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SELECTIVE FLOTATION AS APPLIED TO CANADIAN ORES.

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Canada possesses many deposits of complex ores, such as lead-zinc-iron sulphides, copper-zinc-iron sulphides, copper-iron sulphides, and other combinations of these minerals, many of which contain values in both gold and silver.

The Mines Branch of the Department of Mines, through its Ore Dressing and Metallurgical Laboratories at Ottawa, has made numerous investigations into the treatment of such Canadian ores by selective flotation. The purpose of this review is to bring more directly to the attention of the mining public the information which has been acquired and the progress which has been made in the treatment of ores by this method, and also to point out the assistance the Government is rendering the mining industry by maintaining at Ottawa fully equipped ore dressing and metallurgical laboratories for research on the treatment of ores and metallurgical products. A brief description of the flotation equipment in these laboratories is given in order that the reader may better realize the type of work which can be performed.

The small-scale laboratory equipment consists of a 1000-gramme Ruth flotation machine, a Callow pneumatic batch unit, a Janney machine and a continuous fine-grinding flotation unit which will treat 20 pounds of ore an hour grinding 200 mesh. The large-scale tonnage equipment consists of two different types of grinding mills, viz. one 4½-foot Ha rdinge conical mill and one 6 by 3-foot Marcy rod mill. Both can be operated in closed circuit with a standard Dorr classifier having an interchangeable bowl attachment. Flotation equipment comprises a Greenawalt 8-cell flotation machine and a Callow pneumatic flat-bottom unit with a capacity of one ton per hour. Tables, classifiers, and screens can be cut in as desired, and continuous flow sheets can be built up by using various combinations of these machines. The addition of a unit which will treat 200-pounds of ore an hour is contemplated.

Recent Flotation Practice: The tonnage of complex ores separated by flotation has rapidly increased. The use of chemical reagents to modify the surface of the ore particles, the

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C 212

discovery of soluble chemical flotation reagents, and the economic solution of the fine-grinding problem are the principal factors responsible for the increase. The remarkably successful operation of a number of all-flotation custom plants in treating complex ores is unquestionably due to their efficient research organizations.

A general effort is being made to raise the grade of flotation concentrates and to determine for each mill the point at which the greatest profit can be made, when the lowered freight and smelter charges on the concentrates are compared with the extra cost of concentration and greater loss in tailing. A careful study of this question has shown in a number of cases that it would pay to regrind either the flotation concentrate or middling, as the case may be, and subject them to a second flotation. It is quite obvious that a flotation concentrate for the most part already ground to minus 200 mesh is difficult to regrind unless it is carried as an extremely thick pulp to the grinding mill and a high circulating load maintained in the classifier-ball mill circuit, consequently concentrate or middling to be ground generally goes to a desliming classifier of the drag type; sands from this classifier go direct to the mill, while the overflow goes to a thickener, from which the thickened discharge also goes to the regrinding mill. An alternative arrangement, and one that has proved satisfactory, is the use of the Genter thickener, in which case the product to be ground goes direct to the thickener. The mill, as a rule, is operated in closed circuit with a bowl classifier, and very high circulating loads, up to 1000 per cent, are recommended.

The types of flotation machines are numerous, but when compared to a few well-known types of standard machines, the majority have little merit. In the flotation of lead-zinc ores practice seems to favour the use of sub-aeration machines for lead and standard spitz type machines for zinc. The pneumatic type machine has come very much to the fore with the development of a new type of cell known as the MacIntosh, which uses a revolving blanket. Blanket troubles caused by banking and clogging are eliminated by this type of construction. The cell is especially adaptable for selective separations.

A number of new flotation reagents of great merit have been recently developed. Cresylic acid reconstructed with phosphorous pentasulphide is a new reagent which is highly selective in the presence of sulphides of iron and can be used on copper-zinc or lead-zinc ores. "Flotagen S" and "Flotagen" are two other new reagents, the former functions with cyanide, sodium sulphite, and also with sodium thio-sulphate. Flotagen, which should be added as a solid, can be used only in alkaline pulps and requires sufficient agitation before flotation to allow it to go into solution, although violent mechanical agitation is not necessary. Flotagen S, on the other hand, goes into solution readily and should not be in contact with the ore pulp for more than one minute before flotation. It is not advisable to allow Flotagen to remain in contact any longer than necessary, as there is often a marked decrease in efficiency if the time is prolonged. The use of Xanthate, phospho-cresylic acid, and other soluble flotation agents has been found to have a beneficial effect on the filtration of the flotation concentrate. By the substitution of these reagents for oils it has been found, in some cases, that the tonnage handled by the filters can be doubled.

The length of time that the modifying reagent is allowed to remain in contact with the ore before flotation is a most important factor and until recently has been more or less disregarded. The general practice has been to take care of the time element by adding the reagents to the ball mill. This was effective in most cases but took care only of the flotation of the first mineral. It has been found advisable to provide time contact for certain reagents used for the flotation of the second mineral. Improved results have been obtained and the consumption of reagents lessened by increasing the time of contact before the flotation of the first mineral by the addition of conditioning tanks equipped with agitating devices. Present practice uses up to $2\frac{1}{2}$ hours contact before the lead separation and up to one hour between the lead and zinc separations. The time factor of course, depends entirely on the ore being treated, and some ores require only a few minutes. The claim that violent agitation is necessary during the time contact is, in the writer's opinion, a very mistaken assertion.

It is the experience of those in charge of concentrators for the flotation of heavy sulphide ores that to obtain maximum recovery the ore must be freshly broken, and that only a small supply of broken ore can be carried in the stopes, as the slightest oxidation interferes with the flotation, and heavy losses result. This applies particularly to the taking of samples which are to be used for experimental work. The author has frequently found it necessary to request that new samples be taken and shipped from freshly mined ore because of the difficulty in obtaining any separation on previous samples sent. The second samples, taken as requested, responded readily to selective flotation.

Oxidation may also take place in the grinding mills when heavy sulphides are being treated. Serious difficulty has been experienced in some mills with this problem. The condition is undoubtedly caused by the heat generated at the point of contact by the impact of the balls striking the lumps of sulphide ore. The action can be overcome by increasing the classifier capacity and building up a high circulating load to the mill so as to keep the pulp in the mill only a very short time; by the addition of modifying reagents in the proper amount, or by carrying a more dilute pulp in the grinding mill at the sacrifice of capacity.

There are two different methods of applying selective flotation to an ore: The first, and the one which is more commonly used, is a straight flotation separation made on the original pulp. The second method is by bulk flotation of the two or more minerals and the subsequent separation of them by selective flotation. Some misunderstanding seems to exist regarding the use of these two methods, and attempts are frequently made to compare their respective merits without taking into consideration the particular ore on which they are to be used. Although each method has a distinct field of its own, there is generally little room left for any argument as to which method should be used, provided that sufficient information on the character and behaviour of the ore has been obtained by previous experimental work. An ore containing 2.25 per cent lead, and 6 per cent zinc, providing there are no interfering soluble salts present, calls for a straight selective float. But if the lead was around 1 per cent or less, a bulk float of both the lead and zinc with a selective separation on the bulk concentrate would probably be the better method. In the latter case, if possible, only a part of the zinc should be cut with the lead. This would be possible, in the majority of cases, because of the

better floating properties of the lead, and by not adding the copper sulphate reagent used to promote the flotation of zinc until after the lead had been floated. By this method the tonnage of bulk concentrate to be treated in the time contact tanks would be much less than if the whole of the zinc had been floated. The ratio of lead to zinc would also be higher which would assist the separation to be obtained between the lead and zinc in the subsequent selective separation. In an ore containing soluble copper salts, a bulk flotation of the zinc and lead is practically always necessary. The reason for this is the amount of modifying reagents necessary to overcome the promoter action of the copper salts on the zinc.

In the general practice of bulk flotation the zinc concentrate is obtained as a tailing from the flotation of the lead. In the case of an ore containing a large proportion of pyrites, considerable trouble may be expected from the iron which will tend to float in the bulk concentrate. As a rule it is found impossible to reactivate the zinc mineral after the bulk concentrate has been subject to the time contact treatment with the reagents required to effect the separation. The iron which had floated would remain with the zinc preventing possibly a shipping grade of zinc concentrate being produced.

Heavy sulphide copper ores frequently contain small amounts of zinc. Such an ore having an approximate analysis of copper 1.3, zinc 0.6, and iron 20 per cent or higher, and also containing considerable soluble sulphates, could be concentrated for the copper content alone, the zinc being discarded. The copper concentrate, however, would probably carry 5 per cent zinc. This amount would be objectionable to the smelter. The reagent costs would be too high owing to the presence of soluble salts to attempt to retard the zinc by the ordinary methods. The flotation of a bulk concentrate containing the copper, zinc, and part of the iron would in all probability give a product which it would pay to work on for the production of a marketable zinc concentrate.

On the other hand, a heavy sulphide ore containing copper 2, and zinc 6 per cent lends itself much better to a straight selective flotation. Another point worth drawing attention to is that should the proportion of zinc in the above ore be raised to 10 per cent, the copper concentrate would carry a much greater amount of zinc and the final recovery of zinc might be lower than from the first example.

Some ores contain soluble salts, such as sulphates of iron, alumina, magnesium, copper, etc., in such quantities that even the addition of large amounts of soda ash or lime will not prevent the modifying reagents being destroyed as quickly as they are added, thus preventing them from fulfilling their function. There are in general two ways of attacking such a problem: (1) to endeavour to make a bulk concentrate with the idea of subsequently making the selective separation on the bulk concentrate, and, (2) to wash the ore by crushing and partly dewatering and again bringing the pulp to the required density by the addition of fresh alkaline water. Ores of the above characteristic present a research problem which will tax the ingenuity of the metallurgist to solve. Fortunately mining companies which have been successful in overcoming such difficulties are showing a very fair attitude by publishing their methods.

Owing to separate problems involved in the work from which the information contained in this article has been obtained, the method of

presentation will be confined to giving examples of the separations obtained on different types of ore, and the methods and reagents used to obtain them.

Modifying Reagents: In order that the reader may better understand the methods used in obtaining these separations, a brief description of the uses of some of the more common modifying reagents will be given. For a description of the properties of oils and flotation reagents, see Memorandum Series No. 11, "Selective flotation as applied to Canadian Ores".

Lime - This reagent was one of the first to be used in flotation. It requires a time contact with the ore and, if possible, should be added in the grinding mills. It is used to overcome the deleterious effect of soluble salts which are frequently present in ores, but its principal use is as a depressant for iron sulphides in the selective separation of lead and zinc, copper and zinc, and copper and iron sulphides. It also has a marked depressing action on zinc sulphides in their separation from lead and copper. If added in too large amounts it will also prevent the lead from floating. Chalcopyrite will float in a strongly alkaline pulp, but chalcocite and some of the other copper minerals are affected by the least excess. Therefore, lime should not be used in their flotation. As a general rule, much better results can be obtained in the flotation of these minerals with a combination of soda ash and cyanide than with lime.

Soda ash (sodium carbonate) - This is used for much the same purposes as lime, but its action is much less severe. Its action as a depressant on sulphides of iron is not great, but its great value in flotation arises from its property of increasing the differences between the relative floating properties of two or more minerals, such as zinc and iron sulphides, so that a sharper separation between the two minerals can be obtained.

Sodium bicarbonate - The use of this reagent is preferred in the selective separation of lead and zinc when the ore contains only a minor amount of pyrite or pyrrhotite.

Sodium cyanide - This is one of the most valuable of all modifying reagents, since by its action the surface of the zinc sulphides and iron sulphides are modified so that they do not float. By taking advantage of this action, lead and copper sulphides may be separated from them. As a rule, cyanide is used with either soda ash or lime, generally with soda ash. It is frequently used in combination with zinc sulphate, in which case the solutions of the two salts are brought together just as they enter the flotation pulp so as to form an emulsion of zinc cyanide. Time contact is important when using cyanide, some ores requiring one hour contact, others only a few minutes contact.

Sodium sulphite and thiosulphate - These two salts have similar properties. Sometimes one is the more effective, sometimes the other, depending entirely on the ore. They are used for the same purpose as cyanide and in a number of cases have been found just as effective.

Zinc sulphate - is used as a depressant for zinc in separations of lead and zinc. It is possible that its use might be extended to the separation of copper and zinc, but the writer has never had occasion to try it.

Sodium acid phosphate - is used occasionally as a depressant for zinc in lead and zinc separations.

Sodium hydrosulphite - This can be used also as a depressant for zinc.

Sodium dichromate - This salt is used in the selective separation of lead and zinc with cyanide, on ores which contain large amounts of iron sulphides. It seems to have a beneficial action. It is seldom necessary.

Copper sulphate - is added in quantities of one half to two pounds per ton, as a decided promotor for the flotation of zinc.

Ores consisting of complex mixtures of galena,
sphalerite, pyrite and pyrrhotite, containing
gold and silver.

The problems involved in the treatment of such ores are varied although the recovery of the gold and silver is, in most cases, the major problem. In connection with the recovery of silver some very interesting results were obtained on an ore received from the Slocan district, B.C., where the recovery of silver is the major problem. The particular sample was sent in from a mill which was recovering only 50 per cent of the silver in the feed, by a combination of gravity separation and flotation. The typical flow sheet of the district includes jigs and tables for concentrating the lead. The tailings containing the zinc and a large proportion of the silver are dewatered, reground, and floated, the products being a low-grade lead product, high in silver, and a high grade zinc concentrate.

In order to determine the association of the silver, pieces of galena, blonde, and gangue were picked out and assayed for silver. The pieces were approximately 3/8 inch in size. The results were as follows:

	<u>Assay silver</u>
Galena, crystallized in large cubes	100.7 ozs/ton
Galena, finely disseminated, aggregates of crystals	102.0 "
Zinc blonde Pb 0.16, Zn 63.30 per cent	7.5 "
Gangue Pb trace Zn 1.66 per cent	1.18 "

This shows conclusively that in place the silver, in some form, is associated almost exclusively with the galena. Samples of the ore were crushed to about 10 mesh and a lead concentrate made by tabling. The concentrate contained 70 per cent lead but only 67 ozs/ton silver, as against 100 ozs/ton in the coarse unbroken pieces of pure galena. Other tests showed that the finer the galena was crushed the less silver the lead concentrate contained. It was obvious, therefore, that the silver was being freed by crushing. This was confirmed by the results of the flotation tests, as it was found that the largest proportion of the silver generally reported in the zinc concentrate. There was abundant evidence to show that the silver which was freed from the galena by crushing was in an extremely fine form. This was further borne out by some work on the products from one of the operating combined gravity and flotation mills which showed that 75 per cent of the silver in the zinc flotation concentrate was in the minus 200 mesh material, and that the losses in the tailing were due to very finely divided silver mineral.

It follows that owing to the finely divided state of a part of the silver values, high recovery could not be expected in a mill using gravity concentrators, nor would a high recovery of silver be obtained by the use of jigs or tables. It would also follow that a large amount

of the fine silver would be lost when the jig and table tailings were dewatered preparatory to regrinding for flotation of the zinc. The fact that the tailings from the flotation department treating the reground tailings and fines from the jigs and tables contain high values in silver shows that the fine silver is very hard to collect in the flotation concentrate. This is in accordance with personal experience of the writer who has observed in a number of cases that very fine particles of minerals by themselves are very difficult to oil and collect in the froth. It had also been observed that in a flotation pulp containing relatively coarse grains of minerals, the fine grains exhibited a marked tendency to cluster and cling around the larger ones. This characteristic suggested that in order to collect and recover the finely-divided silver values a greater proportion of large grains must be present so that by the bulk action of the flotation of the coarse grains, the fine grains would be gathered in and collected.

For these reasons separation of the lead and zinc from the original mill pulp by selective flotation was regarded as a solution of the problem. Later, after a method of separating the lead and zinc had been worked out it was found that very high recoveries of the silver were obtained, the largest proportion reporting in the zinc concentrate.

In general, the selective separation of lead and zinc is obtained by the addition to the ore pulp of some reagent that will temporarily deaden the floating properties of the zinc, thus permitting the recovery of a high-grade lead concentrate. In the case of the majority of Canadian ores, the separations are made more difficult by the presence of large amounts of iron sulphides, both pyrite and pyrrhotite, and some reagent must be used which will permanently deaden these two sulphides so that they will not float with either the lead or zinc. The only reagents used which perform both these functions are sodium cyanide, sodium sulphite, and sodium thiosulphate, but the following have been used with some success in temporarily deadening the zinc: zinc sulphate, sodium acid phosphate, and sodium hydrosulphite. The action of zinc sulphate is much more effective when used with sodium cyanide, the two being mixed by bringing the streams of the two reagents together just as they enter the flotation pulp. So far as has been observed by the writer, these reagents when used separately have no effect in preventing the iron sulphides from floating.

These modifying reagents are used in an alkaline pulp, either with soda ash or lime. Lime has a permanent deadening effect on pyrite and pyrrhotite, and also affects galena in the same way but to a lesser degree. Cases have been reported of sodium sulphide being used to effect a separation between lead and zinc, but, with the exception of one instance, no success has been obtained with it in our laboratories. By the use of copper sulphate, the action of these modifying reagents is destroyed, and the flotation properties of the zinc minerals is revived. The length of time the modifying reagent is in contact with the ore is important. As a general rule it may be stated that the effect of a small amount in contact for a longer period approaches the effect of a larger amount in contact for a shorter period. Mills are now erected in which surge tanks are used to increase the time the modifying reagents are in contact with the ore.

The use of pilot tables for indicating the work the flotation machines are doing is recommended wherever possible. If the lead and zinc separations are not perfect enough the flotation concentrate may

be tabled without causing undue losses. The action of flotation coagulates the fine mineral grains so that they settle much more rapidly and there is not the same tendency for the fine mineral to be lost in the overflow of the thickeners.

Examples of selective separation on lead-zinc ores

1. The ore consisted of galena and sphalerite with silver, the sulphides being disseminated throughout the gangue, requiring fine grinding. Analysis:

Lead	7.10 per cent
Zinc	14.06 "
Silver	3.04 ozs/ton
Gold	trace

The ore was ground to 50 mesh for flotation. Small scale tests only were made.

Product	Weight %	Assays			Per cent of values		
		Pb %	Zn %	Ag oz.	Pb	Zn	Ag
Lead conc.	13.6	48.90	8.45	9.06	77.0	8.5	45.6
Zinc conc.	24.5	6.25	46.95	5.30	17.7	84.7	47.9
Tailing	61.9	0.75	1.49	0.29	5.3	6.8	6.6

Reagents used:

Soda ash	4.0 lbs/ton	Added and ground with ore
Cyanide	0.5 "	" " "
Thiocarbamide	0.2 "	" " "
Zinc sulphate	0.5 "	" " "
Pine oil	0.1 "	Added to lead coll
Copper sulphate	1.0 "	Added to zinc coll
Xanthate	0.3 "	" " "
Pine oil	0.1 "	" " "

Both cyanide and zinc sulphate were used to modify the flotation properties of the zinc and pyrite. On this particular ore, the addition of zinc sulphate gave a sharper separation between the lead and zinc than was obtained with cyanide alone. The length of time the ore was kept in contact with the cyanide was found to be an important factor. A mill is under construction based on the results of these tests.

2. The ore was a heavy sulphide type containing gold, silver, galena, sphalerite, pyrite, and pyrrhotite, with very little gangue. It required grinding to approximately 65 mesh for flotation. Analysis:

Copper	0.73 per cent
Lead	18.47 "
Zinc	23.54 "
Gold	0.42 oz/ton
Silver	19.42 "

Small scale tests only were made -

Product	Weight %	Assays				Per cent of values			
		Pb %	Zn %	Au oz	Ag oz	Pb	Zn	Au	Ag
Lead conc.	32.7	52.31	10.97	0.80	46.3	93.0	13.2	63.9	81.3
Zinc conc.	41.1	2.31	47.69	0.16	5.74	5.2	83.3	16.1	12.7
Table conc.	5.2			0.90	9.80			11.4	2.7
Tailing	21.0	1.25	1.25	0.17	2.93	1.8	1.4	8.6	3.3

Reagents used:

Soda ash	6.0 lbs/ton	Added and ground with ore
Thiocarbanilide	0.2 "	" " " "
Sodium cyanide	0.3 "	Five minutes contact
Cresylic acid	0.2 "	Added to lead cells
Copper sulphate	1.25 "	Added to zinc cells
Xanthate	0.4 "	" " "
Pine oil	0.1 "	" " "

A mill, based on the results of these tests, has been built and is in successful operation.

3. The following separation was obtained on dump middlings consisting of sphalerite, pyrite, and pyrrhotite, and only 2 to 3 per cent gangue. Analysis:

Lead	2.45 per cent
Zinc	23.00 "

The following are results of a 5-ton test:

Product	Assays		Approx. recoveries	
	Pb %	Zn %	Pb	Zn
Lead concentrate	16.50	12.40	85 %	
Zinc concentrate	0.30	51.62		91 %
Tailing	0.45	0.85		

Reagents used:

Soda ash	18.0 lbs/ton	Added and ground with ore
Cyanide	0.26 "	Added to lead cells
Acid coal tar creosote	0.21 "	Added and ground with ore
Water gas tar	0.21 "	" " " "
Copper sulphate	2.0 "	Added to zinc cells
Xanthate	0.4 "	" " "

A mill, based on these results, has been built and is in operation.

4. The sulphides in this ore consisted of sphalerite, galena, pyrrhotite and a little pyrite and chalcopyrite, also values in gold and silver. Analysis:

Lead	1.96 per cent
Zinc	5.52 "
Iron	7.42 "
Gold	0.02 oz/ton
Silver	3.28 "

Results of a 5-ton test -

Product	Assays					Approx. recoveries	
	Zn %	Pb %	Fe %	Au oz	Ag oz	Pb	Zn
Lead conc.	4.17	51.09	2.60	0.30	40.0	89.4 %	
Zinc conc.	47.18	1.15			9.0		94.8 %
Tailing	0.30	0.15	9.4				

Reagents used:

Soda ash	2.0 lbs/ton	Added and ground with ore
Acid creosote	0.2	" " " "
Water gas tar	0.2	" " " "
Sodium cyanide	0.15	Added to lead cells
Cresylic acid	0.4	" " " "
Copper sulphate	1.0	Added to zinc cells
No.2 neutral creosote	0.4	" " " "
TT mixture	0.2	" " " "
Pine oil No.5	0.2	" " " "

5. 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 -

Copper	2.35 per cent
Arsenic	0.76 "
Lead	1.26 "
Zinc	14.25 "
Gold	0.03 oz/ton
Silver	3.91 "
Antimony	trace

Results of small scale test -

Product	Weight %	Assays					Per cent of values					
		Cu %	Pb %	Zn %	Au oz	Ag oz	Cu	Pb	Zn	Au	Ag	
Cu-Pb conc.	17.0	12.32	6.62	13.86	0.08	14.76	90.3	93.3	16.3	41.6	70.0	
Zinc conc.	23.0	0.56	0.25	49.6	0.04	3.0	5.5	4.2	80.5	29.1	19.6	
Tailing	60.0	0.16	0.05	0.7	0.015	0.56	4.1	2.5	3.0	28.3	9.5	

Reagents used:

Soda ash	10.0 lbs/ton	Added and ground with ore
Thiocarbunilide	0.25	" " " "
Sodium cyanide	0.24	" " " "
Pine oil	0.10	Added to lead cells
Copper sulphate	1.5	Added to zinc cells
Xanthate	0.3	" " " "
Pine oil	0.1	" " " "

6. This sample consisted of material taken from a tailing dump produced from the concentration of a lead-zinc ore. Analysis -

Lead	0.07 per cent
Zinc	10.06 "

Results of 4 ton test

Product	Assays			Approx. recoveries	
	Pb %	Zn %	Cu %	Pb	Zn
Lead conc.	22.31	10.41	4.23	74 %	
Zinc conc.	0.14	49.09			85 %
Tailing	0.06	0.40			

Reagents used:

Soda ash	2.0 lbs/ton	Added and ground with ore
Acid coal tar crocosote	0.3 "	" " "
Sodium cyanide	0.09 "	Added to lead cells
Cresylic acid	0.6 "	" " "
Copper sulphate	1.24 "	Added to zinc cells
Xanthate	0.25 "	" " "
Pine oil	0.25 "	" " "

7. This sample was a lead-zinc ore carrying silver, the galena and sphalerite were intimately disseminated with pyrite and pyrrhotite.
Analysis -

Lead	3.35 per cent
Zinc	7.08 "
Iron	4.82 "
Silver	9.5 ozs/ton

Results of small scale test:

Product	Weight %	Assays			Per cent of values		
		Pb %	Zn %	Ag oz.	Pb	Zn	Ag
Lead conc.	6.0	46.53	2.74	134.0	80.9	2.2	72.2
Lead midd.	5.9	9.02	6.88	28.5	15.5	5.6	15.2
Zinc conc.	10.9	0.32	55.32	4.5	1.0	82.7	4.4
Zinc midd.	3.5	0.80	15.16	7.44	0.8	7.4	2.4
Tailing	73.7	0.08	0.21	0.88	1.7	2.1	5.8

Reagents used -

Soda ash	5.0 lbs/ton	Added and ground with ore
Thiocarbamilide	0.2 "	" " "
Sodium cyanide	0.24 "	Added to lead cells
Cresylic acid	0.75 "	" " "
Copper sulphate	1.0 "	Added to zinc cells
No.2 neutral crocosote	0.5 "	" " "
Xanthate	0.1 "	" " "

Ores consisting of intimate mixture of chalcopryrite and pyrite and pyrrhotite, with and without appreciable amounts of gold

The successful separation of chalcopryrite from the iron sulphides, both pyrrhotite and pyrite, requires the solution of two problems. The first problem is to free the copper mineral from the iron. Fine grinding is nearly always necessary owing to the intimate association of chalcopryrite with the pyrite. The development within the last few years in the art of fine grinding makes it possible to free economical the two minerals in the finest grained ores. The second problem is to prevent the iron from floating with the copper. In the case of chalcopryrite ore containing no gold, this can generally be done by maintaining a strongly alkaline pulp with lime. Lime has a permanent deadening effect on pyrite and pyrrhotite and very high ratios of concentration can be obtained.

As a general rule, copper ores containing gold present an entirely

different problem, that of recovering the gold in the copper concentrate. The use of lime to retard the iron, except in rare instances, tends to throw the greater part of the gold with the iron tailing. Soda ash or caustic soda is used because it has not the detrimental effect on the gold, but unfortunately it does not deaden the pyrite. Its effect is sufficient to keep the pyrrhotite from floating, but pyrite floats readily. Soda disperses or deflocculates the gangue slimes whereas lime coagulates them. However, in the case of soda, there also seems to be a tendency to coagulate the sulphides.

Sodium cyanide added to a pulp, the alkalinity of which is maintained with soda ash can be used to permanently deaden at least a part of the pyrite, but it also has a modifying effect in some cases on the chalcopryite as well. The chalcopryite can be reactivated by the addition of a limited amount of copper sulphate. By the use of cyanide under these conditions, it is often possible to obtain a higher ratio of concentration without lowering the recovery of the gold.

Examples of selective separations on copper-iron ores, with and without gold values

1. This ore was a complex sulphide consisting of chalcopryite intimately associated with pyrite. The chalcopryite was disseminated throughout the pyrite, the sulphides constituting 80 per cent of the ore. The ore was ground to pass 100 mesh, 75 per cent being -200 mesh.

Analysis -

Copper	3.0 per cent
Iron	40.0 "
Incoluble	7.5 "
Other gangue	4.5 "
Sulphur	45.0 "

Results of tonnage scale test -

Copper concentrate	22.77 per cent copper
Tailing	0.12 " " "
Recovery	96 per cent

Reagents used -

Lime	10 lbs/ton	Added to ball mill
Barretts No. 634	0.6 "	" " "
TT mixture	0.25 "	Added to cells

2. The ore was a copper-gold ore from the Rouyn district, Quebec.

Analysis -

Copper	4.60 per cent
Iron	19.83 "
Silica	50.64 "
Gold	0.04 oz/ton
Silver	0.22 "

Small scale tests only -

Product	Weight %	Assays		Per cent of values	
		Cu %	Au oz	Cu	Au
Copper conc.	35.3	12.14	0.11	95.0	81.1
Copper midd.	8.5	1.39	0.04	2.6	7.1
Tailing	56.2	0.19	0.01	2.4	11.8

Reagents used -

Soda ash	4 lbs/ton	Added to ball mill
Xanthate	0.07 "	Added to cells
Pine oil	0.05 "	" " "

No. 3. This is a copper-gold ore from the Rouyn district, Quebec.

Analysis -

Copper	1.6 per cent
Gold	0.28 oz/ton

Small scale tests only -

Test	Product	Weight %	Assays		Per cent of values	
			Cu %	Au oz	Cu	Au
A	Copper conc.	31.4	4.77	0.76	90.4	85.2
	Tailing	69.1	0.11	0.06	9.6	14.8
B	Copper conc.	22.9	6.50	0.98	94.1	80.06
	Tailing	77.1	0.12	0.07	5.9	19.94

Reagents used -

A	Soda ash	6.0 lbs/ton	Added to ball mill
	No.4 creosote (Barretts)	0.4 "	" " "
	Cresylic acid	0.4 "	Added to cells
B	Soda ash	6.0 "	Added to ball mill
	Cyanide	0.2 "	" " "
	No. 4 creosote	0.2 "	" " "
	Water gas tar	0.05 "	" " "
	Xanthate	0.05 "	Added to cells
	Copper sulphate	0.4 "	" " "

Note: The use of copper sulphate in test B was found necessary in order to reactivate the chalcoprite which had been affected by cyanide.

Ores consisting of intimate mixtures of chalcoprite, sphalerite and iron sulphides.

This type of ore probably presents the most difficult of all selective flotation problems. The use of cyanide to temporarily deaden the zinc and to permanently deaden the iron sulphides seems essential. However, separations have been obtained in our laboratory by floating the copper in alkaline pulp by the addition of lime. For the flotation of the zinc the copper tailing was dewatered to eliminate a part of the lime and the pulp then brought up to the required density by addition of fresh water. At this point soda ash was added and the zinc floated in a soda ash pulp, copper sulphate being used to help reactivate the zinc.

Examples of selective separation of copper and zinc from pyrite and pyrrhotite gangues

1. This was a copper-zinc ore from the Rouyn district, Quebec. The sample contained sphalerite and chalcoprite together with a little gold and silver associated in a heavy sulphide gangue. The copper and zinc sulphides were finely disseminated and fine grinding was required.

Analysis -

Copper	4.15 per cent
Zinc	10.0 "
Gold	0.18 oz/ton
Silver	3.02 "

Small scale tests only:

Product	Weight %	Assays				Per cent of values			
		Cu %	Zn %	Au oz	Ag oz	Cu	Zn	Au	Ag
Copper conc.	17.3	20.66	6.91	0.56	11.96	85.5	11.8	50.3	65.7
Zinc conc.	14.6	1.96	48.57	0.32	1.39	6.9	69.8	24.3	6.4
Zinc middling	8.1	1.30	11.37	0.30	3.80	2.6	9.1	12.7	9.8
Tailing	61.0	0.35	1.56	0.04	0.93	5.0	9.3	12.7	18.1

Reagents used -

Soda ash	10.0 lbs/ton	Added to ball mill
Cyanide	0.5 "	" " "
Thiocarbanilide	0.2 "	" " "
Cresylic acid	0.2 "	Added to copper cells
Copper sulphate	1.0 "	Added to zinc cells
Xanthate	0.3 "	" " "

2. This was a copper-zinc ore from the Rouyn district, Quebec. The sample contained sphalerite and chalcopryrite together with a little gold and silver in a heavy sulphide gangue. The copper and zinc sulphides were finely disseminated and fine grinding was required to free them. The erection of a mill is contemplated in the near future on the basis of the results of these tests. Analysis -

Copper	7.36 per cent
Zinc	6.53 "
Lead	trace
Iron	35.40 "
Insoluble	18.6 "
Gold	0.03 oz/ton
Silver	2.45 "

Small scale tests only -

Product	Weight %	Assays				Per cent of values			
		Cu %	Zn %	Au oz	Ag oz	Cu	Zn	Au	Ag
Copper conc.	28.0	23.74	3.68	0.80	7.04	93.0	16.1	75.4	76.0
Zinc conc.	9.8	1.22	47.38	0.01	0.81	1.7	72.9	3.4	3.1
Zinc midd.	9.7	1.07	3.68	0.01	0.61	1.4	5.6	3.4	2.3
Tailing	52.5	0.53	0.65	0.01	0.92	3.9	5.4	17.8	18.6

Reagents -

Soda ash	4.0 lbs/ton	Added to ball mill
Sodium cyanide	0.3 "	" " "
Thiocarbanilide	0.2 "	" " "
Cresylic acid	0.08 "	" " "
Copper sulphate	0.4 "	Added to copper cells
Copper sulphate	1.5 "	Added to zinc cells
Xanthate	0.3 "	" " "
Pine oil No. 5	0.08 "	" " "

3. The ore contained Gold 0.099 and silver 1.36 ozs/ton: copper 1.89, zinc 4.61, lead 0.45 per cent. The ore was ground to 95 per cent minus 200 mesh for the test work. A large amount of research work has been carried out on this ore by different people. The best results obtained so far are with the following methods, two of which will be described:

Method No.1 - Lime, 3 lbs/ton was added to the ball mill while grinding. Agitated with 1 lb/ton salt for 3 minutes. Floated copper with 0.12 lb/ton Xanthate and 1 drop pine oil. Reclean concentrate with $\frac{1}{2}$ lb/ton salt, 0.02 lb/ton xanthate, and 1 drop pine oil. Agitated with 1 lb/ton lime and $\frac{1}{4}$ lb/ton copper sulphate. Floated zinc with 0.12 lb/ton xanthate and 1 drop cresylic acid. Recleaned concentrate with 0.1 lb/ton copper sulphate, 1 lb/ton lime, and 0.02 lb/ton xanthate.

Method No. 2 - Agitated with 3 lb/ton soda ash and 1 lb/ton sodium sulphite and $\frac{1}{2}$ lb/ton sodium cyanide. Floated copper with 0.12 lb/ton xanthate and 1 drop pine oil. Recleaned with 1 lb/ton soda ash, $\frac{1}{2}$ lb/ton sodium sulphite and 0.01 to 0.02 lb/ton xanthate. Agitated with 1 lb/ton soda ash, $\frac{1}{4}$ lb/ton copper sulphate and 1 lb/ton sodium sulphite. Floated zinc with 0.12 lb/ton xanthate and 1 drop cresylic acid. Recleaned with 1 lb/ton soda ash, $\frac{1}{2}$ lb/ton sodium sulphite, 0.01 lb/ton xanthate, and 1 drop pine oil.

The following results were obtained from the first method, and in order to approximate continuous operation a series test was made. The copper was floated as a rougher concentrate which was recleaned, the tailing from the cleaning being added to another batch of ore before this batch was floated for the copper. The tailing from the copper flotation was floated for zinc, and a rougher zinc concentrate made. This was recleaned and the tailing from recleaning added to the copper tailing from the second copper flotation. The test was run continuously on 15 charges of ore, 1500 grams to the charge. The results follow -

	<u>Copper concentrate</u>			<u>Zinc concentrate</u>			<u>Tailing</u>	
	<u>Assay</u>	<u>% metal</u>	<u>ratio of contained concentrn.</u>	<u>Assay</u>	<u>% metal</u>	<u>ratio of contained concentrn.</u>	<u>Assay</u>	<u>% metal</u>
Cu	17.0	83.0	10.85	1.1	5.05	11.3	0.28	11.95
Zn	3.0	5.97		43.5	83.2		0.6	10.83
Au	0.53	48.5		0.03	26.4		0.06	25.10
Ag	4.90	33.1		1.6	10.4		0.94	56.5
Pb	4.4	90.0		0.0	0.0		0.05	10.0

Example of separation of chalcopyrite and galena

These two minerals are being successfully separated in actual mill operation. Previous to the operation of the selective separation, the two minerals were collected in the one concentrate and shipped to the smelter. The operation of the selective separation to produce separate lead and copper concentrates has increased the net returns per ton of crude by the amount of \$4.65 per ton.

The separation is effected by floating a bulk concentrate containing the lead and copper. This concentrate is subjected to a time contact with the required modifying reagents until sufficient action has taken place to prevent the copper from floating. The pulp is then floated to produce a lead concentrate, a copper concentrate being