flotation Tests:—

Lot No.	Test No.	Product	Weight		Assay Ag. oz./ton	Values Ag. per cent	Remarks
			Grms.	Per cent			
1		Head sample			26·70		3 per cent +100 mesh; pulp neutral. Xanthate 0·4 lb./ton. Pine oil to froth.
	1	Concentrate	89·7	9·0	271·30	91·0	
		Tailing	901·5	91·0	2·65	9·0	
		Totals	991·2	100·0	26·9	100·0	
	2	Concentrate	110·5	11·1	220·60	91·2	3 per cent +100 mesh; pulp neutral. Soda ash 2 lb./ton, xanthate 0·4 lb./ton. Pine oil to froth.
		Tailing	883·9	88·9	2·65	8·8	
		Totals	994·4	100·0	26·8	100·0	
	3	Concentrate	50·9	5·6	420·60	89·4	3 per cent +100 mesh; pulp neutral (40 per cent coal tar, 60 per cent coal-tar creosote) 0·75 lb./ton, 0·2 lb./ton xanthate. Pine oil to froth.
		Tailing	861·2	94·4	2·96	10·6	
		Totals	912·1	100·0	26·3	100·0	
2		Head sample			233·80		−100 mesh; pulp neutral (40 per cent coal tar, 60 per cent coal-tar creosote) 0·75 lb./ton, 0·2 lb./ton xanthate. Pine oil to froth.
	1	Concentrate	117·7	11·7	1600·90	78·9	
		Tailing	884·0	88·3	56·88	21·1	
		Totals	1001·7	100·0	238·0	100·0	
3	1	Head sample			113·10		3 per cent +100 mesh; pulp neutral (40 per cent coal tar, 60 per cent coal-tar creosote) 1 lb./ton, 0·2 lb./ton xanthate. Pine oil to froth.
		Concentrate	81·2	8·0	1086·80	77·8	
		Tailing	930·3	92·0	27·14	22·2	
		Totals	1011·5	100·0	112·2	100·0	

Composite Sample: Lot No. 1, 90 per cent; No. 2, 2 per cent; No. 3, 8 per cent; Heads Ag. 37·75 oz.

Test No.	Product	Weight		Assay Ag. oz./ton	Values Ag. per cent	Remarks
		Grms.	Per cent			
1	Concentrate	91·2	9·0	375·00	88·3	3 per cent +100 mesh; pulp neutral. (40 per cent coal tar, 60 per cent coal-tar creosote) 0·75 lb./ton, 0·2 lb./ton xanthate. Pine oil to froth.
	Tailing	924·0	91·0	4·94	11·7	
	Totals	1015·2	100·0	38·2	100·0	

Screen Test—Lot No. 1—Feed to Flotation—

Mesh	Weight, grms.	Weight, per cent
+ 65	1·2	0·24
− 65+100	13·4	2·68
−100+150	66·4	13·28
−150+200	75·2	15·04
−200	343·8	68·76

SUMMARY AND CONCLUSIONS

The tests show that there is no difficulty in concentrating by flotation the ore of Lot No. 1, which represents 90 per cent of the run-of-mine. The recovery was high, and in practice should exceed 90 per cent of the silver values. It is rather difficult to determine the maximum grade of concentrate that could be obtained in mill operations, from the results of these small-scale tests, but as the concentrates obtained in tests Nos. 1, 2, and 3, were not recleaned, and are only rougher concentrates, it would seem that a concentrate exceeding 500 ounces silver to the ton could be obtained in actual plant operations. The best results were obtained in a neutral pulp with the use of a mixture of coal tar and coal-tar creosote (acid) and potassium xanthate. The coal tar and creosote mixture was added to the ball mill and the xanthate in solution direct to the head of the flotation cells.

The results of the tests on Lots Nos. 2 and 3 show a low recovery. No improvement is shown in the results on the composite sample. In fact, if the recovery made on Lot No. 1 is taken as an average of 90·5 per cent, on Lot No. 2 as 78·9 per cent, and on Lot No. 3 as 77·8 per cent, the calculated recovery on the composite sample should be 89·3 per cent instead of 88·3 per cent as obtained. This checks very closely, showing that the total recovery obtained by concentrating each type of ore separately is practically the same as when the ores are mixed and concentrated together.

From the results of the above tests, it would seem that the concentration by flotation of the ore represented by Lot No. 1 would be a profitable operation, and it would also seem that the ores represented by Lots Nos. 2 and 3 should be shipped direct.

Report No. 229

THE CONCENTRATION OF A COPPER ORE FROM THE PITT MINING CO., PITT LAKE, B.C.

J. S. Godard

Shipments.—A shipment of copper ore, gross weight 218 pounds, was received on May 23, 1925, from the Pitt Mining Co., Pitt Lake, B.C. The shipment represented the low-grade ore from the company's property.

Characteristics of the Ore.—The ore is chalcopyrite associated with pyrite in a quartz gangue. Some silver and a small amount of gold is associated with the sulphides.

Purpose of Experimental Tests.—The purpose of the tests was to concentrate the copper and silver values in the ore into a shipping product.

Sampling and Analysis.—The entire shipment of 218 pounds was crushed to $\frac{1}{4}$ inch and cut twice; 53 pounds was crushed to 8 mesh and cut twice; 14 pounds was crushed to 20 mesh and the head sample cut from this latter quantity. Analysis showed it to contain:—

Gold................	0·035 oz./ton	Copper...............	2·18 per cent
Silver................	0·88 "	Iron..................	7·36 "
Arsenic..............	trace	Lead.................	none
Zinc.................	0·10 per cent	Insoluble.............	69·32 "

EXPERIMENTAL TESTS

A number of flotation tests were made using various reagents, and after grinding to various degrees of fineness. Grinding previous to flotation was done in porcelain pebble jars using grey iron balls and a pulp density of 50 per cent solids. Flotation tests were made in a Ruth machine. In test No. 7, a slightly different procedure was followed. In the previous tests 900 to 1,000 grammes of ore was used in each case, the result being that the pulp dilution in the cleaning of the rougher concentrate was too great, resulting in a high-grade middling product and a low recovery in each test except No. 6. In test No. 7, three lots of ore of 962, 964, and 979 grammes were floated separately, the concentrates being combined for cleaning. The results of the tests, and the reagents used, are given in the following table :—

Test No.	Mesh	Product	Weight per cent	Assays			Per cent of values		Reagent	Amount lb./ton	Added to
				Ag oz./ton	Au oz./ton	Cu per cent	Ag	Cu			
1	65	Concentrate	5·6	10·37	0·09	29·24	58·1	82·1	Lime	5·0	Ball mill.
		Middling	5·3	5·22	....	6·00	27·7	15·9	Xanthate	0·07	Ruth machine.
		Tailing	89·1	0·16	....	0·045	14·2	2·0	Pine oil No. 5	0·10	Ruth machine.
2	65	Concentrate	6·6	98·3	0·05	25·70	70·8	79·9	Water-gas tar, 50 per cent. Coal-tar creosote, 50 per cent.	0·40	Ball mill.
		Middling	9·7	2·41	....	3·63	25·6	16·6			
		Tailing	83·7	0·04	....	0·09	3·6	3·5	Sodium carbonate	3·0	Ball mill.
									Pine oil No. 5	0·1	Ruth machine.
3	−48	Concentrate	4·1	6·90	....	31·56	32·4	58·7	Lime	5·0	Ball mill.
		Middling	6·6	6·76	....	12·17	51·2	36·4	Xanthate	0·07	Ruth machine.
		Tailing	89·3	0·16	....	0·12	16·4	4·9	Pine oil No. 5	0·1	Ruth machine.
		“ + 48	0·3	trace	....	0·10	....	13·3			
		“ + 65	2·2								
		“ +100	13·6								
		“ +150	17·4	0·14	....	0·09	14·9	13·3			
		“ +200	19·9	0·175	....	0·09	21·7	15·0			
		“ −200	46·6	0·22	....	0·15	63·4	58·4			
4	65	Concentrate	4·9	7·20	0·10	30·18	40·1	67·6	Lime	2·5	Ball mill.
		Middling	6·2	5·92	....	9·87	41·7	27·5	QED	0·025	Ruth machine.
		Tailing	88·9	0·18	....	0·12	18·2	4·9	Pine oil No. 5	0·1	Ruth machine.
5	65	Concentrate	4·9	7·20	0·08	31·44	44·2	70·7	Lime	4·0	Ball mill.
		Middling	5·5	5·82	....	9·85	40·1	24·8	Water-gas tar, 50 per cent. Coal-tar creosote, 50 per cent.	0·4	Ball mill.
		Tailing	89·6	0·14	....	0·11	15·7	4·5			
									Pine oil No. 5	0·1	Ruth machine.
6	65	Concentrate	13·1	6·70	....	16·48	88·3	94·1	Soda carbonate	3·0	Ball mill.
		Middling	6·7	1·02	....	0·84	6·8	2·4	Xanthate	0·5	Ruth machine.
		Tailing	80·2	0·06	....	0·10	4·9	3·5	Pine oil No. 5	0·10	Ruth machine.
		“ + 65	1·1	0·11	....	0·09	23·2	1·0	Pine oil No. 5 for clean-up	0·05	Ruth machine.
		“ +100	12·1			0·14		16·3			
		“ +150	14·2	0·05	....	0·06	11·3	7·7			
		“ +200	14·1	trace	....	0·06	....	7·7			
		“ −200	58·5	0·07	....	0·12	65·5	67·3			
7	......	Concentrate	8·6	8·66	....	24·68	85·9	94·0	Lime	5·0	Ball mill.
		Middling	3·1	1·10	....	0·85	3·9	1·1	Xanthate	0·07	Ruth machine.
		Tailing	88·3	0·10	....	0·125	10·2	4·9	Pine oil No. 5	0·10	Ruth machine.
									Xanthate for clean-up	0·01	Ruth machine.

SUMMARY

In tests Nos. 1 to 5 the recoveries are poor, though the grade of the copper concentrate is good. The middling in every case consisted of too large a quantity and was too high-grade to be retained in a mill circuit. In tests Nos. 6 and 7, the recoveries are good in the flotation concentrate, amounting on the average to 87 per cent of the silver and 94 per cent of the copper. The copper in test No. 7 concentrate being considerably in excess of that in No. 6. The middlings are more commensurate to those that would be obtained in actual mill practice.

Grinding to 65 mesh was found to be fine enough for good flotation results.

The gold values are concentrated with the silver and copper.

Potassium xanthate used in an alkaline circuit, preferably lime, gives good results, and it is coming into extensive use in flotation of copper ores. Its action is selective, exhibiting a preference for copper rather than iron. General Naval Stores pine oil No. 5 is an efficient frother.

Report No. 230

THE CONCENTRATION OF A COPPER-LEAD-ZINC ORE FROM ALBERT COUNTY, N.B.

C. S. Parsons

Shipment.—Two samples of copper-lead-zinc ore were submitted by John E. Teahan, Sr., Kerry P.O., N.B. The samples were obtained from a property near New Ireland, Alma parish, Albert county, N.B. Sample No. 1 was of 3 pounds; sample No. 2, of 116 pounds, was received at the Ore Testing Laboratories June 8, 1925, and was taken across a 30-foot face, 5 feet below the surface.

Purpose of Experimental Tests.—The samples were submitted for analysis and also to determine a method of concentration into commercial shipping products.

Character of the Ore.—The ore consisted of sulphides of iron, copper, lead, and zinc, finely crystalline and intimately associated. It also carried values in silver and gold.

Analysis of Samples.—Analysis of the two samples showed them to contain:—

	Sample No. 1	Sample No. 2
Copper	11·86 per cent	2·35 per cent
Arsenic	3·57 "	0·76 "
Lead	2·62 "	1·26 "
Zinc	7·44 "	14·27 "
Gold	0·44 oz./ton	0·03 oz./ton
Silver	14·94 "	3·91 "
Antimony	.....	trace

Experimental Test

The experimental work was conducted on sample No. 2. This sample taken across a 30-foot face, 5 feet below the surface, was considered to be truly representative of the class of ore to be expected from the development of the property. A selective flotation test was conducted making three products, namely a copper-lead concentrate, a zinc concentrate, and a tailing.

One thousand grammes of the ore, previously crushed to 14 mesh, was ground for 40 minutes in a ball mill. The following reagents were used to obtain a separation of the copper and lead from the zinc:—

Soda ash............................	10·0	lb./ton added to ball mill.
Thiocarbanilide.......................	0·25	" " "
Sodium cyanide......................	0·24	" " "
Pine oil..............................	2	drops for lead-copper flotation.
Copper sulphate......................	1·5	lb./ton for zinc flotation.
Xanthate............................	0·3	" " "

The results obtained are given in the following table:—

Product	Weight		Analysis					Per cent of values				
	Grms.	%	Cu %	Pb %	Zn %	Au oz./ton	Ag oz./ton	Cu	Pb	Zn	Au	Ag
Copper-lead concentrate	169·7	17·0	12·32	6·62	13·86	0·08	14·76	90·3	93·3	16·3	41·6	70·0
Zinc concentrate......	231·0	23·0	0·56	0·25	49·6	0·04	3·0	5·5	4·2	80·5	29·1	19·6
Tailing......	600·0	60·0	0·16	0·05	0·7	0·015	0·56	4·1	2·5	3·0	28·3	9·5

SUMMARY AND CONCLUSIONS

The results show that the ore can be concentrated into two marketable products, namely a copper-lead product and a zinc product. The copper-lead concentrate obtained represented 17 per cent by weight of the ore; assayed 12·32 per cent copper, 6·62 per cent lead, 13·86 per cent zinc, 14·76 ounces silver per ton, and 0·08 ounce gold per ton. No attempt was made to reclean the concentrate to eliminate some of the zinc, which would be done in practice. This concentrate contained 90 per cent of the copper values in the ore, 93 per cent of the lead, 70 per cent of the silver, and 42 per cent of the gold.

The zinc concentrate obtained represented 23 per cent by weight of the ore; assayed 49·6 per cent zinc, with very small amounts of copper and lead, and is a good shipping product. This concentrate contained 80 per cent of the zinc values in the ore.

The tailing representing 60 per cent by weight of the ore carried very little values. It would seem, however, from the analysis and loss of values in the tailing that the gold values were to some extent associated with the iron sulphides, which reported in the tailing.

The test is an interesting one, showing what can be done by selective flotation on an intimate mixture of minerals in an ore.

Report No. 231

THE CONCENTRATION OF THE COPPER ORE ("C" ORE-BODY) OF THE HORNE MINE, NORANDA MINES, LTD., ROUYN, QUE.

J. S. Godard

Shipment.—A shipment weighing 115 pounds was received June 15, 1925. It was designated chalcopyrite, and was taken from "C" ore-body of the company's property at Rouyn, Que.

Characteristics of the Ore.—The ore consists of chalcopyrite, pyrrhotite, and pyrite with which is associated some gold and silver. The gangue material is rhyolite.

Purpose of Experimental Tests.—The purpose of the tests was to concentrate the chalcopyrite to the greatest possible extent commensurate with good recoveries of copper, gold, and silver.

Sampling and Analysis.—The entire sample was crushed to one-half inch and cut twice, then reduced to −4 mesh and cut once, further reduced to −14 mesh and cut twice. The last cut was ground to −150 mesh and a head sample cut out. Analysis was as follows:—

Gold..................	0·04 oz./ton	Copper...............	4·60 per cent
Silver.................	0·22 "	Zinc.................	none
Lead..................	none	Iron..................	19·83 "
Arsenic...............	trace	Insoluble.............	50·64 "

EXPERIMENTAL TESTS

Test No. 1

A sample of about 2,000 grammes of ore was cut from 24 pounds at −14 mesh and screened on Tyler standard screens. The products were assayed for gold and copper.

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
+ 20..................................	10·8	0·05	4·03	9·6	9·3
− 20+ 28..............................	14·0	0·04	4·01	10·0	12·0
− 28+ 35..............................	16·5	0·05	4·07	14·8	14·4
− 35+ 48..............................	11·9	0·04	4·08	8·6	10·4
− 48+ 65..............................	9·2	0·05	4·18	8·2	8·2
− 65+100..............................	7·5	0·07	4·95	9·4	8·0
−100+150..............................	5·2	0·08	5·20	7·5	5·8
−150+200..............................	4·8	0·08	5·57	6·8	5·7
−200..................................	20·1	0·07	6·06	25·1	26·2

Test No. 2

Flotation, Tabling, and Screening.—One thousand grammes of ore at −14 mesh was ground wet for 20 minutes in a ball mill with 5 pounds of lime per ton, then floated in a Ruth machine with xanthate 0·07 pound per ton, and No. 5 pine oil, 2 drops. The flotation tailings were tabled on a laboratory Wilfley table. The table tailings were screened.

Product	Weight per cent	Assays			Per cent of values	
		Au oz./ton	Cu per cent	Fe per cent	Au	Cu
Flotation concentrate	25·6	0·12	17·08	28·40	67·9	95·6
Table concentrate	1·9	0·08	0·10	44·40	3·3	....
Table middling	6·4	0·02	0·20	42·40	2·9	0·3
Table tailing	50·4	0·02	0·32	....	22·4	3·5
Slimes	15·7	0·01	0·18	....	3·5	0·6
− 65+100	7·9	0·015	0·62	....	6·3	15·1
−100+150	13·5	0·020	0·57	....	14·3	23·8
−150+200	13·4	0·015	0·41	....	10·6	17·0
−200	65·2	0·020	0·22	....	68·8	44·1
Average tailing	....	0·19	0·32	....	....	....

Tests Nos. 3 to 14—Flotation

The details, results of tests, and the flotation reagents used are given in the following table:—

Summary of Flotation Tests Nos. 3 to 14

Test No.	Mesh	Product	Weight per cent	Assays			Per cent of values		Reagent	Amount lb./ton	Added to	Remarks
				Au oz./ton	Cu %	Fe %	Au	Cu				
3	− 65	Concentrate	17·9	0·15	23·18	29·4	71·2	92·3	Lime	5·0	Ball mill	No other reagents added for clean-up. Froth good.
		Middling	9·8	0·05	1·33	......	9·0	2·9	Xanthate	0·07	Ruth.	
		Tailing	72·3	0·015	0·30	......	19·8	4·8	Pine oil No. 5	0·05	Ruth.	
4	− 65	Concentrate	35·3	0·11	12·14	36·60	81·2	95·0	Soda carbonate	4·0	Ball mill	Froth good. Concentrate, iron-looking in appearance.
		Middling	8·5	0·04	1·39	......	7·1	2·6	Xanthate	0·07	Ruth.	
		Tailing	56·2	0·01	0·19	......	11·7	2·4	Pine oil No. 5	0·03	Ruth.	
5	−100	Concentrate	21·5	0·14	19·82	33·12	55·4	93·7	Lime	3·0	Ball mill	Two lots ore, 1,000 grms. each, floated. Rougher concentrate cleaned together, tails combined. Froth dull—good.
		Middling	8·2	0·08	1·35	30·39	12·2	2·4	Xanthate	0·10	Ruth.	
		Tailing	70·3	0·025	0·25	......	32·4	3·9	Xanthate	0·01	Clean-up.	
									Pine oil No. 5	0·05	Ruth.	
6	−100	Concentrate	15·0	0·19	27·62	29·38	39·5	91·5	Lime	4·0	Ball mill	Froth dull in colour but good, 0·025 lb./ton, pine oil No. 5 added for clean-up.
		Middling	6·4	0·13	2·89	23·52	11·5	4·1	QED	0·05	Ruth.	
		Tailing	78·6	0·045	0·25	......	49·0	4·4	Pine oil No. 5	0·05	Ruth.	
7	−100	Concentrate	22·9	0·16	17·20	35·66	47·1	86·0	Soda carbonate	4·0	Ball mill	Froth bright—good, 0·025 lb./ton, pine oil No. 5 added for clean-up.
		Middling	15·0	0·15	2·47	27·78	29·0	8·1	QED	0·05	Ruth.	
		Tailing	62·1	0·03	0·44	......	23·9	5·9	Pine oil No. 5	0·05	Ruth.	
8	−100	Concentrate	14·5	0·17	28·72	29·38	42·9	89·9	Lime	4·0	Ball mill	Aero brand $Ca(CN)_2$ used.
		Middling	8·2	0·07	3·00	21·29	9·9	5·3	Cyanide	0·3	Ball mill.	
		Tailing	77·3	0·035	0·29	......	47·2	4·8	Xanthate	0·07	Ruth.	
									Pine oil No. 5	0·05	Ruth.	
9	−100	Concentrate	10·6	0·22	27·26	30·59	31·2	63·1	Soda carbonate	4·0	Ball mill	
		Middling	13·4	0·14	7·37	24·54	25·1	21·6	Cyanide	0·3	Ball mill.	
		Tailing	76·0	0·043	0·92	......	43·7	15·3	Xanthate	0·07	Ruth.	
									Pine oil No. 5	0·05	Ruth.	
10	−150	Concentrate	11·2	0·20	26·08	29·81	34·2	63·6	TT	1·0	Ball mill.	
		Middling	18·5	0·07	7·72	22·61	19·7	31·1	Cyanide	2·0	Ball mill	Bakers C.P. NaCN.
		Tailing	70·3	0·043	0·35	......	46·1	5·3	......	......	......	Bubbles small and numerous.

11	−150	Concentrate......	12·5	0·16	31·24	29·96	26·6	88·6	Lime...............	5·0	Ball mill...	Froth good—60 per cent tailings remained on 200 mesh.
		Middling.........	6·4	0·23	3·78	27·62	19·5	5·5	Xanthate...........	0·07	Ruth.	
		Tailing...........	81·1	0·05	0·32	......	53·9	5·9	Pine oil No. 5.......	0·025		
12	−150	Concentrate......	13·4	0·19	30·20	30·16	41·5	84·4	Lime...............	5·0	Ball mill...	Froth good.
		Middling.........	6·6	0·12	7·12	22·57	12·9	9·8	Cyanide............	0·3	Ball mill...	Aero brand $Ca(CN)_2$.
		Tailing...........	80·0	0·035	0·35	......	45·6	5·8	Xanthate...........	0·07	Ruth.	
									Pine oil No. 5.......	0·05	Ruth.	
13	−150	Concentrate......	12·5	0·16	30·98	30·68	37·8	85·0	Lime...............	4·0	Ball mill...	Froth good.
		Middling.........	7·4	0·12	4·80	21·34	16·8	7·8	Cyanide............	0·3	Ball mill.	
		Tailing...........	80·1	0·03	0·41	.....	45·4	7·2	QED...............	0·05	Ruth.	
									Pine oil No. 5.......	0·05	Ruth.	
14	−150	Concentrate......	9·5	0·16	28·58	29·14	24·6	58·6	TT.................	1·0	Ball mill...	Bubbles small and test unsatisfactory.
		Middling.........	10·0	0·12	10·37	24·62	19·4	23·3	Cyanide............	2·0	Ball mill.	
		Tailing...........	80·5	0·043	1·00	......	56·0	18·1	TT.................	0·03	Clean-up.	

SUMMARY AND CONCLUSIONS

1. In the first three flotation tests excellent copper recoveries were made. If the grade of the ore is taken into consideration, the gold recoveries obtained must also be considered as good. The loss of gold in the tailing varied between 20 and 30 cents, which gives a recovery of between 80 and 70 cents. In the latter tests the aim was to produce a high-grade copper concentrate and to determine the recovery of gold obtained with the different percentages of copper in the concentrate. It was found that the recovery of the gold dropped as the grade of the copper in the concentrate was increased. It is hoped that this information will be of assistance in determining the grade of copper concentrate from which the smelter will obtain the greatest profit.

2. From these results it is evident that the gold is not entirely associated with the chalcopyrite but must be either partly free or associated with the other sulphides. The results from tests on samples from "A" ore-body show that the pyrrhotite carries no gold and as pyrrhotite is easily dropped in any alkaline pulp, very little would float in any of the tests reported here. Therefore it follows that the gold is in all probability partly associated with chalcopyrite, partly with pyrite, and partly free, since from the results obtained in a screen analysis made on the flotation tailing from test No. 2, it will be observed that 68·8 per cent of the gold remaining in this tailing is in the −200-mesh product.

3. Crushing finer than 65 mesh did not improve the recovery of the gold or copper.

Report No. 232

THE CONCENTRATION OF A LEAD-ZINC ORE FROM THE KICKING HORSE MINE, FIELD, B.C.

C. S. Parsons

Shipment.—A shipment of 424 pounds of ore was received from the Kicking Horse mine, Field, B.C. It was submitted by the Pacific Mines, Ltd., 607-608 Pacific Building, Vancouver, B.C.

Characteristics of the Ore.—The ore consisted of galena and sphalerite carrying silver values. The sulphides were disseminated throughout the gangue, and required fine grinding in order to free them.

Purpose of Experimental Tests.—Experimental work was requested to determine the amenability of the ore to concentration and the methods of treatment most suitable to obtain marketable products with high recoveries of the contained values.

Analysis of Sample.—

Lead	7·16 per cent
Zinc	14·06 "
Silver	3·04 oz./ton
Gold	trace

CONCENTRATION TESTS

Each lot of ore used for a test was ground separately to 60 mesh. Reagents, as given below, were added and ground with the ore.

Test No. 1

Reagents—

Added to ball mill—density of ball mill pulp	1 : 1	
Soda ash	6·0	lb./ton
Low-grade calcium cyanide (Aero Brand)	2·0	"
Thiocarbanilide	0·2	"
Added to cells—		
Lead cell—Cresylic acid	0·15	"
Zinc cell— Copper sulphate	1·0	"
Xanthate	0·4	"
Pine oil	1	drop

Test No. 2

Reagents—

Added to ball mill—ball mill pulp density	1 : 1	
Soda ash	4·0	lb./ton
Cyanide (Aero Brand)	4·0	"
Thiocarbanilide	0·2	"
Added to cells—		
Lead cell—Pine oil	1	drop
Zinc cell— Copper sulphate	2·0	lb./ton
Xanthate	0·3	"
Pine oil	1	drop

NOTE.—Too much cyanide was used, and interfered with flotation of lead.

Test No. 3

Reagents—

Added to ball mill—ball mill pulp density	1 : 1	
Soda ash	4·0	lb./ton
Cyanide (Aero Brand)	1·0	"
Thiocarbanilide	0·2	"
Zinc sulphate	1·0	"
Added to cells—		
Lead cell—Pine oil	1	drop
Zinc cell— Copper sulphate	2·0	lb./ton
Xanthate	0·3	"
Pine oil	1	drop

SUMMARY AND CONCLUSIONS

The use of zinc sulphate, together with cyanide, gives the best results. Attention is drawn to the fact that no attempt was made to reclean either the lead or zinc concentrate. By doing so the grade of both concentrates could be materially raised.

The separations were obtained without exercising special care in the manipulation of the machines, and there is no doubt that with ore similar in character to the sample received, equally as good, or better, results could be obtained in practice.

A recovery of 80 per cent of the lead in a concentrate assaying 50 per cent lead, and a recovery of 85 per cent of the zinc with a grade of 50 per cent, and a total recovery of the silver in the two concentrates of better than 90 per cent can be expected.

Results of Flotation Tests

Test No.	Product	Weight		Assay			Per cent of values		
		Grms.	Per cent	Pb Per cent	Zn Per cent	Ag oz./ton	Pb	Zn	Ag
1	Lead concentrate...	155·0	15·7	36·03	10·93	6·30	80·0	12·3	35·4
	Zinc concentrate....	250·7	25·3	4·25	45·33	6·25	15·2	82·5	56·9
	Tailing............	584·0	59·0	0·57	1·22	0·36	4·8	5·2	7·7
2	Lead concentrate...	132·1	13·2	41·36	11·84	8·80	65·9	11·5	41·8
	Zinc concentrate....	244·4	24·5	9·67	40·27	5·52	28·5	72·3	48·8
	Tailing............	622·2	62·3	0·75	3·54	0·43	5·6	16·2	9·5
3	Lead concentrate...	135·5	13·6	48·90	8·45	9·06	77·0	8·5	45·5
	Zinc concentrate....	243·7	24·5	6·25	46·95	5·30	17·7	84·7	47·9
	Tailing............	615·5	61·9	0·75	1·49	0·29	5·3	6·8	6·6

Report No. 233

EXPERIMENTAL TESTS ON GOLD ORE FROM THE CONTACT MINES, PAULSON, B.C.

J. S. Godard

Shipment.—A shipment of ore, 117 pounds, was received at the Ore Dressing and Metallurgical Laboratories, June 29, 1925, from the Contact Mines, Paulson, B.C. The shipment was made by Mr. P. H. Fraser, K.V. Mines, Ltd., Vancouver, B.C.

Characteristics of the Ore.—The ore consists of marcasite, with which is associated small quantities of the sulphides of lead, zinc, and copper, as well as gold and silver. The gangue material is siliceous. Platinum was reported to have been found in this ore, but analysis failed to disclose the presence of any of the metals of this group.

Sampling and Analysis.—The entire sample was crushed to ¼ inch and cut twice, then reduced to 14 mesh and cut once, then ground to −150 mesh before cutting the head sample. Analysis showed it to contain:—

Copper................	0·05 per cent	Arsenic................	trace
Lead..................	0·07 "	Silica..................	69·0 per cent
Zinc..................	0·12 "	Silver..................	0·80 oz./ton
Iron..................	10·95 "	Gold..................	0·48 "

Purpose of Experimental Tests.—Experimental tests were desired to determine methods for the recovery of the values. The proximity to a smelter makes a method of treatment that would concentrate the values into a shipping product for the smelter desirable.

Experimental Tests

Test No. 1

Flotation and Tabling—

Two 1,000-gramme lots of ore were cut at −14 mesh, and ground to 100 mesh, then floated separately in a Ruth machine. The rougher concentrates from each lot were combined and cleaned. The tailings were combined and tabled on a small Wilfley table.

Product	Weight per cent	Assay Au oz./ton	Per cent of values
Flotation concentrate	15·1	2·32	72·9
Flotation middling	7·1	0·40	5·8
Table concentrate	1·1	1·86	4·2
Table tailing	47·1	0·11	10·8
Slimes	29·6	0·10	6·3

Analysis of the flotation concentrate gave—silver, 3·88 oz./ ton; lead, 0·46%; zinc, 0·46%; copper, 0·23%; arsenic trace; cobalt nil; nickel, nil; iron, 43·46%.

Flotation reagents—

Soda carbonate		5·0	lb./ton	ball mill
Lot No. 1—	Xanthate	0·6	"	Ruth
	TT mixture	0·10	"	"
	Pine oil No. 5	0·10	"	"
	Sulphuric acid	10·0	"	"
Lot No. 2—	Water-gas tar Coal-tar creosote	1·0	"	ball mill
	Sulphuric acid	10·0	"	Ruth
	TT mixture	0·1	"	"
	Fumol No. 6	0·05	"	"

No additional reagents used for cleaning concentrate.

Tests Nos. 2, 3, and 4—Cyanidation

Three cyanidation tests were made on 600-gramme lots of this ore. The table below gives the details.

No.	Mesh	Dilution	KCN per cent	Assay		Extraction per cent	Consumption lb./ton		Time hours
				Heads	Tails		KCN	CaO	
2	−100	1 : 2	0·10	0·48	0·02	95·8	2·0	4·0	62
3	−100	1 : 2	0·20	0·48	0·03	93·8	1·5	5·1	62
4	−200	1 : 3	0·15	0·48	0·02	95·8	2·2	4·3	62

All grinding done in pebble jar, discharge screened and oversize returned for further grinding.

Tests Nos. 5, 6, 7, 8, and 9—Flotation

Five flotation tests were made on lots of 1,000 grammes of ore at −14-mesh. Grinding was done in pebble jar.

Test No.	Product	Weight per cent	Assay		Per cent of values	
			Au oz./ton	Fe per cent	Au	Fe
5	Concentrate	20·4	1·04	37·33	49·3	79·4
	Middling	9·8	0·16	4·10	3·7	4·2
	Tailing	69·8	0·29	2·25	47·0	16·4
6	Concentrate	16·5	1·16	44·02	38·0	68·7
	Middling	7·2	2·20	20·14	31·5	13·7
	Tailing	76·3	0·20	2·44	30·5	17·6
7	Concentrate	18·2	1·56	44·17	59·9	76·2
	Middling	8·7	0·84	10·13	15·4	8·3
	Tailing	73·1	0·16	2·24	24·7	15·5
8	Concentrate	17·2	1·40	44·42	50·1	72·0
	Middling	9·4	1·15	14·24	22·4	12·6
	Tailing	73·4	0·18	2·22	27·5	15·4
9	Concentrate	13·2	1·24	43·76	34·1	57·8
	Middling	9·5	1·66	22·03	34·9	21·0
	Tailing	77·3	0·18	2·74	31·0	21·2

Reagents—

Test No.	Mesh	Reagents	Amount lb./ton	Added to	Remarks
5	100	Sodium sulphide Hardwood acid creosote No. 27 FPL Hercules tar oil Pine oil No. 5	5·0 0·3 0·1 0·1	Ball mill " " Ruth	Concentrate cleaned in Janney machine without further reagents.
6	150	Barrett's No. 634 Pine oil No. 5 Sulphuric acid	0·4 0·1 15·0	Ball mill Ruth "	
7	65	Barrett's No. 634 Sulphuric acid Pine oil No. 350	0·4 15·0 0·45	Ball mill Ruth "	No results from No. 634 in neutral pulp. Pine oil and sulphuric acid 3·5 lb./ton added for cleaning concentrate.
8	65	P. T. and T. No. 400 Pine oil No. 350 Sulphuric acid	0·28 0·27 18·0	Ball mill " Ruth	Pine oil and sulphuric acid 3·5 lb./ton added for cleaning rougher concentrate.
9	150	P. T. and T. No. 400 Pine oil No. 350 Sulphuric acid	0·28 0·27 18·0	Ball mill " Ruth	Pine oil and sulphuric acid 3·5 lb./ton added for cleaning rougher concentrate.

Test No. 10

Amalgamation and Tabling.—One thousand grammes ore −14 mesh was ground wet to 48 mesh and amalgamated. The amalgamation tailing was tabled.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Fe per cent	Au	Fe
Concentrate	8·2	0·60	44·57	23·2	33·3
Middling	4·0	0·48	41·82	9·0	15·2
Tailing	63·2	0·18	6·73	54·0	38·8
Slimes	24·6	0·12	5·69	13·8	12·7

Head sample—gold	0·48	oz./ton
After amalgamation	0·211	"
Gold amalgamated	56·0	per cent
Recovery—		
By amalgamation	56·0	"
In table concentrate	10·3	"
Total recovery	66·3	"

Test No. 11

Amalgamation at 48 Mesh.—Results of screen test on amalgamation tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Fe per cent	Au	Fe
+ 65	1·2	0·24	86·2	1·4	1·0
− 65+100	7·4	0·32	13·08	11·7	9·1
−100+150	13·3	0·28	13·62	18·3	17·1
−150+200	13·2	0·22	12·09	14·3	15·1
−200	64·9	0·17	9·44	54·3	57·7

Head sample—gold	0·48	oz./ton
After amalgamation	0·203	"
Gold amalgamated	57·7	per cent

Test No. 12

Amalgamation and Flotation.—One thousand grammes ore at −14 mesh, ground wet to 48 mesh and amalgamated. The amalgamation tailing was floated and the flotation tailing given a screen analysis.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Fe per cent	Au	Fe
Concentrate	14·1	0·60	43·30	38·3	64·1
Middling	8·4	0·43	19·18	16·2	16·9
Tailing	77·5	0·13	2·34	45·5	19·0
Screen test—					
+ 65	4·9	0·14	3·74	5·2	7·8
− 65+100	7·2	0·15	3·00	8·1	9·2
−100+150	12·7	0·16	2·55	15·2	13·8
−150+200	8·5	0·18	2·21	11·5	8·0
−200	66·7	0·12	2·15	60·0	61·2

Reagents—

P. T. and T. No. 400	0·28	lb./ton	for rougher	concentrate
Pine oil No. 350	0·45	"	"	"
Sulphuric acid	20·0	"	"	"
TT mixture	0·05	"	"	"
Sulphuric acid	6·0	"	for cleaning	concentrate
Pine oil No. 350	0·27	"	"	"
Pine oil No. 5	0·05	"	"	"

Recovery—

By amalgamation	53·7	per cent
In flotation concentrate	17·8	"
Total	71·5	"

Test No. 13

Amalgamation and Flotation.—The amalgamation tailings were reground for flotation. Flotation heads: gold, 0·21 ounce per ton; iron, 10·22 per cent.

—	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Fe per cent	Au	Fe
Concentrate	13·5	0·64	43·10	40·2	57·0
Middling	10·5	0·42	15·05	20·5	15·4
Tailing	76·0	0·11	3·72	39·2	27·6

Reagents—

P. T. and T. No. 400	0·28	lb./ton	added to	ball mill
Pine oil No. 350	0·45	"	"	"
Sulphuric acid	20·00	"	"	Ruth machine
Pine oil No. 350	0·16	"	for cleaning	concentrate
Sulphuric acid	6·0	"	"	"

Recovery—

By amalgamation	56·2	per cent
In flotation concentrate	18·1	"
Total	74·3	"

Test No. 14

Amalgamation and Flotation.—The amalgamation tailing was reground for flotation. Flotation heads: gold, 0·17 ounce per ton; silver, 0·77 ounce per ton.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Ag oz./ton	Au	Ag
Concentrate	28·0	0·46	2·24	75·0	81·3
Tailing	72·0	0·06	0·20	25·0	18·7

Recovery—				
By amalgamation	gold	64·2 per cent	silver	14·3 per cent
In flotation concentrate	"	26·8 "	"	69·9 "
Total	"	91·0 "	"	84·2 "

Reagents—		
Heavy hardwood oil, S. T. and C. Co.	0·56 lb./ton	to ball mill
Pine oil No. 350	0·27 "	"
Sulphuric acid	15·0 "	to Ruth
Pine oil No. 350	0·27 "	"

Tests Nos. 15, 16, and 17—Flotation at 150 Mesh—

Test No.	Product	Weight per cent	Assay		Per cent of values	
			Au oz./ton	Ag oz./ton	Au	Ag
15	Concentrate	30·0	1·18	2·50	80·0	81·4
	Tailing	69·6	0·13	0·25	20·0	18·6
16	Concentrate	30·1	1·60	2·70	93·2	89·2
	Tailing	69·9	0·05	0·14	6·8	10·8
17	Concentrate	24·3	1·50	2·92	85·9	82·5
	Tailing	75·7	0·08	0·20	14·1	17·5

Test No.	Reagents	Quantity lb./ton	Added to
15	Sodium sulphide	5·0	Ball mill.
	Dupont's flotation oil	0·40	"
	Pine oil No. 350	0·16	"
	Sulphuric acid	18·0	Ruth machine.
	Pine oil No. 5	0·02	"
	Fumol No. 6	0·02	"
16	Soda carbonate	10·0	Ball mill.
	Xanthate	0·80	Ruth machine.
	Pine oil No. 5	0·06	"
17	Barrett's No. 4	0·40	Ball mill.
	Pine oil No. 350	0·16	"
	Pine oil No. 350	0·16	Ruth machine.
	Pine oil No. 5	0·08	"
	Sulphuric acid	16·00	"

SUMMARY

Amalgamation.—Recoveries of the gold by amalgamation at 48 mesh varied from 53 to 64 per cent, with an average of 56 per cent; 14·3 per cent of the silver was amalgamated in test No. 14. The mercury was in good condition after each test.

Cyanidation.—Three small-scale cyanidation tests were made. All three showed good extraction of the gold with moderate cyanide consumption. Using a dilution of 1 : 2 and cyanide of 0·10 per cent on ore ground to −100 mesh was equally as effective as more dilute pulp and stronger cyanide on −200-mesh material.

Tabling.—Tabling the amalgamation tailings did not prove very satisfactory, as considerable gold values remained in the tailings and slimes. Large-scale operations on a classified feed would greatly improve the recovery over that obtained in small-scale work.

Flotation.—A great variety of results were obtained by flotation. The results in the first tests made were disappointing, but this was largely due to part oxidation of the pyrites. Fresh samples of ore were crushed for tests Nos. 14, 15, 16, and 17, with greatly improved recoveries. Test No. 16, using a soda pulp and xanthate, was the most satisfactory. Should an acid pulp be used, a considerable reduction in the amount of the sulphuric acid used could be made on mill-scale operations and freshly broken ore. In the latter tests the froth was good, though the sulphides were slightly sluggish.

CONCLUSIONS

Amalgamation alone is not sufficient to yield good recoveries, but it could be supplemented by tabling the classified amalgamation tailing, or by flotation. Results show that amalgamation should be considered in the treatment of this ore.

Flotation.—Good recoveries can be obtained by flotation. The best results were obtained at 150 mesh. It is a question whether the additional recoveries made at 150 mesh over those at 65 mesh would warrant the extra grinding costs. Flotation could be adopted either alone or after amalgamation.

Cyanidation.—Good recoveries of the values were obtained by cyanidation. However, before adopting this method of treatment, due consideration should be given to the possibility of the ore becoming more refractory, in which case the results by this method would not be so satisfactory.

Report No. 234

THE CONCENTRATION OF A ZINC ORE FROM RENFREW, ONT.

J. S. Godard

Shipments.—Two shipments of zinc ore designated T-1 and T-2 were received at the Ore Dressing and Metallurgical Laboratories during September, 1925, from Renfrew, Ont. Sample T-1 weighed 85·5 pounds and sample T-2, 96·5 pounds. The shipment was made by Messrs. Alderson and MacKay, Montreal.

Characteristics of the Ore.—The ore is a medium-coloured zinc blende in an altered calcareous gangue. Considerable mica is present.

Purpose of Experimental Tests.—The object in conducting the tests was to determine the amenability of the ore to concentration, the grade of the zinc product, and the recovery of the zinc values that could be expected.

Analysis.—The two samples were found to contain on analysis:—

Sample	Lead per cent	Zinc per cent	Silver oz./ton	Gold oz./ton
T-1	nil	11·75	trace	nil
T-2	trace	18·42	trace	nil

Experimental Tests

Flotation tests were made on each sample, using 1,000 grammes of ore, and a laboratory Ruth flotation machine. Reagents used:—

Soda carbonate	5·0	lb./ton added to ball mill
Xanthate	0·6	" " Ruth
Pine oil No. 5	0·05	" " "

The results of the tests are given in the following tables:—

Sample No.	Product	Weight per cent	Assay			Per cent of zinc values
			Zn per cent	Ag oz./ton	Fe per cent	
T-1	Concentrate	24·2	47·69	0·04	....	94·6
	Middling	9·3	4·37	....	....	3·3
	Tailing	66·5	0·39	....	....	2·1
T-2	Concentrate	30·3	57·63	trace	3·48	93·1
	Middling	12·9	8·13	....	....	5·5
	Tailing	56·8	0·47	....	....	1·4

Screen tests on the tailings show the following grinding results:—

Mesh	Weight per cent		Cumulative per cent	
	T-1	T-2	T-1	T-2
+ 48	1·3	....	1·3	....
+ 65	2·9	0·6	4·2	0·6
+100	14·1	13·3	18·3	13·9
+150	13·2	11·8	31·5	25·7
+200	13·7	13·5	45·2	39·2
−200	54·8	60·8	100·0	100·0

SUMMARY AND CONCLUSIONS

Straight flotation produced very good results on ore typified by samples T–1 and T–2. A good grade of concentrate, with high recoveries, is easily made.

Report No. 235

THE CONCENTRATION OF A LEAD ORE FROM THE FRONTENAC MINE, PERTH ROAD, ONT.

J. S. Godard

Shipment.—A shipment of 56 pounds consisting of 5 different samples was received at the Ore Testing Laboratories, September 30, 1925, from the Frontenac mine, Perth Road, Ont. The ore was submitted by Mr. J. M. Forbes, Ottawa.

Characteristics of the Ore.—The ore consisted of galena associated with a small quantity of zinc blende, in a calcite gangue.

Purpose of Shipment.—All samples were to be analysed for lead and zinc, and samples Nos. 3, 4, and 6 were to be given a table test.

Sampling and Analysis:—

Sample No.	Weight lbs.	Lead per cent	Zinc per cent
2	4·0	7·92	0·39
3	14·5	3·42	0·48
4	16·0	2·38	0·50
5	5·0	3·89	0·32
6	16·0	3·37	0·10

EXPERIMENTAL TESTS

One thousand grammes of each of the three samples was tabled separately on a small Wilfley table. Tabling was done on material crushed to pass a 20-mesh Tyler screen.

Results

Sample No.	Product	Weight per cent	Analyses		Per cent of values Pb
			Pb per cent	Zn per cent	
3	Concentrate	3·5	65·14	2·94	77·5
	Middling	34·7	0·52	0·51	6·2
	Tailing	53·7	0·74	....	13·5
	Slimes	8·1	1·04	....	2·8
4	Concentrate	2·5	67·52	2·96	76·1
	Middling	18·0	1·36	1·05	11·0
	Tailing	70·6	0·35	....	11·1
	Slimes	8·9	0·45	....	1·8
6	Concentrate	2·9	74·94	0·44	67·6
	Middling	8·3	2·62	0·07	6·8
	Tailing	79·4	0·97	....	23·9
	Slimes	9·4	0·59	....	1·7

SUMMARY AND CONCLUSIONS

The table concentrates contained from 65 to 75 per cent lead, with recoveries (not including the middlings) of from 68 to 76 per cent. Although these recoveries seem low, it should be borne in mind that the feed was −20 mesh, dry, and unsized, and that the operations were conducted on a small table. In actual practice, the coarse material would be jigged and the finer tabled, which, with the re-running of the middling should recover from 85 to 90 per cent of the lead values in the ore. The slimes from jig and table concentration of the sized and classified feed could be thickened and the values recovered in a small flotation unit. If sufficient zinc were present in the ore to warrant it, a zinc middling product could be cut from the tables, reground, and floated into a marketable zinc product.

Report No. 236

THE CONCENTRATION OF A COPPER-ZINC ORE FROM THE AMULET MINE, ROUYN, QUE.

C. S. Parsons

Shipment.—A shipment of 80 pounds of copper-zinc ore was received from the Amulet mine, Rouyn mining district, Quebec, October 13, 1925. The sample was marked "Composite No. 1a".

Analysis and Characteristics.—The sample contained copper and zinc with gold and silver values, associated with iron sulphides. The copper and zinc sulphides were finely disseminated, necessitating fine grinding to free them. Analysis of the sample showed it to contain the following:—

Copper	4·15 per cent
Zinc	10·00 "
Gold	0·14 oz./ton
Silver	3·02 "

Purpose of Experimental Tests.—The following tests were made with the object of separating the copper and zinc sulphides, and determining where the gold and silver values would report.

EXPERIMENTAL TESTS

A series of four selective flotation tests was run, different reagents being used in each test. The ore was ground to pass approximately 65 mesh. Two amalgamation tests were made for the recovery of the gold. One table test was made for recovery of the gold remaining in the flotation tailing.

Results of Flotation Tests:—

Test No.	Product	Weight per cent	Analysis				Per cent of values			
			Cu per cent	Zn per cent	Au oz./ton	Ag oz./ton	Cu	Zn	Au	Ag
1	Copper concentrate..	22·0	17·20	11·33	0·24	7·80	90·1	24·5	39·4	59·2
	Zinc concentrate....	14·7	1·49	47·55	0·22	3·38	5·2	69·0	24·0	17·2
	Zinc middling.......	13·8	0·79	4·09	0·32	3·80	2·6	5·5	32·9	18·6
	Tailing.............	49·5	0·18	0·20	0·01	0·33	2·1	1·0	3·7	5·6
2	Copper concentrate..	15·6	21·78	11·13	0·20	9·0	82·6	17·1	21·7	46·9
	Copper middling....	4·6	3·71	17·27	0·18	5·38	4·2	7·9	5·8	8·3
	Zinc concentrate.....	12·2	1·41	51·61	0·04	1·56	4·2	62·2	3·4	6·3
	Zinc middling.......	6·8	2·05	13·02	0·20	4·00	3·4	8·8	0·5	9·2
	Tailing.............	60·8	0·38	0·66	0·14	1·44	5·6	4·0	59·6	29·3
3	Copper concentrate..	21·5	17·62	9·91	0·46	10·66	90·7	21·0	71·6	75·1
	Zinc concentrate....	14·3	0·74	41·5	0·02	0·98	2·5	58·1	2·1	4·4
	Zinc middling.......	9·3	1·06	16·15	0·035	1·64	2·3	14·7	2·4	5·0
	Tailing.............	54·9	0·34	1·15	0·06	0·86	4·5	6·2	23·9	15·5
4	Copper concentrate..	17·3	20·66	6·91	0·56	11·96	85·5	11·7	50·7	65·9
	Zinc concentrate....	14·6	1·96	48·57	0·32	1·39	6·9	70·0	13·6	6·6
	Zinc middling.......	8·1	1·30	11·37	0·30	3·80	2·6	9·1	23·1	9·9
	Tailing.............	61·0	0·35	1·56	0·04	0·93	5·0	9·2	12·6	17·6

Results of Amalgamation Tests:—

Test made on head sample—		
Heads, gold	0·14	oz./ton
Tailing, gold	0·11	"
Recovery of gold	21·4	per cent
Test made on flotation tailing from Test No. 2—		
Heads, gold	0·14	oz./ton
Tailing, gold	0·015	"
Recovery of gold	89·3	per cent

Results of Table Test:—

Test made on flotation tailing from Test No. 4—

Product—	Gold oz./ton	Silver oz./ton
Heads	0·04	0·92
Table concentrate	0·79	2·42
Table tailing	0·02	0·96

Recovery of gold .... 68·8 per cent
Table concentrate represented .... 3·7 per cent of feed

DISCUSSION

The tests indicate that a remarkably good separation of the copper and zinc can be obtained, the copper concentrate of test No. 4 having the lowest content of zinc.

With respect to the gold and silver values, the erratic way in which they report in the different products indicates that a part of the gold is present in the ore in the free state. The writer is not inclined to view this erratic behaviour of the gold as due to the different reagents used in the various tests. The gold remaining in the flotation tailing can be reduced to 0·015 ounce per ton by amalgamation, as indicated by the test on the tailing from test No. 2, which gave a recovery of 89 per cent of the remaining gold. The amalgamation of the ore before flotation was not so successful, as a recovery of only 21 per cent was obtained.

Too much weight should not be placed on these amalgamation tests, as it will be necessary to have large-scale tests made to determine the most efficient place to introduce amalgamation, especially taking into consideration the amount of gold reporting in the zinc concentrate, in which product it is not desired. It must also be determined where the gold which is in the middling product will report in actual operation when this middling is returned continuously to the circuit. It is possible that it would be best to amalgamate after copper flotation and before zinc flotation. It is practically impossible to determine these points from small-scale laboratory work.

Referring to the table test made on the flotation tailing from test No. 4: A good recovery of the gold was obtained in a concentrate representing 3·7 per cent of the weight of the flotation tailing and assaying 0·8 ounce per ton, with a tailing assaying between 0·015 and 0·02 ounce per ton.

The presence of free gold in the sample may be due to the samples from which this composite sample was made up being taken close to the surface and becoming contaminated by free gold from the gossan capping, and that the gold will not occur in this state in the ore below surface oxidation. If this supposition is correct, amalgamation to recover the gold at some point in the process could be dispensed with.

SUMMARY

The results of the tests show that the ore is amenable to concentration by selective flotation of the copper from the zinc. High-grade copper concentrate can be obtained with recoveries of 90 per cent of the copper values. This concentrate is fairly high in zinc and a good deal of experimental work could be done to obtain a better selective action between the copper and the zinc so as to lower the zinc content. It would seem that this could be more readily done by re-treatment of the copper concentrates. High-grade zinc concentrate, containing 50 per cent zinc can be obtained and the recovery of zinc in this concentrate should be around 75 per cent.

With respect to the gold values, if the sample submitted is representative of the ore in depth, by supplementing flotation with either amalgamation or tabling, the amount of gold lost in the final tailing can be reduced to about 30 cents per ton. Part of the gold recovered reports in the zinc concentrate. The smelters allow very little for gold in a zinc concentrate, as it is difficult to recover. One of the aims of any further test work should be to prevent the gold and silver values reporting with the zinc concentrate.

Report No. 237

THE CONCENTRATION OF A ZINC-SILVER ORE FROM THE WONDERFUL MINE, SANDON, B.C.

J. S. Godard

Shipment.—A shipment of 235 pounds of a zinc-silver ore from the Wonderful mine, Sandon, B.C., was received at the Ore Dressing and Metallurgical Laboratories October 22, 1925. It was designated as sample of feed to the Alamo concentrator, operated by the Cunningham Mines Ltd. at Alamo.

Characteristics of the Ore.—The ore was tentatively classed as lead-zinc-silver, but on analysis was found to contain only 0·35 per cent lead. The principal gangue mineral is silica, but some graphite as graphite schist, some siderite, and a small quantity of a manganese mineral, presumably, rhodochrosite, are present. Silver amounting to about 8 ounces per ton is associated with the sulphides, principally the zinc.

Sampling and Analysis.—The ore as received, ½-inch size, was thoroughly mixed and cut twice in a Jones sampler. One cut was ground to −14 mesh and cut twice, then further reduced to −48 mesh before cutting the head sample. Analysis showed the ore to contain

Lead	0·35 per cent	Zinc	9·52 per cent
Silver	7·80 oz./ton	Insoluble	61·70 "
Sulphur	7·09 per cent	Manganese, determined qualitatively	

Purpose of Experimental Tests.—The flow-sheet at the Alamo concentrator consisted of grinding, screening on a Callow belt screen, tabling out the lead, dewatering the table tailing and slimes, and flotation of the slimes for recovery of the zinc. The loss of silver was high, and the recoveries of both lead and zinc were poor. Selective flotation was considered as a means of effecting a saving of the silver and increasing the recoveries and grades of concentrates in the case of both the lead and zinc. The possibility of making a zinc concentrate containing the silver and disregarding the small quantity of lead present was also considered. With the above two purposes in view, tests as tabulated below were undertaken:

Test No.	Product	Weight per cent	Assays			Per cent of values		
			Pb per cent	Zn per cent	Ag oz./ton	Pb	Zn	Ag
1	Lead concentrate	10·4	4·01	2·95	13·56	99	3·1	18·5
	Zinc concentrate	21·4	tr.	43·18	28·00	......	94·6	78·5
	Tailing	68·2	tr.	0·33	0·33	......	2·3	3·0
2	Lead concentrate	11·6	3·80	2·81	13·50	99	3·3	20·0
	Zinc concentrate	20·8	tr.	44·50	29·00	......	95·3	77·0
	Tailing	67·6	tr.	0·21	0·36	......	1·4	3·0

The tailings from tests Nos. 1 and 2 were screened:—

Test No.	Mesh	Weight per cent	Cumulative per cent
1	+ 65	0·5	0·5
	+100	9·4	9·9
	+150	11·6	21·5
	+200	13·0	34·5
	−200	65·5	100·0
2	+ 65	0·1	0·1
	+100	1·9	2·0
	+150	6·8	8·8
	+200	10·7	19·5
	−200	80·5	100·0

Reagents Used for Tests Nos. 1 and 2:—

Lead reagents—				
Soda carbonate	5·0	lb./ton	added to	ball mill
Thiocarbanilide	0·2	"	"	"
Cyanide	0·5	"	"	"
Pine oil No. 5	0·04	"	"	Ruth
Zinc reagents—				
Copper sulphate	2·0	"	"	"
Xanthate	0·5	"	"	"
Pine oil No. 5	0·04	"	"	"

Summary.—There is very little difference in results between these tests. Evidently, grinding as in No. 1 is sufficiently fine to make the separation.

Test No. 3

Selective Flotation at 65 Mesh—

Six lots of ore of 1,000 grammes each at −14 mesh were ground and floated separately in a Ruth machine. The lead concentrates were combined and cleaned. The zinc concentrates were similarly treated. All the tailings were combined.

Product	Weight per cent	Assays			Per cent of values		
		Pb per cent	Zn per cent	Ag oz./ton	Pb	Zn	Ag
Lead concentrate	4·5	8·84	1·41	22·36	92·2	0·7	13·4
Lead middling	5·9	0·19	5·27	5·08	2·5	3·2	4·0
Zinc concentrate	16·8	0·14	53·11	33·60	5·3	92·9	75·3
Zinc middling	7·2	tr.	2·91	3·54	....	2·2	3·4
Tailing	65·6	tr.	0·15	0·44	....	1·0	3·9

Reagents: Same as in tests Nos. 1 and 2.

Summary.—Good recoveries were made of the lead, zinc, and silver values. The amount of lead concentrate appeared to be much more than the analysis showed, due to the presence of graphite which floated with the lead. The carbon in the lead concentrate amounted to 5·64 per cent.

Test No. 4

Flotation at 65 Mesh—

The purpose of the test was to concentrate the lead, no attempt being made to concentrate the zinc. Eight lots of ore of 1,000 grammes each were ground separately to 65 mesh and floated separately in a Ruth machine. The cleaning of the concentrate was done as follows. Four rougher concentrates were combined and cleaned making a concentrate, then the remaining four rougher concentrates were treated similarly. Both cleaner concentrates were combined and recleaned, and the concentrates from this recleaned, making in all three middling products.

	Weight per cent	Assays			Per cent of values		
		Pb per cent	Zn per cent	Ag oz./ton	Pb	Zn	Ag
Lead concentrate	4·9	7·98	1·05	21·52	95·4	0·5	14·5
Middling No. 1	6·4	0·11	5·67	4·54	1·7	3·9	4·0
Middling No. 2	1·3	0·25	3·01	3·42	0·7	0·4	0·6
Middling No. 3	0·9	0·98	2·36	4·68	2·2	0·2	0·6
Tailing	86·5	tr.	10·24	6·74	....	95·0	80·3

Reagents—

Soda carbonate	5·0	lb./ton added to ball mill
Thiocarbanilide	0·20	" " "
Cyanide	0·50	" " "
Pine oil No. 5	0·04	" " Ruth

Summary.—It seems impossible to raise the lead content of the concentrate above 9 per cent (test No. 3) by flotation. Graphite again appeared in the lead concentrates. It is darker than the galena when seen in the cells and forms a very thin coating over the bubbles. The carbon in the lead concentrate amounted to 6·56 per cent.

Tests Nos. 5 and 6

Flotation at 65 mesh—

In tests Nos. 5 and 6, the lead was disregarded and the ore treated as a zinc-silver ore.

Test No.	Product	Weight per cent	Assay		Per cent of values	
			Zn per cent	Ag oz./ton	Zn	Ag
5	Zinc concentrate	20·0	45·68	32·94	96·8	91·0
	Zinc middling	10·4	1·30	1·94	1·4	2·8
	Tailing	69·6	0·25	0·64	1·8	6·2
6	Zinc concentrate	16·6	52·71	37·90	92·5	87·6
	Zinc middling	8·6	5·32	6·18	4·8	7·4
	Tailing	74·8	0·35	0·48	2·7	5·0

Reagents Used—

Test No. 5—	Soda carbonate	5·0	lb./ton	added	to ball mill
	Copper sulphate	2·0	"	"	Ruth
	Xanthate	0·4	"	"	"
	Pine oil No. 5	0·08	"	"	"
Test No. 6—	Lime	5·0	"	"	ball mill
	Copper sulphate	2·0	"	"	Ruth
	Xanthate	0·4	"	"	"
	Pine oil No. 5	0·08	"	"	"

Summary.—As far as recoveries are concerned there is little to choose between soda carbonate and lime as a means of producing alkalinity. Slightly better zinc recoveries were obtained with soda carbonate, although the silver results were slightly better when lime was used. The froth in the Ruth machine, using soda carbonate is more active, but if a pneumatic type of cell were used, lime could possibly be utilized at a lower cost.

SUMMARY AND CONCLUSIONS

Selective Flotation.—Good recoveries of all three metals can be made by selective flotation. There was not sufficient lead values in the ore to make a high-grade lead concentrate.

Straight Flotation.—When the lead content is disregarded and the ore treated as a zinc-silver ore, good recoveries of both zinc and silver are obtained. The concentrate is of good grade, averaging 50 per cent zinc and 35 ounces silver per ton, with recoveries of 95 per cent of the zinc and 90 per cent of the silver values. No difficulty should be experienced in duplicating the results of these tests in mill-scale operations.

The reagents used were found entirely satisfactory so no others were tried. If the ore be considered as a straight zinc-silver proposition the alkaline reagent should be added to the ball mill in the closed circuit. The best results in using xanthate should be obtained by adding it in either solid form or in solution to a pump, or some other mixing device, previous to entering the flotation cell. Pine oil if used as a frother should be injected just following the xanthate.

III

REPORTS OF INVESTIGATIONS: ELECTROCHEMICAL AND HYDROMETALLURGICAL LABORATORIES

HYDROMETALLURGICAL TREATMENT OF IRON SULPHIDE ORES FOR THE PRODUCTION OF ELECTROLYTIC IRON, WITH THE RECOVERY OF SULPHUR AND OTHER METALS AS BY-PRODUCTS

R. J. Traill and W. R. McClelland

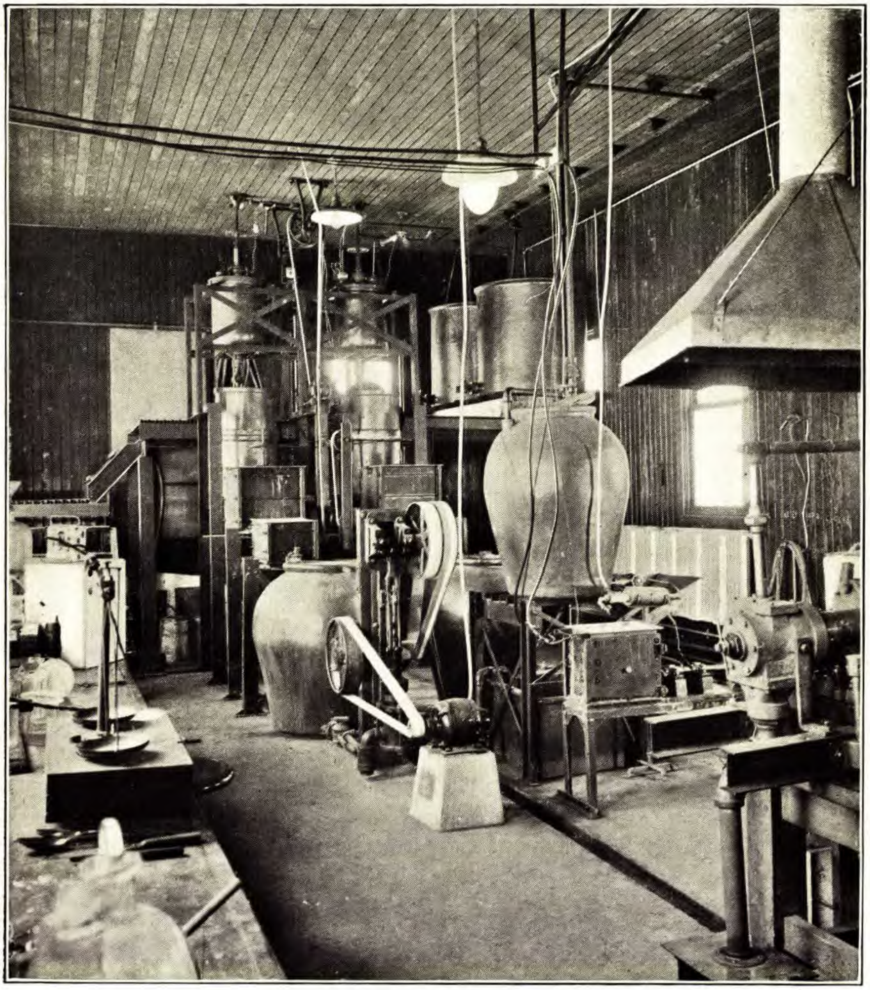
Introductory

The leaching experiments on pyrrhotite and roasted pyrite were continued along the same lines as described in the report for 1924, and were carried forward to the purification of the electrolyte and the electro-deposition of iron. Previous experiments showed quite definitely the importance of having a pure electrolyte to obtain good deposits of iron. Lead appears to be the most harmful impurity, and its almost complete elimination seems necessary, and although copper and zinc are less harmful, they should be eliminated as completely as possible for best results. Tests for the purpose of eliminating these interfering elements were made and will be briefly described.

Some progress has been made in electro-deposition; good deposits have been obtained, and cell operation has been fairly well standardized. The essential factors governing iron electro-deposition would seem to be purity, high temperature, and acidity of electrolyte. The purity is the most difficult to control and although this has been partly accomplished, it has by no means been completely solved and further work is contemplated along this line. With satisfactory electrolyte, proper design of cell, and careful control of operating conditions, it appears evident that current efficiencies averaging 95 per cent are quite possible.

Leaching of Pyrrhotite

Full details relative to leaching pyrrhotite are given in the report of 1924. The method described therein has been followed throughout these tests, and may be summarized here as treating the finely ground (−200 mesh) ore in stoneware agitation jars with a ferric-ferrous chloride solution at a temperature of 95°C. Investigation showed, however, that freshly ground ore reacted much more rapidly than ore which had been ground and left standing for some time. Apparently surface oxidation occurs when the crushed ore is allowed to stand, and this exerts a retarding influence on the leaching of the ore. With oxidized ore the time of leaching is from 50 to 100 per cent greater than with fresh ore and there is a tendency toward greater hydrolysis occurring in the liquor.



Equipment of the electrochemical laboratory for electrolytic iron investigations.

CONVERSION OF PYRITE TO PYRRHOTITE

Pyrite is not readily soluble in ferric chloride and requires a preliminary treatment which consists of roasting in a non-oxidizing atmosphere whereby one atom of recoverable sulphur is distilled off and an artificial pyrrhotite is formed. The furnace and operation are described in the report of 1924. Numerous tests were carried out on Eustis pyrite (Eustis, Quebec) ore and fairly consistent results obtained. The recoveries of sulphur were not so high as desired, but increased recoveries should not be a problem of any great difficulty. The product resulting from this roasting process conforms closely to the formula Fe_8S_9, and it is a much more soluble material than natural pyrrhotite. With the furnace as designed it was necessary to make three passes of the ore to reach approximate total elimination of the one atom of sulphur. The following are typical examples of the results:—

(A)	Charge 30 pounds, —40 mesh.........	41·2 per cent iron.
	Time of each pass....................	1½ to 2 hours.
	Analysis after 1st pass...............	21·43 per cent chloride soluble iron.
	2nd pass...............	40·14 " " "
	3rd pass...............	49·10 " " "
		1·83 " copper.
		0·70 " lead.
	Sulphur recovered....................	3·0 pounds.
(B)	Charge 30 pounds, —40 mesh.........	41·2 per cent iron.
	Time of each pass....................	about 2 hours.
	Analysis after 1st pass...............	20·42 per cent chloride soluble iron.
	2nd pass...............	38·45 " " "
	3rd pass...............	48·9 " " "
	Sulphur recovered....................	3·0 pounds.

The ore used in these tests had oxidized to some extent, and the oxide apparently picked up free sulphur forming an iron sulphide. Further sulphur was lost by condensing in cooler parts of the furnace and through initial conversion with the oxygen in the furnace to sulphur dioxide. The sulphur recovery is less than 50 per cent.

LEACHING ARTIFICIAL PYRRHOTITE

Leaching tests on roasted pyrite or artificial pyrrhotite have been described in the report for 1924. The rate of solubility is about three times that of natural pyrrhotite, probably on account of its spongy condition when obtained from the furnace. The effect of surface oxidation is also noticeable in this material and long standing of the product results in slow leaching rate with attendant hydrolysis. The separation of the residue and leach liquor by filtration is a much simpler problem in this case than it is with natural pyrrhotite.

METAL IMPURITIES AFFECTING DEPOSITION

Most pyrrhotite and pyrite ores contain small quantities of metal sulphides other than iron such as copper, lead, zinc, nickel, cobalt, etc. These metals react in leaching and become constituents of the leach liquor. It was assumed at the outset that these impurities would have a decided effect on the nature of the deposit, and this assumption was proved early in the investigation. Lead is particularly harmful in its effect on the deposition of iron. Its effect varies with its concentration in the electrolyte. In concentrations of 0·5 gramme per litre and over, characteristic highly porous, soft, spongy, and dark deposits are obtained. In concentrations below this, down to 0·004 gramme per litre, the deposits are mossy and soft and spongy, with cracking and scaling at the lower limits. Concentrations around 0·002 grammes per litre apparently have no effect on the deposition, provided the rate of flow through the cell does not exceed 4 to 5 litres per hour per square foot cathode area.

Copper in amounts over 0·05 gramme per litre tends to cause scaling and treeing, but in lower concentrations does not materially affect the nature of the deposit although the presence of copper will detract from the value of the iron for many purposes.

Zinc, the most difficult of the impurities to remove from the electrolyte, fortunately is the least disturbing of the elements in its effect on electro-deposition of iron. In concentrations of from 1 to 2 grammes per litre it has the effect of producing pitted and soft deposits, and is readily co-deposited with the iron. Concentrations below 0·35 gramme per litre have no material effect on the nature of the deposit, nor at this concentration does there appear to be any serious co-deposition of zinc and iron. It has not been definitely determined to what extent nickel and cobalt affect deposition, but it may be said that in low concentrations, 0·05 gramme per litre or less, their effect is negligible.

Hydrogen sulphide present in the electrolyte causes a black coated deposit, but does not otherwise seem to affect the nature of the deposit except from a chemical standpoint, when the probability is that the sulphur content will be above the average. Ferric chloride in concentrations of 1 gramme per litre upward causes cracking of the deposit with subsequent scaling or peeling.

PURIFICATION OF ELECTROLYTE

In electro-deposition work, generally, purification of the electrolyte has proved the most difficult problem, and iron deposition is no exception to this rule. Electrolyte liquors obtained from leaching pyrite or pyrrhotite are essentially the same, consisting of ferrous chloride and the chlorides of any other metals present in the ore. The process of purification of electrolyte should, therefore, be very much the same for both ores, the final result sought being a comparatively pure solution of ferrous chloride. The leach liquor obtained from these ores may contain one or more of the following metal chlorides: copper, lead, bismuth, antimony, arsenic, zinc, nickel, cobalt, selenium, calcium, magnesium. The latter two may be disregarded as having no effect in normal quantities, excepting, perhaps, an effect on crystalline structure. The others cause poor deposition or result in iron of less purity, through co-deposition. In the ores used in this investigation, the abundant impurities have been copper, lead, and zinc, with only traces of bismuth, arsenic, and nickel, consequently the three former impurities have received most attention.

Sponge iron at once suggests itself for the removal of copper and lead. In several experiments sponge iron was added to the ferrous liquor in quantities from two to ten times the amount necessary to cement out the copper and lead. This resulted in the removal of 90 to 95 per cent of the copper, but only about half of the lead. Deposition tests on the liquor thus treated gave characteristic lead-iron deposits, and analyses showed the presence of lead and copper. Therefore this method is not to be considered as satisfactory for removal of small quantities of copper or lead, but might readily be used where the copper content in the liquor is above 3 grammes per litre, 90 per cent of the copper being removed in a marketable form, and the remainder removed by a precipitation process.

Purification

Most of the metals likely to occur in the iron sulphide ores and subsequently in the leach liquors will react with hydrogen sulphide to form insoluble sulphides. Experiments were therefore carried out using hydrogen sulphide, iron sulphide (artificial) and calcium sulphide. Hydrogen sulphide readily precipitates copper and lead in concentrations above 0·02 gramme per litre, quantities smaller than this precipitate with difficulty. Zinc can not be completely removed by hydrogen sulphide and when present in quantities as high as 5 grammes per litre in almost neutral solution (p_H 4) as much as 0·5 gramme per litre may be left unprecipitated. This is probably due to the increased formation of hydrogen ions in solution from the hydrogen sulphide as a result of the sulphide precipitation, resulting in a lesser ionization of the hydrogen sulphide and the establishment of a solubility equilibrium between the zinc sulphide and the solution. The addition of ammonium acetate or sodium acetate to the solution reduced the concentration of the hydrogen ion and the ionization of the hydrogen sulphide takes place to a greater extent, and it is possible by this means to precipitate completely the zinc as sulphide. This method is, however, not considered practical as the amount of acetate required would be quite excessive, resulting in the formation of a complex electrolyte, with the chances of having a high-carbon iron product on electrolysis.

Pure iron sulphide was tried as a precipitant for lead and copper and was found to be fairly satisfactory and might readily be used where these impurities alone occur in an electrolyte. It was added in powdered form, (60 to 80 mesh) to the slightly acidified liquor and resulted in the precipitation of the copper and lead in sufficiently complete amount to make a good electrolyte. Little or no zinc is precipitated by this means. Calcium sulphide as a precipitant has given varying results, and although copper and lead have been fairly consistently and successfully removed from the electrolyte by its use, the same cannot be said of zinc. Liquors containing 0·5 gramme of zinc per litre and upwards have been treated by this method and in all cases zinc has been precipitated, but has never been completely removed. The lowest content yet reached has been 0·15 gramme per litre, but this has only been obtained after two or more treatments with calcium sulphide and with a co-precipitation of iron, resulting in a reduction of the iron content, and the building up of calcium salts in the electrolyte. It would be possible, of course, to recover the precipitated iron and also to eliminate the calcium salts by the addition of ferrous or ferric sulphate in the circuit, but whether this would be an economical procedure has not been determined.

It may be said that the elimination of lead and copper and possibly other heavy base metals, such as bismuth and antimony, presents no great difficulty, but the elimination of zinc still presents a problem.

It would appear, however, from data obtained, that the reduction of the zinc content to 0·3 gramme per litre would be satisfactory, as electrolytes with this zinc content have given iron deposits practically free from zinc.

ELECTRO-DEPOSITION

Electro-deposition tests were carried out in a cell constructed of ebony asbestos wood, 12 inches long, 12 inches deep, and 7¼ inches wide, inside dimensions. The cathode, a cylindrical steel mandrel, is separated from the graphite anode plates by an asbestos cloth diaphragm. The cathode has a deposition surface of approximately 1 square foot, and is rotated at a speed varying from 250 to 350 r.p.m.

Electrolysis is carried out with a hot electrolyte and for this purpose a pre-heater was used to feed hot electrolyte to the cell. The pre-heater was made from a piece of Shelby steel tubing 1 inch in diameter, and 2 feet long, surrounded by an insulated wire resistance element; the liquor from the feed tank was run through the heated tube to the cell. This scheme did not work satisfactorily as it was difficult to obtain a continuous flow of definite temperature. Recourse was made to an immersion type of heater controlled by a rheostat and placed in the cell. This consisted of a piece of pyrex tubing in the shape of a broad U having a heating element of nichrome wire spirally wound, running through it and consuming about ½ kilowatt per hour. This heater worked very satisfactorily and stood up remarkably well. The cell temperature was consistently maintained between 80 and 90°C. with the feed rate about 4 litres per hour of cold liquor. The current density usually employed in the deposition tests was 100 amperes per square foot, at a cell voltage of 4·0 to 4·4. In all some forty deposition tests were made using pure and impure electrolytes. The effect of impurities has already been mentioned, and it will only be necessary to outline here a few facts relative to best conditions for satisfactory iron deposition.

The governing factors in cell operation, given a pure electrolyte, may be stated as follows:—

Temperature of electrolyte should be between 80 and 90°C. Lower temperatures incline toward nodular deposits and lower current efficiency. Acidity of electrolyte is also important, high acidity, p_H 3·2 or higher, results in cracking, treeing, and pitting, due probably to high excess of hydrogen at the cathode. The current efficiency is also greatly reduced, the voltage rising with the acidity, and the resultant iron is extremely brittle because of the high hydrogen content. Iron content of the electrolyte does not appear to be a critical factor, good deposits being obtained from concentrations varying from 150 to 220 grammes of iron per litre. An anolyte overflow analysing half ferric and half ferrous would seem the best practice. If the ferric is allowed to build up beyond this, excessive chlorination results which means loss of ferric chloride. A hydrostatic head must also be maintained in the cathode compartment to prevent ferric chloride seeping through into the catholyte. A near neutral electrolyte, p_H 4·0 to 4·8 tends to hydrolyse readily, a considerable precipitate of iron oxychloride being formed. The presence of oxychloride does not adversely affect the nature of the deposition, in fact, it probably acts as a depolarizer, but it impregnates the diaphragm, setting up a resistance and also results in a loss of iron in the solution.

Summing up, the best operating conditions appear to be: temperature 80-90°C., acidity p_H 3·7 to 3·9; iron content 170 to 185 grammes per litre; absence of ferric in catholyte; and pure electrolyte. With these conditions and proper cell design, current (cathode) efficiencies averaging 94 per cent appear quite possible. The anode efficiency is always somewhat below this, being around 90 per cent, due to slight chlorination and other causes. The following data may be given as an example of a cell run:—

Cathode	Steel mandrel 12 x 3½ inches diameter, rolled surface.
Anodes	Graphite plates, 12 x 12 x ½ inch.
Electrolyte	$FeCl_2$—156 grms. iron per litre.
Feed rate	3·4 litres per hour.
Average temperature	85° C.
Time	5 hours.
Average C.D	106 amperes (124 amps. per sq. ft.).
Average volts	3·9.
Total ampere hours	529·8.
Weight of deposit	518 grammes.
Thickness of deposit	0·030 inch.
Cathode efficiency	93·8 per cent.
Anode liquor	61·2 grammes iron per litre.
Anode efficiency	90 per cent.
Deposit	Silver grey, perfectly smooth, satin surface, hard and brittle, smooth at edges.

The deposit was annealed at about 600°C. and removed from the mandrel as a perfect tube. Analysis was not made on this iron, but other samples from similar eletrolyte showed maximum impurities to be lead, copper and sulphur, each less than 0·01 per cent, with carbon 0·002 per cent and traces of phosphorus and zinc, with iron running 99·94 to 99·97 per cent.

The following briefly summarizes the results of 37 deposits run on electrolyte from pyrrhotite and pyrite:—

Excellent	Smooth and matt	10
Good	Slight treeing at edges	6
Fair	Slightly uneven and coarse	8
Poor	Treed edges and nodular	7
Very poor	Cracked and treed	6

MANDREL TREATMENT

The condition of the mandrel would seem to be an important point. Best results have been obtained with a machine-turned mandrel subsequently rolled to a smooth surface. Difficulty has been experienced in successfully stripping the deposits from the mandrel, and no definite procedure can be laid down in this regard. Six excellent deposits have been stripped as tubes, and several deposits have been stripped incompletely, or as plates. The problem would seem to be a mechanical one rather than one of deposition conditions. Coatings of various types have been tried, such as oils, grease, metal soaps, etc., but most satisfactory results have been obtained with an extremely thin coating of vaseline.

SUMMARY

The results obtained in this investigation would indicate that the hydrometallurgical treatment of pyrite and pyrrhotite is possible from an electrochemical standpoint. Sufficient data, however, have not been obtained to warrant commercial application, and a continuation of the investigation will be made in an effort to clear up some of the uncertain factors.

Report No. 238

A HYDROMETALLURGICAL TREATMENT FOR PYRRHOTITE OF LOW GOLD AND COPPER CONTENT, FROM NORANDA MINES, LIMITED, ROUYN TOWNSHIP, QUEBEC

R. J. Traill and W. R. McClelland

The following tests were carried out on a small shipment of pyrrhotite from the Noranda property with the object of recovering the iron sulphur, gold, and copper present in the ore. The ore consisted of a mixture of pyrrhotite and pyrite with low values in copper, gold, and zinc. The analysis of the ore was as follows:—

Total iron	50·3 per cent
Ferrous iron	33·18 "
Copper	1·20 "
Zinc	0·37 "
Gold	0·10 oz./ton

Apparently about one-third of the iron content of the ore is present as pyrite which is practically insoluble in ferric chloride solution, and for favourable recovery of iron it would be necessary at some stage in the process to convert this pyrite to artificial pyrrhotite. Two possible methods of treatment were tried, both giving about the same results in iron and copper extractions. The results obtained although not entirely satisfactory are interesting and encouraging from a hydrometallurgical viewpoint and warrant further investigation. Sufficient ore was not available at the time to continue the tests. The following is a brief summary of the methods of treatment and the results obtained.

PRELIMINARY TEST—DIRECT LEACHING METHOD

The ore ground to pass 200 mesh was leached with a hot ferric-ferrous chloride solution. The volume of leaching liquor was 4 litres made up of 70·2 grammes per litre ferric iron and 72·8 grammes per litre ferrous iron. The charge to this leach was calculated on the theoretical requirement based on the soluble (ferrous) iron in the ore, plus a 25 per cent excess, and amounted to 528 grammes ore. When the leach liquor was completely reduced the residue was separated by filtration, dried, and weighed. The residue weighed 378 grammes and contained 14·56 per cent soluble iron. This residue was submitted to a non-oxidizing roast using an iron pipe retort at 750°C, the elemental sulphur resulting from the leaching and one atom of sulphur from the pyrite being distilled off. The roasted product weighed 241 grammes and analysed 54·6 per cent soluble iron, showing that the pyrite had been changed from an insoluble to a soluble artificial pyrrhotite. This product was then submitted to a further leaching with ferric-ferrous chloride and a residue obtained which weighed 134 grammes and analysed 17 per cent total iron. Distillation of the elemental sulphur from this residue resulted in a product weighing 73 grammes, of the following analysis:—

Total iron	31·40 per cent
Ferrous iron	16·40 "
Copper	1·16 "
Gold	0·78 oz./ton

It will be noted here that the gold value has been concentrated approximately 1 : 8. The presence of 1·16 per cent copper in this final residue may preclude the practicability of direct treatment of the residue for gold recovery by cyanidation. The high iron and copper content is probably due to the presence of unaltered chalcopyrite and pyrite and a further non-oxidizing roast followed by leaching would probably result in a residue that could be economically treated by cyanidation for gold, or the residues could be sent to a smelter for treatment for recovery of gold and copper. The total extractions from the ore obtained by this method show:—

Iron extraction	91·3 per cent
Copper extraction	86·6 "

DIRECT ROASTING METHOD

In an effort to shorten the process direct roasting of the ore was attempted, with the expectation that the pyrite would be altered to artificial pyrrhotite and that the natural pyrrhotite would remain unaltered. This would seem a feasible theory, but results proved unsatisfactory in that the iron extractions were very low, due it is thought to an alteration of some nature in the pyrrhotite by roasting causing excessive hydrolysis in the leaching. This oxychloride of iron formed by the hydrolysis slowed up the reaction between the ore and the ferric chloride quite seriously, and it was necessary to add a greater excess of ore to bring about complete reduction of the liquor. This resulted in lower percentage extraction and high iron content residues necessitating further roasting and leaching to obtain a favourable recovery of iron or copper. The final result was somewhat similar to that obtained by direct leaching, the residue having a high iron and copper content, and the gold concentrated about 1 : 8. This test was carried out on a larger scale, using the roasting furnace leaching and cementation equipment as described in the report on pyrrhotite treatments, and finally depositing the iron in a diaphragm cell. The following is a brief summary of the results obtained: 13 kilograms (approx. 30 pounds) of the ore was ground to 10 mesh, fed to the rotary furnace and roasted under non-oxidizing conditions at 750°C. One pass through the furnace was made resulting in a product weighing 12·24 kilograms, and analysing:—

Total iron	52·42 per cent
Ferrous iron	48·96 "
Copper	1·28 "

A small amount of sulphur was recovered, but most of the sulphur set free was oxidized in the furnace to SO_2. The roasted product was leached in 3 lots with hot ferric-ferrous chloride solution, each leach having a volume of 50 litres. The leaching action, as already stated, was slow due to excessive hydrolysis. The residue from leaching weighed 7·9 kilograms and the extraction of iron and copper amounted to:—

Iron extraction	59·7 per cent
Copper extraction	64·4 "

The leach residue was subjected to distillation, resulting in the recovery of 1·8 kilograms of sulphur. The residue from distillation weighing 5·3 kilograms and containing approximately 50 per cent total iron, was given a secondary roast to convert any unaltered pyrite to pyrrhotite, and a product weighing 4·78 kilograms was obtained.

Total iron	54·6 per cent
Ferrous iron	53·8 "
Copper	1·13 "

One hundred and fifty-two grammes of sulphur was recovered. The product from the second roast was leached with a ferric-ferrous chloride solution and a residue weighing 2·74 kilograms obtained, which on being subjected to a sulphur distillation roast yielded a final residue of 1·6 kilograms.

Total iron	43·2 per cent
Ferrous iron	38·1 "
Copper	1·14 "
Sulphur	28·5 "
Gold	0·76 oz./ton

The residue obtained, 1·6 kilograms, would show a concentration by weight of 8·15 : 1. The gold content in the head sample amounted to 0·0015 ounce, and in the final product 0·0013 ounce, showing a slight loss which may have occurred in handling. A summary of these results showed:—

Total iron extraction	89·5 per cent
Total copper extraction	88·6 "
Sulphur recovered, 2·7 kilograms	60·4 "

PURIFICATION OF ELECTROLYTE

The leach liquors from the primary leaching of the direct roasted product analysed about 163 grammes per litre ferrous iron, about 0·6 gramme per litre copper and 0·4 gramme per litre zinc, and the leach liquor from the secondary leaching analysed 145 grammes per litre ferrous iron, 0·7 gramme per litre copper and less than 0·2 gramme per litre zinc. In the purification of this liquor an attempt was made to separate the copper by the addition of sponge iron. This, however, was only partly successful, the last traces of copper being difficult to remove, and recourse was had to the use of calcium sulphide to remove the remaining copper. This procedure resulted in an electrolyte of sufficient purity for electro-deposition of iron, the only apparent impurity being zinc, which was present to the extent of about 0·35 gramme per litre.

ELECTRO-DEPOSITION OF IRON

Electro-deposition was carried out in the diaphragm type cell, the cathode being a steel mandrel of approximately 1 square foot area, the anodes being graphite plates. An asbestos cloth diaphragm separated the anolyte from the catholyte. A description of the cell and its operation is outlined in the report on pyrrhotite and pyrite. The following summary of cell operation data serves as a typical example of the tests on this electrolyte:—

Electrolyte—$FeCl_2$	145 grammes per litre Fe.
Rate of electrolyte feed	3·8 litres per hour.
Time	5 hours.
Average current density	103·7 amperes per sq. ft.
Average voltage	3·7.
Average temperature	86° C.
Weight of deposit	490 grammes.
Thickness of deposit	1/32 inch (approx.).
Speed of cathode rotation	274 r.p.m.
Anode overflow, Ferrous	76 grammes per litre.
Ferric	60 grammes per litre.
Current efficiency	90·8 per cent.
Nature of deposit	Silver-grey in colour, surface smooth with a very few small nodules, very finely crystalline, coherent and of uniform thickness. After annealing at 400 to 600 ° C. to remove hydrogen, the deposit was removed from mandrel by rolls as a tube, perfect in shape.

Recovery of Gold in Residue

The residues obtained from leaching were submitted to cyanidation. J. S. Godard carried out the tests and reported results as follows:—

Residue grms.	Solution c.c.	KCN per cent	Assay head	Tailing oz./ton	Extraction per cent	KCN consumed, lb./ton	Time hours
606	2360	0·2	0·76	0·185	75·6	32·7	50

Grinding.—Residues ground wet in a ball mill using grey iron balls. Intermittent screening through 200 mesh. Oversize returned to mill.

Remarks.—Cyanidation in a more dense pulp 1:2·5 using 0·075 per cent cyanide and increasing the time might increase the extraction with a considerable decrease in cyanide consumption. The residues contained 1·14 per cent copper and with a more complete removal of the copper the gold extraction should be better, and the consumption of cyanide much lower. With a smelter in close proximity, cyanidation of the residues would not be necessary, as this product contains sufficient values in gold and copper to be shipped to a smelter for recovery of the values.

SUMMARY

Two methods of treatment are described namely (a) directly leaching the pyrrhotite content, and with subsequent roasting of the pyrite; and (b) directly roasting the mixed sulphides thereby converting the pyrite to artificial pyrrhotite. The latter method proved less satisfactory than anticipated, a new problem arising in the roasting effect on pyrrhotite, necessitating further roasting treatment to secure satisfactory iron recovery. Both methods give practically the same results.

Results show extractions of iron and copper amounting to 89·5 per cent and 88·6 per cent respectively, with 60·4 per cent recovery of sulphur under more or less adverse conditions.

Concentration of gold values in the residue in the ratio of 1:8 results from either method of treatment.

A pure electrolyte can be obtained, suitable for recovery of iron as electrolytic iron.

Report No. 239

A NEW PROCESS FOR THE TREATMENT OF ILMENITE FOR THE RECOVERY OF ELECTROLYTIC IRON AND TITANIUM OXIDE CONCENTRATE FOR PIGMENT AND OTHER PURPOSES

R. J. Traill and W. R. McClelland

Microphotographs by E. A. Thompson

Introductory

Canada has, in the province of Quebec, deposits of ilmenite which compare favourably as to size and grade with any of the known deposits in other countries. These deposits are favourably situated with respect to transportation, to the use of cheap hydro-electric power for their reduction, and to general markets for the disposal of commercial products obtained from them. No industry has been established in Quebec for the utilization of these ores. Production has been confined to shipments of ore to United

States for the manufacture of ferro-titanium and titanium pigment. The latter is a recent development, the result of Norwegian chemical research, and it promises to be a most valuable one, and should develop into an important commercial industry in Canada. The special qualities of titanium white pigment may be briefly summarized as follows:—

1. Greater covering power than zinc oxide or white lead.
2. Chemically inactive with vehicles or other pigments.
3. Not affected by acid fumes or sulphide gases.
4. Remains white under any atmospheric conditions.
5. Is non-toxic.

Possessing such good qualities, it is highly probable that this pigment will replace white lead or zinc oxide to a great extent.

In the present practice of pigment manufacture from ilmenite ores the titanium content alone is recovered, the iron content averaging 30 to 40 per cent going to waste. A process that would recover both the iron and titanium content would therefore be advantageous as it would result in a fuller utilization of the valuable constituents of the ilmenite ores of the country.

A process having this end in view has been investigated in the electrochemical laboratory and the results obtained have been very encouraging and warrant further investigation on a semi-commercial or pilot-plant scale to check and corroborate the results, and to determine cost data. A brief summary of the process follows together with observations respecting important details of operation which should be of material assistance in following these tests up on a somewhat larger scale. The products of this process are electrolytic iron and titanium oxide concentrate suitable for pigment-making purposes. There is a considerable market for electrolytic iron, and its physical properties are now sufficiently well known to assure a steady and increasing demand.

The Process

The ilmenite is reduced by means of carbon in the form of coal, coke, or charcoal, in a furnace in which the temperature and atmosphere can be controlled. The iron oxide is converted into spongy metallic iron, but the titanium oxide is chemically unchanged. The product is passed over a magnetic separator to remove unburnt carbon and dissociated gangue material. The magnetic product, consisting almost entirely of metallic iron in the form of sponge, and titanium oxide is then treated with ferric chloride solution. The iron is dissolved, forming ferrous chloride, leaving an insoluble residue of impure titanium oxide which is separated by filtration or other means, and subsequently treated by an acid or acid sulphate process for recovery of pure oxide of titanium. The ferrous chloride liquor obtained after separation of the insoluble residue may contain impurities, and it is purified at this stage by a precipitation process which eliminates the undesirable elements. Electro-deposition of iron is carried out in a diaphragm cell, the iron being deposited on a rotating, steel-mandrel cathode, or steel plates, and with anodes made up of graphite. With the deposition of iron in the catholyte compartment, oxidation in the anolyte compartment takes place forming ferric chloride, which is returned to the leaching system for action on fresh sponge. The mandrel with the deposit of iron is then given a pre-annealing treatment to drive

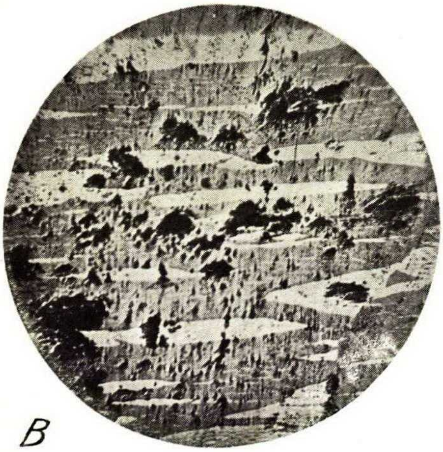
Plate III

A. Polished specimen of Ivry ilmenite ore. The ore is dense black, medium even-grained, granular ilmenite with some hematite, carrying scattered particles of iron pyrite, occasional flakes of black mica, a few grains of smoky quartz, and a little feldspar. It is feebly magnetic, and has a reddish black streak. No magnification.

B. Polished specimen, unetched. Light streaks are hematite in the darker ilmenite. Black spots are gangue. The lamellae of the ore have been displaced by minute faults. Magnification, 260 diameters.

C. Polished specimen, etched with strong hydrochloric acid. Shows three grains in contact differentially orientated. The dark streaks are hematite, etched out, and the light, ilmenite. The dark spots are gangue. Magnification, 260 diameters.

A



B

C

out the hydrogen, and the deposit is removed from the mandrel, by suitable rolling device, in the form of a tube. The tube may be further annealed to suit cold rolling work.

EXPERIMENTAL WORK

Sponge Making

Ore.—The ore used was ilmenite from Ivry, Que., on the Mont Laurier branch of the Canadian Pacific railway. It was submitted by Messrs. McArthur, Irwin, Limited, of Montreal, and contained as shown by analysis:—

Insoluble	7·90 per cent
TiO_2	30·80 "
Fe	40·70 "

Furnace.—In the making of sponge iron the ideal practice would be one embracing continuous operation, and a rotary furnace system would probably best meet this requirement. Although a small oil-fired rotary furnace was available, tests conducted some years ago on metallization of hematite ore in this furnace gave results which were unsatisfactory on account of the difficulty of maintaining a reducing atmosphere, and therefore, recourse was had to a brass-melting, pot furnace, gas-fired, for the making of the sponge iron. The United States Bureau of Mines, at their Seattle station, have developed a rotary kiln furnace, oil-fired, which seems to have given good results on the metallization of iron oxide ores, hematites and magnetites. The furnace with which they conducted their experiments had a capacity of 1,400 pounds of sponge iron per day, and proved to be quite satisfactory for continuous operation. Messrs. Thornhill and Anderson have developed a furnace of the revolving hearth, muffle type for the metallization of iron oxide ores.

Mix, Ore, Temperature, and Time.—The charge to the fireclay pots consisted of three parts ilmenite, 60 to 80 mesh, to one part charcoal, 10 to 20 mesh, by weight, thoroughly mixed. The pots, holding 32 pounds mix, with a slight covering of charcoal were placed in the furnace at a temperature of about 900 to 1,000° C. The heat penetration through such a mixture is necessarily slow, requiring five to six hours to reach centre. When the charge reached about 950° C. the temperature was maintained for about 1½ hours. The pots were allowed to cool in the furnace overnight and the product passed through a 10-mesh screen preparatory to magnetic separation which was made on a Wetherill magnetic separator.

Analysis Sponge Products.—An average test shows analysis of products as follows:—

	Total iron	Metallic iron	TiO_2	C
Magnetic	46·7	40·5	36·9	2·0
Non-magnetic	12·8	2·7	9·3	37·9

Metallization	83·2 per cent
Yield metallic iron	80·3 "

With properly designed furnace the yield and metallization of iron can probably be improved. The most important fact determined in these tests is the necessity for careful control of temperature. At temperatures above 1,000° C. sintering of the mass, or fusion, results, and tests would indicate that such a product is not readily soluble in ferric chloride liquor. Using a rotary furnace, oil-fired, it is probable that the proportion of coal to ore will need to be higher, say 75 : 100, to ensure a completely reducing atmosphere. Much of the excess should, however, be recovered in the magnetic separation, and used in subsequent mixes.

Sponge Leaching

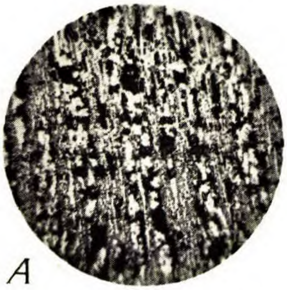
Scale of Tests.—Initial leaching tests were carried out in enamel pails of 6 litres capacity, fitted with agitating devices, the volume of leaching liquor used being 4 litres. Larger leaching tests (50 litres) were carried out in 15-gallon stoneware tanks, similar in design to nitrating kettles, and fitted with stoneware agitators. The leaching liquor used was a mixture of ferrous and ferric chlorides, made up from ferric chloride salt partly reduced with iron, or from liquor obtained as anolyte overflow from the electro-deposition cell. The charge of sponge for leaching was dependent upon the amount of ferric iron in the leaching liquor, sufficient being added to cause complete reduction of all iron to the ferrous state. The charge is calculated from the equation, $2Fe\ Cl_3 + Fe = 3Fe\ Cl_2$.

Temperature of Leaching.—Leaching was carried out with both cold and hot leaching liquor to determine the difference, if any, in time and extraction factors. Leaching at a temperature between 90 and 100° C. results in a quicker reaction between the sponge and liquor, with a higher extraction of metallic iron, recoveries running 98 to 99 per cent. The time factor is reduced by more than one-half. The titanium content is, however, somewhat higher than obtains in cold leaching (0·0375 gramme per litre as against 0·015 gramme per litre in cold leaching). Cold leaching should average about 92 to 95 per cent extraction. It may be stated here that the reaction $Fe + 2Fe\ Cl_3$ is exothermic and that a temperature of about 50° C. would be reached starting with initial temperature of 20° C. It would be difficult to say which treatment causes the greater hydrolysis of ferric chloride, and whether anything is gained by hot leaching.

Filtration.—Suction filtering was used in separating insoluble residue from leach liquor, but proved somewhat slow, especially as filter cake thickened. A combination of settling or thickening and filtering would probably be more satisfactory.

Impurities.—The Ivry ore contained small amounts of copper, lead, nickel, and cobalt, and these elements were found in the leach liquor. As they exert a harmful effect on the electro-deposition of iron they were precipitated as the sulphides by the use of calcium sulphide CaS, in the presence of a small amount of acid removed by filtering and subsequently recovered by known processes.

Selective Leaching and Purification.—It is possible, however, to partly overcome this step by using an excess of sponge material in leaching which would prevent the copper and lead, which are the most harmful from the deposition standpoint, from going into solution. The liquor or electrolyte would then contain only small amounts of nickel and cobalt which would co-deposit with the iron without seriously affecting its general properties. Such a method would of course necessitate further treatment of the residue by excess ferric chloride solution to completely remove the excess of iron and free the copper and lead from the titanium oxide. This



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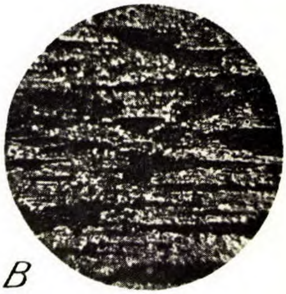

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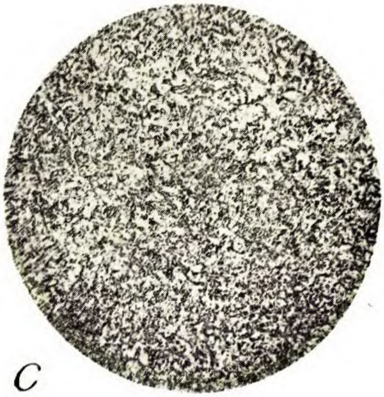

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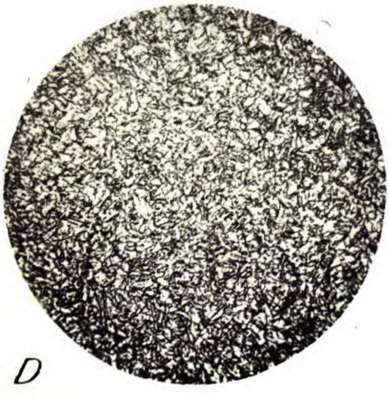

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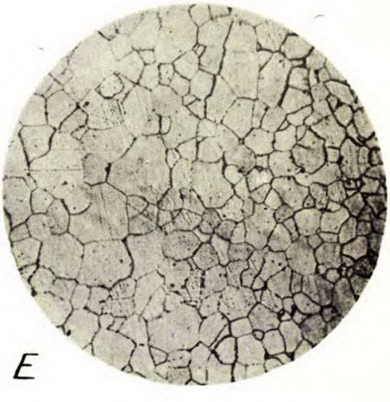

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leach liquor could then be treated for recovery of copper and lead which could be built up through successive residue treatments and recovered more readily in this higher concentration, by cementation or other method.

Electro-deposition

Cell.—The type of cell used in these experiments was a diaphragm cell with an electrolyte capacity of about 12 litres, having a steel mandrel of one square foot plating area as cathode, and two graphite anode plates. The catholyte and anolyte cells are of almost equal capacity and separated by an asbestos diaphragm of U-shape, the cathode being inside the diaphragm and the anodes being placed down the sides and across the bottom close to the diaphragm. Outlets on the anolyte sides of the cell were provided for drawing off the regenerated ferric chloride, and an outlet from the catholyte was arranged to regulate the hydrostatic head necessary for proper operating conditions.

Electrolyte Feed.—The electrolyte was fed into the catholyte compartment at a regulated rate which averaged 4 litres per hour. Satisfactory deposition can only be made at temperatures between 80 and 95° C. This necessitates pre-heating the electrolyte before going to the cell, or arranging a heating system in the cell itself to maintain the temperature required. The speed of the rotating mandrel should be somewhere between 250 and 400 r.p.m.

Preparation of Mandrel.—The preparation of the mandrel is highly important and although various schemes have been tried out, no single method has proved satisfactory with regard to successful stripping of the deposit. The removal of the deposit is of course strictly a mechanical problem and with proper conditions and apparatus should be readily accomplished. The obtaining of good smooth deposits, however, depends upon a smooth surfaced mandrel. Coatings of various types have been tried but vaseline seems to be the most satisfactory. Smoothed mandrels having iron rust coatings have also given good results.

Current Density and Voltage.—Current densities around 100 amperes per square foot have proved quite satisfactory. The voltage depends on the resistance set up by the electrolyte, the diaphragm, and the distance between electrodes. In the design of cell used in this laboratory an average of 4 volts has been obtained at the above current density.

Effect of Impurities Nickel, Cobalt, Copper, Lead.—The presence of nickel, copper, or cobalt in the electrolyte in quantities less than 0·02 gramme per litre has no apparent effect on the deposit, larger quantities tend to cause cracking or stripping of the deposit. The presence of lead in quantities below 0·002 gramme per litre has no ill effect, but quantities above this cause spongy deposition, pitting and treeing, according to the amount present.

Effect of Low Temperature.—Deposition at 70°C. or lower is attended with nodular formations and generally uneven deposits. The voltage required is also higher, resulting in reduced current efficiency.

Acidity.—The electrolyte should be near neutral $p\text{H}$ 3·7 to 3·9 giving best results.

Nature of Deposits.—Deposits should be silver-grey in colour, smooth, and microscopically non-crystalline. Best conditions are pure electrolyte, temperature 85 to 95° C. and careful regulation of C.D., and also acidity not greater than p_H 3·7.

Occluded Hydrogen.—The iron as deposited contains hydrogen, thereby making it extremely brittle. This hydrogen is readily removed by slightly annealing at 300° C. upwards.

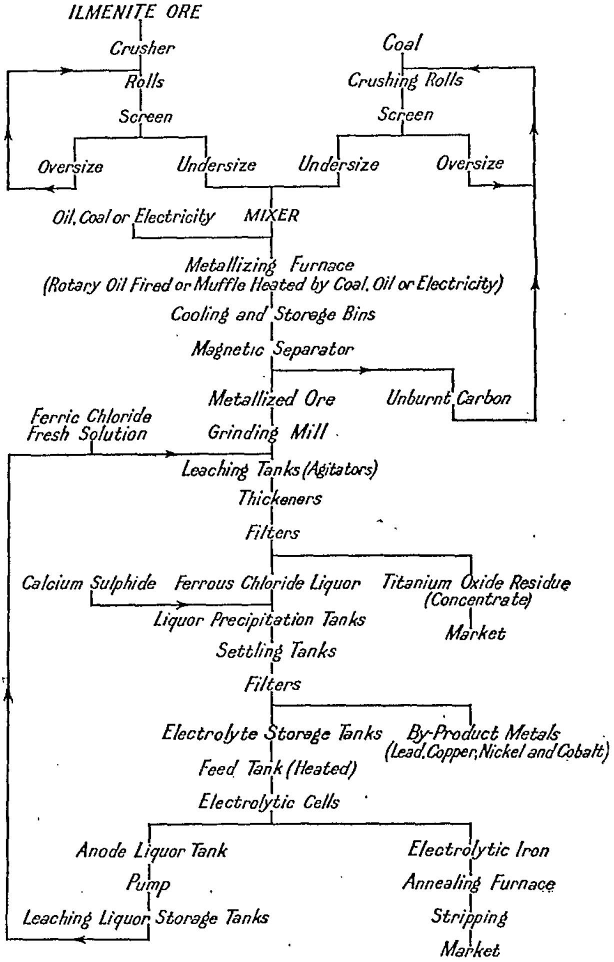

Figure 3. Flow-sheet of process for the production of electrolytic iron and titanium oxide concentrates from Canadian ilmenite ores.

Test Example

Sponge.—The following is submitted as an example of a complete test.

Leaching.—Charge to pot for sponge making consisted of:

Ivry ilmenite	3 parts.
Charcoal	1 part.
Metallization temperature	990° C.
Time	1½ hours.
Product 20-lbs—Magnetic	16¼ lb.
Non-magnetic	2⅝ “
Loss	1⅛ “ (sinter dust, etc.)
Metallic iron in magnetic product	35·7 per cent.
Yield metallic iron	79·5 “
Metallization	82·0 “
Liquor—$FeCl_2$	60·5 grammes Fe per litre.
$FeCl_3$	71·5 “ “
Volume	50·0 litres.
Sponge iron required to reduce $FeCl_3$	71·5 x 50 ÷ 2 = 1,787·5 grammes.
Sponge material added	5,000 grammes = 1,785 grammes iron.
Feed rate	Complete in 10 minutes.
Time of leaching	1¼ hours.
Temperature	21° C. start, rising to 47°—exothermic reaction.
Iron dissolved	1,713·8 grammes (calculated from reactions).
Extraction of iron	96 per cent.
Liquor strength	167 grammes per litre.
Weight of residue	3,415·5 grammes.
Analysis of residue—	
Total iron	16·13 per cent.
Metallic iron	0·25 “
Titanium oxide	53·62 “

NOTE.—Weight of residue 3,415·5 grammes plus calculated weight of iron dissolved, 1,713·8 grammes, totals 5,129·3 against 5,000 grammes feed. This increase in weight may be accounted for as due to hydrolysis of solution, and also to oxidation of undissolved iron in residue during drying.

Assuming this theory to be correct, calculation shows that hydrolysis occurred to the extent of 1 per cent iron content of leaching liquor.

Purification of Electrolyte.—The electrolyte obtained contained copper, lead, nickel, and cobalt, 100 grammes calcium sulphide was added to the cold liquor and after 10 minutes' agitation the precipitate was allowed to settle, and the solution was filtered. The filtered liquor showed a very slight trace of unprecipitated sulphides. The excess of hydrogen sulphide in the liquor was destroyed by aeration. The black precipitate contained: lead, 8·9 per cent; copper, 2·2 per cent; nickel, 1·9 per cent; cobalt, 2·2 per cent; the remainder being iron and sulphur. The liquor strength was 164·2 grammes per litre.

Iron Deposition.—

Cathode	Rotary mandrel, steel, approximately 1 square foot area —speed 350 r.p.m.
Anodes	Graphite plates 12 x 12 x ½ inch.
Liquor	164·2 grammes ferrous per litre.
Time	6 hours.
Current density	Max. 125, min. 100, amperes per sq. ft.
Average density	About 120 amperes per sq. ft.
Total ampere hours	714·1.
Voltage	Max. 4·4, min. 4·2 volts.
Average	4·3 volts.
Temperature	Max. 90° C., min. 80° C. Average, 85° C.
Liquor feed	22½ litres, or 3·75 litres per hour.
Cathode liquor overflow	4¼ litres.
Anode liquor overflow	17¾ litres, approximately 75 grammes per litre ferrous iron.
Weight of deposit	702 grammes.
Thickness	0·043 inch.
Current efficiency	94·3 per cent.
Anode efficiency	92·4 per cent.
Nature of deposit	Silver-white, finely crystalline, smooth, with a few slight knobs, coherent.

Stripping.—

The deposit was annealed at a low temperature preparatory to stripping. Stripping was attempted in lathe using a roller, but, as deposit loosened only in places, it had to be split and removed as sheet.

Analysis of Deposit.—

		Per cent	Quantity used for determination
Carbon	less than	0·001	2 grammes
Manganese	"	0·005	5 "
Phosphorus	"	0·006	10 "
Sulphur	"	0·007	10 "
Silicon	"	0·006	10 "
Copper	"	0·0024	100 "
Cobalt		0·015	100 "
Nickel		nil	100 "
Chromium		nil	25 "
Titanium		nil	25 "
Total impurities		0·0424	
Iron, approximate		99·96	

Cold Rolling Test.—A piece of the sheet, 0·5 inch wide, 1·0 inch long and 0·043 inch thick, was cold rolled without further annealing to a length of 13 inches and width of 0·64 inch, thickness 0·003 inch, showing:—

Increase in length	1,200 per cent
Reduced area	91 "
Reduced thickness	93 "

The rolled strip was quite sound and showed no cracking or holes. Another sample rolled to 0·002 inch thickness was also found to be sound and free from cracks and holes.

Summary of Test.—

1. A sponge was made from ilmenite ore and charcoal with a metallic iron yield of 79·5 per cent.

2. Leaching this sponge results in a 96 per cent extraction of the iron and the formation of a concentrate of titanium oxide (TiO_2) assaying 53·62 per cent.

3. Small amounts of other valuable metals, namely copper, nickel, cobalt, and lead, were separated in a concentrated form for recovery treatment.

4. Electro-deposition of iron carried out with a current efficiency of 94·3 per cent and a good deposit of iron 99·96 per cent pure.

IV

REPORT OF INVESTIGATIONS: PYROMETALLURGICAL LABORATORY

A COMBINED PYRO AND HYDROMETALLURGICAL PROCESS FOR THE TREATMENT OF NICKELIFEROUS PYRRHOTITE AND OTHER SULPHIDE ORES FOR THE RECOVERY OF IRON AND SULPHUR IN ADDITION TO THE OTHER CONTAINED METALS

H. C. Mabee and A. E. Smaill

The small-scale laboratory experiments which were begun in 1924 and described in the Report of Investigations for 1924, page 104, were continued until it was clearly indicated that it would be advisable to enlarge the scale of experiments. The small-scale experiments showed that the most promising procedure after the production of the self-disintegrating matte would be roasting, chloridizing, and leaching. The scale of the experiments was, therefore, enlarged along these lines. This necessitated the construction of a specially-designed chloridizing furnace and the installation of wooden, leaching and filter tanks.

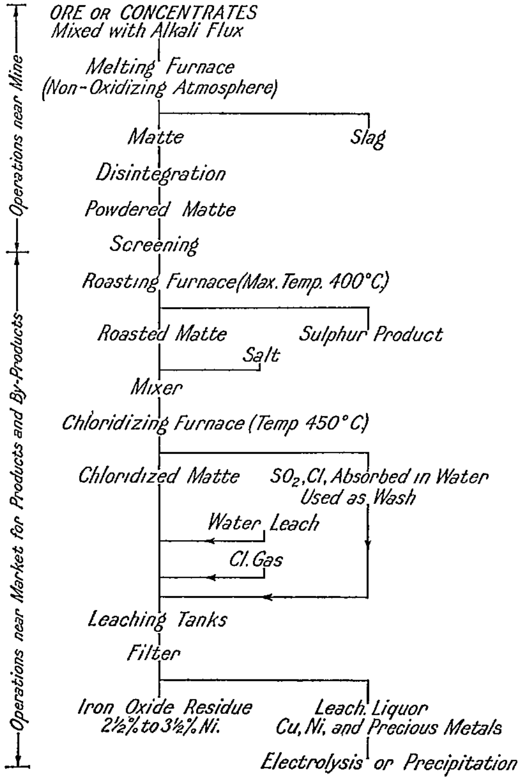

Figure 4. Flow-sheet of process for the recovery of iron and other contained metals from nickeliferous pyrrhotite and heavy sulphide ores.

The flow-sheet, Figure 4, indicates the procedure followed in these tests. A low-grade matte containing practically all the sulphur and metal values of the ore is first produced. The matte after disintegration is roasted, chloridized and then leached, leaving a residue of the matte in the form of a finely divided oxide. It will be seen that the process is simple and direct, and that marketable products should be obtained in very short actual working time. The losses are very low and the recoveries can be made without the destructive effects on agriculture that occur under present methods. The fuel requirements should be low and where electric power is available it can be employed for the entire process, including smelting.

Matte.—When a sulphide ore is melted with an alkali flux under non-oxidizing conditions, a low-grade self-disintegrating matte containing the sulphur and metal values is produced, the slag carrying the non-metallic gangue and other impurities. The matte for these experiments was made by melting the ore in 10-kilogram lots, in a gas-fired crucible. When cool the slag separated cleanly from the matte by fracture. The concentration in the matte produced is as follows:—

Ore head	Per cent of ore in matte	Copper		Nickel		Iron	
		Head	Matte	Head	Matte	Head	Matte
1. Shebandowan	31·5	1·65	5·05	2·97	9·00	24·30	....
2. " (combined)	65·5	4·02	7·55	3·82	6·92	31·7	....
3. "	67·8	4·78	6·89	4·3	5·57	31·7	42·6
4. Frood	34·4	1·45	4·00	1·35	3·8	18·79	56·56
5. Murray (tabled)	68·0	0·92	1·35	1·92	2·95	31·18	51·4
6. "	62·8	0·77	1·02	2·16	2·74	36·42	43·83

The ratios show that this concentration is practically one of elimination of the gangue only:—

Ore	Copper to nickel		Copper and nickel to iron	
	Ore	Matte	Ore	Matte
1. Shebandowan	1 : 1·8	1 : 1·78		
2. "	1 : 0·95	1 : 0·92		
3. "	1 : 0·9	1 : 0·81	1 : 3·49	1 : 3·42
4. Frood	1 : 0·93	1 : 0·95	1 : 6·7	1 : 7·0
5. Murray	1 : 2·08	1 : 2·18	1 : 11·2	1 : 11·9
6. "	1 : 2·8	1 : 2·7	1 : 12·42	1 : 11·6

On exposure the matte readily disintegrates into a finely divided state; about 50 per cent is less than 200 mesh and only about 2 per cent will not pass an 80-mesh screen. There is no segregation of values. The moisture content is less than 3 per cent. An attempt was made to concentrate the nickel and copper values by tabling and magnetic separations but the results obtained were found to be of little practical value. Flotation tests on fresh matte showed a certain amount of concentration but as this method would hardly prove practical the tests were not followed up.

Roasting.—The fine subdivision and the low ignition point of the matte suggest the possibility of reducing the sulphur by combustion in the same manner in which powdered coal is now used. This has not been followed up on a large scale although small-scale experiments were promising and would probably give very uniform results without the use of mechanical stirrers, etc.

The roasting experiments were carried on in a large 4-tier muffle furnace, the muffles being so arranged that the material can be raked from the top muffle down into the next below and so on. The matte was charged in 10-kilogram lots and was stirred at regular intervals. The temperature was kept at about 400°C. until the bottom muffle was reached, where the temperature was not allowed to exceed 600°C. The total time of any batch in the furnace was approximately 4 hours.

Chloridizing Roast.—The roasted material was next given a chloridizing roast to render the copper and nickel more soluble, and further reduce the sulphur. The best experimental results were obtained by roasting in thick beds with moistened matte. To obtain these conditions on a large scale a special chloridizing furnace was designed and constructed. It consists of a vertical cylinder of conical section made of burnt fireclay mixture. The principal internal dimensions of the cylinder are, top 4 inches diameter, bottom 7 inches diameter, height 3 feet. The cylinder is externally electrically heated by means of heavy resistance wires wound vertically, the current being controlled by a suitably tapped transformer. The heating elements are surrounded by a thick heat insulation made of a porous fireclay mixture, burnt in place, with a kieselguhr backing and held in place by a covering of sheet iron. The cylinder is open at the bottom but the top is closed by a cover with openings through which thermocouples can be inserted. Outlet holes in the cylinder connected to suction are provided just below the cover, by means of which the evolved gases are drawn through a water-cooled section, or condenser, fitted with a trap at the bottom, then through two 20-litre wash bottles before going to waste. The liquor caught in the trap is largely hydrochloric acid carrying some iron, nickel, and copper, some of which if not all is due to matte particles being drawn in and dissolved. The absorption of the sulphur gases by the wash bottles is improved by the cooling in the condenser section of the suction system.

Charge.—The total amount in the furnace when fully charged was approximately 15 kilograms. The chloridizing operation was continuous, but at the same time intermittent in that following a pilot charge of 5 kilograms, held up in place with a charge of quartz, space was made for a further addition of a 5-kilogram batch by raking out sufficient at the bottom, the whole column dropping down. Each batch consisted of 5 kilograms of roasted matte well mixed with about 10 per cent of sodium chloride and dampened with sufficient water to cause the material to retain its form when moulded by hand. A thermo-couple was inserted well into the fresh charge and as soon as the temperature rose to about 400°C., raking out at the bottom was started. The time for a batch to heat to this temperature varied with the condition of the furnace, but when in good running condition, was approximately one hour. The water in the wash bottles when saturated with sulphur vapours was found to be a very effective leach for the copper.

Leaching.—The leaching was accomplished in an elevated cypress wood tank, 3 feet in diameter, equipped with mechanical stirrers, and a means for injecting a small quantity of steam when heating was desired. From the leach tank the batch was washed on to a filter bed equipped with air suction. The leach liquors finally reached a portable cypress wood storage tank which could also be used for electrolyzing.

The total amount of chloridized matte leached in the first trial was 77·7 kilograms. This was leached with absorption water from the wash bottles of a previous chloridizing roast followed by a 10 per cent sulphuric acid wash. The resulting dried residues weighed 56·2 kilograms, the analysis of which showed:—

Iron	64·4	per cent
Nickel	3·3	"
Copper	0·36	"
Sulphur	0·21	"

The nickel to iron ratio is 1:19·3. This would make a nickel-steel of 5·17 per cent. The nickel-copper to iron ratio is 1 : 17·45, which would make a steel carrying 5·72 per cent copper-nickel.

The residues are finely divided oxides and a screen analysis shows:—

Mesh	Per cent	
−200	48·6	78·9
−100+200	30·3	
− 65+100	11·1	21·1
+ 65	10·0	

The leach liquors carried the following values in grammes per litre:—

	Copper	Nickel	Iron
1. Water leach	9·10	2·28	0·14
2. Acid wash	4·85	1·02	1·2
3. Water wash	1·72	0·6	0·8

The liquors were evaporated down for convenience in holding over for treatment.

The next trial on matte made from the same ore shows improved results. In this test the chloridizing furnace and the leaching tanks were in operation at the same time. Each batch as soon as withdrawn from the chloridizing furnace was immediately transferred to the leaching tank, thus utilizing the heat of the chloridized matte. Small-scale experiments having shown that the copper extraction was improved by using the saturated absorption water from the chloridizing unit, the leach was purposely confined to this, in order to try out this solvent.

The results of this test are as follows:—

Products	Weight in kilograms	Analysis in per cent			
		Copper	Nickel	Iron	Sulphur
Ore	130·0	0·77	2·16	36·42	20·18
Matte	81·7	1·02	2·74	43·83	26·21
Roasted matte	87·2	1·07	2·91	46·46	8·04
Leached residue	62·0	0·13	2·68	62·73	0·91
Percentage of ore in solution	....	92·0	41·0	....	....

The above percentages expressed in the form of ratios show clearly the absence of material loss in the several operations. The ratios are as follows:—

—	Nickel — Copper	Iron — Copper-nickel	Iron — Copper	Iron — Nickel
Ore	2·8	12·4	47·3	16·8
Matte	2·7	11·6	43·0	16·0
Roasted matte	2·7	11·6	43·4	15·9
Leached residue	20·6	22·3	485·0	23·5

It will be noted that the ratios are fairly constant until the leach is performed.

The ratio of the nickel to iron in the leached residue shows that steel carrying 4·27 per cent nickel may be produced or if the copper is taken into consideration the combined percentage is 4·47.

The residues show considerable improvement over the first test.

The proportion of copper to nickel is 1 : 20·6 which, apart from some trade prejudice is not now considered to be in any way objectionable. The sulphur could be taken care of in steel-making by slagging.

On the other hand the ultimate goal is the total elimination of the copper and sulphur with a higher extraction of the nickel.

Small-scale experiments show that improvements can be made and in the next trial, which is now underway, better results are anticipated.

The residues, used as a pigment, make an iron oxide paint of good covering quality.

Small-scale experiments show that the extraction of nickel can be improved by the use of brine as a solvent.

The leach solutions were combined in a storage tank and are being held for treatment. The wash waters were approximately measured, sampled, and then run to waste. In this trial, no attempt was made to build up the solutions. The analyses are:—

Solutions	Volume litres	Grammes per litre		
		Copper	Nickel	Iron
Combined leach	162	4·96	4·3	1·79
First wash	68	0·26	0·31	0·27
Second wash	71	0·012	0·03	0·08

Small-scale experiments were run on the combined leach solution and these tests indicated that the recovery of the copper and nickel, by electrolysis, will be attended with no great difficulty. It will be necessary, however, to make arrangements and carry on further trials on a scale which will permit possible trouble, not troublesome on a small scale, to develop and be overcome. The following experiments indicate a line that could be followed.

With a carbon anode and copper cathode, 98·2 per cent of the copper in solution was recovered, without attempting an end point, and this was followed by removal of the small amount of iron in solution; the cathode was then changed to nickel and 97·4 per cent of the nickel in solution was deposited. This experiment was repeated in several ways by changing the means for the removal of the small amount of iron, with equally favourable results. A small amount of chlorine is given off but in the above small tests no attempt was made to collect or make use of this. In a large-scale experiment this is one of the features that will be investigated.

Two batches of matte are now ready for treatment, one of 92 kilograms from Murray ore and the other of 25 kilograms from Noranda ore. Small-scale experiments are now being made with these mattes, with a view to taking care of points not attended to in the above tests, and at the same time attention will be given to the precious metal values as well.

It is the intention to combine all the residues and reduce them to steel. The resulting ingot will be subjected to the usual physical and chemical test to determine its qualities.

Conclusions.—The trials reported on indicate the results obtained with the new apparatus. A recovery of 92 per cent of the copper and 41 per cent of the nickel values of the ore has been made in solution, from which they may be easily extracted by electrolysis.

The residues in the form of finely divided oxide represent a recovery of over 96 per cent of the iron value of the ore combined with the remainder of the nickel. The residues are clean and may readily be reduced in an electric furnace to steel carrying about 4·25 per cent nickel. They may also find use in the production of sponge iron or as a pigment.

Experiments now under way have in view the improvement of the above results and the recovery of the precious metal values as well.

V

REPORT ON THE WORK OF THE CHEMICAL LABORATORY OF THE DIVISION

H. C. Mabee
Chemist-in-charge

During the year 1925, 1,109 samples of ores and mineral products were received and reports issued thereon. This involved about 4,850 chemical determinations. Practically all the samples received were directly connected with the experimental test and research work conducted by the officers of the Division in carrying out their investigations. The usual wide variety of ores and products was received and reported upon, as indicated in the following list:—

Aluminum	2	Lithium (mica)	2
Arsenical	16	Molybdenite	69
Bronze scrap	25	Nickel-copper	12
Calcite	8	Nickel-silver-cobalt	16
Copper	8	Platinum-gold	7
Copper-silver	19	Silver ore	17
Copper-zinc-silver	53	Silver-gold	124
Gold ore	117	Silver-lead-zinc	16
Gold-copper	154	Titaniferous iron	8
Lead (metallic)	2	Samples for identification and valuation	22
Lead-zinc-silver	412		

Although these ores represent a fairly wide classification it will be noticed that the greater number are lead, zinc, and copper, with associated minerals, including gold and silver. This is due to the active search for the ores of these metals, samples of which were submitted to the Ore Testing Laboratories for experimental tests.

The above list does not include the determinations made in the electrochemical laboratory in connexion with the electrolytic iron investigations. In this laboratory a considerable amount of analytical work was performed on iron sulphide and oxide ores, the products from these ores, and on control samples of the test operations.

During the year a technical officer of the National Research Council was given the use of the laboratories for their investigatory work on the beneficiation of Canadian magnesites. Every assistance was given in the way of laboratory space, supplies, etc. The chemical work of their officer is not included in this report.

No change occurred in the permanent staff of the laboratories during the year. B. P. Coyne, R. A. Rogers, and L. Lutes performed the regular routine work, and in addition gave assistance to the investigatory staff in the examination and assay of special samples of products from the test operations.

VI

THE CONCENTRATION OF CANADIAN MOLYBDENITE ORES

INTRODUCTORY

W. B. Timm and C. S. Parsons

Articles have appeared from time to time in the technical press descriptive of the methods applied to the concentration of molybdenite ores in various parts of the world. The chief occurrences in Canada, the history of early concentration methods, the uses, etc., of the metal molybdenum, are fully described by V. L. Eardley-Wilmot in his monograph "Molybdenum" Report No. 592, which has been published by the Mines Branch, Department of Mines, Ottawa.

The present article, prepared in memorandum form, will serve as a supplement to the portion dealing with the concentration of the ores, and covers the practical details of the concentration of the different types of Canadian molybdenite ores. It is intended to assist the mill operator in selecting the type of concentration plant best suited for his ore, and to aid the millman in the manipulation of his concentration so as to attain the best results.

Molybdenite, the sulphide of molybdenum, is the chief mineral of the metal, and is found widely scattered throughout Canada, from Manitoba east to the Atlantic seaboard; and from the Rocky mountains west to the Pacific ocean. During the war quite an industry was developed in Canada; a number of concentration plants were erected, and reduction plants were established at Belleville and Orillia; in 1918 the Canadian production of molybdenite reached a total of 378,029 pounds. The rapid decline in the market price at the close of the war was responsible for the closing down of all Canadian properties. However, the market is now steadily rising due to the depletion of war stocks and the progress made in metallurgical research in finding new uses for the metal, so that the future looks promising for the revival of the molybdenum industry in Canada.

CHARACTER AND TYPES OF MOLYBDENITE ORES

From the viewpoint of the operator and the millman who are concerned with the concentration of the ores, Canadian ores may be classified into the following types:—

1. Ores of the large flake variety in which the molybdenite occurs in flakes above half an inch in diameter, the associated mineral being iron sulphides in a pyroxenite-calcite gangue. As an example of this type may be cited the ores of the Spain mine in Griffith township, Renfrew county, Ontario.

2. Heavy pyritic ores, in which the molybdenite is of the medium flake variety, associated mainly with the iron sulphides, pyrite and pyrrhotite, the gangue being as a rule pyroxenite. In this type of ore the flake is mostly above one-eighth of an inch in diameter, and may be present up to two inches in diameter. As examples of this type may be cited the ores of the Bain mine in Masham township, Hull district, Que., and the ores of the Renfrew Molybdenum Mines, Brougham township, Renfrew county, Ontario.

3. Ores of the medium-flake variety in which the molybdenite is more or less disseminated throughout the pyroxenite gangue rock and associated with small amounts of iron sulphides. The flake as a general rule is smaller than the No. 2 type, requiring finer grinding to free it. As an example of this type may be cited the ores of the Joiner property on lots 3 and 4, concession XX, Cardiff township, Haliburton county, Ontario.

4. Ores of the medium-flake variety in which the molybdenite with an appreciable amount of iron sulphides is disseminated throughout a gangue of quartz, feldspar, and fluorite (altered syenite-gneiss). The flake of this type rarely exceeds half an inch in diameter, and is all freed by grinding to from 35 to 48 mesh. As an example of this type may be cited the Moss Mine ores, Onslow township, Pontiac district, Quebec.

5. Ores of the medium-flake variety; the molybdenite occurring in feldspathic quartz veins with sericite. The molybdenite is generally in the form of rosettes not exceeding half an inch in diameter, and can be freed from the gangue by crushing to 35 to 40 mesh. As an example of this type may be cited the ores on the properties of the Molybdenite Reduction Co., in LaCorne and Malartic townships, Abitibi district, Quebec.

6. Ores of the fine flake and amorphous variety; the molybdenite occurring along the fractures and disseminated throughout the quartz gangue. Examples of this type are the ores of the Alice Arm district, B.C., and the deposit at Kakabeka Falls, Ont. Very fine grinding (to at least 80 mesh) is required to free the molybdenite from the gangue rock.

CONCENTRATION PROCESS BEST SUITED FOR CANADIAN ORES

Molybdenite is one of the easier flotative minerals, as it possesses a marked affinity for oils, especially kerosene, to a similar extent to graphite. Due to its flaky nature and its greasy feel, it is readily concentrated by the flotation process, either by the film or by the froth process. Fair results were obtained by the former by simply floating the dry ground ore on water, or by oiling the wet ground ore and bringing it to the surface of the water. By both these methods the molybdenite floated off on the surface of the water, and the gangue sank. These methods were superseded by froth flotation, due to the greater simplicity of this process, the smaller floor space required for plant, higher grade of concentrates and increased recoveries obtained, and other considerations.

General Flow-sheet for Canadian Molybdenite Ores

With slight alterations to meet the requirements for each particular ore, the flow-sheet, Figure 5, will be found to give the most satisfactory results with respect to the grade of concentrates produced, recovery, simplicity and low cost of operation.

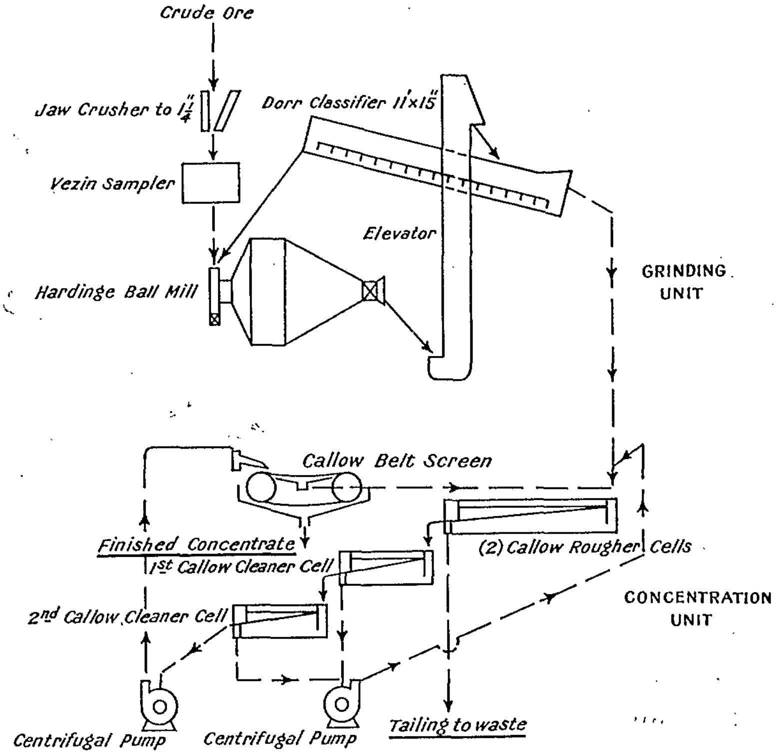

Figure 5. General flow-sheet for the concentration of molybdenite ores.

In the case of ores of the first-mentioned type, the large flakes can be hand-picked, and after crushing the ore is given a preliminary treatment by rolling and screening to obtain as much coarse flake as the operation will warrant. The ore can then be passed to the ball mill and the general flow-sheet followed.

On ores of the second type, a preliminary operation will hardly pay unless a good percentage of the flake is large—above 1 inch in diameter. The general flow-sheet can be followed with few exceptions. More careful manipulation is required to produce a high-grade concentrate on account of the preponderance of iron sulphides present in this type.

For ores of the more or less disseminated type as represented by the third, fourth, fifth, and sixth types described, the flow-sheet can be closely followed with good results. Each particular ore will require certain variations as to grinding, depending at what fineness the flake is entirely freed;

to pulp densities of grinding and flotation operations; to the amounts of reagents used; deflocculation devices, and aperture of the screens used in the final operation.

Most molybdenite ores can be hand-sorted as the mineral is easily distinguished, thus permitting the barren rock to be discarded. Sorting should, therefore, be done whenever possible, so as to give a fair grade of feed to the concentrating plant. As a general rule, the ores can be sorted up to 1 per cent MoS_2 without discarding payable ore, but it would seem that the operation cannot be carried much beyond this point.

CONCENTRATION RESULTS ON THE VARIOUS TYPES OF ORES

Type No. 1

A considerable tonnage of this type of ore was treated during the war when the Ore Testing Laboratories were receiving ores on a custom basis for concentration purposes. As a great number of small individual shipments were received, the concentration results from each were not kept separate. The general procedure with ores of this type was to pick out any large flake that was free of gangue; the ore was then crushed in a jaw crusher, and after removal of the fines any further clean flake was sorted out by passing over a picking belt. The coarse ore was then fed to rolls, and by screening on a 2-mesh screen an oversize product, containing a high percentage of flake, was obtained which was further rolled and screened until the product was of commercial grade. The throughs from the above operations were fed to the ball mill and the regular flow-sheet followed. As a general rule, in ores of this type considerable waste rock can be discarded by hand-picking.

Type No. 2

These ores do not, as a general rule, lend themselves to hand-sorting for the recovery of clean flake. Some flake may be obtained in this manner, but care should be taken not to carry the operation to the unprofitable limit. Barren rock, however, can be discarded and picking operations should be practised to remove waste, so as to give a good feed to flotation. Ores of this type are not so amenable to concentration with the production of high-grade concentrates as the more siliceous types that contain lesser amounts of iron sulphides, due to the tendency of the fine iron sulphides to float and adhere to the molybdenite in the flocculation of the fine flakes. The flake itself in many cases contains fine iron between the laminae, so that concentrates much above 90 per cent MoS_2 cannot be obtained by any mechanical means.

The results of a small-scale test and of a tonnage test on a carload shipment of ore from the Indian Lake deposit, Masham township, Hull district, Quebec, are given below. The ore is of the medium-flake variety in which the molybdenite is associated with a large amount of massive iron pyrite. The gangue is pyroxenite and other lime-silicate minerals.

The richer portions of the ore are almost entirely iron sulphides.

Small-scale Test

Product	Weight		Assay MoS_2 per cent	Per cent MoS_2 values	Remarks
	Grms.	Per cent			
Concentrate—					
+80 mesh	17·7	1·8	89·25	65·1	Total recovery, 88 per cent.
−80 mesh	37·2	3·7	17·00	26·0	
Middling	33·3	3·3	1·94	2·6	
Tailing	891·8	89·2	0·17	6·3	4 lb./ton kerosene.
Feed	980·0	98·0	2·42	100·0	1 lb./ton pine oil.
					A little lime added.

Tonnage Scale Test[1]

Weight of shipment	61,500 pounds
Analysis of sample cut by Vezin sampler	0·83 per cent MoS_2
Average analysis daily sample wet feed to flotation cells	0·92 " "
Analysis of feed by calculation from concentration results	1·00 " "
Content in ore, assuming 1·00 per cent MoS_2 as assay of ore	616·7 pounds
Weight of concentrates obtained	668·5 "
Average analysis of concentrates	78·5 per cent MoS_2
Content MoS_2 in concentrates	524·60 pounds
Clean-up from run (669 pounds at 2·98 per cent MoS_2)	19·9 "
Average analysis of tailing samples	0·12 per cent MoS_2
Content in tailings, MoS_2	72·2 pounds
Recovery MoS_2 from concentration products	87·9 per cent
Recovery figured from formula $R = \frac{100\ (H-T)\ C}{H\ (C-T)}$	88·2 "

Reagents Used.—Kerosene 0·8 lb. per ton; pine oil 0·3 lb. per ton.

Fineness of Grinding.—12·5 per cent on 65 mesh; 50 per cent on 100 mesh.

Conclusions from Results of Concentration Tests.—The above results show that on an ore of this type averaging 1 per cent MoS_2, a concentrate containing 80 per cent MoS_2 can be produced, with a recovery of better than 88 per cent of the molybdenite values in the ore. The ore submitted was taken from near the surface and was somewhat oxidized. This state of the ore did not have any appreciable effect on the recovery, as tailings as low as 0·08 per cent MoS_2 were produced during the run. It may, however, have had some effect on the grade of the concentrate, and it is possible that on fresh ore of this grade a concentrate of 85 per cent MoS_2 could be obtained. The production of a high-grade concentrate from this type of molybdenite ore in which the iron sulphides predominate over the siliceous gangue minerals, is more difficult than from the more highly siliceous ores. More careful control of reagents, pulp densities, deflocculation devices, etc., is required. With proper control, the results given above should be obtained, with the possibility of producing higher grade concentrates on freshly mined ore.

Type No. 3

The dissimilarity of this type as compared with No. 2 is not in the size of the flake but in the absence of large amounts of iron sulphides. The size of the flake generally prohibits hand-picking but waste rock should

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[1] For further details of these tests readers are referred to:—
Mines Branch, Dept. of Mines, Canada, Sum. Rept., 1921, pp. 191-194.
Mines Branch, Dept. of Mines, Canada, "Investigations in Ore Dressing and Metallurgy, 1924," pp. 68-71.

be discarded to give a good flotation feed. The type of ore is very amenable to concentration with the production of high-grade concentrates and good recoveries. The general flow-sheet can be followed very closely. The following results of some small-scale tests, and of a tonnage test, are given on the various classes of ore from the deposits on lots 3 and 4, concession XX, Cardiff township, Haliburton county, Ontario.

Small-scale Test

Four lots of ore were received, representing four types found on the property. In Lot No. 1 the ore was of the medium-flake variety, the flake being much larger and containing more iron sulphides than the other lots. In Lots Nos. 2 and 3, the flake was much smaller being more evenly disseminated throughout the gangue. The ore contained very little iron sulphides. In Lot No. 4, the rock contained an appreciable amount of graphite.

Lot No. 1	0·027 per cent MoO_3	0·68 per cent MoS_2
Lot No. 2	0·027 " "	0·32 " "
Lot No. 3	0·025 " "	0·38 " "
Lot No. 4	0·025 " "	0·12 " "

Lot No. 4 was too low-grade to be worked commercially and no tests were made.

Lot No.	Mesh	Product	Weight		Assay MoS_2 per cent	Per cent MoS_2 values	Remarks
			Grms.	Per cent			
1	48	Concentrate—					
		+80	7	0·7	93·31	81·9	Total recovery 91 per cent.
		−80	31	3·1	2·20	8·5	
		Middling	57	5·7	0·38	2·8	½ lb./ton kerosene.
		Tailing	902	90·5	0·06	6·8	½ lb./ton pine oil.
2	65	Concentrate—					
		+80	3	0·3	80·25	75·6	Total recovery 84 per cent.
		−80	33	3·2	0·58	5·9	
		Middling	86	8·5	0·18	4·7	½ lb./ton kerosene.
		Tailing	891	88·0	0·05	13·8	½ lb./ton pine oil.
3	65	Concentrate—					
		+80	4	0·4	87·25	75·4	Total recovery 83 per cent.
		−80	39	3·8	0·64	5·4	
		Middling	96	9·5	0·20	4·1	½ lb./ton kerosene.
		Tailing	872	86·3	0·08	15·1	½ lb./ton pine oil.

Tonnage Scale Test

This test was on a carload shipment of low-grade ore from the deposit first opened up on the property. The molybdenite flake was up to half an inch diameter in a pyroxenite gangue.

Weight of shipment	59,026 pounds
Analysis of ore	0·308 per cent MoS_2
Analysis of concentrates obtained	89·65 " "
Analysis of tailings	0·115 " "
Recovery of molybdenite values	62·7 "

Reagents Used.—0·75 pound per ton kerosene and pine oil.

Conclusions from Results of above Tests.—The ore is very amenable to flotation, with the production of a high-grade concentrate and good recoveries. A 90 per cent concentrate should readily be made from these ores. The large-scale test was made during the earlier test work on molybdenite ores and the best operating conditions were not practised. Tailings as low as 0·05 per cent MoS_2 should have been produced on this grade of ore, which would give a recovery of 83·8 per cent. The small-scale tests which were run later indicate this possible recovery.

Type No. 4

The flake of this variety of ore is usually below ¼ inch in diameter and fairly well disseminated throughout the siliceous gangue rock which consists of quartz, feldspar, and fluorite (altered syenite-gneiss). An appreciable amount of iron sulphides is present. The nature of the gangue makes the rock more difficult to crush than the other types but as all the flake is freed at 48 mesh, crushing and grinding costs are reasonably low. The ore is very amenable to concentration so that the general flow-sheet can be followed very closely. Sorting out of waste rock should be practised to give a good grade of mill feed. Ores of the former types appear to contain much more molybdenite than ores of the same grade of this type, due to the smaller flake and the more disseminated character of the latter. The following tonnage tests were made on two carloads of ore from the Moss mine, Onslow township, Pontiac district, Quebec.

Tonnage Scale Tests[1]

Shipment No. 1—

One car of ore, net weight	50,600·0	pounds
Analysis of sample cut from dry ore by Vezin sampler	2·21	per cent MoS_2
Average analysis daily sample wet feed to flotation cells.	2·45	“ “
Average analysis of dry and wet samples	2·33	“ “
Content MoS_2, using average analysis	1,178·98	pounds
Amount of concentrate obtained	1,213·0	“
Analysis of concentrate	92·93	per cent MoS_2
Content of MoS_2 in concentrate	1,127·24	pounds
Calculated analysis of tailing	0·10	per cent MoS_2
Recovery MoS_2, from average assay and calculated tailing	95·6	“

Shipment No. 2—

One car of ore, net weight	84,600·0	pounds
Analysis of sample cut from dry ore by Vezin sampler	2·33	per cent MoS_2
Average analysis daily sample wet feed to flotation cells.	2·60	“ “
Average analysis of dry and wet samples	2·48	“ “
Content MoS_2, using average analysis	2,081·16	pounds
Amount of concentrate obtained	2,135·0	“
Analysis of concentrate	93·44	per cent MoS_2
Content of MoS_2 in concentrate	1,994·94	pounds
Analysis of tailing, average of daily samples	0·12	per cent MoS_2
Calculated analysis of tailing	0·12	“ “
Recovery of MoS_2	95·8	“

Conclusions from Results of Concentration Tests.—The ore is very amenable to concentration by flotation. With proper manipulation and under the right conditions, a high-grade concentrate, 93 per cent MoS_2 is obtained with recoveries in excess of 95 per cent of the molybdenite content

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[1] For further details of these tests readers are referred to:—
Mines Branch, Dept. of Mines, Canada, "Investigations in Ore Dressing and Metallurgy, 1924," pp. 68-71.

in the ore. Concentrates assaying over 96 per cent MoS_2 were made during the runs under the best conditions. The concentrates are exceptionally free from deleterious substances; are suitable for the manufacture of molybdic acid, molybdic salts, molybdenum metal, ferro-molybdenum, and should command the highest market price.

Type No. 5

In this type of ore the molybdenite occurs in quartz, feldspathic quartz, and feldspar veins, generally in the form of rosettes, rarely exceeding $\frac{1}{4}$ inch in diameter. It is usually associated with sericite. Very little iron sulphides are present. The chief known occurrences of this type are south of Amos, Que., and in Nova Scotia. Barren quartz and feldspar are easily sorted out, and this should be practised to give a good grade of ore to the mill. The ores are very amenable to concentration, so that the general flow-sheet can be followed closely. Crushing operations do not need to be carried finer than 48 mesh to free the flake. A tonnage test on the ore from the Molybdenite Reduction Company's properties in LaCorne and Malartic townships, Abitibi district, Que., gave the following results:—

Tonnage Scale Test[1]

Weight of ore concentrated	19,757·0	pounds
Analysis of ore	2·02	per cent MoS_2
Content of MoS_2 in ore	399·5	pounds.
Concentrates obtained	365·5	"
Analysis of concentrates	90·80	per cent MoS_2
Content of MoS_2 in concentrate	323·9	pounds
Clean-up of ball mill, etc	1,253·5	"
Analysis of clean-up, etc	4·59	per cent MoS_2
Content of MoS_2 in clean-up	57·54	pounds
Tailings, weight	18,147·0	"
Analysis of tailings	0·10	per cent MoS_2
Content of MoS_2 in tailings	18·15	pounds

Conclusions from Results of Concentration Test.—With an average mill feed of 2·02 per cent MoS_2, a concentrate averaging 90·8 per cent MoS_2 was obtained with an average tailing of 0·10 per cent MoS_2. This gives a recovery of about 95 per cent of the molybdenite content in the ore. Higher grade concentrates and lower tailings were made at intervals during the run. The ore is very amenable to concentration. Its physical character and the absence of appreciable quantities of other sulphides such as copper and iron, make it an attractive milling ore. It is not difficult to grind, the molybdenite being freed at about 40 mesh. These characteristics permit of an exceptionally high-grade concentrate being produced with very high recoveries of the molybdenite values in the ore.

Type No. 6

Ores representative of this type are more difficult to concentrate with the production of high-grade concentrates and low tailings. Much finer grinding is required. The fine flake and amorphous variety is generally associated with a quartz gangue. The quartz is sometimes discoloured by the presence of extremely fine flake. Grinding should not be carried beyond the profitable limit. The general flow-sheet can be followed with variations as to degree of grinding, pulp densities, amounts of reagents

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[1] For further details of these tests readers are referred to:—
Mines Branch, Dept. of Mines, Canada, "Investigations in Ore Dressing and Metallurgy, 1923." pp. 76-77.

used, and to the screens used for obtaining the finished concentrate. The following concentration results are given from small-scale tests made on ores from Kakabeka Falls, Ont., and Alice Arm, B.C.

Ore from Kakabeka Falls

The molybdenite is of the amorphous variety, filling the fractures in the quartz and disseminated throughout the quartz. A small amount of pyrite is present. Analysis showed it to contain: molybdenite, 2·46 per cent; copper none; bismuth, none; arsenic none.

Test No.	Mesh	Product	Weight		Assay MoS_2 per cent	Per cent of MoS_2 values	Reagents
			Grms.	Per cent			
1	50	Concentrate.......	33	3·3	60·97	67·6	1½ lb./ton kerosene.
		Middling..........	38	3·8	9·17	11·6	½ lb./ton crude turpentine.
		Tailing...........	929	92·9	0·68	20·8	
2	80	Concentrate.......	41	4·1	52·44	73·1	3 lb./ton kerosene.
		Middling..........	40	4·0	10·19	13·8	1 lb./ton crude turpentine.
		Tailing...........	919	91·9	0·42	13·1	
3	80	Concentrate.......	40	4·0	58·87	76·6	3 lb./ton kerosene.
		Middling..........	97	9·7	4·93	15·5	1 lb./ton pine oil.
		Tailing...........	863	86·3	0·28	7·9	

Conclusions from Results of Concentration Tests.—The ore requires grinding to at least 80 mesh to give low tailings. The concentrate made from these small tests is below marketable grade but by screening on 100 mesh, a portion of the fine iron and silica adhering to the flakes will be removed. A 70 to 75 per cent MoS_2 concentrate should be obtained from this class of ore. The recovery should be about 80 per cent.

Ore from Alice Arm, B.C.

The molybdenite is of the fine flake and amorphous variety, filling the fractures and disseminated throughout the quartz gangue.

Test No.	Mesh	Product	Weight		Assay MoS_2 per cent	Per cent of MoS_2 values	Reagents
			Grms.	Per cent			
1	65	Concentrate.......	31	3·1	66·51	77·6	1 lb./ton kerosene.
		Middling..........	20	2·0	10·72	8·3	½ lb./ton pine oil.
		Tailing...........	949	94·9	0·39	14·1	Little lime added.
		Feed.............	1,000	....	2·63	....	
2	65	Concentrate.......	51	5·1	60·45	93·2	1 lb./ton kerosene.
		Middling..........	41	4·1	2·36	2·9	½ lb./ton pine oil.
		Tailing...........	908	90·8	0·14	3·9	Little lime added.
		Feed.............	1,000	....	3·28	....	

Conclusions from Results of Concentration Tests.—The ore requires grinding to 65 mesh to obtain low tailings. The concentrates made are below marketable grade but by deflocculation and screening to remove adhering silica and pyrite a 70 to 75 per cent concentrate should be obtained from this class of ore. The recoveries should be between 85 and 90 per cent. The copper content of the ore was about 0·2 per cent, and the concentrates assayed 0·15 per cent copper.

NOTES ON FLOTATION OF MOLYBDENITE ORES

1. The ore should be ground to pass practically all through 35 mesh for flotation. Large thick flakes will be lost in the tailing unless ground to this mesh. The fineness of grinding will depend on the mesh at which the flake is freed and on the character of the flake. A large thin flake will float with more ease than a small thick flake. The degree of grinding should be determined by small-scale laboratory tests.

2. The reagents best suited for the flotation of molybdenite ores are pine oil as the frothing oil and kerosene as the collecting oil. The best quality steam-distilled pine oil, such as the General Naval Stores No. 5, and the British American Oil Company's "Lampolene" brand of kerosene, are preferable. The amounts of these oils used need to be determined for each individual ore, and will depend on the character of the ore, its molybdenite content, the nature of the flake, whether thin, flat, or heavy, thick flakes, etc. For an ordinary clean ore of 1 per cent grade the average amounts of reagents will be: pine oil about 0·25 pound per ton and kerosene 0·5 to 0·75 pound per ton. If an excess of pine oil is added, the flotation circuit will build up with too much dead oil, giving a voluminous froth with very little carrying property, resulting in the lowering of the grade of the concentrate and increasing the loss in the tailing. The oil should be added to the ball mill, especially the kerosene, a portion of the pine oil can be added to the ball mill and a portion to the head of the flotation cells. The addition of other reagents, such as lime and soda ash, is not necessary on an ordinary clean ore, although they have the effect of raising the grade of the concentrate, this is at the expense of a higher tailing. The use of these reagents increases the amount of kerosene necessary to produce low tailings. In cases where the ore is very badly oxidized and the circuit becomes built up with soluble salts, destroying the effectiveness of the flotation oils, it becomes necessary to use alkaline salts to counteract the acidity of the ore.

3. The proper density of the pulp in the ball mill and in the flotation circuit should be determined for each individual ore. The best results are usually obtained with a pulp density of between 45 and 50 per cent solids in the ball mill. The correct oiling effect on the molybdenite to give the best results is not obtained by too thick or too thin a pulp. The proper pulp density can be gauged by the character of the froth on the flotation cells. Using Callow pneumatic cells, the froth should consist of lively breaking bubbles, the molybdenite appearing on the convex faces near their domes, and all coming off in the first foot or so of the cells. If on the other hand the froth is voluminous, heavy, and slow-breaking, and the molybdenite appears in the troughs between the bubbles and is carried down towards the discharge end, the proper conditions do not prevail.

4. In order to produce a high-grade concentrate care should be taken to keep the flake at the coarsest size possible at which it will readily float, and at the same time, free from any attached particles of gangue or other mineral. In order to prevent freed flake from returning with the classifier oversize, the classifier can be equipped with air and water sprays. By permitting the feed to plunge down into the settling area, thus creating a boiling action which tends to bring the flake to the surface; by using water sprays on the rakes to wash out the flake from the oversize; and by the use of an air spray to blow the flake towards the overflow, it is possible

to reduce the amount of flake returning to the mill with the oversize by approximately 50 per cent.

5. In the flotation of molybdenite there should be ample rougher cell capacity to make a clean tailing to go directly to waste, and to provide for any emergencies such as increased grade of the ore, the adjustment of flotation conditions, and the return of large amounts of middling products from the cleaner cells and the finishing screens. The cleaner cell operations should be crowded, so as to take off as high grade products as possible. To obtain high grade concentrate it is better to have two cleaners, one following the other, than only one. A much higher grade product will be sent to the finishing screen.

6. In the case of some ores which are exceptionally clean, the finishing screen can be dispensed with, but as most ores contain some pyrite and other minerals, which in a fine state of division will float and become entangled with the molybdenite, the screen is necessary. The mesh of this screen will depend on the fineness of the flake. For ores other than No. 6 type described above, an 80-mesh "Ton-cap" screen is used. For ores of the No. 6 type, 100-mesh screens are necessary. The use of the finishing screen for increasing the grade of the concentrate depends on the coagulating effect of kerosene on fine particles of molybdenite, thus permitting them to form a larger unit and stay on the screen while the fine pyrite and gangue passes through. A series of analyses made of the concentrate before and after passing over the screen showed that the use of the screen raised the grade 5 to 10 per cent in MoS_2 content.

This coagulating effect, or flocculation, of the molybdenite has a tendency to include some fine pyrite and other minerals with it, and has to be broken up to a certain extent before the cleaner concentrate passes onto the screen. This can be done by feeding fresh water with the cleaner concentrate to a centrifugal pump and pumping to a feed-well for distribution onto the screen. The control of the amount of water is important. An excess breaks up the flocculation of the molybdenite permitting too much fine molybdenite to pass through the screen and be returned to the flotation circuit. Too little will not permit the fine pyrite and gangue to become disentangled, thus lowering the grade of the concentrate. The use of the centrifugal pump between the last cleaner cell and the screen is to re-flocculate the molybdenite in a pulp of clean water, so that the fine pyrite and gangue will be delivered onto the screen free from entanglement in the flocculated molybdenite, and so pass through the screen.

Market Requirements

The market requirements call for 85 per cent molybdenite in ores and concentrates. During the war a concentrate assaying over 65 per cent MoS_2 was received by the reduction plants, provided that the other minerals contained in this low-grade concentrate were mostly iron sulphides. As a rule this was the case, and permitted straight reduction to ferro-molybdenum without the addition of iron, with the production of the ferro-alloy containing 70 per cent molybdenum, and less than 0·4 per cent sulphur. It was on this basis of analysis that the ferro-molybdenum produced in Canada was sold. Molybdenum concentrates should, therefore, contain over 65 per cent MoS_2 for the reduction to ferro-molybdenum to be used for steel purposes. Small amounts of copper and of other metals, below

0·5 per cent, should not be harmful, as the amounts of the metal used for steel purposes is generally under 0·5 per cent molybdenum, so that the amount of copper and other metals taken up by the steel from the ferro-molybdenum will be infinitesimally small.

For the manufacture of molybdic acid, molybdenum salts, and molybdenum metal, the concentrates should be as high grade as possible.

Concentrates assaying over 90 per cent and as high as 95 per cent MoS_2 are preferred for these purposes.

Canadian molybdenite ores as a rule are exceptionally free from harmful impurities and the concentrates produced from them are high grade. This is especially true of the eastern deposits. The western deposits are more apt to contain copper and other impurities.

Market Quotations

The New York quotations for molybdenite ores and concentrates are 75 to 80 cents per pound of MoS_2 content in ores and concentrates containing 85 per cent molybdenite and over. Against this price there is a United States tariff of 35 cents per pound molybdenum content, or 21 cents per pound of MoS_2 content, which makes the Canadian market price 54 to 59 cents per pound, less transportation charges. European quotations are 45 to 50 shillings per unit which, reduced to Canadian currency, is 50 to 55 cents per pound, or about equal to the United States price, after the custom tariff is deducted.

Present Outlook

It would seem that for the immediate future, holders of molybdenite properties could not figure on much advance over the present price, namely 50 to 55 cents per pound at Canadian points of shipment. The price of molybdenite is controlled by the price of other metals used for a similar purpose in the manufacture of ferro-alloys. Although there are certain cases where the use of molybdenum has proven its superiority over others of the rarer metals, up to the present this use has not consumed any large amounts. The price will, therefore, be governed by such metals as vanadium and other alloys of steel having a fairly large production. A factor governing the price will be the ability of the United States operators to produce at the present market price. The greatest obstacle to Canadian production at the present time is the United States tariff. Production is being retarded by the uncertainty of the market, and by the different viewpoints of the mine owners and buyers of molybdenite concentrates. The mine owners desire to be assured of a contract for their output at a reasonable price before proceeding with the expenditure of any great amount of money in developing their properties. The users of molybdenum desire to be assured of a uniform and continuous supply at the lowest market price, before changing over to molybdenum for alloy steel manufacture. The revival of the industry in Canada will depend on the assurance of a steady demand for the concentrates at a price indicating profitable operations.

Limitations to Workable Deposits

A 1 per cent molybdenite ore is a $10 ore with molybdenite at 50 cents per pound. On a basis of 50 tons a day, or even as low as 25 tons a day, operating expenses should not exceed $7.50 per ton, on any fair-sized deposit. In the case of one company operating during the war, it was shown that with molybdenite at $1 per pound, the workable limit was 0·3 per cent MoS_2, on a basis of 100 tons daily. As shown from the concentration of the ores given above, the process is simple, the majority of Canadian ores are readily amenable to concentration, grinding in most cases does not have to be carried finer than 48 mesh, so that concentration charges are low. The chief item of cost is mining and development. Owing to the character of the deposits and veins, considerable waste has to be broken and sorted out. Much development has to be done in waste rock, so that mining costs will be as high, if not greater than lode mining. Capital expenditure for mining plant and equipment will be similar to lode mining for the same tonnage mined, but the cost of the concentration plant will be lower, as the process as a general rule is simpler.

THE CONCENTRATION OF CANADIAN FLAKE GRAPHITE ORES

C. S. Parsons

INTRODUCTORY

The recent revival of interest in graphite due to improved market conditions has prompted the writing of this article. The history of graphite mining in Canada should be carefully studied by any company contemplating entering this field. Many failures in the past have been due to inefficient methods of concentration, but a number have resulted also from the erection of extensive milling and plant equipment before the extent and grade of the ore-bodies were determined; and from insufficient knowledge of the market and trade requirements.

TYPES OF CANADIAN GRAPHITE ORES

Two main classes of graphite ores are known to occur in Canada: disseminated crystalline flake graphite, and the more or less massive crystalline graphite which occurs in veins or pockets. The deposits of the latter variety which is commonly called "crystalline graphite," have never been considered commercially important, and for this reason their concentration will not be discussed in this article.

In Canada the average run-of-mine ore of the flake variety will contain 10 to 15 per cent graphite; the average American ore of this type contains only between 3 and 7 per cent.

FACTORS GOVERNING THE VALUE OF A GRAPHITE ORE

In determining the value of a graphite ore, granted that there is a sufficient tonnage available, the governing factor is the size of the flake. Since there is a ready market only for large flake, the value of the ore depends on the amount of No. 1 and No. 2 flake recoverable rather than on the total graphite content of the ore. It is possible that a 10 per cent ore may be more valuable than a 15 per cent ore if more No. 1 and No. 2 flake can be recovered from it. However, certain ores may contain more large flake than others but still be of less value, due to the hardness and nature of the associated gangue. A hard siliceous gangue is difficult to free, causing a large amount of flake to be ground up in the process of separation.

CONCENTRATION

Before the introduction of froth flotation, the concentration of graphite presented many difficulties. It may be conservatively stated that the losses in the tailings exceeded 50 per cent. The actual production of a high-grade graphite concentrate by flotation would be a simple matter, as it is one of the most easily floated minerals, were it not for the fact that the trade demands certain qualities, and pays the highest price for No. 1 flake (which is flake remaining on a 90-mesh screen). The finer sizes have a limited market and at most times it is impossible to dispose of them at any price. The main problem is, therefore, the design of a flotation plant that will destroy as little as possible of the large flake in the ore. The efficiency of the mill will depend more on the amount of large flake produced than on the total recovery of graphite.

Refining

It can be readily understood that there is a point beyond which it would not be practical to raise the grade of a concentrate produced by continuous operation in a concentrating mill. Therefore, to determine when concentration should be stopped, and refining in batch lots resorted to, is essential. In the refinery each operation is under exact control and the size of the flake is more easily preserved.

General Principles in Design of Concentrating Mill

There are a few general principles which should be kept in mind when designing a flow-sheet for a mill to concentrate graphite ore. The ore should be crushed only to the point where it is possible to obtain a reasonably clean tailing. The concentrate produced by the primary flotation should contain a minimum of free gangue. In recleaning this concentrate the middling products will consist of true middlings and particles of gangue which have become mechanically entangled with the concentrate. The greater part of the entangled gangue can be eliminated by operating the first cleaner cells in closed circuit with the roughers. The flow-sheet should be flexible enough to grade the true middlings into two general classes. These two classes are: first, particles of gangue to which are attached small pieces of graphite; second, particles of graphite to which are attached smaller pieces of gangue. These two classes naturally grade into each other. Just where the division between these two classes should be made will depend on final results in the refining department. After the division has been effected, the first class should be returned to a separate grinding circuit, and the second class treated as final concentrate and delivered to the refinery, where the work of freeing the attached pieces of graphite can be effected more efficiently.

Flow-sheet of an All-flotation Plant

A flow-sheet of a 200-ton mill which operated three years on a hard, 7 per cent ore is shown as Figure 6. This mill was one of the first flotation plants to treat a hard, siliceous graphite ore. Flotation replaced a system of buddle concentration. A study of this flow-sheet will show that the entire concentrate from the flotation cells is passed through two pebble mills for regrinding. When the mill was first built only one regrinding mill was used, but the additional recovery of graphite overloaded the capacity of the refining plant, and an agitation was immediately started by the refinery superintendent to have a higher grade product delivered to the refinery. A second regrinding pebble mill was then installed. The graphite from this mine was used entirely for lubricating purposes, and even a grade of 88 per cent carbon had to be further refined.

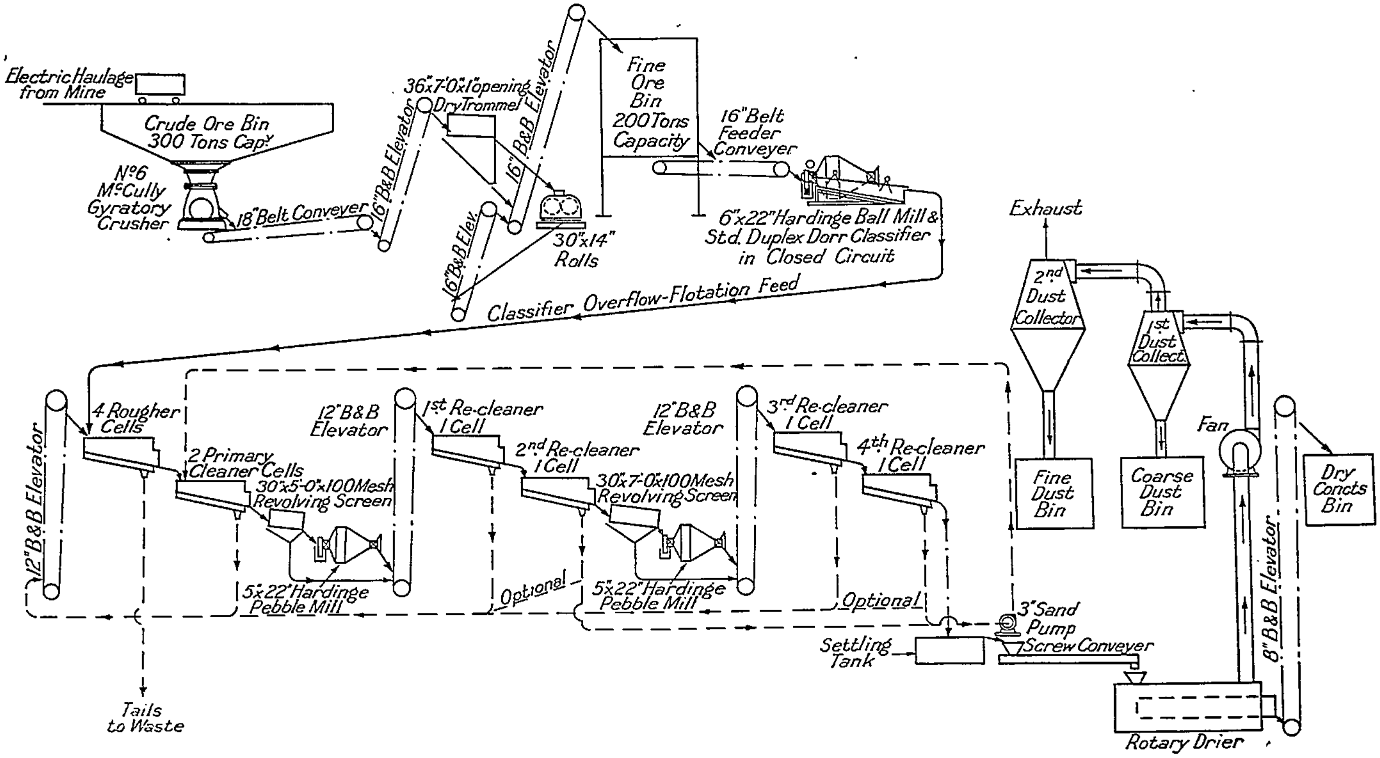

Figure 6. Flow-sheet for graphite mill using an all-flotation process.

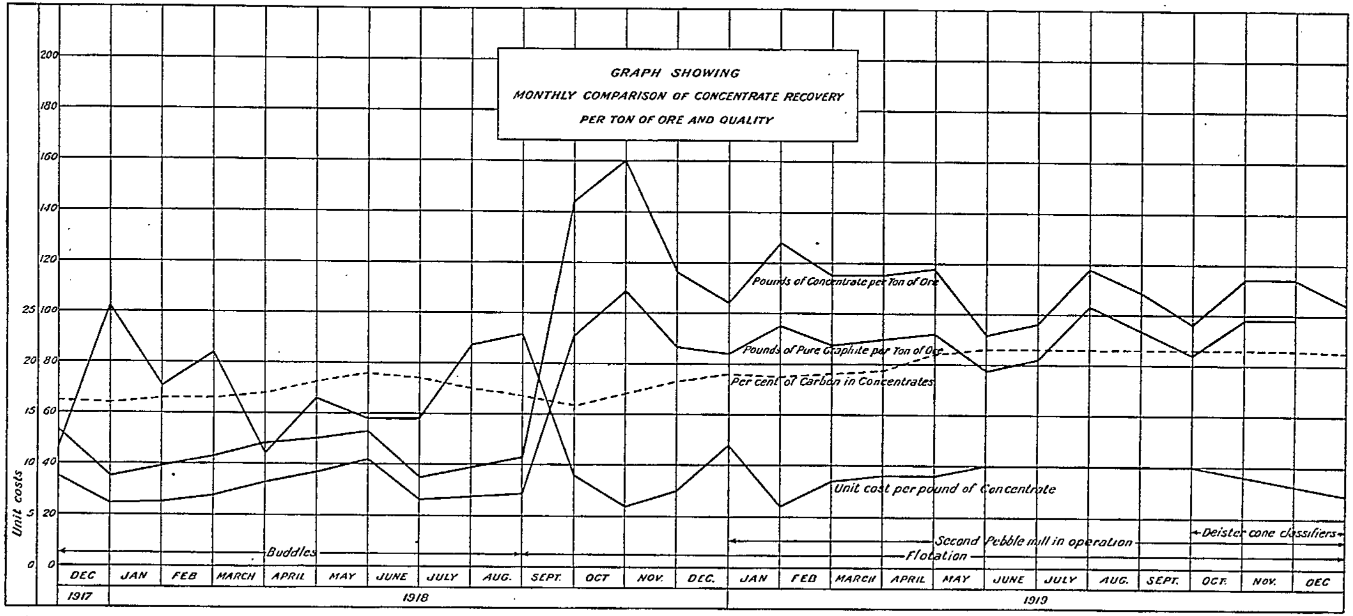

Figure 7. Graph showing comparison of costs using different systems of concentration.

A series of curves is given in Figure 7, showing a comparison between the buddle system, flotation using one regrinding mill, and flotation using two regrinding mills. These curves give the number of pounds of pure graphite (carbon) recovered from a ton of ore; the weight of concentrate recovered from a ton of ore; the grade of concentrate and percentage of carbon contained; and the unit cost per pound of concentrate produced. A comparison between the use of buddles and flotation with reference to the amount of +60-mesh pure graphite (carbon) is given in Table I.

TABLE I

A Comparison of Average Results Obtained on a 6 to 7 per cent Hard Graphite Ore

—	Buddles	Flotation —one regrinding mill	Flotation —two regrinding mills
Pounds of concentrate recovered from one ton of ore........	45	120	116·0
Per cent recovery on 60-mesh screen........................	60	43	25·0
Per cent carbon content of +60-mesh product..............	60	73	88·0
Pounds of pure graphite recovered, + 60 mesh.............	16	35	25·5

The flow-sheet used in this mill (Figure 6) is not in accordance with the principles previously given. The clean, coarse flake should have been separated from the concentrate before passing to pebble mills for regrinding. One of the most practical methods of doing this is to table the concentrate first. This flow-sheet is given to illustrate how readily any additional grinding will destroy the coarse flake, and to draw attention to the remarkable improvement in recovery and lowering of costs by flotation.

FLOW-SHEET OF PLANT USING FLOTATION AND TABLING

A flow-sheet, Figure 8, is given of a mill in which tables were used on the flotation concentrate. In this case the sands and part of the middlings were run to waste in the table tailing, instead of being reground and returned to the circuit. The grade of concentrate delivered to the refinery depended largely on the operation of the tables, so that in order to raise the grade it was necessary to turn more graphite into the table tailing, thus causing a loss of considerable large flake. The recovery was low, probably not exceeding 75 per cent, but this was offset by the large proportion of No. 1 crucible flake recovered. It will be observed from the flow-sheet that the flotation concentrate was not reground as in the previous one, so that although the recovery was low, the proportion of No. 1 crucible flake produced was large and the refinery actually produced 70 per cent No. 1 flake of a grade exceeding 92 per cent carbon.

THEORETICAL FLOW-SHEET

A flow-sheet, Figure 9, has been worked out according to the general principles of design given. This flow-sheet starts with grinding in closed circuit with either an Akins or Dorr type classifier, the grinding being carried only to the point where clean tailings can be made from the rougher

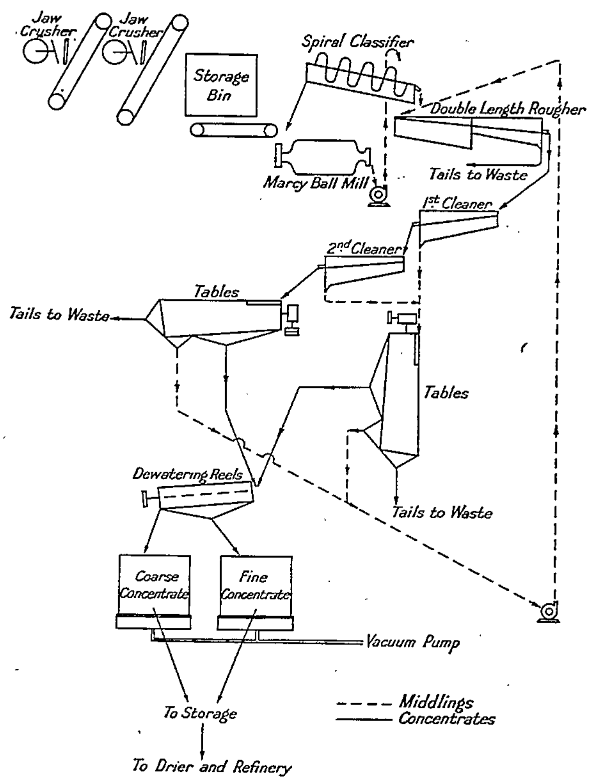

Figure 8. Flow-sheet for graphite mill using flotation and tabling.

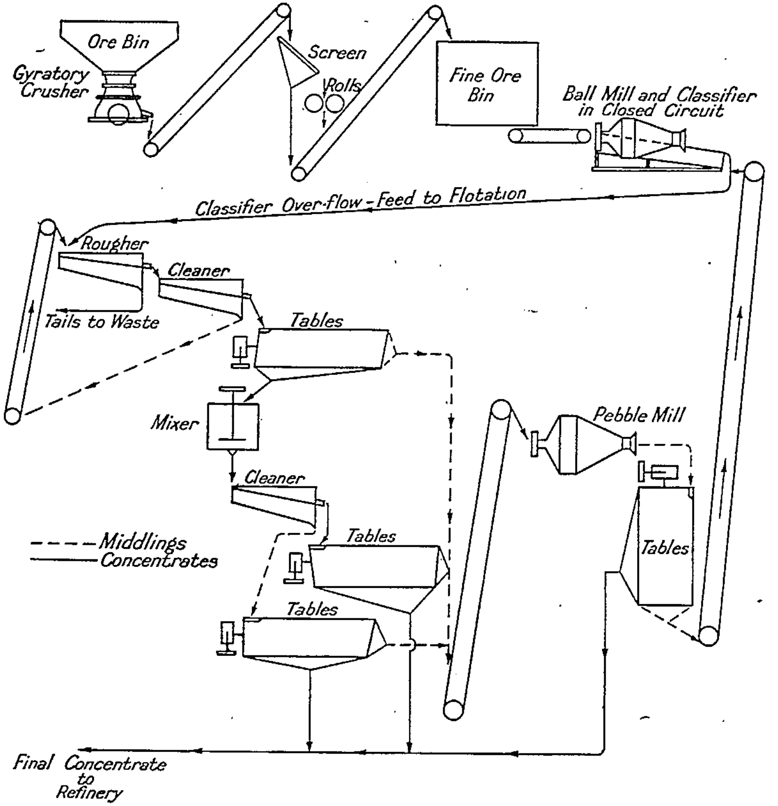

Figure 9. Theoretical flow-sheet for flake graphite ores based on experiments.

cells. There is a possibility that it would be practical at this stage to table the classifier returns to prevent the coarse flake contained in the oversize again passing through the ball or rod mill. The rougher concentrate is cleaned once in a cleaner cell and the cleaner tailing returned to the rougher feed. The cleaner cell should be operated to drop only entangled sands. The concentrate from the cleaner cell goes to tables which as far as possible should be operated to separate only the sand particles containing relatively small flakes of graphite. The concentrate from the tables is dewatered and sent to a second cleaner cell, while the table tailing goes to a pebble mill for regrinding. The second cleaner cell will drop some coarse, free flake as well as flake of various sizes with attached pieces of gangue. The second cleaner tailing is then tabled and a finished concentrate produced. By this tabling operation the clean coarse and fine flake dropped from the flotation cell is recovered and not returned for further grinding in the pebble mill, whereas the flake of various sizes containing attached gangue is sent to the pebble mill for regrinding. The second cleaner flotation concentrate is again tabled; this tabling may not be actually necessary, but it acts as a safeguard against the inefficient operation of the flotation cells. The discharge of the regrinding pebble mill is tabled, and a finished concentrate made, and the sands and entangled particles of graphite and gangue are returned to the head of the rougher cells.

It is believed that a flow-sheet designed along these general lines will give the most efficient results on flake graphite ores. On soft and easily disintegrated ores it will not be necessary to carry the flotation and tabling so far. The particular advantage claimed for this flow-sheet is that besides eliminating the coarse flake from the circuit as soon as it is freed, the fine flake produced in the refinery will be of a much higher grade than that obtained from the two flow-sheets previously given.

Interpretation of Small-scale Tests

The correct interpretation of the results of small-scale laboratory test work requires explanation. The scope of this paper does not allow a lengthy discussion on this subject, but it has been thought advisable to draw attention to the great difference in results obtained by crushing in small laboratory ball mills and crushing in large standard sized mills. One example will be given to illustrate this difference. An exhaustive number of laboratory tests were run on a certain ore to ascertain the quantity of large flake that could be recovered. These tests showed a recovery of more than 50 per cent of the flake on a 60 mesh screen, the product assaying 85 per cent carbon. The results from the mill designed from these tests showed a recovery of only 30 per cent on 60-mesh, assaying 85 per cent carbon. This large difference was due to the action of grinding in large ball mills.

Some Notes on the Types and Operation of Machinery in General Use

Grinding.—The ball mill is in general use, and there is little doubt that this is the most practical and satisfactory machine for the final reduction of the ore. The conical type of mill is most favoured because it is claimed that the flake when freed floats out of the mill more quickly than with the cylindrical types. However, the writer has seen equally as good results obtained in the ordinary cylindrical mill when used with either a diaphragm or open trunnion discharge. The lubricating nature of graphite has been found to reduce the rated capacity of a ball mill 25 per cent, a factor that should be taken into consideration when determining the size of the mill to install. From extensive tests carried out by the writer it was found that less flake was destroyed by grinding in a dilute pulp of from 45 to 55 per cent solids (as much as 10 per cent additional flake reporting on a 60-mesh screen). The rod mill is suggested as possibly giving more efficient grinding than the ball mill. The Ore Dressing and Metallurgical Division of the Mines Branch contemplates carrying out a number of comparative tests with the rod mill and ball mill.

Closed Circuit Grinding with Classifier and Ball Mill.—An excess of water must be avoided in order to obtain efficient results in a flotation plant. Hydraulic classifiers should not be used in the circuit. It has been claimed that they are better than the Dorr or Akins types for preventing the return of the large flake with the oversize to the ball mill, but it has been found in three different mills that any advantage gained in this way is offset by the increased loss of coarse flake in the flotation tailing, due to high dilution of the pulp in the cells.

Flotation Machines.—The Callow pneumatic cell has been more extensively used than any other for the flotation of graphite, but there is no reason why just as satisfactory results cannot be obtained from any other standard type of machine. The type of Callow cell found to be the most satisfactory for graphite is the short 8 by 2 feet cell with sloping bottom. A short, narrow, flat-bottom cell of the same size should also work well. The long, 19 by 3 feet flat-bottom cell or the 40 by 3 feet cell, are not recommended for the average Canadian flake ore.

In the operation of a pneumatic type of cell it has been found advantageous to use a system of baffling in the cell. The sides of the cell are built up with boards to a height of 12 to 14 inches, and a series of three vertical cross baffles of different heights are built in the cells so that the froth descends in cascades and flows through an outlet. In the rougher cell, the outlet is at the feed end, the side boards being cut away 12 to 20 inches from that end, allowing the froth to discharge over the sides. In the cleaner the froth is carried direct through an opening cut in the front or discharge end of the cell. These baffles are carried down to within 6 inches of the blankets and are arranged on top so that there is approximately a 3- to 4-inch drop between each baffle. The use of the baffles makes the operation of the rougher cell much simpler, and by their use in the cleaner cell, the grade of concentrate can be raised 5 to 10 per cent.

Flotation Reagents.—Kerosene and steam distilled pine oil are the two oils which should be used. The use of kerosene, however, requires careful manipulation, as the least excess will tend to kill the froth in the cells. For this reason some operators have had trouble using it, but the writer has operated with it for a number of years and never experienced trouble. Hardwood creosote oil has been used along with the kerosene to overcome the raw oil effect of the kerosene, but its use is not to be recommended, as it is accompanied by a greater loss of graphite in the tailing. Additional reagents such as lime and soda ash are seldom necessary.

Tables.—Any table made by a reputable manufacturer should be satisfactory. Some operators use fine silk bolting cloth over the launder along the low sides of the table, for dewatering the graphite concentrate. The frame holding the silk is attached to the side of the table so that the bumping action of the table keeps it free. In most cases the riffling of the table will have to be altered to take care of the type of feed to the table.

Filters and Dewaterers for the Concentrate.—Continuous suction filters have not as a rule been satisfactory, due to the physical character of the graphite. In most cases it has been found impossible to reduce the moisture content of the concentrates below 30 per cent. The lubricating qualities of graphite, together with the tendency of the thin flat flakes to seal the pores of the filter cloth prevent a cake from being built up and held in the filter until the proper time for discharging. A number of more recent tests made by the United Filters Corporation on a sample of concentrate from one of the Buckingham mills have shown more favourable results. The results of these tests are as follows:—

Test No.	Pulp ratios by weight	Vacuum on cake, inches	Vacuum on dry, inches	Moisture in cake, per cent	Cake thickness, inches	Capacity, sq. ft. per hour, dry graphite
1....................	3 : 1	18	18	20·9	2	216·0
2....................	8 : 1	11	16	20·5	1¼	36·0
3....................	20 : 1	15	18	20·5	⅝	24·0
4....................	16 : 1	8	17	20·1	9/16	16·0
5....................	5 : 1	5	18	20·6	⅞	42·5
6....................	10 : 1	21	20	22·5	⅞	60·0
7....................	20 : 1	21	21	24·5	3·8	23·8

In each test, with the exception of No. 4, the pulp in the filter was air agitated. The hourly capacity of the filter varies, due to the speed at which the filter is revolved, the vacuum at which the cake is built, and the pulp ratio. It has been suggested that if the +90 flake were screened out before filtering, much better results could be obtained. The suggestion is practical, and it would appear that better results can be expected.

The system most generally used is a series of settling tanks which are allowed to fill alternately. After settling, the water is decanted and the graphite allowed to drain through a diaphragm of burlap, or other porous material. There is no reason why a Dorr thickener cannot be used. The only objection to the Dorr tank that the writer has heard, is that a heavy froth forms on the top of the tank, but this can be taken care of by building a high retaining ring inside the tank and by occasionally skimming off this froth, which in some cases consists of very high-grade fine flake.

Driers.—The type of drier commonly used is the direct-fired rotary. There are two disadvantages to a drier of this type: contamination by ash from the fuel, and the dust loss. The dust loss can be avoided by the use of dust collectors. In one instance, the installation of a dust collector saved 1,200 pounds of graphite a day. Indirect types of driers, such as the Lowden, or Ruggles-Cole, although the first cost is high, are probably the more satisfactory.

Finishing Plant.—The process of refining must be developed for the particular concentrate to be treated. The most important factors which will affect the treatment scheme to be used are such physical characteristics as, diameter, thickness, hardness, and toughness of the flake, and the nature of the impurities to be removed, whether soft or hard. Coarse, hard impurities are more difficult to handle and the amount of work the refinery should do on this class of product should be balanced against the amount of work which can be done effectively in the concentrating mill. A good deal of information has come to the writer's attention pertaining to different methods of refining, but the information obtained is not considered reliable enough to draw definite comparative conclusions. One large American company which has been in the graphite industry from the beginning, uses a system of batch ball mills and dry screening. A large, successful Canadian company uses burrstone mills and dry screening. Both claim they have tried other methods, and that their present practice has proven the best. The writer favours the use of the burrstone mill. Both the operation of the ball mill and the burrstones must be in the hands of experienced and capable men, as the personal element governs the efficiency of the operation in both cases. An experienced operator will recover 50 per cent more No. 1 flake than an inexperienced one.

The writer recommends that when considering a new installation for a finishing plant, a simple system of burrstones and screens of the Hummer type be tried at first. Any elaborations on this system can be developed gradually as experience with the particular ore shows the way. It will be found advisable to house the finishing plant and drier in a separate building from the concentrating mill, on account of the dust from the dry process of finishing.

THE ORES OF WESTERN QUEBEC—THEIR CHARACTER AND METALLURGICAL TREATMENT

W. B. Timm

The more recent developments in western Quebec have shown the ores of this district to be rather complex in character, requiring more extensive and more difficult metallurgical treatment than was at first expected. The first impression obtained from the initial discoveries, was that the ores of the extension of the Kirkland Lake and Porcupine belts in Quebec would be similar in character to the Kirkland Lake and Porcupine ores. Developments have since shown that although the ores along the southern margin of the belt are only slightly different, the ores of the more recent discoveries within the belt are vastly different and more complex in character, requiring a different metallurgical treatment.

CLASSIFICATION OF THE ORES

From a metallurgical viewpoint the ores may be classified into the following types:

Ores in which iron pyrite is the predominating sulphide. In this class of ore the pyrite is generally disseminated throughout the vein matrix and the gold values are closely associated with it. Small amounts of chalcopyrite may be present but as a rule to no great extent.

Ores in which arsenopyrite is the predominating sulphide. In this class of ore the gold occurs in the fractures of the quartz and is associated to a lesser extent with the arsenopyrite. Iron pyrite may also be present.

Ores in which pyrrhotite is the predominating sulphide. In this type is found a number of different minerals, depending on the degree of replacement of the pyrrhotite by chalcopyrite, pyrite, and zinc blende. On the same property ore-bodies are found ranging from massive pyrrhotite with very small amounts of chalcopyrite, to those in which the chalcopyrite predominates; others of disseminated ore, with chalcopyrite and pyrrhotite in their associated gangue rock; in some others zinc blende predominates. As a rule, the gold values are not associated to any extent with the pyrrhotite but with the other sulphide minerals, chiefly the chalcopyrite. It is this latter type of ore and its sub-types that are of prime importance, the abundance of which is the reason for the establishment of a smelting industry in the district.

EXAMPLES AND METALLURGICAL TREATMENT

The ores of the Arntfield group in Boischatel township and adjacent claims and those of the Huronian Belt Company's claims in Rouyn township, may be taken as examples of the first type, namely those in which iron pyrite is the chief sulphide mineral. This type of ore should present no very difficult metallurgical problem. If the gold values are consistent in being associated with the iron pyrite it would seem that amalgamation followed by close table concentration would be the method applied to their treatment. The concentrates could be reground and cyanided or shipped to a smelter. Should these ores contain copper values, table concentration could be supplemented by flotation and the entire concentrates shipped to a smelter. Should the gold values be to some extent associated with the gangue minerals the above process would have to be supplemented by cyanidation, providing there is remaining after concentration sufficient values to warrant this process.

The ores of the Edna Bathurst claim in Rouyn township, of the O'Brien mine in Cadillac township, of the E. J. Thompson group in Cadillac township, and of the Sisco mine in Dubuisson township, may be taken as examples of the second type in which arsenopyrite is the predominating sulphide. The gold occurs in blue quartz along the fractures, and to some extent associated with the arsenopyrite and pyrite. The recovery of values from this type of ore should not be difficult. The presence of so much free gold will make amalgamation a primary step in the process of treatment. Whether the ore will be concentrated to obtain an arsenopyrite concentrate before cyanidation or cyanided directly after amalgamation will depend on the market for arsenic, and on the effect of the presence of arsenopyrite on cyanide operations. Some arsenopyrite ores are amenable to cyanidation, others are more refractory.

As examples of the third general type in which pyrrhotite is the predominating sulphide mineral, may be cited those ores which are commanding so much attention at present. These include the ores of the Horne mine, Rouyn township; the Amulet, and the Waite-Montgomery mines, Duprat-Dufresnoy townships; the Alderson-MacKay and Chance groups, Boischatel township; the Abana group, Desmeloizes township; and a number of others. On some one or other of these claims may be found ores with the following characteristics:—

Heavy sulphide ores, ranging from massive pyrrhotite with little chalcopyrite to high-grade copper ore (chalcopyrite predominating over the pyrrhotite). Within this range the high gold values do not always follow the high copper values, but in many cases occur in the ores of the low or medium copper content.

Disseminated ores, where the minerals, chalcopyrite, pyrite, and pyrrhotite, are disseminated throughout their associated gangue rock.

Heavy sulphide ores, ranging from high-grade copper ore (chalcopyrite predominating over the pyrrhotite) with little or no sphalerite to high-grade zinc ores (sphalerite predominating over the pyrrhotite) with small amounts of chalcopyrite. Here again within this range, the higher gold values do not always occur with the rich copper ores.

Disseminated ores of chalcopyrite, sphalerite and pyrrhotite, these minerals being disseminated throughout their associated gangue rock.

From the above classification of the ores of the third general type it will be seen that their treatment will present to the metallurgist many difficult problems. Almost all the ore-bodies or deposits vary in some respect from one another in the character of their ore, so that each will require careful study to determine its amenability to the general scheme of treatment of the ores as a whole. The general scheme of treatment for the ores of this type will no doubt embody the erection of concentrating plants at the various mines with a custom smelter favourably located to receive ores and concentrates.

The concentrating ores will consist of the disseminated ores of the district, and ores of the heavy sulphide sub-types, low in copper, or containing both copper and zinc values requiring the separation of the minerals. These will constitute in time the greater bulk of the ore mined, and to meet this situation the smelter will be required to take a considerable tonnage in the form of flotation concentrates.

The concentrating ores may be summarized as follows:—

Disseminated ores of chalcopyrite and pyrrhotite with, perhaps, a certain amount of pyrite. These will require concentration to separate the chalcopyrite and pyrite, which as a rule carries considerable gold values, from the pyrrhotite and worthless gangue rock. This is a problem of gravity concentration and flotation, or vice versa, or simply straight flotation to recover the copper and gold values in a concentrate suitable for smelting operations. Certain of this class of ores containing a considerable excess of free silica are found in the district, and such ores will be in demand for converter flux, and are therefore more valuable in the raw state than in the concentrated form.

Disseminated ores of chalcopyrite, sphalerite, and pyrrhotite, with perhaps, a certain amount of pyrite. These will require concentration to separate the various constituents into three products, namely, a copper-gold concentrate of the chalcopyrite and pyrite, if it be present, a zinc concentrate, and a pyrrhotite-gangue tailing. This is a type of problem in respect to the solution of which selective flotation has made such rapid strides in the last few years, that there would seem to be no serious difficulty in applying it to this class of ores.

Heavy sulphide ores, chiefly pyrrhotite, low in chalcopyrite, and containing in some cases considerable pyrite and also considerable gold values, associated with the chalcopyrite and pyrite. In the case of ores of this class the economic point between the direct smelting and concentrating of the ores will need to be determined. This will probably lie in the vicinity of 4 per cent copper. Whether concentration is feasible will also depend on the recovery of the gold values. Where the ore is pyrrhotite and chalcopyrite, a separation of these two minerals can be made with the production of a copper concentrate carrying the gold values, with good recoveries in both copper and gold. Where a considerable amount of pyrite is present and this mineral carries a certain percentage of the gold, the separation is more difficult.

Heavy sulphide ores containing chalcopyrite, sphalerite, and pyrrhotite, with perhaps, a certain amount of pyrite. With ores of this class the determining factors between smelting and concentrating ores will be the zinc content in the copper ore, and the copper content in the zinc ore. Between these two points the ore will require concentration, and the method most likely used will be selective flotation making a copper concentrate, a zinc concentrate, and a pyrrhotite tailing. Small-scale tests have shown very favourable results in the concentration of this class of ore.

SMELTING OPERATIONS

The discovery of what would seem to be large tonnages of smelting ore, within a comparatively small radius, has decided those directly interested to establish a smelter in the district favourably situated to the principal deposits. Concessions with regard to townsite, smelter site, and smoke nuisance seem to have been definitely settled agreeably to the operators and the Provincial Government. The design of the smelter is in progress, if not complete. The railway from O'Brien, on the Canadian National railway, will be ready by the end of the year, when the equipment will be brought in and construction work started. The first unit is reported to be of 500 tons, although later reports, no doubt due to the favourable developments within recent months, have increased this to 1,000 tons daily capacity. The product of the smelter will be blister copper, which it is the intention to ship to the eastern states for refining, at least until refining works are established in eastern Canada near a source of cheap hydro-electric power. For the reduction of copper-gold ores and copper-gold concentrates, two methods of treatment may be successfully applied, namely blast furnace and reverberatory smelting. A careful study of the character of the ores, the percentage of coarse ore to fines and concentrates, the tonnage of each to be treated, available ore reserves, installation, and treatment costs, etc., are points to be considered in determining which of these two processes is the more economical method. It is very probable that both blast furnace and reverberatory smelting will eventually be used, the blast furnace being used for the heavy pyritic coarse ores and the reverberatory for the fines and products of the concentration plants, flue dust, etc. The blast furnace and reverberatory matte will be blown to blister copper in basic converters. Highly siliceous ores containing copper and an excess of free silica, such as the Chadbourne ore, will be used for converter flux.

The commencement of operations may only include a blast furnace plant, advantage being taken of the sulphur content in the heavy sulphide coarse ore to supply a great deal of the heat for reduction. Pyritic or semi-pyritic smelting may first be practised, using a minimum of coke, around five per cent, for reduction. Lime rock for fluxing purposes will have to be brought in from the most favourably situated localities with respect to freight rates, and its content of available free lime. Limestone, coke and coal will furnish incoming freight for the railway.

If the initial plant consists of blast furnace smelting, as great a burden of the heavy coarse sulphide ore as possible will be carried. The fines broken from mining operations, the concentrates from the concentration plants, and the flue dust will require sintering to agglomerate the fine particles together for blast furnace operation. The more of this sinter that is used in the charge, the more coke will be required for reduction. If reverberatory smelting is practised, the fines, concentrates and flue dust with the proper amount of flux necessary to give a free flowing slag, will be pre-roasted in furnaces of the McDougal type, and the calcines fed directly to the reverberatory fired with pulverized coal.

It would seem that some system of banking the various types of ores and concentrates will be necessary to obtain uniform operation. The smelter operator must know the composition of the various ores well ahead, so as to make up his charge and keep it uniform from day to day, otherwise, he will experience all kinds of trouble. This is of especial importance in blast furnace operations where a slight variation in the alumina-iron content of the ores causes trouble in the otherwise easy operation of the furnace. A custom's smelter for treating the ores and concentrates of the district will be called upon to purchase all types of ore. Some will contain fairly high zinc values and the percentage of this metal in the charge will have to be kept constant; others will contain arsenopyrite to some extent; all these points make the banking system necessary for most efficient operations.

Little so far has been reported on the establishment of reduction works for the treatment of zinc ores and concentrates. This is probably due to the fact that no very large bodies of zinc ore have been opened up. Recent discoveries, however, indicate that sphalerite will be one of the commercial minerals of the district and is present in varying amounts in a number of the deposits. In some cases it would seem to be in commercial amounts, necessitating its separation from the copper and its treatment in a zinc reduction works. It is not likely that zinc ore and concentrates will be treated in the district, probably they will be shipped out to works in the Eastern States or to Belgium, or to a source of cheap power on the St. Lawrence, or Ottawa rivers, or some other favourable point where the electrolytic process for the recovery of the metal in the form of spelter can be more successfully operated. Rumours have been current of the establishment of a zinc refining plant on the Saguenay in connexion with the power development there. If such works materialize it would offer a favourable point for disposal of this class of ore and concentrates from western Quebec. There is also a brisk demand for zinc ore and concentrates to supply the needs of the Belgian works, to which Canada shipped a large quantity of its zinc concentrates during 1924.

SUMMARY

It would appear that the vicinity of the important copper-gold discoveries in western Quebec gives promise to be the centre of a large metallurgical industry where the ores of the district will be reduced in smelting works, the establishment of which will assist materially in the opening up of the country. It will be an incentive to greater mining activity in providing the small operator with a smelter to which he can ship his high-grade ore while his property is in course of development. It will open up a market for farm produce to supply the needs of the mining communities and thus enhance agricultural development in the vicinity. Lumber will be required for construction and mining purposes. In order to save the pulpwood in the vicinity of smelter works, it will require to be cut as soon as possible. Although the character of the ores present difficult metallurgical problems they are not greater than have been surmounted in the past. One has only to review the history of such mining camps as Sudbury, Cobalt, Rossland, Kimberley and others to see evidence of the success made by the operating companies on ores which seemed equally as difficult of treatment, if not more so, than the Quebec ores. In each case the ores have been successfully treated, and moreover, in the case of the Quebec ores, much valuable information on the treatment of similar ores in other parts of the world is already available.