FLOTATION & CONCENTRATION OF AURIFEROUS ORES OF N. CANADA

THE FLOTATION

and

CONCENTRATION OF CERTAIN COMPLEX

AURIFEROUS ORES OF NORTHERN CANADA,

(with especial reference to the
Porcupine and Kirkland Lake Areas.)

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CONTENTS.

THE FLOTATION and CONCENTRATION OF CERTAIN COMPLEX AURIFEROUS ORES OF NORTHERN CANADA (with especial reference to the Porcupine and Kirkland Lake Areas.)

GENERAL INTRODUCTION.

While flotation is firmly established as a means of concentrating ores of the base metals, such as copper and lead, this is far from being the case in the extraction of gold from its ores. This state of affairs may quite possibly be due to the satisfaction felt by mill-men in various parts of the world with the comparatively recent improvements in the cyanidation process as a means of gold recovery. To be sure, there are some mills **which** employ flotation for concentrating gold ores, but the tonnage so treated is very small compared to that treated by cyanidation.

A considerable amount of research work on the flotation of gold ores has been done within recent years at McGill University in the Mining and Ore Dressing Laboratory. While the results as a whole have not been entirely satisfactory, they have been sufficiently encouraging to warrant further research on the subject.

The equipment of the laboratory is very well suited for an exhaustive research on flotation. One of the most important features of this equipment is a complete model mill unit consisting of a grinding circuit with a classifier, two types of flotation apparatus and a table. In addition to this, there is a single cell Minerals Separation machine and ample facilities for cyanidation work. The model mill unit enables tests to be made under conditions approximating very closely those existing in actual ore-dressing plants, although, as a matter of fact, results obtained in these tests can usually be bettered in practice since a closer regulation of conditions and quantities is possible due to larger scale of operations.

The apparatus above mentioned has already been used in Graduate School work by A. J.P. Walter, M.Sc., who investigated certain of the ores etc. in his experimental work. The flotation part of R.E. Legg's research also was performed on the single cell M. S. machine.

As already stated above, there are certain commercial mills employing flotation for the treatment of gold ores. Such plants are, however, the exception - especially in the more important gold mining camps such as the Witwatersrand, the Porcupine and the Kirkland Lake districts.

A rather striking application of flotation occurs in the treatment of Copper - Gold ores at Tul Mi Chung, Korea[1]. The ore is very complex, copper, gold, silver, bismuth, molybdenum, iron, arsenic, zinc and lead minerals all being present. Research work was started in 1915 and various methods of treating this ore were attempted. Tabling and amalgamation were

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1. Concentration of Gold Copper Ores by Froth Flotation at Tul Mi Chung, Korea. R.J. Lemmon T.I.M.M., Vol.33, 1924.

tried and discarded. Flotation, followed by treatment of the tails on Overstrom tables next held the field. This method gave a satisfactory copper recovery, but the gold extraction was low. It became obvious that fine grinding was necessary and it was noticed that the gold losses were proportional to the unrecovered pyrite. Following further careful research the plant adopted all flotation practice, which gives satisfaction at the present time. The gold recovery has been raised from 58% to 76%, while the copper recovery is 92%.

The Mill Feed at this plant contains:-

1.2% Copper

3.5 - 8.0 dwt. Gold/Ton, and the

Concentrates assay:-

24 - 25% Copper.

100 dwt. Gold/Ton.

6 oz. Silver/Ton.

Plans have been made to re-treat the old table tailings by flotation, employing sub-aeration type machines. The average recovery expected is 1.50 dwt. Au per Ton, and 0.07% S.

Turning to Canada, owing to the unpleasantly high losses in the cyanidation tails,- the Wright Hargreaves Gold Mine, Kirkland Lake, in 1925, undertook some research work.[1] "The chief source of loss lies in the tellurides which are found in the lower levels of the mine. These tellurides are unaffected by cyanide, or nearly so. Hence this is an increasing source of loss, as the mine gets deeper. The relation of values

1. Treatment of the Telluride-Bearing Gold Ores of the Wright-Hargreaves Mines, Ltd. Mueller, Grant & Heath. T.A.I.M.M.E., Feb. 1926.

as distributed between cyanide soluble gold and cyanide insoluble gold is rather constant, so that the tailing to be produced can usually be foretold from the assay of the mill head. Grinding and milling conditions being normal, about 6.7% of the value would be represented by telluride losses. With a base tailing as represented by regular solution losses + coarse material + ordinary cyanide losses equal to $0.40, the daily tailing from the plant would be calculated; for example:-

Mill Head $12.00, Tailing $0.40 + 0.067 x $12 = $1.20 [1.]

" " 20.00 " $0.40 + 0.067 x 20 = $1.74 "

After considerable experimental work, the Wright-Hargreaves Company decided to treat the underflow from the classifiers by flotation in Callow Cells, the concentrate being subjected to treatment by bromo-cyanide. The flotation tails are then returned to the cyanide circuit and treated in the usual manner.

Flotation is also used in other mills for the treatment of gold ores, in widely scattered places. Among these mills may be mentioned the Shawmut-Belmont,[2] the Belmont-Surf Inlet, The Mother Lode, California. Certain Colorado mills and mills in the Kalgoorlie gold field[3] of Australia.

1. Treatment of the Telluride-Bearing Gold Ores of the Wright-Hargreaves Mines Ltd. (previously quoted)
2. R.E. Legg, M.Sc. Thesis, 1924. "The Application of Gravity Concentration and Flotation as accessory to Cyanidation in the Treatment of a Porcupine Ore."

3. Truscott, Text Book on Ore Dressing, p. 483.

Before discussing the recent experiments by the writer, it might be well to give a resume of the work done by previous research fellows in Mining at McGill, on this subject.

R.E. Legg[1]. approached the flotation of gold ores as an accessory to cyanidation, whereas the writer proposes that the cyanidation of gold ores be replaced by flotation. His results are particularly interesting on account of data obtained this year in certain tests on low grade Hollinger Ore. Legg encountered difficulty in obtaining a gold extraction greater than 80%, while he obtained 89% extraction of the pyrite. The cause of this low gold extraction he claims is the coarse free gold, as he obtained colours of free gold on panning the flotation tails. Panning and flotation gave a 97.2% extraction and a tailing assaying about 0.01 oz. Au./ Ton.

Snijman[2] found that lots of ore graded by screening, contain sulphides in direct proportion to the percentage weights of the sizes. This holds good for the flotation tails of Porcupine Ore, according to Legg. If this were the case also for the concentrates, then liberation would be proportional to the grinding. Amalgamation tests on an ore ground to 82% minus 200 mesh gave 87.8% extraction.

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1. R.E.Legg, M.Sc.Thesis,1924 (previously quoted).

2. J.J. Snijman, Coarse vs. Fine Grinding in Gold Recovery by Cyanidation. M.Sc.Thesis,McGill University, 1924.

Roasted flotation concentrates yielded 90% extraction with high cyanide strength, (4 lbs./Ton) when agitated for 24 hours. Using 2.4 lbs. of cyanide and agitating for 26 hours the extraction was only 80%. 1.55 lbs. of lime were used in both cases.

In the case of flotation tails, a 92.7% extraction was obtained using 1 lb. of cyanide and 24 hours agitation. The importance of the time factor in extracting gold, especially when metallics are present, was clearly shown in these tests.

Legg also carried out tabling tests, and found that the gold values in the three products were proportional to the sulphide content, due to the fact that the stamp battery residues contained most of the coarse gold due to classification. The table middles and tails were mixed, then floated, an extraction of 88% of the gold being obtained.

The total gold extraction by tabling and by the flotation of table middles and tails was 96.5% to 97.6%. Cyaniding the roasted concentrate gave a 99.5% extraction with 36 hours agitation. Only 91.5% extraction was obtained on the unroasted concentrate.

Legg suggests floating the ore, then tabling the tails, the concentrates from both these operations being classified, and ground in closed circuit, before being sent to a filter. They would then be roasted and finally cyanided. It is interesting to note that both Walter and Legg suggest tabling the flotation tails. In practice this would probably prove too costly to be adopted, as very nearly the whole original feed tonnage would have to be so treated. It is reasonable to assume that the concentrate would be but a very small proportion of the feed. Another reason against the use of tables for re-treatment of the flotation tails is the low efficiency of tables treating such an ore ground to the size necessary for successful flotation.

A.J.P. Walter[1] carried out a large number of tests using both the model mill, then equipped with Callow cells only, and the single-cell Minerals Separation machine. He studied in considerable detail the effects of various reagents and mechanical adjustments of the apparatus. His results are particularly interesting as they afford a chance to compare the performance of the Callow and Minerals Separation machines.

With rich ore from the Lakeshore Mine (assaying 2.8 oz Au per ton) he obtained an extraction of 87.5% with tails assaying about 0.5 oz. Au per ton, and concentrates 6.0 to 25.0 oz. Au. per ton.

Most of Walter's work was done on Wright-Hargreaves ore, but his results were not particularly encouraging and certainly not as good as those obtained by cyanidation. Certain mechanical manipulations, such as changing the blanket on the Callow cells, were found to aid the percentage reduction of the

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1 A.J.P. Walter, "Reduction of Telluride Ores", M.Sc Thesis, McGill University, 19[illegible].

values in feed to values in tailings and to lower the values remaining in the tails. The percentage reduction is the relation the gold values in the feed minus the gold values in tails bear to the gold values in the feed, expressed as a percentage. It is an index of the actual extraction, and is useful when this is not available:

<u>e.g.</u> Feed assays 0.76 oz. Au./ Ton.

Tails " 0.0825 " " "

$$\% \text{ reduction} = \frac{0.76 - 0.0825}{0.76} \times 100$$

$$= 89.0\%.$$

Recovery, or extraction, is taken to mean the total amount of gold obtained in the various products, other than the tails, expressed as a percentage of the total gold fed to the circuit. A more dilute pulp yielded a cleaner concentrate, but the extraction suffered.

In the ore treated the proportion of floatable mineral is very small, hence in order that the oil be of sufficient concentration to enable it to function properly and without waste, the ore pulp should have a rather low liquid-solid ratio, 3 or 4 to 1. A high liquid-solid ratio will mean that only the most floatable mineral will become suitably oiled and floated, thus yielding a clean froth, but the tails will consequently carry unpleasantly high values. The high liquid-solid ratios in Walter's work are on account of the extra water necessary to keep the 150 mesh screen from blinding.

The addition of lime in certain quantities, viz. up to 0.01# CaO per Ton of Solution, gave a better percentage reduction and lower tails.

Sodium Silicate gave increased extraction when used in amounts up to 3 lbs. per ton of Solution.

Gas Tar oil did not prove particularly successful for though lower tails were obtained the percentage reduction was lower, due to lower feed values. Grinding to 95% minus 200 mesh was used in this case.

The best results he obtained occurred when the froth on the rougher cell was 4½ inches deep. (See tests 17-21). The tails assayed about 0.1 oz. Au. per ton, and gave a percentage reduction of over 83%.

In conclusion, Walter suggests that work on this subject be carried out, using a Minerals Separation machine instead of a Callow; that the effect of Sodium Silicate be studied in large scale tests and, thirdly, that tests be made on the tabling of the flotation tails.

T H E O R Y.

In this section the writer proposes to deal briefly with the Theory of Flotation. It seems needless to go into the matter in detail as it has already been treated in a masterly manner by writers of renown, but as the description and discussion of the experimental work which follows necessarily assumes a knowledge of the subject, it is desirable to present here a summary statement of the Principles of Flotation.

For the benefit of any reader who may desire further information on the theories involved, a selected bibliography is appended to this section.

H. L. Sulman defines flotation as "ore-dressing processes wherein metallic sulphide or other particles suspended in an aqueous ore pulp are recovered therefrom by an air-bubble froth."[1]

The majority of ores, except those of iron, consist of particles of metallic compounds often quite high in value, disseminated through gangue material (rock) of little or no value, and the economic use of the ore involves the concentration of the valuable material by the elimination of as much as possible of the worthless rock.

When the valuable particles are fairly large the usual method of concentration is to crush the ore to whatever degree is necessary to free the constituent minerals from the rock, and then to separate the metallic compounds from the rock by methods

1. "A Contribution to the Study of Flotation", H.L. Sulman, T.I.M.M. Vol. XXIX, 1919-20.

depending on differences in specific gravity as in the vast majority of cases the minerals have a much higher density than the gangue.

However, the successful use of gravity concentration decreases directly as the size of the mineral particles to be so treated. The lower economic limit is reached when the ore is ground finer than about 100 mesh. Flotation can be successfully applied to ores as coarsely ground as 48 mesh and from that size down to an extremely fine state of sub-division, so it is obvious that the fields of these two processes supplement one another and fortunately overlap to some extent.

The actual operation of flotation is carried out somewhat as follows: The ore from the primary crushing circuit is ground to the final size usually in ball mills, operating in closed circuit with classifiers. The common practice is to add the oil and frequently the modifying agents into the ball mill, thus allowing intimate contact with the ore particles. The comminution may take place with or without water, usually the former, depending on practical consideration, changing for each ore. In order to carry out the flotation of the valuable minerals there must be an aqueous ore pulp, the liquid-solid ratio varying from 2.5:1 to 6:1. The next stage is the aeration of the ore pulp, which may be carried on by violent agitation, as in the mechanical type of flotation apparatus, or by the injection of streams of air bubbles under low pressure as in the pneumatic type of machine. This aeration allows the air bubbles to attach themselves to certain mineral particles, as explained later, which rise to the surface, there forming a mineral-bearing froth sufficiently strong to withstand the weight of the mineral. The gangue particles which do not attract the air bubbles, sink to the bottom and are eventually wasted, or removed as tailings. The froth is skimmed off and forms the concentrate.

The minerals most readily floated are the metallic sulphide and native metals, though there are many others that exhibit this property , but to a less degree in most cases. The floatable minerals exhibit the tendency to float to a different degree, characteristic of each mineral. Hereafter in this thesis, in explaining flotation, the word sulphide is used as a general term to cover all floatable minerals.

The difference in degree of floatability of sulphides is mentioned in the preceding paragraph. In ordinary flotation, one concentrate and one tailing is produced, but by accentuating the differences in floatability by refinements of operation, such as the addition of agents, we may obtain two or more concentrates of different character from complex ores before finally discarding the tailing. These agents may act either chemically or physically, and the effect produced on the sulphides may be either permanent or temporary. A very close regulation of their use is generally necessary.

Differential flotation offers a wide field of experiment for complex sulphide ores.

In order to understand the theory put forward to cover the various phases of flotation, it is necessary to comprehend the molecular theory of the constitution of solids and liquids as commonly accepted. There is no need to give details of this wellknown theory in a work such as this.

As the degree of sub-division increases the ratio of the surface of a particle to its volume increases also, until at a certain point, approximately 80 mesh, the influence of gravitational forces is less than that of the surface or molecular forces acting on that particle. Hence when finely divided ores are treated, gravity plays a negligible part in flotation phenomena, while molecular forces are all important.

Various substances exhibit different degrees of wetting and it is upon this phenomenon that an explanation of flotation may be attempted. Wetting is defined by Truscott as the spread of a liquid over the surface of a solid to the displacement of air or another liquid; air is the medium usually displaced. A substance which is completely <u>immersed</u> in any particular liquid is not necessarily <u>wet</u>, and on its removal it may remain wet, or wet to a certain degree, or may even appear to repel the liquid from its surface, and thus to remain dry.

"Wetting which depends on the degree of adhesion established at the plane of contact between liquid and solid is therefore a condition of wide variability." [1]

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[1] H. L Salman, opus cit.

Different minerals are wet to different degrees, for instance oxides and most gangue minerals such as silica, are wet quite readily by water, whereas metals and sulphides do not exhibit this property to nearly the same degree.

The extent to which wetting takes place depends on the relation of the surface tension to the interfacial tension. In this paper it will suffice merely to define these terms and the reader is referred to Edser's "General Physics", and to Sulman's paper qlready quoted for more detailed information. Surface Tension is due to the lack of molecular force above the surface; hence the unbalanced or unsatisfied forces of the surface layers of molecules tend to deflect and link these molecules together. Interfacial tension results from the unbalanced molecular attraction at the surface. Adhesion is molecular attraction between interfaces hence it creates a pressure. Consequently when interfacial tension is high the adhesion is small and vice versa.

Hence wetting is dependent on the matter of the surfaces of contact and on the relationship of the interfacial tension to the surface tensions.

Imagine a drop of water on the surface of a solid otherwise in contact with air, as in Fig. A Let δ_1 represent the surface tension of the water; δ_2 that of the solid and δ_{12} the interfacial tension between solid and water. The contact angle marks the equilibrium between these forces. The tendency of each of these three forces is to contract the surface from which it derives. The solid surface being rigid, it is obvious that neither the solid face nor the solid-water interface can be reduced by actual contraction of the solid. As a result, the point of contact is displaced along the solid surface, this amounting to virtual contraction.

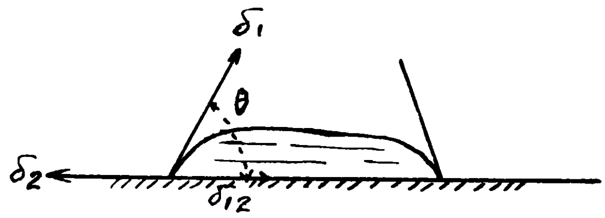

Fig. A

If the surface tension be the greater, the water is pulled over the solid meeting it at an acute angle. In such a case the solid is said to be wetted by water, or to display a preference for it. (Fig. *B*). An obtuse angle is formed if the interfacial tension is greater. The solid exhibits a preference for air in this case (Fig. *C*).

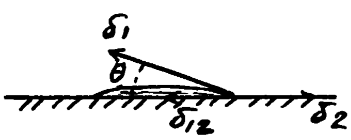

Fig. B.

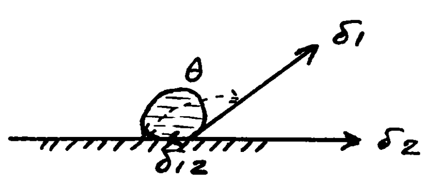

Fig. C

The forces are not drawn to scale

The equilibrium reached in wetting may be expressed quantitatively by the equation:-

$$\delta 12 \times \delta 1 \quad \cos\theta = \delta 2.$$

This formula is limited in its application as it contains two unknowns.

The contact angle exhibits both an upper and lower limit, the difference between these two values being known as the Hysteresis of the contact angle. For instance, water will not advance until the maximum value has been exceeded, nor retreat until the minimum angle has been reached. In Fig. *D* at (a) the normal contact angle θ is shown while at (b) the maximum and minimum values θ_1 and θ_2 are given. The hysteresis is represented by $(\theta_1 - \theta_2)$.

Fig D.

Hysteresis is a measure of the anchoring of the water surface to the solid surface, and is doubtless ude to an interlocking of the water and solid molecules.[1]

Very large contact angles, greater than 90° are indicative of non-wetting and preference for air. Angles less than 90° indicate wetting.

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1. Contribution to Study of Flotation, Sulman, T.I.M.M. Vol.29 page 114.

"It is therefore evident that contact angle and hysteresis though indicative of ease or difficulty of wetting and some guide to floatability, do not directly disclose the essential relations in flotation. The same may be said of wetting itself."[1]

Flotation is dependent on the relation between the water tension and the interfacial tension whereas non-wetting is contingent on the relation between the solid tension and the interfacial tension. It has been found by experiment that the interfacial tension can be varied by chemical action, or by electrostatic charge. The floatability of a mineral is enhanced by increasing its surface tension, and vice versa.

The condition of the surface of a mineral has a great deal to do with its floatability. Grinding has been found to have considerable effect, doubtless due to strains set up within the mineral and to alterations in its surface energy. Long delay or exposure between comminution and floating frequently renders successful flotation difficult or impossible, partly due to oxidation and partly to the changes in surface energy. Freshly ground surfaces exhibit a tendency to wet easily. Hence gangue minerals are very thoroughly wetted when the usual practice of wet crushing is followed, and consequently show even less tendency to float. Sulphides act similarly, but to a less extent.

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1. Text Book on Ore Dressing, Truscott, p. 492.

We are now ready to discuss the formation of froths and the aeration of the ore pulp with the consequent flotation of the mineral and sinking of the gangue.

The presence of molecules of a foreign substance at the surface of a liquid influences the surface tension of the liquid, increasing or decreasing it depending on whether the foreign molecules are negatively or positively adsorbed. Adsorption is the result of surface tension and occurs when foreign molecules are dissolved in a liquid. It is either positive or negative according as the foreign molecules tend to concentrate at the surface, due to greater attraction between water molecules than between a water molecule and a foreign molecule, or as they tend to concentrate at the centre of the liquid when the attractions are the reverse of positive adsorption. Sulman[1] goes into this question in considerable detail.

The property of frothing which a liquid acquires is due to lowered surface tension caused by an adsorption.

A bubble film of pure liquid is unstable since the slightest external force such as drainage of liquid by gravity, evaporation, etc. would rupture it, e.g. a bubble formed at the surface of pure water bursts immediately.

Imagine a substance to be positively adsorbed in a pure liquid, σ_l is lowered. If a unit area of such a film be stretched, the film will be thinned and the molecules in the middle of the film will be drawn into the outside layers consequently diluting the foreign molecules. The surface tension

1. Study of Flotation, H.L.Sulman, T.I.M.M.,Vol.29, p.94-97.

of the film will therefore be increased, thus tending to resist rupture. Similarly the surface tension will be decreased if the film contracts. Hence the film is elastic, and stable equilibrium results. "Low surface tension alone will not give film stability .... Contamination is necessary."[1]

Different substances exhibit different powers of adsorption, so that if two solutes are present in the same solute, the one tending more strongly to be adsorbed displaces the other from the surface layer. Thus, in order that mineral particles are not to be displaced from the surface by frothing agents, the latter must be sufficiently adsorbed to form a strong and elastic film, but not so much as to prevent adsorption of the mineral. The frothing agent is added to confer elasticity on the water film, as the mineral particles alone cannot do so sufficiently. It will be seen that a joint adsorption is required. The soluble portion of many oils serve very efficiently as frothing agents, and the insoluble part of the oil adsorbed at the mineral surfaces decreases their adhesion for water. Hence a single oil may act as both frothing agent and as filming agents in enhancing the floatability of sulphide particles. Various substances such as phenols, cresols, pine oils, terpenes,etc. are used as frothing agents. A satisfactory frothing agent must produce a large bubble surface and still leave the bubble film with sufficient mineral adsorptive energy.

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1. Study of Flotation, H.L. Sulman, T.I.M.M. Vol. 29,p.97.

Completely insoluble oils are unsatisfactory for flotation, but nevertheless they may be well employed when mixed with cresol or other frothing agents. This insoluble portion when adsorbed at the mineral surface increases the contact angle with the air-water surface, thus enhancing the floatability of the mineral and strengthening the mineralized film. This oil film on the mineral, also increases the hysteresis of the contact angle, thus "stiffening" the froth.

The froth consists of mineral slime, several particles thick, and some larger particles. These coarse particles alone will not form a froth, but may be raised to the surface on being attached to air bubbles and there show a tendency to become dislodged. Additional rising air bubbles present these particles from sinking entirely and again raise them to the surface where they may be held by a loose froth system.

The immiscible oil thus serves as a collecting agent and must consequently be thoroughly distributed throughout the ore pulp. Since the oil film coating the mineral is very thin a very small amount of oil is required, usually much less than 1% of the ore by weight. Greater quantities of oil hinder flotation, tending to produce granulation or aggregation of the mineral particles as in the Cattermole system, due to the adhesion for oil.

Similarly the air bubbles must be minutely subdivided and widely distributed throughout the pulp in order that each mineral particle may have ample opportunity for contact with the air and of becoming attached thereto.

There are used in flotation practice many substances termed modifying agents whose function is to increase the difference in floatability between the sulphide and gangue particles. These agents usually act by increasing the wetting of the gangue by reducing the gangue-water contact angle. Deflocculation indicates complete wetting and some reagents accomplish their work in this manner. Other modifying agents act by adsorption thus promoting the wetting of the gangue.

The fineness of grinding has a considerable effect on flotation as finely divided mineral particles are readily adsorbed and also adsorb immiscible oil.more readily than the larger crystalline particles. This necessitates flocculation and the provision of sufficient "slime" to agglomerate the larger particles thus causing their flotation. For further information concerning the theories of flocculation and deflocculation, the reader is referred to Sulman's paper already mentioned.

As no differential flotation was carried out in the experimental work connected with this thesis, there is no need to go into the matter in detail.

A selected bibliography is herewith appended.

Selected Bibliography on the Theory of Flotation.

1. A Contribution to the Study of Flotation,

   H. L. Sulman, T.I.M.M. XXIX,1919-20.

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   Edser, 4th Rep. British Association on Colloid Chemistry.

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   Tucker & Head, T.A.I.M.M.E.

7. Reduction of Telluride Ores,

   A.J.P. Walter, McGill M.Sc. Thesis, 1926.

INTRODUCTION TO EXPERIMENTAL WORK.

DESCRIPTION OF ORES.

1. Hollinger Low Grade Ore:-

A great variety of minerals, both metallic and non-metallic, occur in the ore bodies. Among these minerals may be mentioned native gold, silver and copper, pyrite, pyrrhotite, chalcopyrite, sphalerite, galena, molybdenite, arsenopyrite, various tellurides, oxides, scheelite, silicates, carbonates, etc.

Most of the gold occurs in the native state, chiefly associated with pyrite. Free gold also occurs less commonly with other metallic ores, or alone in the rock matte. It also occurs as telluride of gold, but such occurrences are rare in the Hollinger, although at one place petzite was found carrying 11% of gold and 26% of silver.

Pyrite is the chief metallic sulphide, and is common to all deposits. It usually occurs in the form of very small cubes, but occasionally is massive. It is found in the country rock as well as in the veins. Much of the gold is locked up in the pyrite, assaying being the only means of distinguishing the auriferous pyrite from the non-auriferous. The ore carries 5 to 15 per cent iron pyrites, the average being about 5.5 percent. Combined silica in the ore varies from 49 to 52 per cent, while quartz constitutes about 40 percent.

The ore used in the tests assayed about 0.255 oz. Au. per ton. For further details as to the ore, the reader is referred to the Annual Report of the Ontario Department of Mines, Vol. 33, part 2, 1924, "The Porcupine Gold Area" by A.G.Burrows.

2. Kirkland Lake Ores:-

The ores consist of a broken coarsely crystalline quartz with other minerals deposited in the fractures. Iron pyrites is the most abundant sulphide, being found in both the ore and the wall rock. It is usually well crystallized, though occasionally fine grained. Chalcopyrite occurs chiefly where the ore is gold bearing. Galena, sphalerite and molybdenite also occur in small quantities.

Gold bearing solutions have evidently circulated through fracture planes and have enriched the minerals by deposition of gold in these fractures. Movement took place, and the gold bearing minerals were crushed, polished or slickensided. A later deposition of gold followed.

The gold occurs native or free, as auriferous pyrite and as tellurides. As mentioned in the introduction it is these tellurides that complicate the problem of treatment of the ore. The most common telluride is altaite ($PbTe$), telluride of lead, and is usually accompanied by visible gold. Calaverite, ($AuTe_2$) gold telluride, has been recognized; it assays 40.6 percent gold, and is sparsely distributed throughout the ore. Kalgoorlite, a telluride of mercury, silver and gold also has been recognized. A mercury telluride, coloradoite, associated with native gold, altaite, pyrite and chalcopyrite is found in Vein No.2, Lakeshore Mine. Petzite, Hessite and other tellurides have also been reported as occurring in the Kirkland Lake deposits.

The Wright-Hargreaves ore which was used in the flotation tests assayed about 0.56 oz. Au. per ton, and is a typical Kirkland Lake district ore.

The Lakeshore ore is of a much higher grade, and is inclined to be "spotty" on account of the large amount of free gold occurring therein. Otherwise it is similar to other ores of the camp.

The gangue is composed of quartz, greywacke, syenite, conglomerate, porphyry, calcite, etc.

An excellent account of the Kirkland Lake camp is to be found in Annual Report, Ontario Dept. of Mines, Vol.32, Part 4, 1923, "Kirkland Lake Gold Area," by Burrows and Hopkins.

DESCRIPTION OF APPARATUS.

1. Single Cell M. S. Machine. (see Plate 1).

This piece of apparatus (Fig.1) was used for carrying out the small scale tests. It consists of an agitation chamber $4\frac{1}{2}$" x $4\frac{1}{2}$" x 14" high, internal dimensions. The froth collecting chamber or spitzkasten is a wedge-shaped box attached to the agitation chamber about 5" below the top of the latter and extending about the same distance below it. The apex of the wedge is at the bottom. From a point about 2" above the apex of the frothing chamber a 3/4" pipe leads to the bottom of the agitation chamber. About half-way up the agitation chamber is an aperture $\frac{1}{2}$" x 2" connecting it to the frothing chamber. This aperture is equipped with downward sloping baffles to start the pulp in a downward direction on its arrival in the frothing chamber. The oiled mineral will be floated to the top by the bubbles, while the unoiled gangue will continue on the downward path thence through the pipe back to the agitation chamber.

The impeller, placed inside the agitation chamber, has eight bronze blades, connected at the base by a circular bronze plate. The rapid rotation of the impeller (1200 r.p.m.) causes a two-fold action,-one, the circulation of the ore pulp as outlined above and, secondly, the aeration and thorough agitation of the pulp by forming a vortex in the agitation box into which air is sucked and broken up into bubbles. The impeller is attached to a vertical shaft which is belted to a $\frac{1}{4}$ H.P. electric motor.

The finely ground ore and water is fed into the agitation box, together with such oils and reagents as are desired. The action of the impeller causes the pulp to be thrown violently against the sides of the chamber from which it cascades, entrapping air which is then minutely subdivided and thoroughly mixed with the pulp at the centre of the chamber. The mixed and aerated pulp passes from the agitation box to the frothing chamber by the aperture with the baffles taking the course described above. The floated mineral is skimmed off by hand as a concentrate. There is a screw plug at the bottom of the frothing chamber which permits the removal of the tailings at the conclusion of the test.

The ore is ground in a small ball mill, 6" long by $8\frac{1}{2}$" diameter, by means of steel balls of diameters from 1 to $1\frac{1}{4}$ inches.

The following is the usual procedure when carrying out a test on the single-cell M. S. machine:-

A weighed amount of ore, usually 500 or 1000 grams is put in the ball mill with a known volume of water, such as to give a 1:1 pulp approximately, together with whatever reagents or oils are desired, and the whole is ground for a certain time. After grinding, and removing from the ball mill, the pulp is charged to the M.S. machine and the pulp brought up to the correct level by the addition of water. This gives about a 6:1 dilution when using 1000 grams of ore. The test in the flotation machine is continued as long as a sufficient amount of mineral is obtained in the concentrate or as long as conditions demand. The mineral laden froth is skimmed off during the test while the tails are removed at the end of the test, by means of the plug in the bottom of the frothing chamber. The products, concentrates and tails are dewatered by a suction filter, dried in a steam oven, weighed and assayed after suitable preparation.

The small scale tests such as are obtained by the above procedure are not accurate, but they give very excellent indications of the action of various reagents, and are of very great help in gathering data for planning large scale tests. One thing is certain that results obtained in these small scale tests can always be improved on in larger scale runs, and still more so in actual practice.

2. The Model Mill. (Plates 2, 3,4,5, Fig. 2.)

The ore is fed into the ball mill by means of an endless belt which forms the bottom of a hopper of 100 lbs. capacity.

The belt is intermittently driven in a forward direction by a somewhat complex lever, pulley and cam system. As the belt moves forward it carries a ribbon of ore with it which drops into the hopper of the ball mill. The amount of ore fed into this hopper with each movement of the belt is regulated by a micrometer screw.

A revolution counter is attached to the eccentric thus enabling the weight of feed for any number of revolutions to be calculated. This permits checking the amount fed per hour and thus ascertains whether there is slippage of the feed belt. It was noticed that the belt driving the pulley slipped a good deal. This slip could be eliminated by using a chain and sprocket drive.

The ore is given its final grinding by means of a ball mill, 13 inches in diameter and 20 inches long. Six bars or lifters have recently been placed inside the ball mill to increase its grinding capacity by giving a more positive lift to the charge. The mill rests on four $2\frac{3}{4}$" rollers by means of its flanged ends. One pair of these rollers is driven by a belt to the main shaft and causes the ball mill to rotate at approximately 40 r.p.m.

The ball load consists of about 170 lbs. of manganese steel balls, $1\frac{1}{4}$" & 2" in diameter.

The ore pulp is fed into the mill by means of a scoop feeder, 2" wide. The ground pulp is discharged through a screen screwed to the end plate. Various sizes of screens can be used. The under-size is discharged into a funnel which forms one leg of an air lift by which the pulp is conveyed to the Akins classifier. Former practice was to feed the undersize by gravity directly to the flotation machine. Water is added inside and/or outside the screen, to prevent blinding and to make the pulp sufficiently dilute to enable the air lift to handle it. The small amount of oversize obtained with the coarse screen used with the classifier in circuit is collected in a pan and returned by hand at intervals into the ball mill. This oversize was formerly returned to the mill by air lift.

The Akin's classifier consists of a semicircular trough of galvanized iron, 74 long by $7\frac{1}{2}$ inches in diameter. Running lengthwise is a $1\frac{1}{2}$" shaft to which is attached by means of $\frac{1}{4}$" stove belts $7\frac{1}{2}$" long a galvanized iron spiral of 2" pitch. The width of the spiral blades is 1". Small sections of the spiral are removed to allow some of the water to overflow instead of being dragged up with the oversize. The spiral revolves at about 25 r.p.m., and is driven by level gears and belt from the main shaft. The inclination of the classifier can be altered by raising the angle iron framework at the oversize discharge end. The feed enters the classifier about 5" from the undersize discharge or lower end, the fines or undersize being discharged over a lip into a funnel and a $\frac{3}{4}$" pipe leading to the M.S. machine into which the pulp flows by gravity. The coarse material or oversize settles rapidly, and is slowly conveyed against the water flow to the upper end of the machine by means of the revolving spirals which make a close fit with the bottom of the classifier. The oversize is discharged into a launder returning it to the ball mill for re-grinding. A considerable amount of experimental work was performed in adjusting the classifier and for each test it was necessary to obtain a balance, i.e. to ascertain that the classifier was discharging the same amount of solids as undersize that was being fed into the ball mill, and at the desired dilution. This consumed a good deal of time and required much careful adjusting of water, feed per hour and inclination of classifier to the horizontal. The classifier inclined at its usual angle about 15° holds 4250 cc. of water.

Oil and reagents are distributed to suitable parts of the circuit by means of two oil feeders, one mounted on the mill frame, the other on the M.S. machine frame. Each consists of 4 cast iron wheels, 8" diameter and 1" wide at the periphery. These are mounted on a common axis, being submerged in whatever oil or liquid be held in the glass container underneath each wheel. The film of liquid thus adhering to the rim is scraped off by copper wires flattened at the contact with the wheel, and drops of liquid run down the wire to gather at the end before falling into tin pipes leading to the desired place in the circuit. The inclination of the wires and their point of contact with the wheel are adjustable. The point of contact is above the axis of rotation of the wheels for obvious reasons.

The main flotation unit used this year in the model mill circuit is a 10-cell Minerals Separation machine of modified type designed by Assistant Professor Erlenborn. The outstanding feature of this machine is the parallel flow of pulp maintained from the bottom of the frothing chamber to the bottom of the agitation box of the same cell. The pulp is passed on to the next cell by means of a square tube of adjustable height i.e. variable weir-overflow, extending down the front right-hand corner of the agitation box to the bottom of it, thence through the partition into the agitation chamber of the next cell (see fig. 2). It will be seen that the ore pulp can only be passed on to the next cell in series as the preceding cell is filled to the desired level and overflows into the communication tube. The pulp flows by gravity aided by the suction of the impeller into the next cell. A great advantage of this design is that the feed can be stopped at any time without disturbing the balance of the circuit. It is obvious that a much more thorough agitation can be thus obtained. In order to make the regulation of the pulp level more positive, an adjustable gate valve is placed over the orifice at the bottom of the frothing chamber which leads by means of a pipe to the bottom of the agitation box.

The unit consists of 2 agitation cells followed by 8 flotation cells, all this being mounted on a wooden frame. The agitation boxes of all the cells are 5,7/8" x 5 7/8" by 12½" deep. The frothing chambers are 5,7/8" wide by 9" long, and 22" deep. The pipe leading from the spitzkasten to the agitation chamber is a short section of rubber hose slipped over ¾" nipples. The communication pipe or weir overflow is ¾" x ¾" with a smaller pipe making a snug sliding fit inside it to enable the height to be adjusted.

The impellers are 8 bladed, of the Howard type, made of bronze. The spindles are mounted on 2 bearings and are driven at 1200 r.p.m. by 4" flanged pulleys with 1" belts taking power from a 5 H.P. electric motor.

Launders of various types can be used for collecting the concentrate which is skimmed off by hand.

The flotation tails may be pumped directly to a large storage cone by a small centrifugal pump or may be sent to a small Wilfley table by means of an air lift.

The Wilfley table is mounted on the frame of the model mill, and has proved itself to be of considerable use as an indicator. During several tests a few flakes of free gold were noticed on the table when the M. S. tails were passed over it, but the amount of concentrate obtained was small and low grade. The table concentrate may be cellected in a tub and the tails pumped to storage.

The concentrates from the M.S. machine were at first sent by gravity to a small cleaner flotation cell of the Callow type attached to the framework of the model mill. This cell raised the grade of the concentrate considerably, but caused quite appreciable losses so the practice was discontinued. The Callow cell tails were returned to the head of the Minerals Separation unit. The residual material in the cleaner cell gave a high assay.

The model mill is supported by an angle iron framework so designed that the different pieces of apparatus are at different levels and gravity plays a prominent part in the flow of pulp from stage to stage. The various units are driven by a 6 H.P. motor.

An exeellent detailed description of the model mill is given by A.J.P. Walter in his Thesis on the Reduction of Telluride Ores, 1926.

Flow Sheet of Model Mill.

Ore ground to minus ¼" is fed into the ball mill by the feeder belt and the scoop-feed of the mill. The heavy Barrett #4 oil is usually fed here. The discharge of the ball-mill has to pass through a screen attached to the mill. The oversize is discharged at the open end of the screen and is returned to the front of the ball-mill for regrinding. The undersize was, until Test 7, fed direct to the 8-cell Minerals Separation machine. Recently an Akin's Classifier was built and installed in the circuit so that now the undersize from the screen, of coarser mesh than previously used, is fed to the classifier by means of an air lift. Water is added in the interior of the screen. The classifier, which is adjustable as to slope, makes two products, the oversize which is dragged to the top of the classifier by the spiral and is returned to the ball-mill for further comminution, and the undersize which is fed into the flotation machine by means of gravity. The light pine oil and such reagents as are used are fed into the circuit at the classifier undersize discharge.

The pulp obtains a thorough agitation in the first two cells of the modified Minerals Separation machine before passing to the eight flotation cells. The concentrate from these cells may be handled in a number of ways:-

(1) It may be collected as one concentrate.

(2) " " " " " " ", and sent to a Callow cell for cleaning, the tails from the Callow being returned to the head of the Minerals Separation circuit.

(3) The concentrate from the various cells may be collected separately, or grouped in any way desired.

(4) The concentrate from any of the boxes may be kept as a final concentrate, and the rest returned to the head of the machine for cleaning.

The tails from the Minerals Separation machine may be pumped directly to storage by a centrifugal pump, or may be sent to a small Wilfley table by means of an air lift. The table tails are then sent to storage by pump, and the concentrates are collected in a bucket.

(See Flow Sheet Fig. (3) for further details.)

PROCEDURE FOR LARGE SCALE TESTS.

The ore, previously crushed to minus ¼ inch is placed in lots of 20 lbs. to the feed hopper. The electric motor driving the mill is then started and the air compressor regulated to give the desired pressure. Next the wash water to the ball mill discharge is turned on, and the oil feeder is set in operation; following this the feed is started, the time and tachometer reading being noted. When the undersize discharge from the classifier is at the desired dilution and is equal to the input to the ball mill from the feeder, the circuit is considered in balance and the Minerals Separation machine is started so that flotation may be begun. The balance is determined by collecting a classifier undersize discharge sample for a known length of time in a 1000 c.c. graduate. The specific gravity of the pulp is then determined and the pounds of ore discharged per hour calculated. If the ball-mill discharges through a screen, the undersize proceeding directly to the flotation unit, there is no need of balancing the machine as accurately. The balancing process takes a great deal of manipulation and considerable time for each test.

The next step is to balance the flotation machine, which is accomplished by judicious manipulation of the slide valves at the bottom of the spitzkasten and of the weir overflow or communicating pipes. It will be seen that one cell is regulated at a time. Before starting the motor driving the impeller, the flotation cells are partly filled with water to prevent the spindles from "springing" owing to lack of resistance at their lower extremities. The pulp takes about 30 to 45 minutes to pass entirely through the Minerals Separation machine and to build up the circuit.

Samples of the feed are cut every ten minutes while Minerals Separation tailings samples are cut every five minutes. In order to follow the trend of the test the duration of the run is divided into three approximately equal parts and the various samples for each part kept separate. The concentrate sample is usually cut from the total concentrates after they are dried.

Screen analyses of each lot of feed samples are made on the Bell screening machine.

Specific gravity samples of the flotation feed are taken at frequent intervals by collecting a fairly large volume of pulp in a 1000 c.c. graduate, the time required for collection being noted. The total net weight of pulp is noted, and the percentage of solids calculated from the following formula:-

$$D = \text{percentage of solids in pulp}$$

$$= \frac{100\,S\,(Y - 1)}{Y\,(S - 1)}$$

$$= \frac{100 \times 2.65\,(Y - 1)}{1.65\,Y}$$

$$= 160.5\,\frac{Y - 1}{Y}$$

where $Y = \dfrac{\text{net wt. of pulp}}{\text{volume of pulp}}$

S = specific gravity of the ore = 2.65.

liquid to solid ratio = $\dfrac{100 - D}{D}$

The amount of solids fed to the flotation machine per hour is calculated as follows:-

D x weight of pulp = weight of solids in grams.

$$\frac{\text{wt. of solids} \times 60 \times 60}{453.6 \times T} = \text{lbs. of ore fed per hour,}$$

where T = time in seconds required for collecting sample.

The various samples are filtered, dried, weighed and prepared for assaying. Where a balance of the whole test is desired the weights of all products and samples are recorded carefully, so that the total weight of the products may check with the weight of ore fed to the machine.

It requires at least 3 men to carry out a test successfully on the model mill and flotation circuit. For any test to be really satisfactory to any degree, at least, 200 lbs. of ore should be fed, usually involving a duration of 4 or 5 hours. Doubtless much more reliable results would be obtained if tests were of 10 or 15 hours duration.

ASSAYING.

As a great deal of time was spent in assaying, an outline of the methods used seems in order. All samples were ground to 100% minus 100 mesh.

The charges used for various grades of ore and the method of fusing them follow:-

1. Low Grade Tails (Hollinger)

1. <u>Low Grade Tails (Hollinger).</u>

<u>Charge</u>:-

Ore	4. assay tons.
PbO	$2\frac{1}{2}$ " "
Soda	$3\frac{1}{2}$ " "
Borax	15 grams.
Flour	2 "
Salt cover .	

Charge was thoroughly mixed and placed in Battersea J Crucible by means of paper "pokes" or cones. Two of the buttons thus produced were scorified together, after adding silver. The resulting lead button was cupelled, and the silver bead parted the gold remaining, being annealed and weighed. The weight of gold in milligrams was consequently divided by 8 to give oz. Au. per ton.

2. <u>Feed (for all ores)</u> :-

<u>Charge</u>:-

Ore...............	1 A. T.
PbO	1 " "
Soda	1 " "
Borax	3 gms.
Argol	$2\frac{1}{2}$ "
Salt cover.	

Fuse in 20 gram crucibles, then cupel resulting button after adding necessary silver. Part, anneal and weigh.

3. Concentrates (for all ores):-

(a) Charge:-

Ore.....	1/10 to 2/10 A.T. (depending on grade)
Pb. ...	45-50 to 90 gms.
Silica..	pinch
Borax...	"
Silver..	as necessary.

Scorify, cupel, part, anneal and weigh.

(b) Optional method, giving a larger bead.

Ore......	1/2 A.T (roasted)
PbO 1-1/2	"
Soda ...	6 gms.
Silica..	5-6 "
Borax ..	10 "
Flour ..	3 "

Salt cover.

The 1/2 A.T. of ore was first roasted, then mixed with the charge, fused in 20 gms crucibles, silver added, cupelled, parted and weighed.

4. Minerals Separation Tails (Kirkland Lake Ore)

Charge:-

Ore	2 A.T.
PbO.......	2 " "
Soda	2 " "
Borax	6 gms.
Argol.....	2 " for fusion in gas furnace.
"	3½ " " " " oil muffle.

Salt cover.

Fuse in 30 gram crucibles, add silver, scorify if necessary, cupel, part, anneal and weigh.

The silver was added in the form of small lengths (approximately 1/8") of chemically pure wire, which was buried in the lead button before scorification or cupellation as the case may have been.

STUDY OF FLOTATION OF HOLLINGER LOW GRADE ORE (#199).

Six large scale tests were carried out with this ore, the results obtained were quite encouraging and indicated that flotation could be successfully applied to the concentration of such an ore. The economic possibilities of this method of treatment are considerable, especially as flotation equipment occupies much less space than cyanidation equipment per unit of capacity, less supervision is required, reagents and supplies are cheaper and doubtless the grinding costs could be lowered.

It must be remembered, of course, that flotation is merely raising the grade of the ore, the tails may be wasted, but the gold has still to be extracted from the concentrate. This would involve considerable additional equipment in the form of raasting, regrinding, leaching, precipitation and smelting apparatus for recovering the gold. Hence the saving effected by the concentration process would be partly offset by the cost involved in refining the enriched product.

Certain of the tests show a very high extraction and an exceedingly low tailings loss as compared with present Porcupine practice.

The first two or three tests were run chiefly with a view to testing the mill and the re-designed Minerals Separation flotation units, and to gain experience in manipulation. The manner in which these tests were carried out has already been described. The Akin's Classifier had not yet been built, hence the ball-mill discharge screen undersize was fed directly to the flotation unit. Had this classifier been in the circuit, it is very possible that better results would have been obtained as more uniform conditions of the pulp in the Minerals Separation machine would naturally have resulted from Classification. It would be of considerable interest to run a test duplicating one of those already performed, but including the classifier in the circuit.

TABLE 1.

Test No.	Total Ore Fed (lbs.)	Duration of Test (hrs)	Ore Fed per hr. (lbs)	Liquid: Solid Ratio of M.S. Feed	% -200 mesh	Assays oz.Au.per Ton Feed	Conct.	Tails	% Reduction.	Remarks.
1	40.3	1 hr. 20mins.	27.4		77.9	0.24	3.05	0.0125	94.7	Cleaner cell conct. values given here.
2	127.4	4 hr.	27.33	6.7:1 3.2:1	78.5	0.255	2.575	0.02	92.4	ditto
3	55.2	1 hr 8mins	53.3	7.4:1 5.6:1	73.7	0.255	2.475	0.0125	95.1	ditto
4	235.45	4 hr. 58mins	47.3	4.03:1	73.2	0.375	3.50	0.0178	95.7	" reoiled
5	238.45	4 hr. 30 mins.	52.6	3.14:1	71.2	0.295		0.015	96.5	Concts fr. diff.cells collected separately re-oiled
6	234.6	5 hr. 36mins.	40.6	3.84:1	75.1	0.310		0.015 +0 0.009	95.0 +0 97.0	ditto

NOTE:- Barrett #4 oil was fed to Ball-mill at rate of about 1.25 #/Ton of Ore, while G.N.S.#5 was fed at the rate of 0.1#/Ton.of Ore.

For further details see test sheets in Appendix. The percentage reduction in the last column serves as an indication of the extraction. The reason for using this term, which is not new, is fully dealt with in the following note on extraction and percentage reduction.

In tests5 and 6 it will be noticed that there is a large error in the balance apparently giving a very low extraction. In cases where no balance can be obtained, the extraction cannot be calculated on the feed value with any degree of fairness. The most reliable indicator of the gold recovery in such cases is given by the percentage reduction figure which is based on the ratio of values of the tailings and feed. It is obvious that in test 6, for instance, that the extraction of 85.6% which is based on the total value of gold in the feed is erroneous; with tails of such low value as 0.009½ oz. Au./Ton a true extraction less than 97% is inconceivable. The actual extraction must be calculated from the weights and values of the tails and concentrates. That calculated from the assay and amount of ore fed to the circuit may be termed the theoretical extraction and, as above stated, is unreliable unless there is no error in the balancing of the test. The extraction given for test 5 may be accepted as accurate as the error in balance is negligible.

Test 1. The results of this test are of no particular value owing to the small amount of ore fed. Two facts are worthy of note, however, viz.: the low value of tailings, 0.0125 oz. Au. per Ton, and the high percentage reduction 94.7%.

Test 2. The percentage reduction in values is lower though both the tails and feed values are higher. This test is more reliable as it is of longer duration, hence allowing more ore to pass through the machine.

There are various factors which affect a test, some of which are constant, no matter how long the test may last. Others may tend to gradually increase until in a long test they may have a considerable effect, whereas in a short trial they may be negligible. With the first type, the long test distributes the irregularity over a longer period, and thus secures more representative and true results. For instance, it will be noticed in the later tests of this series that the tails are higher at the beginning of the trial. This is due to the fact that the pulp dilution is very high at the beginning owing to the water added to the Minerals Separation machine when it is started up. For reasons mentioned previously this dilute pulp tends to produce high tails. As the circuit becomes built up and conditions constant, the pulp attains its correct dilution and the flotation machine is able to work at its maximum efficiency under the conditions of that test. Also it is common experience that it requires some time for the oil to build up in the flotation circuit, hence for a long test this condition is less noticeable. This offers another explanation for the higher tails obtained at the beginning of tests.

The second accumulative factor is illustrated by the crowding or over-feeding of a unit in closed circuit. For instance the small Callow cell was used to clean the concentrate from the Minerals Separation machine. If the amount of concentrate was too great for the Callow to handle properly, as was probably the case, it would consequently produce a good clean concentrate, but its tailings would tend to be high. These tails would be returned to the Minerals Separation circuit and enrich that. It is easy to carry the illustration a step farther to when eventually the M. S. tails become enriched. This would only become noticeable for a long test. On the other hand, when all the units are working within their capacity and conditions are constant, it seems reasonable that a test of long duration will produce more reliable results.

Test 3 shows a much smaller loss in the tails, and consequently higher percentage reduction. The amount of ore treated in this test was small, hence the results are not as reliable as they might otherwise have been. It would seem, however, that slightly reducing the fineness of grinding has no harmful effect in fact the result seems favourable.

Test 4. This test is rather encouraging and served as a basis for the remaining two tests of the series. Re-oiling seemed to have a good effect on the circuit. It will be seen that the material remaining in the Cleaner cell assayed quite high. The percentage reduction in values is satisfactory. The failure to obtain a good balance in this test is as much due to material lost and possibly to other cause such as faulty assaying or sampling. This will also account for the poor extraction if the lost material is mainly concentrate.

Test 5. The Callow cell was not used at all in this test, a concentrate being taken off from the different cells of the Minerals Separation machine. A study of the log sheet of this test shows that most of the values are extratted in the first cell, most of the remaining values in the next two cells, and so on. The last 3 cells gave a very low grade concentrate, if it may be termed such, the intention being to obtain as clean a concentrate as possible in the first cells and to raise the extraction with the last cells. The grinding was slightly coarser, but seems to have had no ill effect. A good balance was obtained. This may perhaps be partly due to the fact that there was no loss from the Cleaner cell. These results are particularly gratifying in view of the fact that a high rate of feed was maintained to the ball-mill with consequently coarser feed to the flotation circuit.

<u>Test 6</u> is a repetition of Test 5, but with a lower feed rate to the mill, thus raising the fineness of grinding slightly. The liquid-solid ratio is slightly higher in this test also.

The effect of the dilution or liquid-solid ratio of the pulp has been dealt with to a certain extent in the introduction. The most suitable ratio for flotation is 3 or 4 to 1, as a rule. With dilute pulps the only way of getting a sufficient amount of floatable mineral is to pass the pulp through the flotation circuit very rapidly. This would prevent thorough agitation and contact of the oil, etc., and mineral. Hence the grade of tails would be raised. The effect of the pulp dilution is noticeable in table 1, though it would not be fair to assign the whole result to this cause.

The dilution of the pulp also has a strong effect on the capacity of the flotation machine, for if we assume that the pulp, whatever its dilution, will pass through the machine at the same rate, it is obvious that a greater tonnage of solids will be handled if the liquid to solid ratio is low.

If the efficiency of a flotation machine is measured by the amount of mineral it extracts from the feed, it will be seen from the previous remarks that the liquid-solid ratio may have a profound influence.

The tails showed very low values, consequently the percentage reduction in values was high. The error in balance accounts for the low theoretical extraction to a certain extent. The most interesting feature of this test is the very low tailings loss which promises well for the possibility of flotation as a means of extracting gold from low grade pyritic ores.

It is interesting to note that no reagents other than oil were used in the above flotation tests. There is every possibility that with further research as to reagents, etc. some very low tailings and even high extractions would result.

The tails from the Minerals Separation machine were sent to the Wilfley table, which was used merely as an indicator or check. In large scale tests it is the practice to improve the operations in any manner possible, suggested by observation. The table served as an indicator of the action of the Minerals Separation machine. For instance, when the speed of the impeller decreased due to power drop, a broad band of mineral appeared on the table shortly afterwards, whereas before there had been none. For the most part very little mineral showed on the table. Most of the table concentrate proved to be metallic iron, probably from the ball mill. A few small flakes of gold were observed on the table during different tests, but it was shown that the table affected the extraction but slightly. In commercial practice a large number of tables would be needed for this operation as nearly the full tonnage fed to the mill would have to be so treated. Consequently it would seem that flotation followed by tabling would not be a very satisfactory nor economical method, even if tables were efficient for handling fine material of this type.

From the foregoing tests it will be seen that a good extraction has been made and that the tailings loss is very low, but another problem arises now, namely, the treatment of the concentrate. This product as it comes from the flotation machine is, as a rule, rather low grade, assaying $1\frac{1}{2}$ to 2 ozs. Au. per ton as an average, or making a concentration ratio of 5 or 6 to 1, which is very low. Dealing with a large mill of say 5000 or 6000 tons per day capacity, it would mean that 1000 tons of concentrate would have to be handled per day. If this is to be roasted preparatory to further treatment, a tremendous plant would have to be erected. Clearly some means must be devised for "stepping up" the grade of the concentrate, and thus reducing the tonnage to be handled.

With this end in view two tabling tests were run on concentrates from Test 5.

Test 5A. The object was to effect a table concentration of the concentrate obtained from the 1st Minerals Separation cell. Ten lbs. of this product was first mixed with water (wetted) then fed by hand onto the table.

A concentrate and tailing was made and collected.

Amount of ore fed .... 10.0 lbs.

" " concentrates obtained..		2.72 lbs.
" " tails "		7.25 "
loss		0.03 "
	10.0 lbs.	10.00 lbs.

Assays and Balance:

	Oz. Au./Ton		Wt.		Units
Feed	2.08	x	10	=	20.80
Conct....	6.66	x	2.72	=	18.10
Tails	0.046	x	7.25	=	3.38
					21.48

The error in balance is doubtless due to richness of the concentrate and the free gold contained therein.

It was attempted to make as high grade a concentrate as possible.

Test 5B. The object was as in 5A, using only concentrate from Minerals Separation cells 6, 7 and 8.

Table was fed as in test 5A, but this time an attempt was made to get clean tails.

Amount of Ore fed .... 8.0 lbs.

" " Concentrates obtained..		0.312 lbs.
" " Tails "		7.50 "
" " material lost		0.188 "
	8.00 lbs.	8.00 lbs.

Assays and Balance.

	Oz. Au./Ton		Wt.		Units
Feed	0.18	x	8	=	1.44
Conct.	2.79	x	.312	=	1.18
Tails	0.0975	x	7.5	=	.731
					1.911

The two above tests give some indication that tabling of the concentrate might be successful in practical work. That is, the 6 to 1 flotation concentrate might be tabled producing a high-grade final concentrate while the tails are returned to the flotation circuit for re-treatment. This would doubtless prove more satisfactory than attempting to make a final tailing by tabling.

Discussing the possibilities of tabling the concentrate in a rather theoretical way and using tests 5, 5A and 5B as basis for calculations, we arrive at some interesting conclusions:

Test 5.	Wt.	% Wt.		Oz Au/Ton	Units
Conct from 1st Cell	= 22.06 lbs	54.9	x	2.08	114.2
" " 2 & 3rd "	= 3.56 "	8.92	x	3.10	27.65
" " 4 & 5th "	= 3.46 "	8.77	x	0.56	4.69
" " 6,7 & 8th"	= 11.32 "	27.41	x	0.18	4.94
TOTAL................	40.40 "	100.00			151.48

hence, the average grade of concentrate will be 1.515 oz. Au./ Ton or $30.30 / Ton.

In the first three cells 63.8%, say 64%, of the total weight of concentrates is obtained:

Wt.		% Wt.		Oz.Au./Ton.		Units
22.06	=	86.0	x	2.08	=	179.0
3.56	=	14.0	x	3.10	=	43.4
25.62		100.0				222.4

The average grade of concentrate from the first two cells is consequently 2.224 oz. Au./Ton, or $44.50 per Ton.

According to the results of Test 5A. this can be concentrated up to 6.65 oz. Au. per Ton, or $133.00 per Ton, the weight of this concentrate being 27% of the weight of the table feed.

In the last five cells 36% of the total weight of concentrate is obtained. The average grade of this, calculated as for the others is, 0.27 oz. Au.per Ton or $5.40 per Ton. Test 5B. shows that by tabling we can raise the grade to 2.75 oz. Au. per Ton, or $55.00 per Ton, the weight of this concentrate being 1.4% of the original concentrate.

This concentrate may be tabled with that produced by the first three Minerals Separation cells, that is, 65.4% of the weight of Minerals Separation concentrates now assay over 2.22 oz. Au. per Ton, and by tabling as shown in Test 5A, a concentrate assaying 6.65 oz. Au. per Ton is obtained. The weight of this high grade concentrate is found to be 18.3% of the weight of the original total flotation concentrate.

The table tails can either be returned to the flotation circuit or possibly in the case of the lower grade tails, as in Test 5B, be wasted without further treatment.

It will readily be seen that the amount of tailings to be roasted and subjected to subsequent leaching and smelting processes, is but a small fraction of the original flotation concentrate.

The possibilities of this method of retreating the concentrate are quite great, as the above tests and calculations indicate. Nevertheless it is not wise to draw positive conclusions nor to make definite statements based only upon such meagre evidence as is offered by the two tabling tests noted previously. The retreatment of the concentrate offers a wide field for further experimental work. A certain amount has been already done by A.J.P. Walter, to whose thesis 1926 the reader is referred for information.

The reader is also referred to R.E. Legg's 1924 research into the recovery of gold by flotation and tabling. His best extraction of gold by flotation was 84%. By panning the flotation tails he obtained particles of coarse gold, too large for flotation, thus raising the gold recovery to 97%; the final tails containing about 0.012 oz. Au. per ton. He also carried out some cyanidation tests on the flotation tailings, recovering about 93% of the gold in the tails.

In addition to the flotation of the ore, Legg tried tabling it using a sand feed assaying 0.38 oz. Au. per ton. From this he obtained products giving the following assays:

Product	Gold Assay Oz./ Ton.	Total Sulphide Recovery.	Total Gold Recovery.
Concentrate	3.77	77.8%	79.65 %
Middles	0.12	4.42	3.45
Tails	0.08	17.78	16.90
		100.00	100.00

The middles and tails from the above test were combined, giving an assay of 0.085 oz. Au. per ton, then floated; with the following results:-

	Sulphides	Gold
I. Recovery by:		
1. Table	77.8%	79.65%
2. Flotation (Test 6)	10.2	16.8
Total Recovery	88.0%	96.45%
II.		
1. Table	77.8%	79.65
2. Flotation (Test 7)	16.1	18.0
Total Recovery	93.9%	97.65%

Thus by a combination of tabling and the flotation of the table middles and tails, an extraction of 97% is possible This applies only to the sand as the slime material, containing 13.5% of the total gold, was first removed. The slimes could probably be satisfactorily treated by flotation.

It would have been interesting to compare results had Legg tabled the flotation concentrates.

From the above discussion it would appear that tabling can be considered seriously as an accessory to flotation in the extraction of gold from a Porcupine Ore. Just what should be its place in the flow sheet of a large Mill is a problem, but the writer feels confident that the tabling of the concentrate would prove most satisfactory from a practical point of view.

Balance for Test 5, illustrating methods used in this phase of a test:

Feed.

Weighed Feed to Ball Mill	238.45	lbs.
Feed Samples	2.02	"
Calculated Feed to Minerals Separation	236.43	"
Feed to M.S. by adding wt. of parts	228.22	"
Loss (probably in B.M. scoop when washed out, or accumulated in Ball Mill)	8.21	"

		$ Au/Ton			
Au. in Feed	=	5.90 x 228.22	=	.....	1347.00 units

Concentrates

M.S. Conct (1)	=	41.60 x 22.06	=	......	918.00
" " (2 & 3)	=	62.00 x 3.56	=	......	220.50
" " (4 & 5)	=	22.20 x 3.46	=	......	76.70
" " (6,7 & 8)	=	3.60 x 11.32	=	......	40.75
" Residue	=	1.80 x 13.69	=	.....	24.62
		54.09			1280.57

Tails.

Tails (1)	=	0.375 x 73.6	=	......	27.60
" (2)	=	0.25 x 53.3	=	......	13.34
" (3)	=	0.25 x 47.1	=	......	11.78
		174.0			52.72

Total Tails & Concts.	1333.29	units
Loss	13.71	"
	1347.00	"
Error in balance .... =	0.92%	

$$\text{Extraction} = \frac{1280.57 \times 100}{1347.0}$$

$$= 95.3\ \%$$

Assays were run on the +200 mesh tails and on the -200 mesh tails with a view to determining whether the losses lay in the coarse or fine material. It was found that the +200 mesh product assayed $0.40 Au. per ton while the fine material ran only $0.15 per ton in Gold. It was therefore seen that the losses are in the coarser material. The proportion of this product (+200 mesh) is, however, quite small, less than 30%.

Calculating back from these assay values and the screen analyses, it was found that the M. S. tails assayed 22.97 cts. per ton in gold, which checks closely with the valus obtained by assaying the tailings samples.

Calculated from + and - 200 mesh:

	%	Assay $Au./Ton.		
- 200 product	72.0 x	0.15	=	10.80
	28.0 x	0.40	=	11.20
	100.0			22.00

The value of the table concentrates must be added to above value as these tails were tabled, whereas the assay samples were taken from the discharge of the flotation machine before tabling.

Table Concts. assayed $9.00 Au./Ton.

2 x 9.00 = 18 units

Divide these units into 174.0 lbs. (wt. of tails)

= 0.965 cts. to be added

to + and - 200 mesh tails,

hence total value of tails is:-

- 200 mesh product	10.80
+ 200 " "	11.20
Table concentrates	0.97
	22.97 cts. Au./Ton.

STUDY OF THE FLOTATION OF WRIGHT-HARGREAVES LOW GRADE ORE (#201).

Four large scale and numerous small scale tests were carried out on Wright-Hargreaves ore. The results obtained were far from satisfactory, and many interesting problems arose as a result of these tests. In view of Walter's work last year on this ore using Callow cells, this series of tests afford an excellent basis for the comparison of pneumatic and mechanical flotation machines.

Cyanidation tests were carried out on the tailings of some of the large scale flotation trials. The results of these runs will be fully discussed in a later section of this thesis.

The large scale tests will be dealt with first. These were all of considerable length, in only one case was less than 100 # of ore fed to the mill. This was the first series of tests in which the classifier and the lifters in the ball-mill were used. A great deal more care was necessary in balancing the mill, but a more uniform flotation feed was obtained. Several grinding tests had been run previous to any attempt at flotation, hence most of the mechanical adjustments, etc. were known, allowing good conditions to be maintained during the flotation trials.

The following table #2 gives a summary of this series of large scale tests:-

TABLE #2. Ore #201.

Test No.	Ore Fed per Hour	% minus 200 mesh	Liquid Solid Ratio	ASSAYS (Oz.Au./Ton) Feed	Conct.	Tails	% Red. Values in Feed to Values in Tails.	Remarks.
7	43 & 36(-)	89.6	4:1	0.56	5.83	0.09 to 0.12	84.0 to 78.5	B #4, G.N.S.#5, Na_2S & K.Xanthate all used. Reoiling in 3rd & 5th cells. Classified M.S. feed.
8	33.9	88.38	3.5:1	0.56	4.60	0.065 to 0.105	88.4 to 81.2	No reagents other than oil used. Pine oil used for re-oiling in 6th M.S. cell.
9	36	90.86	3.84:1	0.76	5.8	0.0825 to 0.10	89.0 to 86.9	Duplicate of Test 8, except Na_2S & K.Xanthate were added.
10	38.8 to 50.0	72.06	2.4:1	0.71	5.3	0.165 to 0.115	76.8 to 84.0	Considerable difficulty in balancing circuit. Coarser grinding. Reagents added. Note low L:S ratio.

Test 7.

At the beginning of the test the ore was fed at the rate of 43 lbs. per hour, this was later cut down to 36 lbs. per hour which gave more satisfactory conditions. The ball-nill did not seem to be able to grind the larger tonnage to sufficient fineness. The concentrate formed was quite high, grade, but seemed to contain a large amount of gangue. The flotation tails were tabled, but no mineral, other than metallic iron was observed thereon, an especially close watch was kept for "colours" of gold, but none were seen.

The percentage reduction from values in feed to values in tailings was rather low. The tailings assays showed quite a wide variation, that of the sample for the first part of the test being lowest. The results of this test agree closely with those obtained in #9, in which the conditions were very similar, the chief difference lying in the higher feed value. There are also slight differences in the fineness of grinding and liquid-solid ratio of the pulp.

Test 8.

This and the next succeeding test are duplicates except that additional reagents other than oil were used in Test 9. It would seem that the results produced by oil alone are superior to those obtained when additional agents are employed It would reduce the cost in practice if agents are not necessary or desirable, but in some of the small scale tests it will be noticed that reagents are desirable when floating the cyanide tailings.

The grade of concentrate obtained in this test was lower than in any of the others, but this is more than compensated by the leaner tails produced. The higher value of the tails occurred in the first third of the run. In the last two thirds of the test the tails assayed 0.065 oz. Au. per ton., the lowest value obtained in this series of tests. The constancy of the values of the tails during the latter part of the test shows that this is the most reliable test on this ore.

During the latter part of the test a very small amount of pine oil (G.N.S.#5) was added in the sixth cell. This improved the appearance of the concentrate from this and the succeeding boxes.

<u>Test 9.</u>

Sodium Sulphide and Potassium Xanthate were added as shown in the detailed tables, but seem to have had no beneficial effect in this test. It was later found out that Sodium Sulphide retards the flotation of gold, but aids the recovery when gold is associated with pyrite[1]. This fact may partially account for the poorer results obtained in this test. The higher feed value may also account for this. The percentage reduction in values in this test is quite high, the highest obtained as a matter of fact, though it is partly due to the higher feed values while the tailings loss remains approximately the same. This tailings loss which tends to be practically constant in three of the four tests may be termed the absolute

1.- Notes on the Flotation Process, Lord & Snyder. Southwestern Eng. Corp'n.

loss. In tests 7 and 9, the tailings have approximately the same value, viz:-

	Tails.	Feed
Test 7	0.09 to 0.11 oz.Au./Ton	0.56 oz Au.per Ton.
Test 9	0.03 to 0.106 "	0.76 "

There is no essential difference, so it would seem that the tails are not directly proportional to the feed value. This is important as the ores of the Kirkland Lake district incline towards "spottiness" or erratic values. Particular attention was paid to the crushing, and mixing of the ore preparatory to feeding to the mill to insure that each bag of ore would be as representative as possible of the whole lot. The grinding and other conditions, such as reagents, were the same in both tests.

Test 10.

Considerable trouble was experienced in getting the mill circuit to balance as a thicker pulp (classifier discharge) and coarser grinding was desired. This was eventually accomplished, but there was only a small amount of ore left to complete the test. That the results are not good may be accounted for by several facts:-

1. The grinding may be too coarse to liberate the values sufficiently for flotation. (72.06% - 200 mesh).
2. The liquid-solid ratio may have been too low; i.e. the pulp too thick for successful flotation.
3. The reagents may have had a harmful effect as noted in the discussion of the previous test.

The assays of the feed did not check particularly well, that for the first part of the test being 0.66 oz. Au. per ton, while that for the latter part was 0.75 oz. Au. per ton. The tails obtained in the first half assayed higher than those obtained in the rest of the test. This is probably due to the fact that the best conditions had not been reached until the latter half of the run. The percentage reduction in values is lower as may be expected from the higher tailings.

The much greater rate of feed per hour is due to the attempt at coarser grinding which necessitated crowding the grinding and classifier circuit, lowering the water and raising the inclination of the classifier slightly.

The small amount of ore fed during this test was due to the fact that the supply was inadequate. This was unfortunate as a longer test would have been desirable, especially so as conditions even at the end apparently were not yet constant as evinced by the tailings assay.

It is interesting to compare these last two tests as they differ widely in many respects. Dealing first with the feed, the value of it is about the same, but the rate of feeding is considerably higher in the second test. The fineness of grinding in the two tests can hardly be compared; in Test 9 the percentage of minus 200 mesh product is about 91%, while in test 10 it is 72%. This raises an interesting point, the tailings value is still rising in test 9 whereas in test 10 the tails decrease in value, and at the end of both tests are approximately the same. Hence it is not at all certain that such fine grinding is necessary for economical extraction.

Economical extraction is stated here as opposed to actual extraction, for it is obvious that if it costs say 75 cents a ton to obtain an additional extraction of 50 cents, this higher extraction is clearly not economical. Hence for economical extraction such fine grinding may not be necessary.

Another point worthy of comparison is the ratio of liquids to solid in the ore pulp of the last two tests. It may be that a pulp of such low dilution is desirable with an ore containing such a small proportion of floatable mineral. Before definite statements can be made as to what is the best liquid to solid ratio for such an ore, a great deal more research should be done on this subject, and the possibility of obtaining some truly interesting results is great.

As there is, as yet, no definite knowledge of the capacity of the flotation circuit, another field for experimenting is opened up. The higher tails in the last test may have possibly been due to the high feed rate; this, however, does not seem very probable as the feed rate was only increased during the last part of the test, and this is the time during which the lower tails were made. The real explanation may be the direct opposite of the above, the machine may not have been working up to its capacity with the low feed and thick pulp, hence its efficiency was impaired. It is evident that the thick pulp is not detrimental to flotation, however. It is possible that the higher liquid to solid ratio in test 7 may account for the higher tailings. Compare this with test 8. The pulp is thicker and the tails lower. It is not safe to place the credit entirely with the pulp dilution, however.

No actual balance was attempted on any of these tests unfortunately, but approximate balances, neglecting all residues in fact all products except flotation concentrates and tails, have been calculated for these tests. It is assumed that each group of tails represent one-third of the total tails, which is not necessarily correct.

Test 7.

Feed	Wt.		Oz.Au./Ton.		Units.
Feed	160.0 lbs.	x	0.56	=	89.6
Tails 1	53.3	x	0.09	=	4.80
" 2	53.3	x	0.12	=	6.40
" 3	53.3	x	0.11	=	5.87
			Tails		17.07
Concts.	9.5	x	5.83	=	55.5
			Total	=	72.57

Test 8.

Feed	106.3 lbs.	x	0.56	=	59.5
Tails 1	35.4	x	0.105	=	3.82
" 2	35.4	x	0.065	=	2.30
" 3	35.4	x	0.065	=	2.30
			Tails		8.42
Concts.	10.1	x	4.60	=	46.5
			Total	=	54.92

Test 9.

Feed	Wt.		Oz.Au./Ton.		Units.
Feed	143.0 lbs.	x	0.76	=	108.7
Tails 1.	47.6	x	0.0825	=	3.93
" 2.	47.6	x	0.0975	=	4.65
" 3.	47.6	x	0.10	=	4.76
			Tails	=	13.34
Concts.	12.75	x	5.8	=	74.0
			Total	=	87.34

Test 10.

Feed	93.0	x	0.71	=	66.1
Tails 1.	46.5	x	0.165	=	7.68
" 2.	46.5	x	0.115	=	5.36
			Tails	=	13.04
Concts.	8.65	x	5.3	=	45.90
			Total	=	58.94

Some interesting comparisons are offorded between A.J.P. Walter's work and the above tests. Generally speaking it may be said that the Minerals Separation or mechanical type of flotation machine seems to give better results on this ore than pneumatic type - as exemplified by the Callow cell. This may be due to the fact that re-oiling is possible in the former and better opportunity is afforded for thorough contact of oil and mineral by mechanical agitation. Another point in favour of the mechanical type is the much "tougher" froth produced, the Callow cell making a more evanescent, weaker froth. This naturally would appear to be a drawback as far as the flotation of metallic gold is concerned.

Choosing Walter's best tests for comparative purposes we find that in no case were as low grade tailings produced as in Test 8 of this year's work. In most cases the percentage reduction in values obtained with the Callow is lower than that obtained with the Minerals Separation type. A much higher rate of feed per hour was also used this year.

A summary of Walter's best results is given on the following page in Table 3, and should be compared with Table 2.

Except where specially noted Barrett No.4 and Pine Oil, G.N.S.#5, were used in the above tests.

The first four of Walter's tests (9 - 12 incl.) are not particularly noteworthy except insofar as they show the relation of the rate of feeding to the extraction. Test 13 shows the result of finer grinding. The low tails produced in test 14 may be caused by the addition of a small amount of lime, but it should be noted that the feed value is much lower. The same remark applies to Test 15 in which Gas Tar oil was used to replace Barrett #4. The high liquid-solid ratio may have the effect of lowering the extraction. Similarly with Test 19.

TABLE 3.

Test No.	Feed in lbs. per hour.	% minus 200 mesh.	Liquid :Solid Ratio.	ASSAYS Oz. Au. per Ton Feed	Conct.	Tails	% Redn Values in Feed to Val. in Tls.	Remarks.
9	23.2	-	4.0:1	0.78		0.12	84.4	100 mesh discharge screen
10	34.5	78.23	6.9:1	0.545		0.16	70.7	Ditto.
11	31.3	79.90	3.4:1	0.52		0.13	75.0	
12	26.95	86.05	3.2:1	0.45		0.11	75.5	
13	26.95	95.14	5.6:1	0.72		0.15	79.25	150 mesh discharge screen
14	26.95	77.3	5.3:1	0.57		0.10	82.4	Tails for Cy.
15	23.2	95.19	7.2:1	0.38		0.085	77.6	Gas Tar Oil CaO G.N.S.#5.
16	26.95	77.3	5.3:1	0.57		0.10	82.4	Tails for Cy
17	23.2	75.35	4.85:1	0.55		0.09	83.6	Higher for 100 mesh screen.
18	18.2	90.53	5.9:1	0.60		0.10	83.3	Higher for 150 mesh screen.
19	18.2		9.6:1	0.90		0.14	84.4	Tar oil
20a.	18.2		7.45:1	0.70		0.13	81.4	Sodium Silicate Gas Tar GNS#5.
20b.	18.2			0.75		0.11	85.3	Ditto, lime.
21	18.2		6.0:1	0.59		0.09) 0.10)	84.7	B #4 GNS#5 CaO.

Tests 17 and 18 were run with a higher froth in the rougher cell, which seemed to produce a favourable effect. Tests 20a and 20b. show the effect of Sodium Silicate. The high tails may be a result of the higher feed value as the percentage reduction was fairly good.

Test 21 was run to obtain a complete balance in which Walter was successful. This would seem to be his best test, taking all things into consideration. The tailings assayed quite low, and the extraction quite high, namely 84.3%.

The tests run this year on the Minerals Separation machine were all of longer duration or at higher rates of feed than Walter's on the Callow cell, hence more ore passed through the circuit. This should make this year's tests more representative, and for that reason perhaps more reliable.

It is obvious that the Minerals Separation circuit has a much greater capacity than the Callow. As a matter of fact the maximum capacity of the Minerals Separation machine has not yet been determined, it is at present limited by the ability of the grinding and classifying circuit to produce material of the desired fineness.

A number of small scale flotation tests were carried out on tailings produced by cyanidation of the Wright-Hargreaves ore, both from the Company's mill and in the laboratory. The results of these trials are to be found under the section on Cyanidation.

STUDY OF THE FLOTATION OF THE LAKESHORE HIGH GRADE ORE (#200).

Very little experimental work was done on this ore, as the term was drawing to a close, but one large scale test and four single-cell Minerals Separation tests were carried out - the former ~~latter~~ with not particularly encouraging results. The small scale tests were performed on the tailings from cyanide tests on Ore #200 by the Fourth Year Class in Mining. The most interesting result of these tests is the success in ~~assaying~~ by previous elutriation the copious, dirty froth otherwise produced in floating the ore, thus agreeing with the similar tests on the Wright Hargreave's cyanided tails. In one test a separate concentrate was made from the dirty, whitish froth, which proved to be quite high grade, assaying 4.6 oz Au. per ton. It is quite possible that this dirty froth is largely made up of tellurides, though microscopical and chemical examination is necessary before this can be definitely stated.

The combination of cyanidation and flotation reduces the tails to a quite low value. The gold extraction from the cyanide tailings varies from 43.4 to 53.1% by flotation, which is quite satisfactory when the low value of the feed, about 0.25 oz. Au. per ton, is taken into consideration. That this could be greatly improved is beyond doubt as so little work was done to get the best adjustments.

A tabulation of the more important data from the small scale test follows. For complete details the reader is referred to the Appendix.

TABLE 4.

Test No.	Assays Oz.Au./Ton.			% Of Total GOLD in	
	Feed	Tails	Conct.	Tails	Conct.
25	0.2536	0.15	2.20	56.0x	44.0x
26	0.2795	0.16	2.60	54.4	45.6
27	0.2885	0.17	2.40	56.6	43.4
28	0.2378	0.117	4.60	46.9	29.5
		2nd. Conct.	1.40		23.6

x Extraction based on Collected amounts of products. Several grams of Concentrate were lost. See Note on Test 25 below.

The results given above for Test 25 are very probably inaccurate as several grams of concentrate were lost before a leak in the collecting pan was noticed. The percentage of total gold in the concentrate should consequently be considerably higher. This and Test 28 give the best results, both as regard low tailings value and high extraction, hence it would seem that grinding in oil produces favourable results. The better results for Test 28 are probably due to the greater concentration of oil and thicker pulp during grinding, also to the greater time of contact

In Test 26 elutriation has the effect of producing a much better looking froth, but neither the tailings assay nor the extraction is particularly good.

Agitation with Potassium Xanthate and Sodium Sulphide does not seem to improve the extraction from the tailings. This test is probably the poorest of the four in regard to results. It should be noted that the feed value in this test is the highest in this series.

The results of the large scale flotation test on the Lakeshore Ore were decidedly unsatisfactory. The concentrate produced was exceedingly high in value, viz. 43.13 oz. Au./Ton, but small in quantity which may have caused a certain degree of salting of the tails.

In order to build up the froth, the concentrate from the first two boxes instead of being skimmed off, was distributed between the next three cells, while the froth from the sixth and eighth cells was returned to the seventh. Hence concentrate was collected from the third, fourth, fifth and seventh boxes.

The increasing grade of the tails from 0.532 to 0.630 oz. Au./Ton during the test, indicates a building up somewhere in the circuit, which is probably due to the insufficient quantity of concentrate removed. The liquid to solid ratio should be suitable for successful flotation. It is possible that the small amount of oil fed accounts for the poor results to a certain extent.

With only one test carried out, it is absolutely impossible to draw definite conclusions with regard to the suitability of this ore for flotation. Four tests made on this ore last year by A.J.P. Walter, using the Callow cells, gave very similar results. Walter decided that the extraction was made in the rougher cell and suggests insufficient oil and mechanical imperfections as the probable cause of the low extractions obtained. His tests on this ore were merely preliminary trials designed to provide experience in the manipulation of the apparatus.

It is apparent from the above remarks that more experimental work is required in the flotation of this ore before definitely deciding that it is impossible of successful treatment by this method.

STUDY OF CERTAIN CYANIDATION TESTS ON ABOVE ORES.

In order to get an idea of the comparative merits of flotation and cyanidation, and combinations of both processes, as a method of treating Kirkland Lake ores, a series of cyanidation tests was planned in conjunction with the flotation work. It was realized that lack of time would prevent the writer from personally carrying out these trials, but Prof. Bell very kindly offered to supervise a group of Fourth Year students who chose cyanidation as the subject of their Ore Dressing theses. Messrs. Dewar, Findlay and Tatley carried out all the cyanidation tests mentioned in this section, and performed all the assaying in connection therewith. The writer has, however checked the extraction values, the figure given in the tables being calculated from the feed and tailings assays. Certain small scale flotation tests were also performed on tailings from cyanide tests with an object of finding out whether flotation should precede or follow cyanidation to give the greatest extraction.

Table 5 on the following page, summarizes the results of the cyanidation tests. Several tests, notably Nos. 2, 4, 5 and 7, have no direct connection with flotation work, but are included as a matter of interest and for the sake of completeness.

The extraction in test #1 is very low, doubtless due to the fact that the operators had no previous experience in this kind of work.

TABLE No. 5.

Cy. Test No.	Ore Used	#/Ton of Solution KCN	#/Ton of Solution CaO	Liquid :Solid Ratio	% -200 mesh	Time of Agit-ation	Assays oz.Au./Ton Feed	Assays oz.Au./Ton Tails	Extr -act ion %	Remarks
1	Wright-Hargreaves	1.95	0.44	2.37:1	96.5	24	0.468	0.16	66.0	
2	Wright-Hargreaves	1.90	0.50	2.33:1	90.82	24½	0.446	0.091	79.5	
3	Tls fr.Flo -tation Test No.7	1.95	0.77 0 0.34	2.33:1	85.26	18	0.1007	0.036	64.2	
4	Wright-Hargreaves	1.80	Na_2CO_3 9.5	2.33:1	92.6	24	0.460	0.143	73.4	Sodimm Carbonate used for Alkali.
5	Wright-Hargreaves	1.90	0.34 0.46	2.43:1	91.5	24	0.488	0.120	75.5	
6	Lakeshore	2.0	0.60	2.42:1	87.24	40	2.39x	0.27	88.6	Milled in Cyanide
7	"	2.0	0.55	2.44:1	94.02	36	1.52	0.20	87.0	Milled in Water
8	Tls. fr. Flot. Test No.11	1.90	0.35	2.45:1	86.42	20¾	0.68	0.235	65.5	

x = Assay of feed to Ball Mill. This ore is very spotty and in spite of great care in mixing and preparation it is difficult to obtain two lots of ore of the same assay.

The figures given above are all copied directly from the Test Sheets of the Fourth Year Group which performed the tests. The extraction was calculated from the feed and tailings values and corresponds to the percentage reduction figure given in the flotation tests.

The tails from large scale flotation Test #7 comprised the feed for cyanide Test #3. The extraction is low, but this may be due to the fact that in spite of the floatability of the tellurides their proportion to total gold in the tails is probably higher than that in the feed. Hence cyanidation would make a poor recovery on such material. A very careful microscopical analysis of the tailings is necessary to determine in just what form the values occur in the tailings.

Test #4 is of no particular value except to note the effect of sodium carbonate instead of lime as the alkali. The very great amount used is due to miscalculation; as good, if not better results could doubtless be obtained using a more nominal amount, say 2 or 3 lbs. per ton.

Tests #6 and #7 show really good recoveries of gold, especially when the telluride content is taken into account. The former is particularly interesting, as flotation tests were carried out on the tailings. The extraction by cyanidation is much higher than that produced by flotation on Lakeshore ore. (See large scale flotation test #11). The feed in the flotation test was slightly higher in value than that in the cyanidation test, but the tails were disproportionately higher. To repeat, it is not fair to condemn flotation as being unsuitable for treating this ore on the results of a single rather unsatisfactory test. It is interesting to note that the fourth year students calculate that 70% of the gold was extracted during the milling (grinding).

Test #7 shows that quite high extraction can, however, be obtained even if the ore is not milled in cyanide.

The cyanidation of flotation tails again yields a low recovery, very nearly the same in both tests #3 and #8. The tails produced are still very high in the latter test. These should be compared to those given in table 6, dealing with small scale flotation tests on cyanided tailings of both Wright Hargreaves and Lakeshore ores.

Fifteen small scale tests were carried out on some cyanide tailings from the Wright Hargreaves mill. These were shipped to the University several years ago in a moist state, and have been kept that way ever since, being sealed in old carbide tins. Ten small scale tests were also performed on fresh cyanide tails resulting from Fourth Year tests on Wright Hargreaves ore. It is interesting to note the difference in behaviour between the stored and fresh tailings, the former being amenable to flotation whereas the latter needed a preparatory treatment before they could be floated. Various reasons for this difference of behaviour suggested themselves. Did the storage, with possible oxidation, have any effect on the floatability? Did the containers have any chemical effect on the stored tails, rendering them more floatable? Was the difference due to the fact that the fresh tails had been recently thoroughly aerated in the Pachuca tanks increasing the tendency of the gangue to float? Could lime or cyanide, if any remained with the fresh tails, have caused the uncontrollable, dirty froth produced? Again, differences in grinding may have been partially the cause. It is quite possible that keeping the tailings moist for so long

TABLE 6.

Flot'n Test No.	Ore	Assays. Oz. Au. per Ton Feed	Tails	Concts	% Extractn.	Remarks.
1	Tests 1-15	0.1025	0.05	3.55	51.85	) For discussion of
2	were run	0.0980	0.05	3.40	49.70	) these tests see
3	on moist tailings	0.0936	0.04	2.40	58.15	) following pages of) this thesis.
4		0.0961	0.04	2.40	59.40	)
5	shipped	0.1537	0.10	2.20	36.70	)
6	from	0.15665	0.04	4.10	75.35	Not Reliable see text.
7	Wright -	0.09295	0.04	1.90	58.20	
8	Hargreaves	0.0960	0.04	2.60	57.50	
9	Mill	0.0945	0.045	2.15	51.00	
10		0.0991		2.20	56.40	See text & test sheets
11		0.0852	0.04	2.05	53.00	for tailings data.
12		0.0843	0.04	2.95	53.30	
13		0.0864	0.0375	2.0	57.20	
14		0.0843	0.030	2.2	53.40	
15		0.0879	0.060	6.9	32.10	
16 to 19	Tails from) Cy. Test #1)					) Omitted as no products were) collected or assayed. See) text and test sheets.
20	"	0.1258	0.105	3.20	20.4	About 50% of total gold
21	"	0.1291	0.090	2.85	21.3)	in feed lost in material
22	"	0.1319	0.090	2.80	17.8)	removed by elutriation.
23	"	0.1329	0.085	4.45	36.8	Negligible loss by elutri -ation.
24	"	0.1618	0.075	1.50	53.6	
25	Tls. from	0.2536	0.150	2.20	44.0	See note under Table 4.
26	Cy. Test #6	0.2795	0.160	2.60	55.6	
27	"	0.2885	0.170	2.40	43.4	
28	"	(0.2378	0.117	4.60	29.5) 53.1	) Two Concentrates were
		(		1.40	23.6)	) made in this test.

would have the action of allowing any air bubbles attached to mineral particles to escape, and the water present for so long a time would naturally tend to wet all the minerals present to as great an extent as possible. Both types of tailings were tested for lime and cyanide, but these were not found to be present in appreciable amounts.

It would have been interesting to determine what effect lime and cyanide have on the flotation of such ores. This could be done by agitating different lots of ore with lime in one case, cyanide in another, mere aeration in a third case, and combining all conditions in the last case. Such a series of tests would doubtless give valuable insight into this problem, and give some very interesting results, but unfortunately there was insufficient time to put this idea into practice this year.

Pursuing the idea that the fineness of grinding might have affected the floatability of the mineral, elutriation of the fines was attempted (Tests 20-24). The first of these trials were very crudely performed and are not very reliable. They accomplished the purpose of removing the slime, but too much of the other material was also elutriated. Just as good results were obtained if only slimes, amounting to but a few grams, were siphoned off. This naturally caused a much smaller loss in gold values through elutriation. Judging by the results obtained it would seem that grinding can be so fine as to injure the flotation of the mineral. Walter in his thesis found that the extraction was directly proportional to the time, and hence to the fineness of grinding. He neglects to describe the nature of the froth obtained with the finest grinding. It would have been interesting to know if he had difficulty from an uncontrollable froth.

It is quite possible that the peculiar froth obtained from the fresh tailings is not due to grinding conditions, but the fact that when the very fine material is elutriated, the froth becomes normal and satisfactory, and can surely be taken as an indication that this slime must have some effect on the nature of the froth.

Generally speaking the flotation of the cyanide tails gave but fair results. The gold extraction did not exceed 60%, and the tailings could not be reduced below 0.03 oz. Au. per ton. The flotation feed or cyanide tails from the Wright Hargreaves mill assayed 0.156 to 0.0843 oz. Au. per ton, the average value being about 0.094 oz. Au. per ton. The cyanide tails produced by the fourth year Mining Class carried about 0.125 to 0.16 oz. Au. per ton. The low extraction obtained in the treatment of the last tails is due largely to the loss in gold in the elutriated material. The concentrate is slightly higher grade, but the tails assay 0.085 oz. Au. per ton as an average value.

The small scale tests may be divided into two broad classes,- first those on the mill tailings and secondly those on fresh tailings from local cyanide tests.

The first group consisted chiefly of an investigation of the action of certain addition agents. An elaborate series of tests for each reagent, though perhaps desirable, was not carried out due to lack of time. Hence it is not possible to give definite statements as to the effect of various concentrations of each agent. The reader is referred to the test sheets in the Appendix for data.

Referring to the first two tests, it is natural to expect that grinding in oil would increase the extraction. That this is not the case, as indicated by the tests, is due to the small amount of oil used in the second test.

The action of both Sodium Sulphide and Potassium Xanthate is favourable as shown by tests 3 and 4 respectively. The froth, however, is still of a poor, weak character. The extraction is the highest obtained in the whole series. The note on page 61 on the action of Sodium Sulphide should be recalled here. The cyanide has removed all the free gold, hence there should be no gold loss due to the action of the Sodium Sulphide which will have full opportunity to exercise its good effect on the sulphide minerals and on the tellurides. It is highly probable that most of the gold is in the form of tellurides in the cyanide tailings since they are not cyanide-soluble.

The appearance of the froth was greatly improved by the addition of both Potassium Xanthate and Sodium Sulphide, a really rich well-mineralized froth being obtained.

In Test 5 the extraction is very low. This may be accounted for by the high tailings value which may be the result of faulty assaying. The total gold value in this and the following test is very much higher than in any of the others.

Test 6 should not be seriously considered. More reliance may be placed on the results of Test 8, which duplicated the procedure of this trial. The acid does not seem to have any decided effect, either favourable or otherwise. This remark applies also to the other reagents as the difference in extraction is very small, and the tails and concentrate assays of one test approximate respectively those of other tests.

The remark on Test 9 may require some explanation. The Wright-Hargreaves Company is credited with having reported that they found a lime froth suitable for flotation of the tellurides. The appearance of the froth as obtained in Test 9 was decidedly unfavourable, as it was evanescent and bore no mineral. This state of affairs was only corrected eventually by the addition of a large amount of oil, which had the effect of apparently killing the lime froth. The time occupied by this test, and many others for that matter, is altogether too great for commercial consideration.

The remaining tests are not outstanding and the discussion of the previous tests will pretty well cover all points arising in the others. In tests 12 and 13 respectively, F.P.L.#26, a hardwood creosote oil, and T. T.(thio - carbanilid dissolved in ortho-toluidine), were used. Their action seems to have produced no results out of the ordinary.

Agitation with Potassium Xanthate and Sodium Sulphide, to give better contact, seemed to have no particular effect.

The use of Pine Oil (G.N.S.5) as the only frothing and collecting medium does not seem to be very effective. The high grade of the concentrate should be noted. The action is due to the fact that pure oil is soluble, hence tends to produce a weak and somewhat voluminous froth with but little carrying power.

Tests 16 to 19 do not require a great deal of discussion beyond what has already been given at the beginning of this section. The floated material, though skimmed off, had very little weight and appeared to be chiefly viscous bubbles with a small amount of solids in a state of very minute subdivision. It was not thought worthwhile to assay the products at the time, a decision that is now rather regretted as it would have thrown some light on the nature of the float.

Tests 20 - 22 inclusive show a very poor gold recovery though the concentrate assays about 3 ozs. Au. per ton. The flotation tails average somewhat less than 0.09 oz. Au. per ton. The reason for the low gold extraction figure is that it is based on the total gold in the feed. It should be noted that over 42% of the total gold value lies in the elutriated material which was discarded. The concentrates contain about 60% of the gold in the flotation feed.

In test 23 the low gold recovery is doubtless due to the small amount of oil fed. Test 24 shows a decided improvement all around. The extraction is better, and the grade of the concentrate is lower, while that of the tails is really quite low. It should be noted that the feed is higher in value than in previous tests.

Before really satisfactory results can be obtained, a thorough microscopical study should be made of the tails to determine in what form the gold is contained. This would enable reasonable methods for treatment to be planned. A great deal more experimental work would have been done on the flotation of cyanide tailings had not time been lacking. This is a very desirable subject for further research.

TABLE 7.

Case No.	Cyanide Test No.	Flot'n. Test No.	Ore	Assays Oz. Au. per Ton Orig.Feed	Final Tails	%Extractn based on Feed &Tail Values.	Remarks.
1	1	24	Wright-Hargreaves	0.468	0.075	84.0	Cyanidation, tls.floated in single cell M.S
2	3	7	"	0.56	0.036	93.5	Large Scale Flot'n. Test #7, followed by Cyanidation.
3	6	28	Lakeshore	2.39	0.117	95.2	Cyanidation, Tls.floated in single cell M.S
4	8	11	"	2.8	0.235	91.6	Large Scale Flot'n. Test #11 followed by Cyanidation.

The above table affords us an interesting though incomplete comparison of the relative effectiveness of flotation followed by cyanidation, and cyanidation with subsequent flotation treatment of the tails.

In Cases 1 and 3 cyanidation was followed by flotation of the tails, the latter operation being performed on the single cell Minerals Separation machine. Only the best results obtained are recorded above. The poor results obtained on Wright Hargreaves are are disappointing; it must be remembered, however, that the flotation of cyanide tailings from the Wright-Hargreaves mill produced tails that assayed as low as 0.03 oz. Au. per ton. This would bring the total extraction to about 95%. It is quite possible that as good results could be obtained on the fresh cyanide tails. The mill extraction by cyanidation is also higher than that obtained in the laboratory tests

The combination of cyanidation and flotation seems to be successful in treating rich Lakeshore ore, especially if the processes are carried out in the above order, flotation seeming to have the ability of extracting the cyanide insoluble gold after cyanidation.

The series of tests carried out on Lakeshore ore is particularly interesting as it enables comparisons to be made of both flotation and cyanidation alone, and of the combinations of these two processes. As above stated cyanidation followed by flotation gives most promise of being a commercial success.

Might have written

GENERAL SUMMARY AND CONCLUSIONS.

That flotation can successfully compete with cyanidation in the production of low grade tails and high gold extraction from a low grade Porcupine ore, has been proved by certain of the tests discussed above. Some additional work remains to be done in the raising of the grade of concentrate; suggestions as to possible tabling methods have, however, been made. In the first few tests it was noticed that quite a high grade concentrate can be made by re-floating or cleaning the concentrate. There is every possibility that this method would prove commercially successful.

Walter's conclusions, for the most part, agree with the writer's on the possibility of treating Kirkland Lake ore by flotation. There are strong indications that this will be successful, but sufficient work has not yet been done either to condemn or promote flotation as a means of gold recovery.

Probably the solution of the Kirkland Lake problem of the recovery of the tellurides lies in a method involving cyanidation followed by flotation of the tailings. This treatment has given the lowest tailings and highest extraction obtained by any method, this being especially true in the case of high grade ore. The writer wishes to insist on the necessity of further work on this phase, particularly the carrying out of additional large scale tests which should give a good indication of the commercial possibilities of such a process.

The mechanical type of flotation machine seems to give slightly better results, fulfilling predictions made by Prof. Erlenborn and others.

Further work on this subject should be directed towards a study of suitable reagents, fineness of grinding, pulp dilutions, amount of oil and what oils are most effective. This research should be based on, and guided by microscopical study and chemical analysis of the tailings in order to ascertain in what form the unrecovered gold values occur. These tests could be first carried out with the single cell Minerals Separation machine. The best results obtained in any series should form the basis for a thorough and carefully executed trial, using the model mill circuit, treating several hundred pounds of ore in a test.

Since assaying the various products of the different tests, consumes such a large proportion of the whole time, comparatively little actual research work is done. A great deal more would be accomplished if the services of a competent assayer were continually available. This would also no doubt be conducive to more accurate results.

No mention has been made in any part of the thesis of the means of extracting the gold from the flotation concentrate. This is beyond the scope of this paper, hence the reader is referred to the theses of R. E. Legg and A.J.P. Walter, previously mentioned. Both these writers treat this subject in considerable detail. Some interesting data on the treatment of gold concentrate by cyanidation is given in the Mining and Scientific Press, May and June, 1911 in a series of articles by J. W Hutchinson, dealing with the operations of the Goldfields Consolidated mill.

Henry T. Airey

ACKNOWLEDGMENTS.

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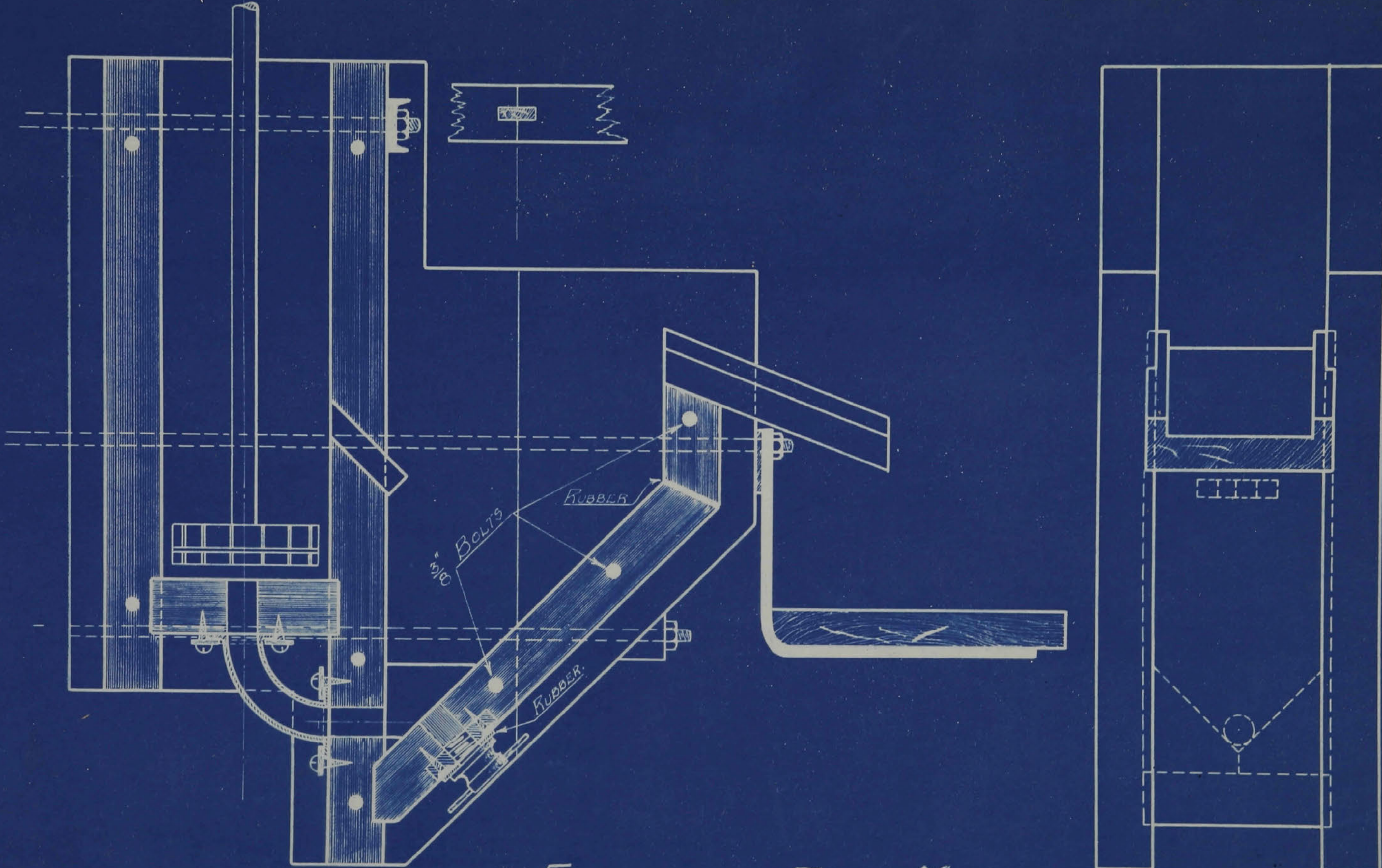
Thanks are also due to Messrs. Dewar, Findlay and Tatley, Senior Mining students, for carrying out the cyanidation tests under Professor Bell's direction, and to the staff of the Metallurgical Department, who willingly gave assistance and advice in the assaying work.

APPENDIX.



FLOTATION TEST MACHINE

MINERALS SEPARATION TYPE

Fig. 1

ORE DRESSING LAB. McGILL UNIVERSITY.

Scale – 3 in. = 1 Foot.

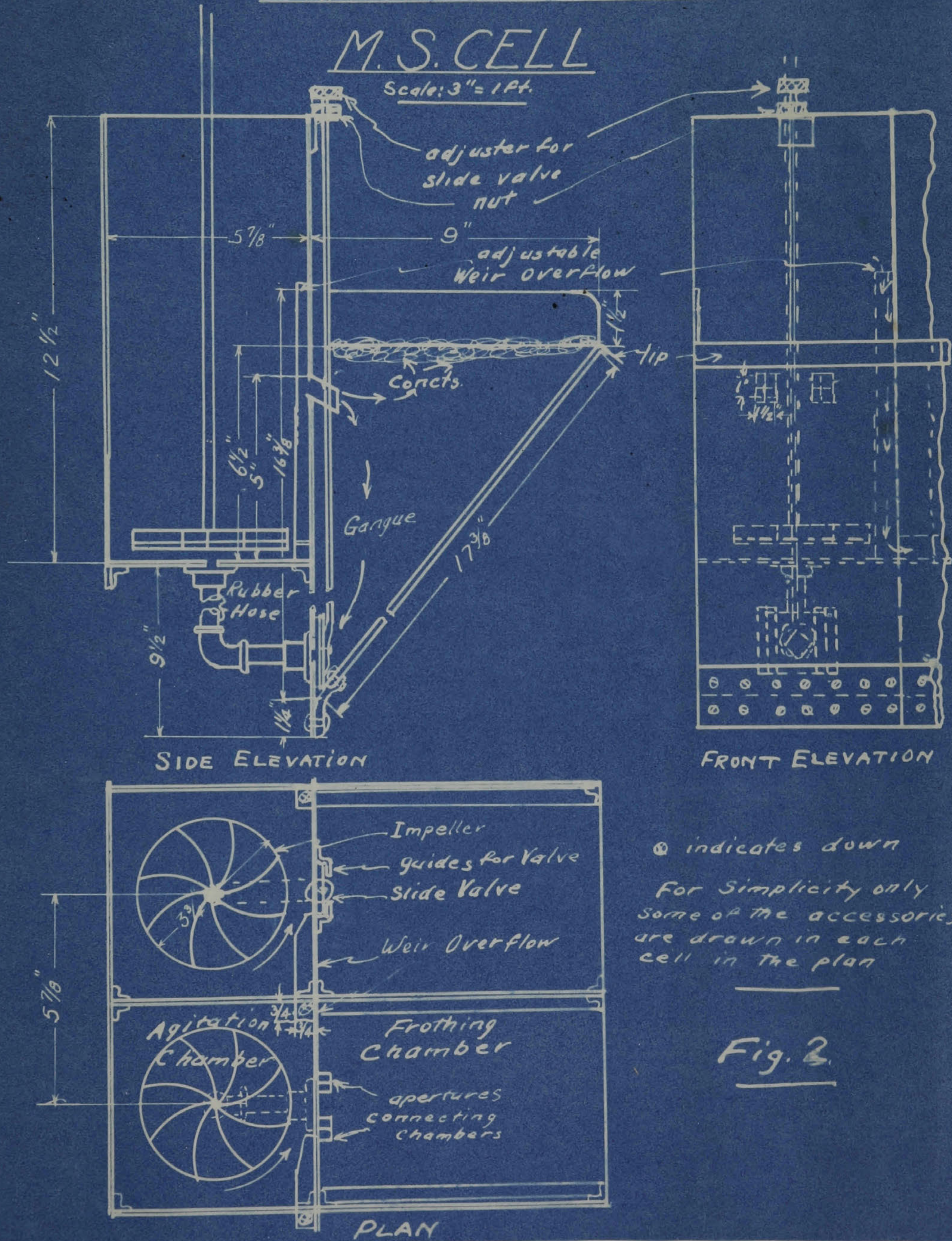
C.L.D.

1926-27

MINING DEPT. McGILL UNIVERSITY.

DIAGRAM
OF
PARALLEL-CONTROL MODIFIED
M.S. CELL

Scale: 3" = 1 ft.



⊗ indicates down

For Simplicity only some of the accessories are drawn in each cell in the plan

Fig. 2.

FLOW SHEET OF MODEL MILL

MINING DEPT. McGILL UNIVERSITY.

1926-27.

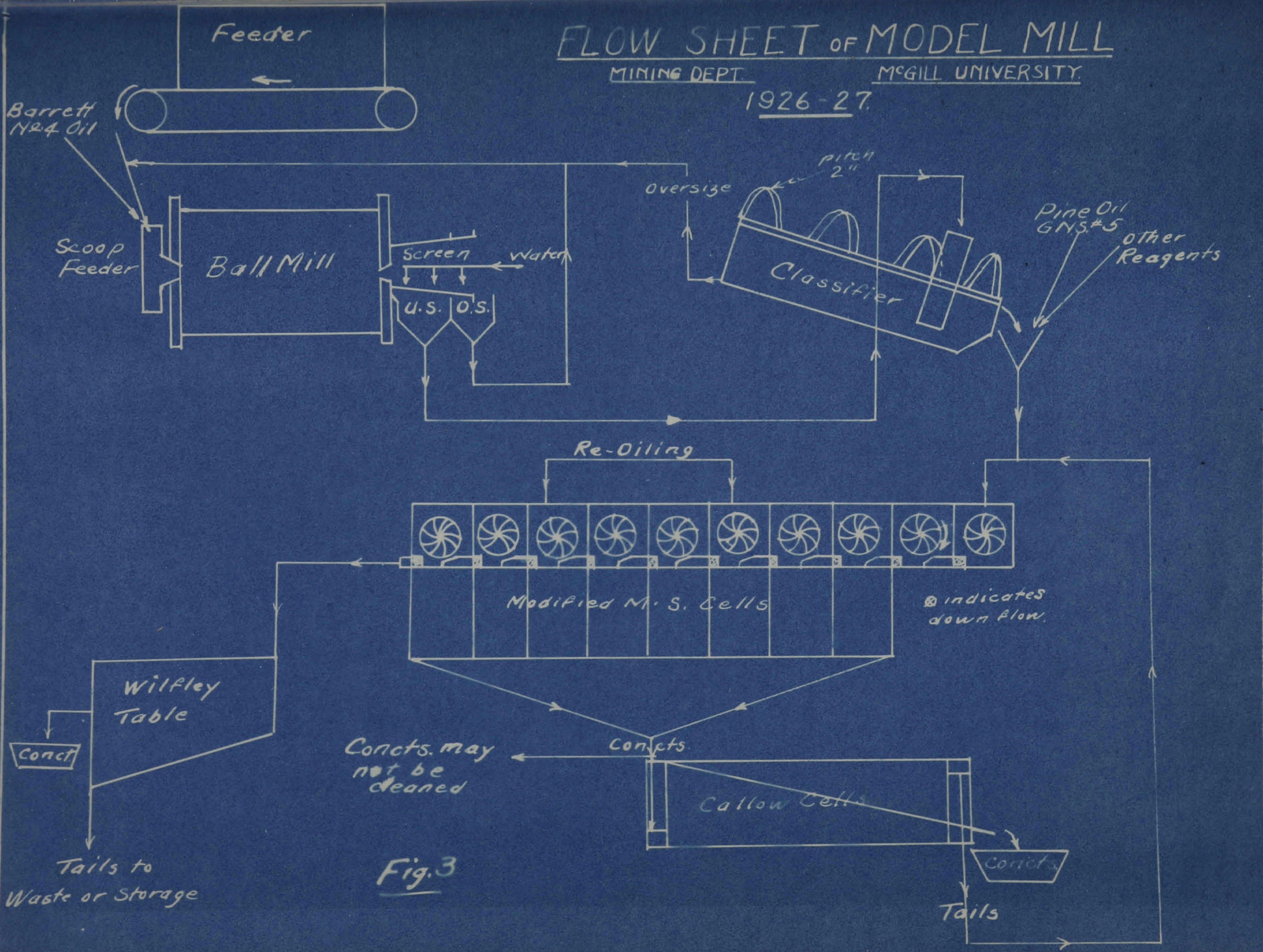
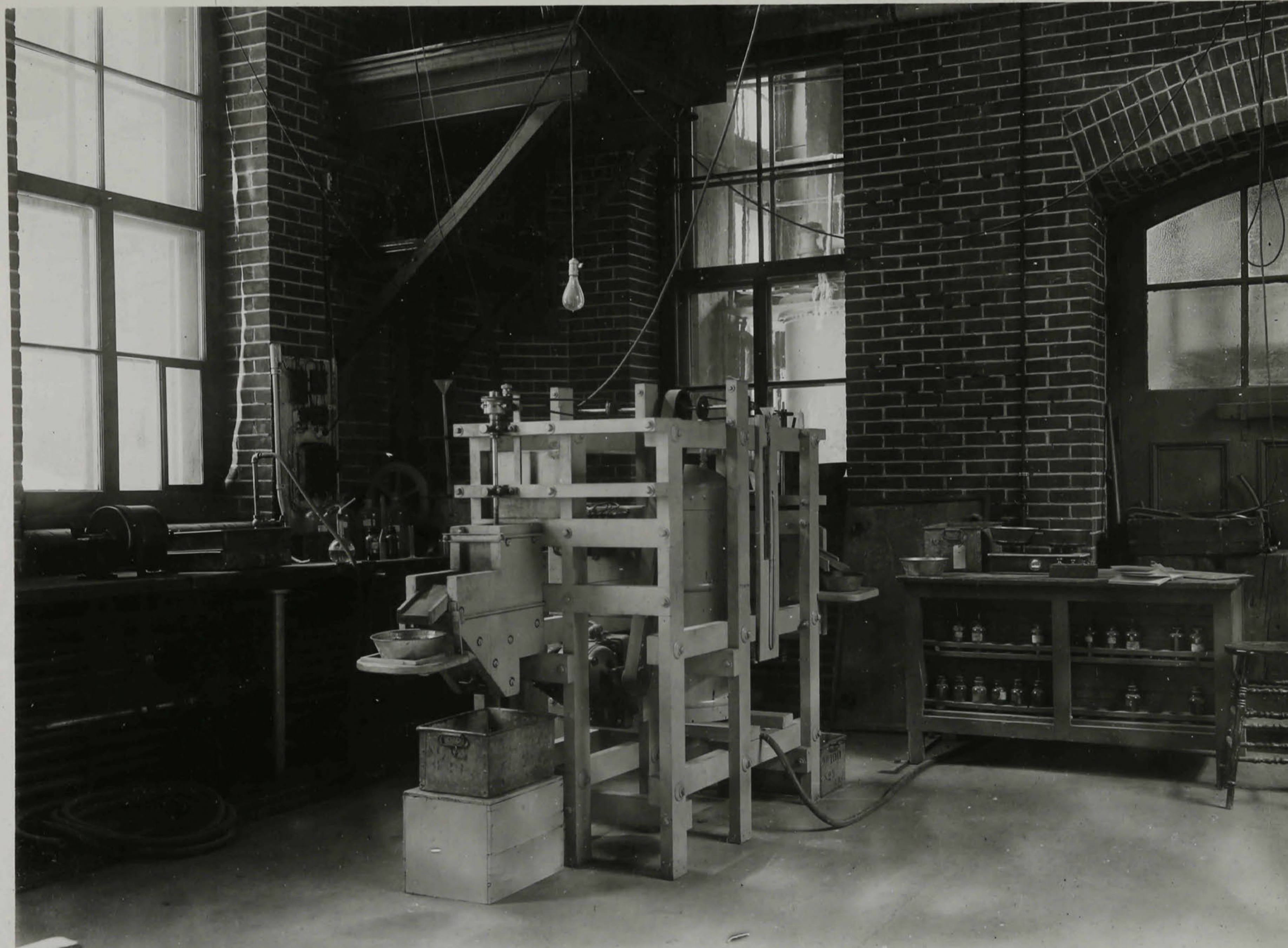
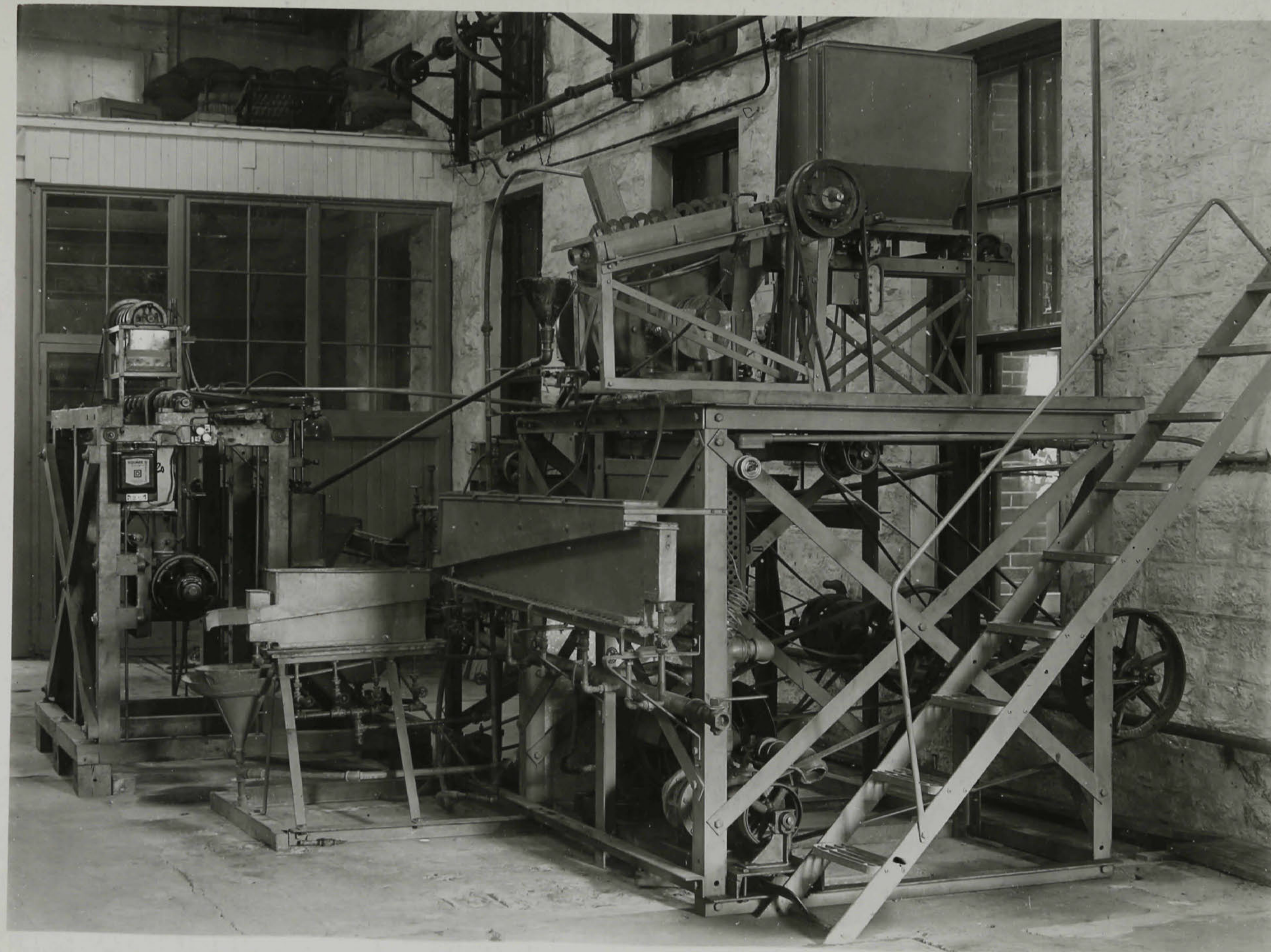


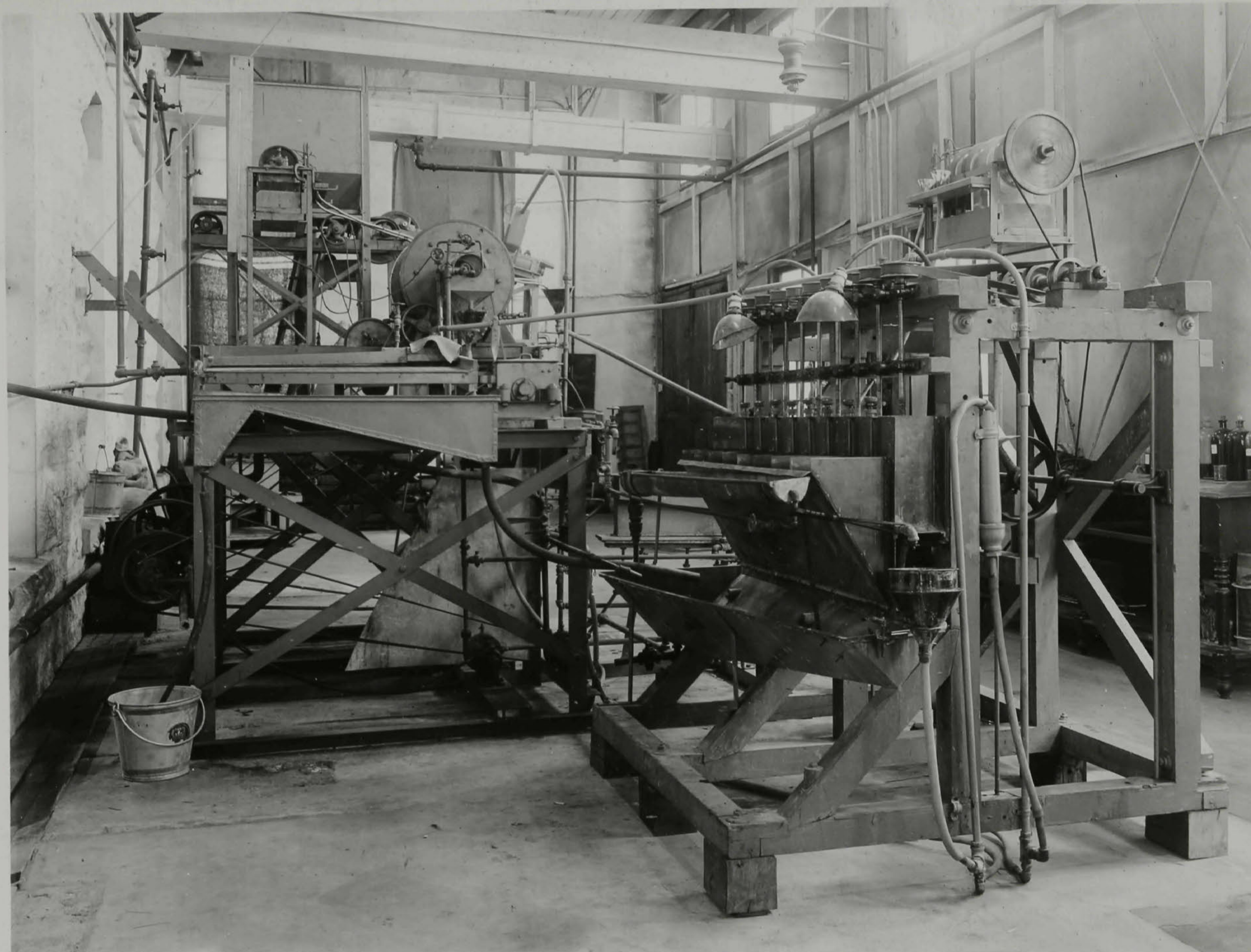
Fig. 3



Single Cell M.S.
Plate 1

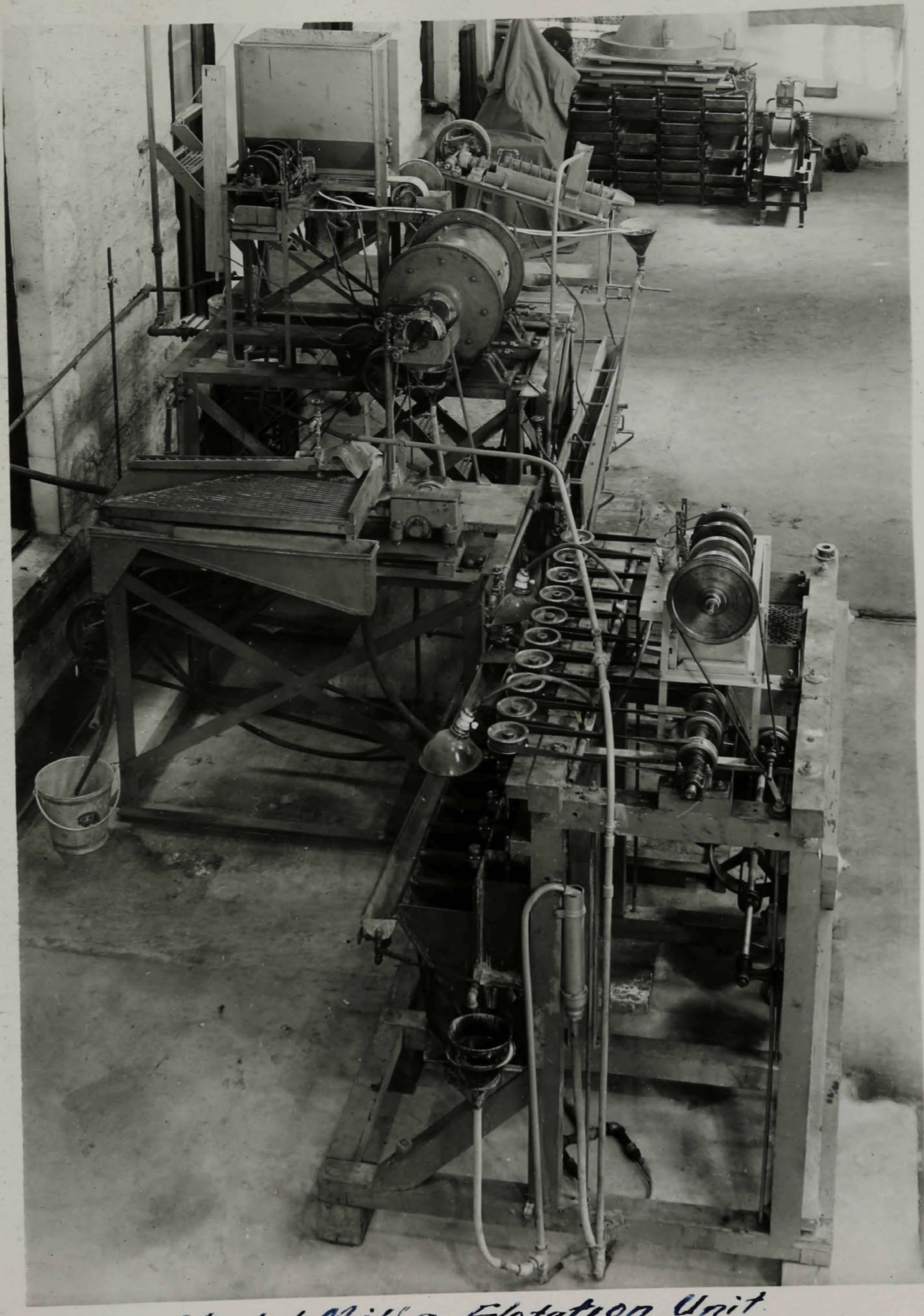


Model Mill and Flotation Unit
Plate 2.

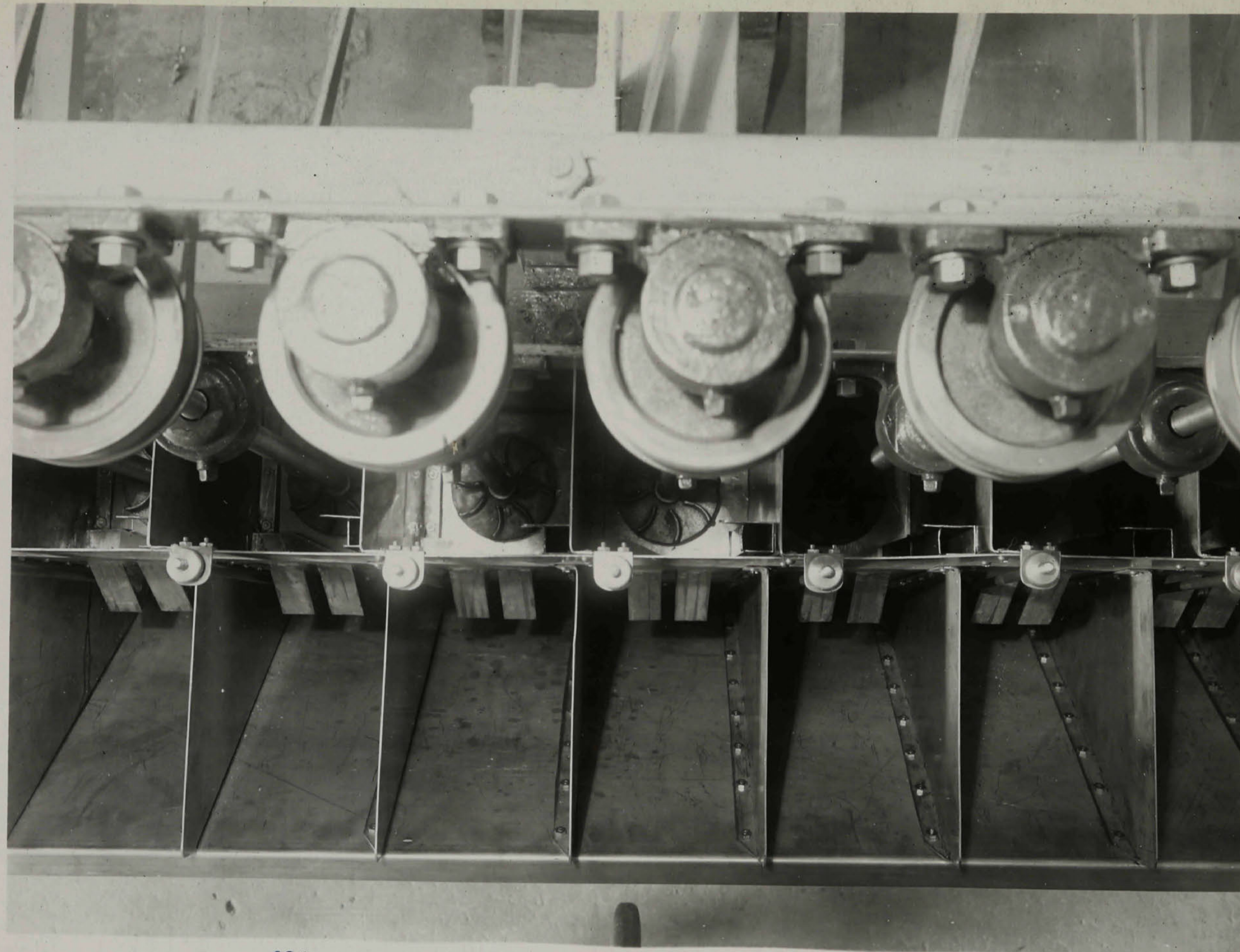


Model Mill and Flotation Unit

Plate 3.



Model Mill & Flotation Unit
Plan
Plate 4



Plan View of Parallel-Control, Modified
M.S. Flotation Cells.
Plate 5.

Test No. 1.　　Ore No. 199.　　Date:- Oct.21/26.

Duration of Run:- 1 hour 20 mins.

Object of Test:- Concentration.

Liquid Solid Ratios		Air Pressure	Height of Froth	Reagents	~~Lbs. per ton of Solids~~	Lbs. per ton of Solids
Feed				B.#4	10 dr./min.	= 1·51
Tails				G.N.S.#5	2½-3dr./min	= 0·424
Assay Samples						
Micrometer Setting			9			
Feed per hr, indicated			27.4#			
Revolution of Counter						
Feed per 1000 Revs.						

Products	Weights	% Weights	Oz. /ton	Oz./ton x weight	Screen Analysis of Feed			
					Grade	%	Grade	%
M.S.Feed			0.240					
" Conc.			0.700		+20		+100	8.31
Tails			0.0125		28		150	11.24
C.Conc.	2318 gm.		3.050		35		200	2.27
					48		-200	77.98
					65		-300	
Total								
Feed	40.32#							

% Error in balance =　　% Extraction =

% Reduction from values in feed to values in tailings = 94.7

Remarks:- This test was run largely to try out the machine and to gain experience in manipulation.

Test No. 2 · Ore No. 199 Date:- Nov. 3/26.

Duration of Run:- 4 hrs. 0 min.

Object of Test:- Concentration.

Liquid Solid Ratios		Air Pressure	Height of Froth	Reagents	~~Lbs. per ton of Solids~~	Lbs. per ton of Solids
Feed	6.7:1 ~~3.2:1~~	9		B #4	12 dr/min.	1·81
Tails				G.N.S.#5	$2\frac{1}{2}$ dr/min.	0·424
Assay Samples						
Micrometer Setting			9			
Feed per hr. indicated			27.3 #			
Revolution of Counter						
Feed per 1000 Revs.			16.9 #(m)			

Products	Weights	% Weights	Oz. /ton	Oz./ton x weight	Screen Analysis of Feed			
	(Wt. of Sample)				Grade	%	Grade	%
M.S. Feed	767 gms.		0.255					
"Conc.	93		0.70		+20		+100	6.38
Tails	2525		0.02		28		150	9.98
C. Conc.	3480		2.575		35		200	2.64
Tails	66 gr.				48		- 200	78.48
					65	2.41	-300	
Total								
Feed	127.74 #		0.255					

% Error in balance = % Extraction =

% Reduction from values in feed to values in tailings = 98.0

Remarks:-

Wash Water to B.M. = 1.050 = 42.# / hr.

~~B.M. outlet Screen 65 mesh.~~

Test No. 3	Ore No. 199		Date:- Nov. 4/26.
Duration of Run:- 1 hr. 08 mins.			
Object of Test:- Concentration and effect of Coarser Grinding.			

Liquid Solid Ratios		Air Pressure	Weight of Froth	Reagents	~~Lbs. per ton of Solids~~	Lbs. per ton of ~~Solution~~ solids
Feed	1.41:1 5.66:1	9		B #4	12 dr./min.	0.93
Tails				G.N.S.#5	2 " /min.	0.145
Assay Samples						
Micrometer Setting			FULL Throw			
Feed per hr. indicated			53.3 #			
Revolution of Counter						
Feed per 1000 Revs.						

Products	Weights gms.	% Weights	Oz. /ton	Oz./ton x weight	Screen Analysis of Feed Grade %		Grade %	
M.S.Feed	742		0.255		+20		+100	8.65
~~" Conc.~~	~~2122~~		~~0.825~~		28		150	12.30
~~Tails~~	~~1264~~		~~0.0125~~		35		200	2.48
C.Conc.	2264		2.475		48		300	73.70
					65	2.45	-300	
Total								
Feed			.255					

% Error in balance = % Extraction =

% Reduction from values in feed to values in tailings = 97.0

Remarks:-

B.M. WASH WATER = 1.10 = 53.5 #/hr.

Test No. 4	Ore No. 199	Date:- Nov.17/26.

Duration of Run:- 10.54 a.m. to 3.52 p.m. 4 hrs. 58 mins.

Object of Test:- To study effect of re-oiling.

Liquid Solid Ratios		Air Pressure	Weight of Froth	Reagents to B.M.	Lbs. per ton of Solids	Lbs. per ton of Solids
Feed	4.03:1	8" Hg.		B #4.	13 dr./min.	1.13
Tails				G.N.S.5	1 "/min.	0.084
Assay Samples						
Micrometer Setting			FULL Throw			
Feed per hr. indicated			47.3 #			
Revolution of Counter			8086 Total			
Feed per 1000 Revs.			29.1#			

Products Samples	Weights lbs.	% Weights	Oz. /ton	Oz./ton x weight	Screen Analysis of Feed			
M.S.Conc.t	.338		2.710	0.915	Grade	%	Grade	%
Tails	3.92		0.0178	2.165	+20		+100	8.92
	4.25		0.030	2.435	28		150	12.38
~~Clean R.~~			3.336	3.150	35		200	2.84
" C	16.82		3.50	58.90	48		- 300	73.18
M.S.Res.	12.87		0.12	1.545	65	2.62	-300	
Total				69.10				99.94
Feed	235.45#		0.375	87.9				

% Error in balance = 21.15 % Extraction = 73.8%.

% Reduction from values in feed to values in tailings = 95.7

Remarks:- *See Notes in Text Re. Extraction.*

B.M. Wash Water O.S. 1.10 = 53.5 # / hr.

Re-oiling in 4th M.S.cell B #4. 4 drops / min.
" " " 6th " " 2 " / " } *0.524#/Ton.*

Sampling started at 11.35.

Tails pumped to large Cone for storage.

Test No. 5	Ore No. 199	Date:- Dec.2/26.

Duration of Run:- 4 hrs. 30 mins.

Object of Test:- To test cap'y of M.S.

Liquid Solid Ratios		Air Pressure	Weight of Froth	Reagents	~~Lbs. per ton of Solids~~	Lbs. per ton of ~~Solution~~ Solids
Feed		8" Hg.		B #4(B.M.)	15-18 dr.	/min. = 1·29
Tails				G.^{N}.S. 5.	2-2½ " /	" 0·227
Assay Samples						

Micrometer Setting	Full Throw			
Feed per hr. indicated	52.65			
Revolution of Counter				
Feed per 1000 Revs.	34.4 #			

Products	Weights (~~Dry~~)	% Weights	Oz. /ton	$ ~~xxx~~ Oz./ton x weight	Screen Analysis of Feed Grade	%	Grade	%
M.S. 1	22.06 lbs.		2.08	920.00				
" 2-3	3.56		3.10	220.50	+20		+100	9.55
" 4-5	3.46		0.56	76.70	28		150	13.35
"6-7-8	11.32		0.18	40.75	35		200	3.62
" Tails			.015	52.07	48		- ~~2~~300	71.20
M.S. Res.	13.69		.90	24.62	65	2.17	-300	
Total	~~8~~			1334.64				
Feed	238.45		.295	1347.00				

% Error in balance = 0.92 % | % Extraction = 95.3 %

% Reduction from values in feed to values in tailings = 95.0

Remarks:- B.M. Wash Water O.S. 1.10 = 53.5#/ hr.

Re-oiling in 4th M.S. Cell 3 dr / min. to 6 dr. / min. = ·236 #/Ton

" " 6th " " 4 dr./ min. = ·314 " "

~~Pine Oil discontinued at 1.20 p.m.~~

Callow cell not used for cleaning.

Test No. 6 Ore No. 199 Date:- Dec.14/26

Duration of Run:- 5 hrs. 46 mins.

Object of Test:- Concentration (duplicate of Test 5 with lower feed per hour.)

Liquid Solid Ratios		Air Pressure	Height of Froth	Reagents	~~Lbs per ton of Solids~~	Lbs. per ton of Solids
Feed	3.84:1	8½ "		B #4	15 dr./min.	1.53
Tails				G.N.S.5	1-2" 1"	0.095
Assay Samples						
Micrometer Setting			13			
Feed per hr. indicated			40.61			
Revolution of Counter						
Feed per 1000 Revs.			25.66			

Products	Weights lbs.	% Weights	Oz./ton	Oz./ton x weight	Screen Analysis of Feed			
					Grade	%	Grade	%
Tails	173.94		0.015 to 0.0099	46.16				
M.S.1	34.31		1.50	1015.00	+20		+100	7.41
" 2 & 3	3.34		2.23	148.70	28		150	11.62
" 4 & 5	1.82		1.19	43.10	35		200	3.61
"6,7 & 8	6.69		.32	42.40	48		- 200	75.1
M.S.Res'd	14.50		.137	38.40	65	2.16	-300	
Total				1333.76				
Feed	234.6		.31	1503.0				

% Error in balance = 11.4% % Extraction = 85.6

% Reduction from values in feed to values in tailings=

Remarks:- B.M. O S. Wash water 53.5 #/ hr.

B #4 to 4th Cell 3 dr. / min. = 0.306 #/Ton Solids

6th " 4 " " = 0.406 " " "

Pine oil stopped at 1.55 p.m. See Notes in Text re. Extraction.

M.S. motor drew mean of 5.8 amps.

17.5 lbs. discrepancy between feed and total wt. of products.

Test No. 7 Ore No. 201 Date:- Feb.25/27.

Duration of Run:- 5 hrs. 10 mins.

Object of Test:- To test A~~t~~kins Classifier, and to effect concentration by Flotation.

Liquid Solid Ratios		Air Pressure	Height of Froth	Reagents	Lbs. per ton of Solids	~~Lbs. per ton of Solution~~
Feed	4:1	9"		Na_2S	1. #	
Tails				K. Xanthate	0.1#	
Assay Samples				G.N.S.5	0·290 # 3 dr./min.	
Micrometer Setting			12& 10	B.#4	1·29 # 12 " "	to Clas.Disch.
Feed per hr. indicated			43& 36-	"	0·220 # 2	" 3rd M.S.Cell
Revolution of Counter			1568 &6571	"	0·332 # 3	" 5th " "
Feed per 1000 Revs.			54.9 & 38.3			

Products	Weights lbs.	% Weights	Oz. /ton	Oz./ton x weight	Screen Analysis of Feed Grade	%	Grade	%
Tails 1			.09		+20		+100	
" 2			.12		28		150	
" 3			.11		35		200	11.4
Concts.			5.83		48		300	31.9
					65		-300	57.
Total								
Feed	16.0		.56					

% Error in balance = No attempt at Balance % Extraction = 80.35%

% Reduction from values in feed to values in tailings =

Remarks:- Ball Mill seemed unable to Grind 43#/hr. sufficiently fine, a good balance was obtained in the circuit feeding 35#/hr. This was the first time the Classifier and the lifters in the B.M. had been used in a flotation test. Considerable slip in Feed belt - should be chain driven.

Test No. 8	Ore No. 201	Date:- Mar.14/27.

Duration of Run:- 3 hrs. 32 mins.

Object of Test:- Concentration using no reagents other than oil.

Liquid Solid Ratios		Air Pressure	Height of Froth	Reagents	Lbs. per ton of Solids	~~Lbs. per ton of Solution~~
Feed	3.5:1	9"		B #4.	1·43# 12 dr./min	to Ball Mill
Tails				"	0·246# 2 " / "	"3rd M.S. Cell
Assay Samples				"	0·366# 3 " / "	6th M.S. "
Micrometer Setting			10			
Feed per hr. indicated			33.85			
Revolution of Counter						
Feed per 1000 Revs.						

Products	Weights lbs.	% Weights	Oz. /ton	Oz./ton x weight	Screen Analysis of Feed			
					Grade	%	Grade	%
Tails (1)			.105		+20		+100	1.62
(2)			.065		28		150	
(3)			.065		35		200	9.96
~~Conct.~~	~~10.1~~		~~4.6~~		48		300	88.38
					65		-300	
Total	106.3		.56					
Feed								

% Error in balance = | % Extraction =

% Reduction from values in feed to values in tailings =

Remarks:- Water added before classifier = 105 lbs./ hr.

~~Table not used at all.~~

Pine oil added in 6th M.S. cell seemed to improve appearance of conct. Samples were cut so as to divide run into 3 nearly equal parts. ~~Three screen analyses made. 2nd is given above.~~

Test No. 9 | Ore No. 201 | Date:- Mar.15/27.

Duration of Run:- 4 hrs. 1 min.

Object of Test:- Duplicate of Test 8 except reagents used.

Liquid Solid Ratios		Air Pressure inches Hg.	Height of Froth	Reagents	Lbs. per ton of Solids	Lbs. per ton of Solution
Feed	3.84:1	9"		B #4	1.38# 12 dr./min.	to B.M.
Tails				"	0.226# 2	" 3rd Cell.
Assay Samples				"	0.446# 3	" 6th "
Micrometer Setting			10	G.N.S.5	0.215# 2	" 6th "
Feed per hr. indicated			36	"	0.108# 1	Class.Discharge
Revolution of Counter				K.Xanth.	0.1	
Feed per 1000 Revs.			20.0	Na_2S	1.0	

Products	Weights	% Weights	Oz./ton	Oz./ton x weight
Tails 1			.0825	
2			.0975	
3			.10	
Concts.	12.75		5.8	
Total				
Feed	143		.76-	

Screen Analysis of Feed

Grade	%	Grade	%
+20		+100	0.90
28		150	
35		200	8.22
48		300	90.86
65		-300	

% Error in balance =

% Extraction =

% Reduction from values in feed to values in tailings =

Remarks:- Sampling as in Test 8.

Water before Class " " " " viz. 105 # / hr.

Table not used at all.

~~Very good froth in 1st two cells.~~

Very little pine oil desirable.

Three screen analyses made; 2nd is given above.

Test No. 10. Ore No. 201. Date:- Mar.16/27.

Duration of Run:- 2 hrs. 6 mins.

Object of Test:- To study effect of coarser grinding.

Liquid Solid Ratios		Air Pressure	Height of Froth	Reagents	Lbs. per ton of Solids	~~Lbs. per ton of Solution~~
Feed	2.4:1	9		F.F. B #4	1.15# 12 dr./min.	to B.M.
Tails				"	0.195# 2 " "	"3rd M.S.cell
Assay Samples				"	0.195# 2 " "	"6th M.S.cell
Micrometer Setting		10,11,12,& 13		G.N.S.#5	0.087# 1 " "	"Class.Discharge
Feed per hr. indicated		mean = 43		K.Xanth.	0.1 "	" "
Revolution of Counter				Na_2S	1.0 "	" "
Feed per 1000 Revs.						

Products	Weights	% Weights	Oz. /ton	Oz./ton x weight	Screen Analysis of Feed			
					Grade	%	Grade	%
Tails 1			.165		+20		+100	1.82
2			.115		28		150	4.40
Concts.	8.65		5.3		35		200	9.34
					48		300	72.06
					65	2.24	-300	
Total								
Feed	93		.71					

% Error in balance = % Extraction =

% Reduction from values in feed to values in tailings.=

Remarks:- See notes in Text on this test.
Water before Classifier = 90 lbs./ hr.

Two screen analyses made; the ore given above is the second.

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	1 W.T.
DATE	Jan. 18. 1927
ORE NO.	W.T.
PULP RATIO	1 : 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1200 gms. Ore fed to B.M. with 1200 gms water and ground for 5 mins.

Grinding started 10:52
stopped 10:57

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
11.05						Ore chg'd M.S.		
11.06	B #4	10				No Min'l Froth		
11.10	GNS #5	2				Weak Fr; little Minl		
11.13	B #4	3				slightly better Fr.		
11.18	B #4	3				as above.		
11.25	B #4	4				Stronger fr; more minl.		
11.30	B #4	2				little minl. Ev. Fr.		
11.39	B #4	1				" " " "		
11.48	B #4	3				" " " "		
11.57	B #4	1				" " " "		
12.00	B #4	3				" " " "		
12.05						TEST STOPPED		
						Conct.	WT 1C	14
						Tails	WT 1T	947

REAGENT	%	LBS. P.T.
B# 4		1.094
GNS 5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.
	WT 1C	01.5	3.55	0.0532							51.85
	1T	98.5	.05	0.0493							48.15
				.1025							100.00

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	2 WT.
NOTES: 1200 gms. Ore ground in B.M. for 10 mins with 1200 cc. water. Then 10 drops of B #4 added and grinding cont. for 5 mins	DATE	Jan. 18. 1927
Grinding started: 2.30 p.m. 2.46	ORE NO.	W.T.
" stopped : 2.40 " 2.51	PULP RATIO	1 : 1
	R.P.M. IMPELLER	
	PULP TEMP. START	°
	PULP TEMP. FINISH	°
	PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
2.57						Ore chg'd to M.S.		
2.58	G.N.S.#5	1				Slight froth		
2.59	B #4	3				" "		
3.01	G.N.S.#5	1				" "		
3.10	B #4	2				" "		
3.16	B #4	3				slight improvement		
3.31	B #4	3				same		
3.42	B #4	4						
3.57						Test stopped		
						Conct.	W.T.2C	14
						Tails	W.T.2T	957

REAGENT	%	LBS. P.T.
B#4		0.811
GNS 5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.			
	W.T. 2C	1.44	3.40	0.0488							49.7			
	2T	98.56	0.05	0.0492							50.3			
		100.00		0.0980							100.0			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	3 W.T.
DATE	Jan. 19. 19 27
ORE NO.	W.T.
PULP RATIO	1 : 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1200 gms. are ground in B.M. for 10 mins. with 1000 cc. water with 10 cc. Sodium Sulphide (1#/ton).

Grinding started 3.20
" stopped 3.30

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
3.35						Ore chgd to M.S.	Very Evan-	
3.50	B #4	10				Poor Froth but	escent, Minl.	
3.51	GNS #5	2				slightly better	shows	little
3.55	B #4	3				than in previous	tendency	to stay in Froth.
4.05	B #4	7				Tests.		
4.24	B #4	5						
4.40	B #4	8				Slight Improvement		
4.45	B #4	3				" "		
4.49	B #4	3				Worse - no minl		
5.04	B #4	3				" "		
5.10						Test stopped		
						Conct.	WT 3C	22
						Tails	WT 3T	944

REAGENT	%	LBS. P.T.
B #4		1.32
GNS 5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.			
	W.T.3C	2.275	2.40	0.0545							58.15			
	3T	97.725	.04	0.0391							41.85			
		100.000		0.0936							100.00			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	4 W.T.
DATE	Jan. 20 1927.
ORE NO.	W.T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1200 gms. ore ground with 30 cc. K. Xanthate and 1200 cc. water in B.M. for 10 mins.

Grinding started 9.40 a.m.
" stopped 9.50

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
9.55						Ore chgd. to M.S.		
10.10	B #4	15						
10.10	GNS 5	2						
10.15	B #4	5						
10.20	B #4	5				Stronger Froth than previously.		
10.25	B #4	5				Slightly better " "		
10.38	B #4	5				" "		
10.46	B #4	5						
11.00	B #4	6						
11.00	"	5				Little Mineral		
11.18						Test stopped		
						Concts	W.T.4C	23
						Tails	W.T.4T	945
								968

REAGENT	%	LBS. P.T.
K.Xanthate	30cc	= .105#' To:
B #4		1.56
GNS.5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	O% Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT
	W.T.4C	2.38	2.40	.0570						
	4T	97.62	0.04	.0391						
				.0961						

PERCENT OF TOTAL

Au.			
59.4			
40.6			
100.0			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	W.T.5
DATE	Jan. 20 1927
ORE NO.	W.T.
PULP RATIO	1 : 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1200 gms. Ore ground for 10 mins. with 1# Na_2S/Ton and 0.105# K. Xanthate/Ton with 1000 cc. water.

Grinding started 10.46 a.m.
" stopped 10.56 "

REAGENT	%	LBS. P.T.
K.Xanthate	30cc.	0.105
Na_2S	10cc.	1.00
B #4		1.41
G.N.S. 5		0.175

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
11.22						Ore fed M.S.		
11.37	B #4	10			(	Fair Froth, well		
11.40	B #4	5			(	mineralized		
11.43	B #4	5						
11.47	GNS 5	2				Good Froth		
11.50	B #4	5				" "		
11.53	B #4	5						
11.55	GNS 5	2				Rich Froth		
12.00	GNS 5	2				" "		
12.02	B #4	5				" "		
12.09	B #4	5				Little Mineral		
12.15	B #4	5				No "		
12.30			Test stopped			Concts.	W.T.5C	25
						Tails	W.T.5T	952

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	OZ. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	~~WEIGHT~~	PERCENT OF TOTAL Au.			
										977				
	W.T.5C.	2.56	2.20	.0563							36.7			
	5T.	97.44	0.10	.0974							63.3			
				.1537							100.0			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	W.T.6
DATE	19
ORE NO.	W.T.
PULP RATIO	1 : 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1200 gms. Ore with 10 c.c. 1:9 H Cl solution ground in B.M. for 5 mins. (1000 c.c. water in B.M.).

Grinding started 11.05 a.m.
" stopped 11.10 "

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
11.15						Ore fed to M.S.		
11.15	B #4	10				Agitated (little Water)		
11.30	B #4	5				Very poor Froth		
11.32	B #4	5						
11.35	GNS #5	2				Slight Froth Evanescent		
11.37	B #4	5				Ev. Fr. improvement		
11.39	GNS 5	2				Little Mineral		
11.44	B #4	5				Better Froth.		
11.47	B #4	5						
11.50	B #4	7						
11.57	B #4	8				Poorer Froth little Minl.		
1204	B #4	5						
12.12	H Cl.		20			No mineral, froth clean		
12.18	B #4	5				" " " "		

REAGENT	%	LBS. P.T.
H Cl	10% sol. 10cc.	
B #4		1.87
GNS 5		0.117

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.
	W.T.6C.	02.875	4.10	0.1178							75.35
	6T.	97.125	0.04	0.03885							24.65
		100.00		0.15665							100.00

Test stopped 12.24

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY.

FLOTATION TEST NO. W.T.7

NOTES: 1200 gms. Ore with 20 c.c. 1:9 H Cl solution, ground in B.M. for 5 mins with 500 c.c. water

DATE	Jan. 27 1927
ORE NO.	W.T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
10.35						Ore charged M.S.		
10.35	B #4	10				Agitated 15 mins.		
10.50	B #4	5						
10.52	B #4	5						
10.55	GNS #5	2				Fair Minl Fr. still Evanescent		
10.57	B #4	5				" " " " "		
11.05	B #4	5				" " " " "		
11.12	B #4	5				" " " " "		
11.20	GNS #5	2						
11.30	B #4	5						
11.36	B #4	8				Slight improvement		
11.41	B #4	7						
11.46	B #4	5				Less Min'l, Good Froth		
11.52	H Cl.		20			No " " "		

REAGENT	%	LBS. P.T.
H Cl	10	20 c.c.
B #4		1.87
GNS 5		0.117

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	OZ. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.			
	W.T. 7 C	2.65	1.9	.0540							58.2			
	W.T. 7 T	97.40	.04	.03895							41.8			
				.09295							100.0			

Test stopped 11.55

W.T. 7C conct. 26 gms. }
W.T. 7T Tails 957 " } 983 gms.

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	8
DATE	Jan. 31 1927
ORE NO.	W.T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: Repetition of Test 6 - 1200 gms. Ore ground in B.M. for 5 mins. with 10 cc. 1:9 H Cl. and 750 cc. water.

Grinding started 9.31 a.m.
" stopped 9.36 "

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
9.44						Ore chgd. M S.		
9.45	B #4	10				Agitated 15 mins.		
10.00	B #4	5				Poor Froth		
10.02	B #4	5				" "		
10.05	GNS #5	2				"Fluffy" Froth		
10.08	B #4	5				Skimming started		
10.10	GNS #5	2						
10.15	B #4	5						
10.18	B #4	5				Better froth more mineral		
10.20	B #4	7				" "		
10.27	B #4	8				" "		
10.34	B #4	5				Little Mineral		
10.43	H Cl.		20			No mineral		
10.48	B #4	5				" "		

REAGENT	%	LBS. P.T.
HCl	10	[illegible]
B#4		1.87
GNS5		0.117

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.
	W.T. 8C	02.12	2.6	.0551							57.5
	W.T. 8T	97.88	.04	.0409							42.5
				.0960							100.0

Test stopped 10.54 a.m.

Conct. W.T. 8 C. 21 gms.
Tails W.T. 8 T. 970 " } 991 gms.

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	9
NOTES: 1200 grms. Ore ground in B.M. for 5 mins with 750 cc. water and 1 gm. CaO. (Lime alone N.G. in spite of W.H.assertions, according to these results below); Started 10.51 (Has oil killed lime?) (Time too long - Stopped 10.56 (for common practice.)	DATE	19
	ORE NO.	
	PULP RATIO	: 1
	R.P.M. IMPELLER	
	PULP TEMP. START	°
	PULP TEMP. FINISH	°
	PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
10.02						Ore charged M.S.		
11.12						No Froth		
11.12	G.N.S.#5	3				Slight Froth;no mineral		
11.16	B #4	5				" " very little "		
11.20	B #4	5				" " " " "		
11.28	B #4	5				" " " " "		
11.30	G.N.S.#5	2				" "more mineral		
11.35	B #4	5				Fair " " "		
11.42	B #4	5				" " " "		
11.48	B #4	5				less "		
11.57	B #4	5						
12.02	B #4	8				Good Fr; more "		
12.12	B #4	7				Poorer Fr;less "		
12.23	B #4	5				" " V.little"		

REAGENT	%	LBS. P.T.
CaO		0.66
B #4		1.72
G.N.S.5		0.146

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz%. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.
	W.T.9C.	2.24	2.15	.0482							51.0
	W.T.9T.	97.76	.045	.0463							49.0
				.0945							100.0

Test stopped 12 33

Tails W.T. 9 T 959)
Concts. W.T. 9 C 22) 981

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO. 10.

NOTES: 1200 gms. ore ground in B.M. for 5 mins. with 750 c.c. of water, 15 c.c. sodium sulphide and 45 cc. K. Xanthate.

Grinding started 2.33 p.m.
" stopped 2.38 "

DATE	Jan. 31. 1927.
ORE NO.	
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
2.45						Ore chgd M.S.		
3.00	B #4	10				slight froth		
3.05	B #4	5				rich froth		
3.08	B #4	5				skimming started		
3.12	G.N.S.5	3				good rich froth		
3.15	B #4	5				" " "		
3.20	B #4	5				" " "		
3.25	G.N.S.5	3				" " "		
3.33	B #4	5				fair froth, little mineral		
3.39	B #4	5				" " very " "		
3.45						Test stopped		
						Concts.	W.T.10C	25
						Tails.	W.T.10T	.961
								.986

REAGENT	%	LBS. P.T.
Na_2S	15 cc.	1.5#
K.Xanthate	45 cc.	0.152
B # 4		1.25
GNS 5		0.175

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.
	W.T.10C.	2.535	2.2	.0559							56.4
+300	W.T.10T.	367	.0675	.0247							24.9
-300	W.T.10T.	619	.03	.0185							18.7
				.0991							100.0

+300 = 37.2 % of Total Wt. of Tails.
-300 = 62.8 " " " " "

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY.

FLOTATION TEST NO. 11.

NOTES: 1200 gms. ore ground in B.M. for 5 mins with 10 cc. 1:9 HCl; 30 cc. K.Xanthate; 10 cc. Sodium Sulphide and 750 cc. water.

Grinding started 9.39 a.m.
" stopped 9.44 "

DATE	Feb. 1st, 1927.
ORE NO.	W.T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
9.50						Ore charged M.S.		
9.53	B #4	10				Agitated 15 mins.		
10.05	B #4	5				Slight froth.		
10.08	B #4	5				" "		
10.12	GNS 5	3				Good "		
10.20	B #4	5				Fair "		
10.25	GNS 5	3				" "		
10.28	B #4	7				" "		
10.34	B #4	6				" "		
10.40	B #4	8				" ", little	mineral	
10.51	B #4	6				Very "	"	
11.00						Test stopped		
						Conct.	W.T.11C.	22
						Tails	W.T.11T	963
								985

REAGENT	%	LBS. P.T.
K.Xanthate	30 cc	.105
Na_2S	10 "	1.00
HCl.	10%	10 cc.
B #4		1.63
G.N.S.5		0.175

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT
	W.T.11C.	2.24	2.05	.0460						
	W.T.11T.	97.76	.04	.0392						
				.0852						

PERCENT OF TOTAL

Au.			
53.0			
47.0			
100.0			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	12
NOTES: 1200 gms. ore ground in B.M. for 5 mins with 10 c.c. Na_2S; 30 c.c. K.Xanthate; 750 c.c. water.	DATE	Feb. 1st. 1927.
	ORE NO.	W.T.
	PULP RATIO	: 1
Grinding started 10.42	R.P.M. IMPELLER	
stopped 10.47	PULP TEMP. START	°
	PULP TEMP. FINISH	°
	PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
11.05						Ore charged M.S.		
11.05	F.P.L.#26	5				No froth.		
11.10	F.P.L.#26	5				Slight "		
11.15	F.P.L.#26	5				Strong rich froth		
11.20	F.P.L.#26	3				Fairly " "		
11.25	F.P.L.#26	5				" " "		
11.30	F.P.L.#26	8				Very rich froth		
11.38	F.P.L.#26	3				Fair froth		
11.55	F.P.L.#26	4				Fair ", little mineral		
12.00						Test stopped		
						Concts.	W.T.12C.	15
						Tails	W.T.12T.	962
								987

REAGENT	%	LBS. P.T.
K.Xanthate		.105
Na_2S		1.00

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.
	W.T. 12C.	1.52	2.95	.0449							53.3
	W.T. 12T.	98.48	0.04	.0394							46.7
				.0843							100.00

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	13.
NOTES: 1200 gms. Ore ground for 5 mins. in B.M. with 10 c.c. Sodium Sulphide; 30 c.c. K. Xanthate; 750 c.c. water.	DATE	Feb. 1st. 1927
Grinding started 2.24 p.m.	ORE NO.	W.T.
stopped 2.29 "	PULP RATIO	: 1
	R.P.M. IMPELLER	
	PULP TEMP. START	°
	PULP TEMP. FINISH	°
	PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
2.35						Ore charged M.S.		
2.45	T.T.	5				No froth		
2.50	T.T.	5				" "		
2.53	T.T.	5				Good rich froth		
3.27	T.T.	3				Not much mineral		
3.35						Test stopped		
						Concts.	W.T.13C	24
						Tails	W.T.13T	950
								974

REAGENT	%	LBS. P.T.
Na_2S		1.00
K.Xanthate		.105

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.			
	W.T.13C	2.47	2.0	.0494							57.2			
	W.T.13T	98.53	.0375	.0370							42.8			
				.0864							100.0			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	14.

NOTES: 1200 gms. ore ground with 750 c.c. water in B.M. for 5 mins.
Then agitated in M.S. with 10 c.c. Na_2S and 30 c.c. K.Xanthate are added after.

Grinding started 10.20 a.m. stopped 10.25 a.m.
Agitation " 10.30 " " 11.30 "

DATE	Feb. 2nd. 1927.
ORE NO.	
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

REAGENT	%	LBS. P.T.
Na_2S		1.00
~~K.Xanthate~~		.105
B#4		1.75
GNS.5		0.175

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
10.30						Ore charged M.S.		
11.35	B.#4	10				No froth		
11.44	B.#4	5				Little "		
11.48	B.#4	5				" "		
11.52	G.N.S.5	3				Fair rich froth		
11.55	B.#4	5				" " "		
12.00	B.#4	5				Good " "		
12.05	B.#4	5				" " "		
12.08	G.N.S.5	3				Fairly good "		
12.15	B.#4	5				" " "		
12.26	B.#4	7				Fair froth, little mineral		
12.34	B.#4	9						
12.42	Test stopped					Concts.	W.T.14C.	20
						Tails	W.T.14T.	962

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	~~WEIGHT~~ 982	Au. PERCENT OF TOTAL			
	W.T.14C	2.04	2.2	.0449							53.4			
	W.T.14T	97.96	0.03	.0394							46.6			
				.0843							100.0			

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	15.

NOTES: 1200 gms. ore ground in B.M. for 5 mins. with 10 c.c. Na_2S solution and 30 c.c. K. Xanthate solution with 750 cc.water

Grinding started 10.33 a.m.
stopped 10.38 "

DATE	Feb. 4th 1927
ORE NO.	W.T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
9.43						Ore charged		
9.45	GNS #5	3				slight froth no mineral		
9.48	GNS #5	2				Very evanescent fr. little mineral		
9.53	"	2				" " " " "		
10.05	"	2				" " Very " "		
10.10	"	2						
10.15						Test stopped		
						Concts.	W.T.15C	4
						Tails	W.T.15T	975
								979

REAGENT	%	LBS. P.T.
G.N.S. 5		0.341

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL Au.
	W.T.15C.	.409	6.9	.0282							32.1
	W.T.15T.	99.59	0.6	.0597							67.9
				.0879							100.0

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	16.
DATE	Feb. 16th, 1927.
ORE NO.	W.T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1500 gms. wet Tls. from 4th Yr. Cyanide Test agitated for 1 hr. with 1500 c.c. water; 25 c.c. Na_2S and 25 c.c. K.Xanthate. The ore was then charged to the M.S. cell.

Agitation started 10.06 a.m.
" stopped 11.06 "

Soln. is acid, no KCN.

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
11.15						Ore chgd. M.S.-lime froth		
11.19	B #4	10				Copious white viscous Fr.		Concts. & Tails discarded.
11.23	GNS 5	2				" dirty "	"	
11.25	B #4	10				" " "	"	
11.27	HCl.		20			" " "	"	
11.30	HCl.		30			" " "	"	
11.32	HCl.		45			" " "	"	
11.49	B #4.	10				" " "	"	
11.52	K.Xanth.		10=	.05gm.				
11.58	Pulp volume reduced & water added.							
12.01	B #4	5				Froth reduced no mineral.		
12.04	B #4	5				" sl. increas. "	"	
12.15						Test stopped.		

REAGENT	%	LBS. P.T.
Na_2S		1.0
K.Xanthate		0.1
H Cl.	10	
B #4		1.25
G.N.S.5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL			

ORE DRESSING LABORATORY MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	17
DATE	Feb. 16th. 1927.
ORE NO.	W.T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1500 gms. wet Tails from 4th Year Cyanide Test agitated 2 hours with 1500 c.c. water; 25 c.c. Na_2S and 25 c.c. K.Xanth-ate.

Agitation started 10.36 a.m.
" stopped 12.36 "

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
2.30						Ore fed M.S. lime fr.		
2.33	B. #4	10				Copious dirty froth no min'l.		
2.50	B. #4	10				" " "	"	"
3.05	H Cl.		20			less " "	"	"
3.18	B. #4	10				as above		
3.22	$K_2Cr_2O_7$		25			Fr. possibly grayer in colour		
GNS5 →		2				Very copious froth (dirty)		
3.22						Test stopped.		
						Concts.	WT.17C.	
						Tails	WT.17T.	
						DISCARDED.		

REAGENT	%	LBS. P.T.
Na_2S		1.0
K.Xanthate		0.1
H Cl	10	20 cc
$K_2Cl_2O_7$		0.1
B #4		0.936
GNS 5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	19.
NOTES: 1250 gms. moist ore put in M.S. Cell and agitated for 20 mins.	DATE	March 1st, 1927.
	ORE NO.	201
	PULP RATIO	: 1
	R.P.M. IMPELLER	
	PULP TEMP. START	°
	PULP TEMP. FINISH	°
	PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
10.40						Ore charged to M.S.		
11.00	B #4.	10				Very copious froth (dirty).		
11.20						Test stopped.		
						DISCARDED.		

REAGENT	%	LBS. P.T.
B # 4		0.312

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL			

ORE DRESSING LABORATORY MINING DEPARTMENT, McGILL UNIVERSITY.

FLOTATION TEST NO.	20.
DATE	March 1st. 1927.
ORE NO.	201 T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1250 gms. moist Cy. Tls. pulp (as in 19) put in B.M. for 7 mins. with 250 c.c. water. Then washed and diluted - allowed to settle for 20 secs. and elutriated. More water added - allowed to settle for 20 secs, and more slimes drawn off. Then pulp charged to M.S.

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
12.05						Ore charged to M.S.		
12.06	B #4	5				Evanesct.Fr.(clean)		
12.10	Na_2)		20			" " "		
	K.Xanth.)					" " "		
12.35	B #4	5				" " "		
12.36	GNS 5	1				Thick Froth, little minl.		
12.38	B #4	10				" " more	"	
12.40	GNS 5	1				" " "	"	
12.43	B #4	10				" " "	"	
12.50						Test stopped.	C.	6
						M.S.	Tls.	368
						El.	"	455
								829

REAGENT	%	LBS. P.T.
Na_2S		2.0
K.Xanthate		0.2
B #4		0.624
G.N.S. 5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz% Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT
	Tails	44.425	.105	.0466						
	" El.	54.850	.102	.0560						
	Concts.	.725		.0232						
		100.0		.1258						

PERCENT OF TOTAL

Au.		
36.6		
44.0		
20.4		
100.0		

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY.	FLOTATION TEST NO.	21

NOTES: 1250 gms. moist Ore put in B.M. for 7 mins. with 700 c.c. water. Then diluted and allowed to settle for 20 secs. in bucket. The slime was then syphoned off to a marked depth. The settled sand etc. was then fed to the M.S. cell.

DATE	Mar. 1st. 1927
ORE NO.	201T
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
3.30						Ore fed to M.S.		
4.00	B. #4	10				slight fr. no mineral.		
4.02	GNS 5	1				better " little "		
4.05	B. #4	10				" " " "		
4.08	B. #4	5				no "		
4.13	GNS. 5	1				" "		
4.16	B. #4	5				dirty froth		
4.20						Test stopped	C.	9
						M.S.	T.	489
						El.		441
								939

REAGENT	%	LBS. P.T.
B#4		0.624
G.M.S.5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL			
	Tails	52.1	.09	.0468							36.2			
	" El.	47.0	.115	.0550							42.5			
	Conct.	.958	2.85	.0273							21.3			
		100.0		.1291							100.0			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO. 22.

NOTES: 1450 gms. of moist ore put in B.M. for 5 mins. Then diluted after placing in a bucket. Allowed to settle 30 secs. Then syphoned off to a pre-determined amount. The remainder was again diluted, allowed to settle 20 secs and syphoned to same point. Then charged to M S.

DATE	March 3rd. 1927.
ORE NO.	201T.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

REAGENT	%	LBS. P.T.
Na_2S		2.0
K.Xanthate		0.2
B #4		1.56
GNS 5		0.0584

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
11.15						Ore charged to M.S.		
11.15	K.Xanth.)	20				Agitated for ½ hr.		
1	Na_2S)							
11.45	B #4.	10				Poor Froth, no mineral.		
11.50	GNS 5.	1				Slight " little "		
11.53	B #4.	10				Fair " more "		
12.10	B #4.	10				" " little "		
12.12	GNS 5	1				" " more ".		
12.17	B #4	10				" " little "		
12.33	B #4	10				" " very " "		
12.40						Test stopped	C	8
						M.S.	Tails	357
						1st.El.	Tls	429
						2nd El.	"	167
								961

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	~~WEIGHT~~	PERCENT OF TOTAL
	Tls.	37.2	.09	.0335							25.5
	" El.	62.0	.137	.0850							56.7
	Concts.	.834	2.8	.0234							17.8
		100.03		.1319							100.0

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	23.
NOTES: 2000 grms. wet ore (1000 gr. dry) ground in B.M. for 5 mins. L : S = 3:1. Volume halved. Allowed to settle for 6 mins after making l : s = 6 or 7:1. Then syphoned off till ½ vol. Charged to M.S. cell. Ore used: Tails from cyanide test on W.H. ore by IV Year.	DATE	March 10th, 1927.
	ORE NO.	201CT.
	PULP RATIO	: 1
	R.P.M. IMPELLER	
	PULP TEMP. START	°
	PULP TEMP. FINISH	°
	PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
3.50						Ore chgd. to M.S.		
3.51	B #4	10				Froth, no mineral.		
3.55	B #4	10						
3.55	K.Xanth.	)	20			Fair froth with a		
	Na_2S	)				little mineral.		
4.00	B #4	10				" "		
4.13	B #4	5				little mineral		
4.20						Test stopped		
							Conct.	9
							Tails	807
								816

REAGENT	%	LBS. P.T.
B#4		1.094
Na_2S		2.0
K. Xanth		0.2

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL
	Tails	98.89	.085	.0839							63.2
	Concts.	1.11	4.45	.0490							36.8
		100.0		.1329							100.0

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO. 24.

DATE March 10th, 1927

ORE NO. 201 CT.

PULP RATIO : 1

R.P.M. IMPELLER

PULP TEMP. START ° FINISH ° MEAN °

NOTES: 500 gms. ore, allowed to settle for 11 mins. agitated previously with 20 c.c. K. Xanthate and Na_2S for 15 mins., elutriated. Then charged to M.S. cell.

1 : s = 5:1 before elutriation.
1 : s = 2:1 after " (Same ore as in Test 23.)

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
4.25						Ore charged to M.S.		
4.26	B #4	10				No mineral		
4.29	B #4	10				Little "		
4.29	GNS 5.	1				More "		
4.30	"	1						
4.34	B #4	10						
4.36	GNS 5	2				Improved fr. more mineral		
4.41	B #4	10				" " " "		
4.42	GNS 5	1				" " " "		
4.46	KX&Na_2S		10					
4.48	B #4	10						
4.50	GNS 5	2				Vol.Fr. a little mineral		
4.55	B #4	5				Not much mineral	Concts	25
5.05	Finis.						Tails	386

REAGENT	%	LBS. P.T.
B#4		1.72
GNS5		0.203
Na_2S		1.0
K.Xanth.		0.1

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL
										411	
	Tails	93.93	.075	0.0704							46.4
	Concts.	6.07	1.5	0.0914							53.6
				.1618							100.0

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	25.
DATE	April 4th 1927
ORE NO.	200 c.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: 1200 gms. wet ore ground for 5 mins. in B.M. with 500 cc. water and 20 drops B #4 and K.X and Na_2S.

Ore is tails from Cyanide test on Ore #200 by 4th Year.

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
10.00	Ore charged to M.S.					Dirty copious froth.		
10.05	K.Xanth & Na_2S		10			" " "		
10.15	B.#4	5				" " "		
10.23	B.#4	10				Copious froth, some mineral.		
10.33	B.#4	10				Good mineral Brg. Froth.		
10.45	B.#4	10				" " "	"	
10.55	GNS 5.	2				" " "	"	
11.20						Test stopped	Concts.	52
							Tails	973
								1025
	Some conct. lost owing to leak in pan.							

REAGENT	%	LBS. P.T.
K.Xanth.		0.1
Na_2S		1.0
B #4		1.094
G.N.S.5		0.0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL			
	Concts.	5.06	2.2	.1115							44.0			
	Tails	94.94	0.15	.1421							56.0			
				.2536							100.0			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY

FLOTATION TEST NO.	26 .
DATE	April 4th, 1927.
ORE NO.	200 (C)
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

NOTES: Two lots of 500 gms. ore ~~started 10.33~~, agitated half an hour with 10 c.c. of K Xanthate and Na_2S solution, diluted to 3000 c.c., allowed to settle 15 mins. ~~then~~ elutriated

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
2.15						Ore charged to M.S.		
2.15	B #4	10				Fine bubble froth, some mineral		
2.20	"	10				Good fr., considerable "		
2.35	"	10				" rich froth.		
2.40	"	10				" " "		
2.48	"	10				" " "		
2.54	GNS 5	3				Copious fr. not much mineral		
2.59	B #4.	5				Poor fr. very little "		
3.12	H_2SO_4		10			" " " " "		
3.12	GNS 5	2				" " " " "		
3.20	"	2				" " " " "		
3.25	H_2SO_4		10				El. Tls.	4
							Tails	849
						Test stopped....	Concts.	44
								897

REAGENT	%	LBS. P.T. Or
K. Xanthate		0.1
Na_2S		1.0
H_2SO_4	2 cc.	4.
B #4		1.72
G.N.S.5		0.263

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	~~WEIGHT~~	PERCENT OF TOTAL
		4.91	2.6	.1275							45.6
		95.09	0.16	.1520							54.4
				.2795							

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO. 27 .

NOTES: Two 700 gm. lots of wet ore agitated with 10 c.c. of solution for half hr. (about 1½:1 pulp). Then charged to M.S. machine.

DATE	19
ORE NO.	200 c.
PULP RATIO	: 1
R.P.M. IMPELLER	
PULP TEMP. START	°
PULP TEMP. FINISH	°
PULP TEMP. MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
11.50						Ore charged		
11.50	B #4	10				Copious dirty froth		
11.55	"	10				" " " some mineral		
11.05	B #4	10				"	more	"
11.09	"	10				"	"	"
11.30	"	10				"	"	"
11.35	GNS 5	2				"	some	"
11.45						Test stopped	Concts.	42
							Tails	855
								897

REAGENT	%	LBS. P.T.
K. Xanthate		0.1
Na_2S		1.0
B #4		1.56
G.N.S.5.		0-0584

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au.	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL			
	Conct.	4.69	2.40	.1250							43.4			
	Tails	96.31	0.17	.1635							56.6			
				.2885							100.0			

ORE DRESSING LABORATORY, MINING DEPARTMENT, McGILL UNIVERSITY	FLOTATION TEST NO.	281
NOTES: 1500 gms. Wet ore ground in B.M. for 25 mins.with 25 dr. B #4.	DATE	April 4. 1927
	ORE NO.	200 c.
	PULP RATIO	: 1
	R.P.M. IMPELLER	
Grinding started 3.24	PULP TEMP. START	°
	FINISH	°
	MEAN	°

TIME H. M.	REAGENT	DROPS	C. C.	WEIGHT	R.P.M.	NOTES	SAMP. No's	WEIGHT
3.55						Ore chg'd to M.S.		
"				Very copious dirty froth			1st Conc. 16	
4.04	Kx&Na_2S		10	Good rich		"		
4.06	B #4	10		"	"	"		
.23	"	10		"	"	"		
.35	KX&Na_2S		10	Evanescent		"		
.42	B #4	10		Good rich		"		
.53	B "	10		"	"	"		
5.00	G.N.S.5	2						
10						Test Stopped	2nd Conc.	49
							Tes.	972
								1037

REAGENT	%	LBS. P.T.
B #4		1.25
G.N.S.5		0.0584
Na_2S		2.0
K. Xanth.		0.2

SCREEN ANALYSIS OF FEED

GRADE	%	GRADE	%
+ 20		+ 100	
28		150	
35		200	
48		− 200	
65		TOTAL	

TIME MIN.	PRODUCT	% WEIGHT	Oz. Au	WEIGHT	%	WEIGHT	%	WEIGHT	%	WEIGHT	PERCENT OF TOTAL
	1st Conct.	1.54	4.6	.0708							29.5
	2nd "	4.72	1.4	.0560							23.6
	Tls.	94.74	0.117	.1110							47.9
				.2378							