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A. Preface

Within recent years the flotation method for concentrating ores has been intensively developed and extended, with the result that processing costs on many important ores have been considerably lowered. Exploitation of some mineral deposits would not be economically feasible if flotation methods were not used.

Fifty years ago the possibility of concentrating ores by flotation was regarded as remote, if indeed that process was at all considered. It made progress with other minerals before gold ores were treated, but it has been widely applied to the latter since 1925. It has displaced gravity concentration in recovering gold from certain complex ores which are not amenable to cyaniding. Finally, it has made amalgamation and cyaniding more profitable in many instances when worked in conjunction with them.

The object of this work is to conduct various investigations on the flotation of low-grade, auriferous pyritic ores into a concentrate that can be further cyanided, at the same time obtaining a very low-grade tailing that can be discarded to waste.

Flotation is conducted in this work by batch testing. According to *A. M. Gaudin*, flotation testing by batches is widespread because it predicts in a broad way the results of plant operation. He says that, in flotation work, the agreement of large-scale operation with laboratory tests is much better than the agreement between large-scale operation and miniature-scale tests in most chemical, metallurgical, and ore-dressing operations. This is because the number of particles involved, even in a charge as small as 500 g., is so large that the laws of averages hold almost as well as in the field of molecular dynamics.

The tests are conducted along the lines dictated by experience with ores of similar composition and texture. According to *Gaudin*

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the work should be considered routine only when the chemical, mineralogical, and textural analysis resemble those of a well-known ore; when even one of these analysis is distinctly different from experience, the work will involve an unusual amount of original research.

Experience obtained in examining and testing ores from all over the world indicates that no two ores are exactly alike. Consequently, aside from a few principles regarding flotation and the use of reagents, it is generally agreed each ore must be considered a problem for the metallurgist to solve before any attempt is made to go ahead with the selection and design of a treatment plant.

B. Theoretical part

I. Historical survey of gold metallurgy

From very early times the ancients were attracted by the beautiful colour, the brilliant lustre, and the indestructibility of gold. *Gowland* [1] points out that on account of its distribution in the sands and gravels of rivers, and its distinctive appearance, it must have been the first metal to attract the attention of prehistoric man in most regions of the world. He also observes, however, that it could not have been used even for ornaments until the art of melting have been invented, and this could hardly have happened until man had passed the Stone Age culture and entered the Bronze Age. No objects consisting of gold, he says, have been found with undoubted Stone Age remains.

The earliest mining and metallurgical operations of which traces remain were those carried on in Egypt in dealing with the ores of gold. The vast quantities of gold amassed by the Pharaohs were the envy of contemporary and later sovereigns. Though much was imported, received by way of tribute, or captured in warfare, the Egyptian mines themselves were reasonably productive. Over one hundred ancient gold workings have been discovered in Egypt and

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the Sudan, though within the limits of Egypt proper there appear to have been gold mines only in the desert valleys to the east of the Nile near Koptos, Ombos and Apollinopolis Magna. Of one of these mines—possibly near Apollinopolis—a plan has been found in a papyrus of the fourteenth century B. C., and the remains of no fewer than 1,300 houses for gold miners are still to be seen in Wadi Fawakhir, half way between Koptos and the Red Sea. The Ancient Egyptian word for gold is "nub", which survives in the name "Nubia", a country that provided a great deal of the precious metal in ancient days. The gold mines in Nubia and other parts of the Egyptian empire seem to have been very efficiently designed and controlled, though with a callous disregard for the human element employed. This is a picture which is drawn for us by *Diodorus Siculus* [2] (following Agatharchidis of Cnidos):

"In the furthest part of Egypt, on the confines of Arabia and Ethiopia, there is a place containing many mines of gold, which is procured by numerous workmen with vast hardship and expense. The soil being naturally black, and containing many veins and strata of marble, extremely white, and thus distinguished from the circumjacent materials, the superintendents set over the mine-works prosecute the search with a multitude of labourers. The ground containing the gold they first heat with long-continued fire, and so render full of fissures, before they apply manual labour to it; but the rock that is soft and capable of yielding to labour is cut down with stone-cutters. The entire operation is directed by the engineer, who looks out for the proper stone, and marks it out for the labourers. Of those appointed to this task, such as are of the strongest break down the marble-like rock with iron pickaxes, applying no art to their labour, but mere strength, and thus cut galleries, running not in a straight line, but guided by the direction of white veins. These men, in consequence of the crooked course of the galleries, work in darkness, and therefore carry lamps fastened upon their foreheads; and frequently changing their posture, according to the arrangement of the veins, they break down and bring to the floor fragments of the cut rock. Meanwhile the boys throw up the broken mineral as it falls little by little, and carry it up into the open air at the mine's mouth. Here those above

thirty years old receive from them a fixed measure of the broken ore, and pound it in stone mortars with iron pestles, until they reduce it to the size of a vetch. From these the granulated ore is taken by the women and the older men, who have many hand-mills set in a row, and, standing two or three together at the handle, they grind the measure given to them as fine as flour.

Last of all the skilled workmen receive the ore ground fine, and complete the operation. They have a board placed somewhat sloping, on which they throw a small quantity of the dust, and pouring water over it they rub it. Then the earthy particles are dissolved by the water, and run off, owing to the slope of the board; but those containing the gold remain upon it in consequence of their weight. Repeating this frequently, first of all they rub the dust gently with their hands, afterwards they press it with coarse sponges lightly, taking up in this way the loose and earthy part, until gold-dust is left behind unmixed. Finally, other workmen, taking from them the collected dust, according to weight and measure, place it in earthen crucibles, mixing in a certain proportion of lead-ore and lumps of salt, to which they add a little tin and barley-bran. Then they fit on the cover of the crucible, luting it down carefully with clay, and bake it in a furnace five days and nights continuously. Then taking it out, and leaving it to cool, they find nothing of the other materials left in the crucible, but get the gold quite pure, but slightly diminished in weight. The discovery of these mines dates very far back; probably they were found out by the ancient kings".

Remains of all implements have been found but their exact age is doubtful.

Alluvial auriferous sand was also treated, a distinction being made between the gold obtained in this way and that extracted from the mines. The latter was called "nub-en-set", i. e. "gold of the mountain", while alluvial gold was named "nub-en-mu", i. e. "gold of the river". Auriferous sand was placed in a bag made of a fleece with the woolly side inwards; water was then added and the bag vigorously shaken by two men. When the water was poured off, the earthy particles were carried away, leaving the heavier particles of gold adhering to the fleece.

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Fig. 1. Clay crucible found at El Argar

Fig. 2. An Egyptian Metallurgist

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Among the pictorial rock carvings of Upper Egypt there are several illustrations of the gold-extraction process mentioned above. The earliest indications appear to be certain inscriptions on monuments of the Fourth Dynasty (4000 B.C.) depicting gold washing [3]. Certain Stelae of the Twelfth Dynasty (2400 B.C.) in the British Museum refer to gold washing in the Sudan, and one of them appears to indicate the working of gold ore as distinguished from alluvial [4].

In somewhat later times, in the collection of alluvial gold, the sands were washed down over smooth sloping rocks by means of running water, and the particles of gold, sinking to the bottom of the stream, were entangled and caught in the hair of raw hides spread on the rocks—thus anticipating the modern blanket practice.

For many years, when auriferous sands were washed, the aid was also invoked of what *Baron Born* called in 1786 the "elective affinity" of mercury for gold when mixed with impurities. The ease with which gold-amalgam can be collected, in spite of its being less dense than gold itself, is due to the fact that it is wetted by mercury.

It is also of interest to remember that the earliest dawn of the science of chemistry was heralded by the study of the properties of gold, and by the efforts which were made to invest other matters with these properties. From the fourth to the fifteenth century, chemistry, which was first called "Chemia" and then "Alchemy" was defined as the art of transmuting base metals into gold and silver, almost all the labours of the philosophers being intended to aid directly or indirectly in solving this problem.

The extraction of gold by amalgamation was practised to some extent in ancient times [5], but no details of the methods are known. In the middle ages, rich ore and concentrate were ground or rubbed with mercury in wooden bowls using iron-shod mullers rotated by crude machinery. Later, batches of ore were ground fine with mercury in the arrastra, using stone drags on a stone-paved bottom. In recent times the amalgamation of gold has been almost entirely associated with the development of the stamp mill.

In 1848, chlorination was suggested independently by *Percy* and *Plattner* [6], though *Plattner* made the first commercial application of the process to the arsenical gold ores of Reichenstein, Silesia. The chlorination process is now obslete.

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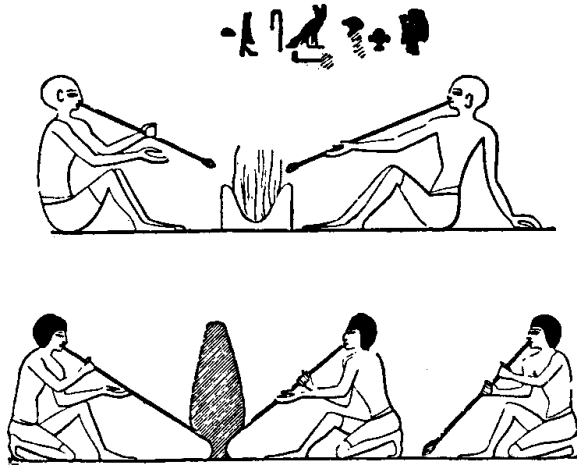
Fig. 3. Egyptian Goldsmiths' Workshop in the Pyramid Age

Fig. 4. Metal-Workers' Workshop in Egypt

Before the introduction of the cyanide process in 1887, the gold in the amalgamation tailings could be recovered only by using chlorine, but this process was expensive to operate, not very efficient, and could not be applied economically to the treatment of lower-grade ores. Throughout the world vast tonnages of ore and tailings were lying untreated and waiting for some economic method to be devised for recovering their gold contents. The new *Rand goldfield* was in a particularly difficult position. Many of the mines had yielded handsome returns from the free milling ores of the oxidised zone, but most of them were now getting down into the

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sulphide ore bodies, and the percentage of the total gold which could be recovered from these ores by amalgamation dropped seriously. In fact, it is not an exaggeration to state that the gold-field, hitherto worked with so much success, was now faced with virtual extinction. Such was the position when the cyanide process was discovered in Scotland. The whole future of the gold-mining industry throughout the world was to be profoundly influenced by it. Many years before the advent of the cyanide process, it was known that gold could be dissolved in strong solutions of potassium cyanide, but it was not until 1887 that three Scots, *MacArthur* and the *Forrest brothers* [7], discovered how to apply the solvent properties of cyanide solutions to the treatment of gold ores. The *Cassel Gold Extracting Company* was formed in 1884 for the purpose of introducing a modified chlorination process for treating gold ores. *Charles Tennant*, a director of this company, commissioned *J. S. MacArthur*, a chemist employed by the *Tharsis Sulphur and Copper Company*, to investigate and report on the proposed method. *MacArthur's* work showed that the process was not practicable, and he and two doctors of medicine, the *Forrest brothers*, devoted all their spare time to experimental work in an attempt to discover an improved process which would recover all the gold from all types of ores. After a year's intensive effort, the great discovery was made in 1887 that very dilute solutions of potassium cyanide could dissolve most of the gold in a normal ore. This was followed in the next year by the equally significant discovery that the gold could be precipitated from its solution, quite simply, by treatment with finely-divided zinc. These two discoveries were destined to revolutionise the metallurgy of gold and greatly to accelerate the development of Africa, Canada, Australia, and other gold-mining countries. Metallurgists of the time were sceptical about the practicability of the claims, for in those days potassium cyanide was an expensive substance available only in comparatively small quantities, and it was relatively unstable and highly poisonous. However, *MacArthur* convinced the directors of the *Gold Extracting Company* that his process was technically sound and practicable. The company changed its name to the *Cassel Cyanide Company*, erected plant for the manufacture of cyanide, and trained about

forty metallurgists to go out to the various goldfields to introduce and prove the success of the new process to the mining world. The first plant was erected in New Zealand in 1889, a second at the famous *Robinson* mine on the *Rand* in 1890, and in the following year plants were built in America. It was at once realised that the process was a complete success, and within a few years plants were erected all over the world. The costly chlorination process was soon abandoned everywhere. It is certain that, but for the cyanide process for recovering gold and the *Cassel Cyanide Company's* process for manufacturing cyanide at a reasonable price, many of the famous gold-mines would not be operating to-day. The adoption of the cyanide process caused a great increase in the demand for cyanide. At first the *Cassel Cyanide Company* supplied an impure potassium cyanide for gold extraction, but with the acquisition of *Castner's* patents and with the gradual introduction of improvements in the manufacturing process it became possible to market the cheaper and, weight for weight, more efficient sodium cyanide. To-day the sodium cyanide supplied to the mines is a fine white product and the purest cyanide obtainable (96—98 per cent NaCN).

In recent years cyanidation and amalgamation have been the basic methods employed to extract gold from its ores. Relatively little use has been made of flotation. This has been in no small part due to the fact that no new goldfields of importance have been developed since flotation attained its full industrial stature. As new goldfields are developed it is likely that flotation will play an increasing part in the treatment of certain ores, either in place of, or in conjunction with, cyanidation [8]. Cyanidation largely replaced amalgamation, and flotation is rapidly encroaching on the field of cyanidation [9]. In the five decades since cyanidation was first introduced the process has been unchanged in basic conception, although much improvement, reflected in increased efficiency and lower costs, has been made in the mechanical equipment. In the same period the flotation process has also made noteworthy advances in the development of new reagents, as well as improvements in both technology and equipment. There still remains, however, an incompletely-explored opportunity for combining the basic features of cyanidation and flotation to achieve results unattainable

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by either process alone. In this field, research has gone forward steadily and many notable achievements have already been made [10].

II. Mineralogical occurrence of gold

Gold is very widely distributed in the rocks of the earth's crust, and is found in varying quantity in rocks of widely different character in almost all parts of the world. The metal is associated with rocks of all geological ages from the oldest to the most recent.

Gold occurs native, as a telluride, or in an unknown form in close association with base-metal sulphides, particularly pyrite, chalcopyrite, and arsenopyrite. In many ores the gold is actually contained within the crystals of one or other of the aforementioned minerals, while in other ores these minerals accompany the gold but are not themselves gold-bearing.
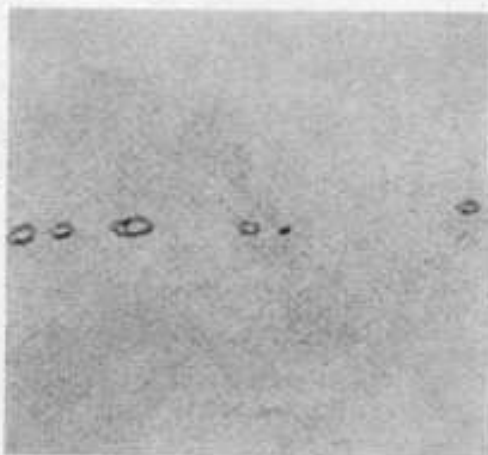

Fig. 5. Inclusions of gold in pyrite, × 1,000. The largest particle is about 4 microns long. (After McLachlan)

1. Gold in pyrite

In seeking to explain the behaviour of gold in pyrite during different treatments, various theories as to the form in which it exists have been propounded. The idea that gold occurred chemi-

cally combined with pyrite has been considered. The balance of evidence seems to be in favour of the theory that gold exists in pyrite in the metallic state. Although the metal is generally invisible in undecomposed crystals of pyrite, it becomes visible when such crystals are oxidised either by air and water in nature, by means of nitric acid, or by being roasted or subjected to deflagration with nitre. As a result of such decomposition, particles of bright, lustrous gold, angular and ragged in shape, but of considerable size, often become apparent. These particles may be separated from the oxides of iron by washing; and the use of nitric acid, followed by panning, is frequently resorted to in order to detect gold in pyrite. Moreover, although usually invisible, gold can sometimes be seen in unroasted pyrite. As long ago as the year 1874, *Richard Daintree* and *J. Latta* [11] found specimens of cubical pyrite in which gold could be seen under a microscope gilding the cleavage planes of the crystals. Again, *G. Melville-Attwood* [12], on examining crystals of auriferous pyrite from California in 1881, found that the faces of the crystals were gilded in some places, and that here and there little specks or drops of gold occurred, partially imbedded in the pyrite. These films were too thin to be detected by an ordinary lens, so that it did not seem surprising that such impalpable material could not be taken up by mercury. *Louis Janin Jr.* [13], found, in a porphyritic gangue from the Republic of Colombia, crystals of pyrite which had gold in small globules on their surfaces. It has long been known, too, that crystals of pyrite are often found adhering to an amalgamated plate, the particles of gold on their surfaces having been amalgamated. It seems likely, in view of all these facts, that most if not all of the gold is in the metallic state. It has been contended that it is disseminated mechanically through the mass of pyrite, but it seems more probable that the interior of the crystal is not auriferous, the deposition of gold being superficial, so that the enrichment of the pyrite is confined to its crystalline faces and cleavage planes. Strong evidence of the richness of the outside of the crystals of pyrite in the Banket ore of the Transvaal is given by *Caldecott* [14]. According to *Head* [15], gold may occur in pyrite in relatively large fragments, or down to specks 5 microns in diameter on crystallographic faces. *H. L. Smyth*,

W. Lindgren, and *W. J. Sharwood* [16] examined a number of specimens and found that under the microscope the gold could be seen at the surface of the pyrite or deposited in minute cracks in it, but never entirely enclosed within it.

2. Relation of the manner of gold occurrence to metallurgy

The occurrence of gold on crystallographic planes in pyrite has an important bearing on the metallurgy of the low-grade, auriferous pyritic ores, since all such ores have been found to posses certain outstanding characteristics in common. These features may be briefly stated as follows [17]:

1. The gold occurs as minute flakes on the crystallographic planes of the pyrite.
2. The gold flakes are of very small size, and individuals 5 microns in diameter have been observed.
3. The pyrite in which small amounts of gold occur in finely-divided form is of the crystalline variety.
4. The outstanding characteristics of crystalline varieties of pyrite are: a) pronounced density or compactness with corresponding absence of porosity, b) extreme brittleness, which results in the production of "hackly" or irregular particles when crushed, and c) a marked degree of resistance to oxidation. This latter characteristic is a complement of a), since it implies opposition to the entrance of such oxidizing agents as air, gases, and solutions.
5. The existence of gold possessing a weak susceptibility to the attractive forces of a magnetic point. This property may be attributed to the existence of a film or coating of some iron compound that has not yet been determined.

Chief among the arguments substantiating that gold occurs in pyrite in the metallic form is the experience of those operators who, by continued experimentation, learned that quite often finer grinding of the pyrite resulted in increasing the percentage of gold extracted by cyanidation. Exceptions to this experience are on

record, and in certain instances it has been reported that the amount of gold extractable by straight cyanidation could not be increased beyond a definite point by grinding excessively fine. The size of the gold particles observed shows that they may or may not be too small to be liberated by grinding the pyrite to a size within the range of economical limits. If the gold particles in a pyritic ore were of a uniform size of 5 microns (theoretical 2,200-mesh) it would be necessary to grind the pyrite to that size to liberate every one of the gold particles. Each gold particle becomes an important consideration in an ore containing between 0.1 and 0.2 ounce of gold per ton, which is less than 0.001 per cent. Obviously such a procedure is not economically feasible.

Mention of fine grinding and its relation to the liberation and recovery of gold should be supplemented by a reminder of possible evils of this practice. The results of a microscopic study of gold losses at a certain plant indicated that the surfaces of gold particles had been contaminated and obscured by continued grinding in the presence of very fine slime [18]. Although the dimensions of the gold particles are the dominant factor in determining the size to which the pyrite must be ground to liberate the gold, the possible ill effects of surface contamination should be considered. This particular phase of fine grinding has not been given the study it merits, but it seems quite probable that the kind of gangue mineral, and its physical properties, play an important part in determining the extent to which mineral surfaces may be contaminated by excessively fine grinding.

The characteristics of pyrite crystallinity, with their associated habits of compactness and lack of porosity, have a direct bearing on the extraction of gold by cyanidation. The extraction of gold by cyanidation obviously depends on the contact of cyanide solution with the gold particle. Since it is not feasible to grind some pyrites finely enough to liberate all the gold, the extent to which unexposed gold particles can be brought into contact with solutions is governed by the porosity of the pyrite. Since careful study of reground cyanide residues has resulted in the finding of gold particles, it seems logical to suspect that at least some of them had not been reached by the cyanide solution.

It is a fact that some gold particles occurring in pyrite have been weakly but definitely susceptible to the attraction of a magnetised point. The hypothesis is that this property is due to the presence of a coating of some compound. Such a coating would be of importance from the standpoint of the rate of dissolving the gold particles in cyanide solution and of floating them.

Tests of several of these "magnetic" gold particles indicate that they are decidedly resistant to the solvent action of cyanide solutions.

The evidence presented as a result of microscopic studies of gold in pyrite shows the bearing of the relationships and physical characteristics described on certain of the problems involved in treating some of the low-grade ores containing auriferous pyrite. Continuation of the work to include other varieties and types of pyritic ores, if they can be obtained, would greatly amplify our knowledge and increase our understanding of the nature of these ores. It should also result in more intelligent selection and application of methods of treating such ores.

3. Choice of ore-treatment method

Choice between flotation, cyanidation, amalgamation, or a combination of these methods for recovering gold depends primarily on the character of the ore. The gold that is not recovered by a particular process is refractory to that process. An investigation to determine how the refractory gold is associated in the ore constituents usually indicates the needed improvement in the process, or additional methods for bettering recovery. Each ore is an individual problem until the association of the gold has been determined. When we know the problem through the interfering minerals or elements, the solution often is only the application of part of a former known practice as an auxiliary or additional feature. In the treatment plan for most gold ores, it has been the custom to try each of the usual recovery methods or combinations thereof, accepting the best results on a basis of the lowest cost. A better plan includes knowledge of the association of the gold with the various minerals from which it is to be recovered. This

affords reasonable foresight into the problems that should be expected and permits of some provision for avoiding probable difficulties.

Attention is being directed [19] to the microscopical examination of the ore to a much greater extent than formerly. By it a true idea of the associated minerals and of their frequency and amount, as well as of the nature, characteristics and mode of occurrence of the gold, can be obtained. In some measure, this may determine the general lines of treatment of the ore, according to whether the gold is free or encased in pyrite or other mineral. Mineralogical examination should be followed by chemical analysis and a determination of the physical properties, both of which are complementary to a knowledge of the minerals in the ore. They do not suffice in themselves to determine a line of treatment, and should not be given undue weight in the summing up.

Amalgamation is suited only to the recovery of coarse free gold. Cyanidation is best adapted to the extraction of fine freed or partly-freed native gold [20]. Flotation is suited primarily to gold intimately associated with the sulphides. In ores in which cyanidation and flotation are more directly in competition [21], cyanidation is favoured by coarse dispersion of gold particles in the sulphides or by the existence of free fine native gold. Conversely, flotation is favoured by fine dispersion of the precious metal in the sulphides. High grade of ore favours cyanidation because of the lower tailing made.

III. Flotation

The apparent specific gravity of a material of porous structure (such as coke) is much lower than the real specific gravity of the solid, owing to air that is entrapped in and around the particle, and frequently a relatively heavy material, even though coarse, may be made to float on water.

Some substances, notably the metallic sulphides, possess the peculiar property of assuming a false specific gravity due, apparently, to their lack of adhesion to water and their ability to cling to a film of air or other gas. If properly introduced into a moving

stream of water, the valuable sulphide may be floated away from the much heavier gangue. This ability of the metallic sulphides, to cling to an air bubble and to be in this way transported to the top of the stream and thus separated, forms the basis of the flotation process.

Fig. 6. Photomicrograph of galena particles in ethyl xanthate solution clinging to a small air bubble, which in turn is suspended from a larger air bubble 1 mm. in diameter. (After H. Rush Spedden)

1. Early attempts

The earliest known recognition of the difference in wettability of minerals by water is recorded in British Patent 488/1860 granted to *William Haynes*. His invention consisted in first triturating dry-ground ore with one-fifth to one-ninth as much fatty or oily agent and then agitating and triturating this mass with water, whereby the combined sulphides and fatty or oily agent were segregated from the combined earthy matter and water.

United States Patent No. 348,157/1885 granted to *Carrie J. Everson* describes a very similar operation, from which, however, improved results were obtained by modifying the water with acids,

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acid salts, or neutral salts. *Everson's* and subsequent patents, up to *Delprat's* U. S. Patent 735,071/1903, involved buoyancy of the oil and selection by oil of certain minerals from an acidulated pulp as their cardinal thoughts; this was the era of bulk-oil flotation typified by the Elmore bulk-oil process. *J. S. Elmore* [22] discovered that residuum oil, if mixed with ore pulp, would froth and rise to the surface, carrying with it sulphides and metallic particles. *H. F. Collins* [23] suggested that the reason was that smooth, lustrous particles were wetted by the oil, but that the oil would not adhere to rough porous particles which absorbed water.

The following table [24] summarizes the important steps in the evolution of flotation from crude beginnings to a fool-proof, elaborate, and scientifically amazing application of chemistry to ore concentration.

Table 1. Important steps in the evolution of flotation

Approximate date	Name of inventor	Invention
1860	Haynes	Usefulness of differences in wettability of minerals by water and oil
1886	Everson	Acidulated pulps desirable
1902	Froment; Potter, and Delprat	Gas as a buoyant medium
1905	Schwarz	Sodium sulphide to recover oxidized base-metal minerals
1906	Sulman, Picard, and Ballot	Reduced amount of oil; introduction of gas by violent agitation
1909	Greenway, Sulman, and Higgins	Soluble frothing agent
1913	Bradford	Copper sulphate as activator for sphalerite
1913	Bradford	Sulphur dioxide to depress sphalerite
1921	Perkins	Specific organic collectors
1921	. . . .	Alkaline circuits
1922	Sheridan and Griswold	Cyanides to depress sphalerite and pyrite
1924	Keller	Xanthate
1925	. . . .	Soaps for flotation of non-metallics

Flotation is today universally practised, and its field of application is being extended daily. The span of the last half century has seen the patenting of adiversity of flotation processes and machines. However, little is known of it as an exact science, although various investigators have been, and are, doing much to place it on a more scientific basis. This, of course, is a very difficult undertaking when one appreciates how ore deposits were formed, and the vast number of mineral combinations existing in nature.

The term flotation was formerly used for all processes of concentration in which levitation in water of particles heavier than water was obtained. Thus, if some particles were retained in an oil layer or at the interface between an oil layer and a water layer, the process was spoken of as bulk-oil flotation; if the particles were retained at a free water surface as a layer one particle deep, the process was skin-flotation; and if the particles were retained as a foamy layer several inches thick the process was froth-flotation. Froth-flotation is the only process that has survived the test of time, and the term flotation is now used universally to describe froth-flotation.

The present modern or "chemical flotation" period originated with the discoveries of *Perkins*, as disclosed in his U. S. Patent 1364043, in 1921, of the effectiveness of definite chemical compounds of nonoleaginous nature as promoters for sulphide minerals. These compounds were organic trivalent nitrogen and divalent sulphur compounds of slight solubility, such as alphanaphthylamine and thiocarbanilide. These compounds do not include the completely water-soluble xanthates patented by *Keller* and *Lewis* in 1924 and disclosed in U. S. Patents 1554216 and 1554220.

2. Flotation reagents

In order to accomplish desired results by flotation, a variety of chemical compounds are needed. These may be classified, according to their function, as follows:

        a) Frothers
        b) Collectors
        c) Addition agents

a) Frothers

Production of a persistent froth of desired selectivity and durability is of prime importance in successful flotation. In this regard, it is to be noted that homogeneous liquids do not froth; only aqueous solutions froth appreciably. Frothing takes place when air is bubbled through a liquid if the air-liquid surface tension is capable of a small but measurable, rapid change. A good frothing agent must be capable of passing readily into the interface between water and air, so that it must be partly water-repellent and partly water-soluble. In other words, it must be "heteropolar", i. e. possess polar and non-polar groups. In this regard, polar, water-insoluble saturated hydrocarbons, such as kerosene, give no froth. Unsaturated hydrocarbons give a light froth because they are non-polar. Organic compounds such as pine oil, alcohols, phenols, and certain fatty acids, give appreciable froth because they contain both polar and non-polar groups. In this latter type of compounds, one part of the molecule has an affinity for water and the other has an affinity for air, or conversely, an aversion to water. It follows that, in order to be effective, frothers must be neither too soluble nor too insoluble. In addition, to ensure selectivity in flotation, frothers must not ionise appreciably.

Dean and *Hersberger* [25] give the following properties of an ideal frothing agent:

1. It must form in low concentrations a copious but not-too-persistent froth.
2. It must be insensitive to hydrogen-ion concentration of the pulp, i. e. it must froth equally well in an acid or an alkaline medium.
3. It must be insensitive to salts, even in high concentration.
4. It must be absolutely non-collecting to both sulphides and non-sulphides.
5. Its frothing properties must not be affected by collecting reagents, including soap.
6. It must readily emulsify and disperse any insoluble collecting agent likely to be used.

Typical examples of frothing agents used for gold ores are pine

oils, cresylic acid, "Aerofloat", hardwood and coal tar creosotes. The first three are the most used at the present time.

It is learned that not all bubbles act in the same way when put to work recovering mineral particles. It has been proved that, in recovering small particles of mineral, large bubbles are not nearly so efficient as an accurately-controlled, large number of small bubbles. This is because it is surface area, not the size of the bubbles, which determines the amount of mineral to be carried to the surface.

b) Collectors

These are organic materials that attach themselves to certain minerals in preference to other constituents when present together in a water pulp. The power of water to wet these minerals is thereby diminished and floatability is increased.

Insoluble agents are brought into contact with, and ultimate adherence to, the mineral particles by mechanical stirring. Where the reagent is soluble, the phenomenon is mostly brought about by chemical adsorption. Typical examples of the former class are oleic and other fatty acids, castor oil, petroleum oils, creosotes, tars, etc. Those of the latter group are alphanaphthylamine, thiocarbanilide, metal xanthates, pyridine, quinoline, etc. The completely soluble chemicals of this class are sometimes called promoters.

In order that promoters may be effective, they must comprise two parts, one part non-polar (hydrocarbon) and the other polar; and they must be capable of adhering, by virtue of adsorption or chemical reaction, to the surfaces of specified minerals. Although resembling frothers to the extent of being heteropolar, promoters differ from frothers in that the polar part of an effective promoter must have a particular affinity for a particular mineral. (The polar part of a frother should have affinity for water only.) Thus, it is found that in the case of promoters for sulphide minerals, for example, the hydroxyl group in the polar part of a frother is replaced by the less water-avid sulph-hydrate group.

In general, the promoters may be considered as falling into the three following classes:

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Anionic type
Cationic type
Nonionic type

Anionic-type promoters

Commonly-used promoters of this type include:

1. Xanthates. The most powerful of the promoting agents. They are the reaction products of an alcohol, sodium or potassium hydroxide, and carbon disulphide. The general reaction may be written:

$$ROH + CS_2 + MOH = H_2O + ROCSSM$$

The structural formula is

$$C=S \begin{array}{c} \diagup OR \\ \diagdown S-M \end{array}$$
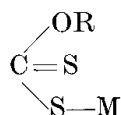
in which R stands for an alkyl hydrocarbon radical such as one of the ethyl, propyl, butyl, or amyl groups, and M stands for monovalent (alkali) metal.

In general, the xanthates with ethyl and propyl groups are less powerful but more selective than those with groups of greater molecular weight. Xanthates usually give superior results in slightly alkaline circuits.

2. Aerofloat, or disubstituted dithiophosphoric acid and salts of such acids. They are the reaction product of phosphorus pentasulphide with alcohols and with phenols. They were first introduced under the name "phosphocresylic acid", from the fact that they were made from the cresylic acid of commerce, and phosphorus pentasulphide. Later, the name was changed to "aerofloat", and as it was recognised that the reaction product contained a large amount of dithiophosphoric acid or of a dithiophosphate, their generic name became dithiophosphates.

· The reaction that yields dithiophosphates is believed to be as follows:

$$4\,ROH + P_2S_5 = H_2S + 2 \quad \begin{array}{c} S \diagdown \quad \diagup OR \\ P \\ HS \diagup \quad \diagdown OR \end{array}$$

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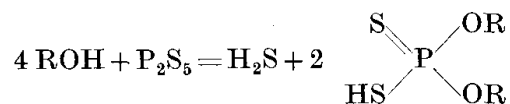
Dithiophosphates are structurally similar to xanthates, the pentavalent phosphorus atom taking the place of the tetravalent carbon atom and binding two alcohol or phenol groups in place of one alcohol group.

These reagents are less sensitive, and have less effect on pyrite, than the xanthates, but are often used in conjunction with the xanthates for the flotation of ore containing pyrite.

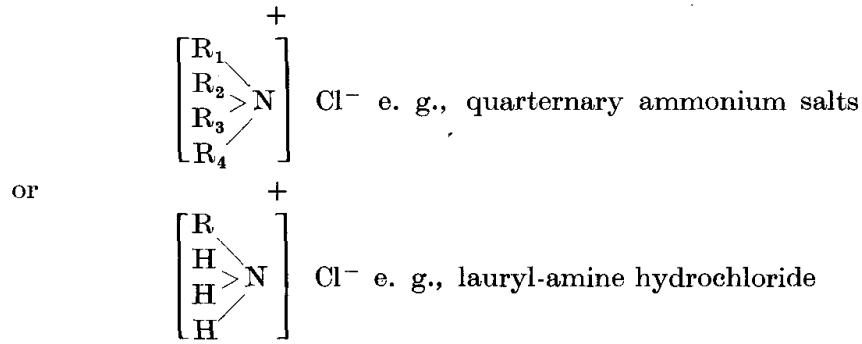3. Thiocarbanilide, substituted thioureas. These are the reaction product of aniline and carbon disulphide.

Structural formula
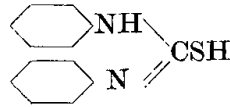
Thiocarbanilide has the distinction of being the first non-frothing chemical collector to be introduced on a commercial scale, but it has been replaced to a large extent by xanthates and dithiophosphates owing to the fact that its insolubility in water renders its use difficult.

4. Fatty and aromatic carboxylic acids, such as oleic acid. Structural formula $R \cdot COOH$

Cationic-type promoters

Wetting agents of the cationic type may be represented by the following structural formula:

$$\begin{bmatrix} R_1 \\ R_2 \\ R_3 \\ R_4 \end{bmatrix}^{+} N \quad Cl^{-} \quad e.\ g.,\ \text{quarternary ammonium salts}$$

or

$$\begin{bmatrix} R \\ H \\ H \\ H \end{bmatrix}^{+} N \quad Cl^{-} \quad e.\ g.,\ \text{lauryl-amine hydrochloride}$$

Cationic-type wetting agents as a class are characterized by the fact that they are surface-active materials containing both polar and non-polar groups. In solution, these compounds ionise to give

a large positive ion and a negative halide or acetate ion. The positive ion generally has a heavy hydrocarbon loading.

The point of most interest in cationic-type wetting agents, so far as flotation in concerned, is the fact that such reagents are specific promoters for silica and silicate minerals. Their selective action towards silica and silicates may almost be compared with the action of xanthates or aerofloat promoters toward sulphide minerals.

Nonionic-type promoters

These are promoters that do not give rise to ions and yet are effective. Of the nonionised collectors, those having a sulphide or disulphide polar group are used in practice for the recovery of sulphide minerals.

c) Addition agents

These are used to modify the normal floatability of minerals with promoters and frothers. They may be subdivided into three general groups:

1. Regulating agents. Their function is to control the alkalinity of the pulp, also to counteract interfering effects of detrimental slimes. In modern froth flotation, alkaline circuits are used almost exclusively for the treatment of sulphide ores and most nonmetallic ores as well. For any particular ore, there is a definite range of pH at which optimum results are obtainable. Therefore, proper control of this alkalinity is of prime importance. Reagents commonly used to obtain desired alkalinity are lime and soda ash. Soda ash is preferable for the treatment of gold ores, because lime tends to depress free gold and must be more carefully controlled than soda ash.

2. Activating agents. Their function is to assist the flotation of those minerals which do not float well without them or have been thrown down by a depressor. The outstanding example of such function is the use of copper sulphate to float sphalerite.

3. Depressing agents. The function of a depressing agent is to assist the selectability of one mineral from another. Originally,

before the development of specific depressants, it was impossible to float minerals of one kind from others whose floatability was closely similar. The commonest depressors are lime, sodium cyanide, and zinc sulphate. Lime, in particular, throws down pyrite, and if it is added in sufficient quantity, subsequent addition of xanthate has no effect. It is possible that the effect of lime on pyrite is one of coating the mineral with a reaction product after dissolving the surface of the mineral to some extent.

Examples of other depressants include the "600" series reagents of the *American Cyanamid Co.* for depressing graphite and carbonaceous slates when floating gold ores. These reagents are primarily derivatives of lignin.

3. Theories of flotation

While flotation has progressed mainly on empirical lines, theoretical considerations as to the reasons for the observed phenomena are becoming better known and further advance has thus been made more probable. No theory has yet been advanced which is universally acceptable, or applicable to the known facts. The following theories, however, have been brought forward during the last forty-five years:

a) The Electrical or Cataphoresis Theory.—According to this, the mineral particles carry an electrical charge of an opposite sign to that on the air bubbles, to which they accordingly become attached. The gangue particles bear a charge similar to that on the bubbles, and are consequently repelled.

b) The Gas Theory.—This postulates an adsorbed gas layer on the mineral particle, to which the air bubble readily adheres.

c) The Chemical Theory.—This assumes that flotation reagents react chemically on the surface molecules of the minerals.

d) The Adsorption Theory.—According to this the collector molecules are adsorbed on the surfaces of the minerals which the collector can cause to be floated. The forces involved are too weak to act on gangue particles. Grouping of the atoms of the collector molecules is assumed.

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e) The Contact Angle Theory.—This postulates that flotation is related to the hysterisis of the contact angle. The smaller the contact angle the less the probability of floating, and this becomes equal to 0 when the contact angle is equal to 0°.

4. The physiochemical concept of the mechanism of flotation

In the attempt to explain the mechanism of flotation, opinion was divided between adsorption and chemical reaction. There is much evidence that both adsorption and chemical reaction play their part in every flotation separation. The physiochemical concept of the mechanism of flotation takes into consideration such properties of physical chemistry as surface energy, surface tension, adsorption, contact angle, polarity, surface condition, and reactivity, as well as surface chemistry [26].

A mineral particle becomes attached to an air bubble when its surface is hydrophobic, and conversely a particle with a hydrophilic surface will not become attached to an air bubble. This fact is apparent from a theoretical consideration of the contact angle relationships at the mineral surface-water-air boundary, and it has been verified experimentally in a large number of cases [27]. Two types of bubble attachment have been observed in actual flotation practice. In one type the mineral particle is located in the liquid-air interface, with part of it actually projecting into the air space. This is the type of attachment which would be predicted from theory. In a second type of attachment the mineral particle is entirely in the liquid but is contiguous to the bubble. In this case it is probable that the mineral is held to the adsorbed interfacial layer between the air and the liquid. In either case the floated mineral has a hydrophobic surface, whereas the unfloated one is hydrophilic. Practically all the minerals as they occur in the natural state possess a hydrophilic surface. It is the function of the collector to convert the surface of the mineral to be floated from a hydrophilic to a hydrophobic state. At the same time it must not alter the hydrophilic surface of the gangue. There has been considerable controversy over the actual mechanism by which the collector

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exerts its amazing selective action. One school contends that in most cases there is a stoichiometric chemical reaction between the collector and the mineral, occurring only on the surface layers of the insoluble mineral and resulting in a hydrophobic compound. The other contention is that the collector is adsorbed and oriented on the mineral surface to form a hydrophobic film.

The chemical reaction hypothesis is stated as follows by *Taggart, Taylor,* and *Knoll* [28]: "All dissolved reagents which, in flotation pulps, either by action on the to-be-floated or on the not-to-be-floated particles affect their floatability, function by reason of chemical reactions of well recognised types between the reagent and the particle affected". "Chemical reactions of well-recognised types" is a description the meaning of which is bound to differ from one metallurgist to another; it seems, however, that, practically speaking, the hypothesis amounts to a statement that metathesis takes place between the mineral and the agent, yielding a less-soluble reaction product that precipitates at the mineral surface.

The adsorption hypothesis adopted by *Wark* [29] may be stated as follows: "All ions dissolved in a flotation pulp liquor adsorb at mineral surfaces. At each mineral surface, the adsorption of each dissolved ion is specific; i. e., it depends on the dissolved ion and on the mineral; this specific ion adsorption is also a function of the concentration of the dissolved ion under consideration and of that of other dissolved ions. If and when a sufficient proportion of the mineral surface is covered by the effective collector ions, the particle becomes floatable".

Some Russian work [30] on the widely used xanthate flotation agents explains their action as due to interaction between the anions of the xanthate and the dissolved cation of the galena, chalcopyrite, pyrite, blende, or similar minerals under treatment. This leads to the precipitation of the heavy metal xanthate and to its selective attachment to the sulphide particle in the pulp. In the flotation of calcite, barytes, fluorspar, and similar polar non-metallic minerals, the action is more complex.

Collectors follow the general composition rule of surface-active agents in that they possess polar (hydrophilic) and non-polar (hydrophobic) portions in the molecule [31]. The polar portion
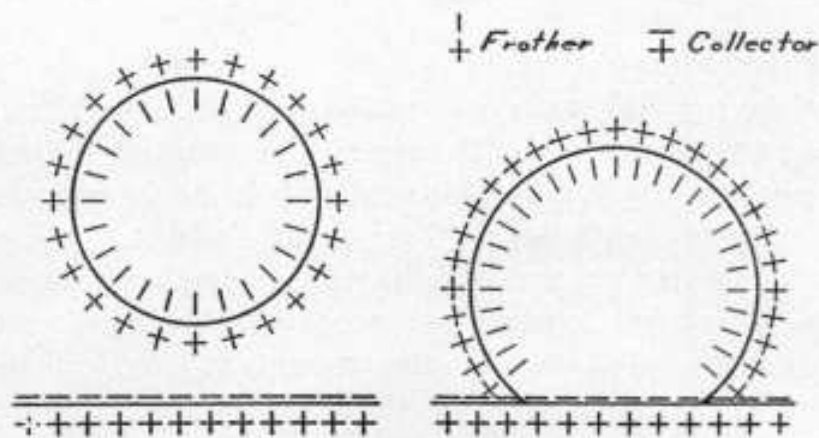
Fig. 7. Mechanism for illustrating a collector and a surface

attaches by chemical or adsorption means to the particle surface, leaving the non-polar radicals extended into the water surrounding the particle. If then a bubble of air contacts this surface, there arises a possibility for the reduction of the total surface of the liquid. Reference to figure 7 will demonstrate this fact.

In figure it may be seen that the bubble is surrounded by frother molecules and the surface by collector molecules, both being polar-nonpolar in nature.

a) Fig. 8 b)
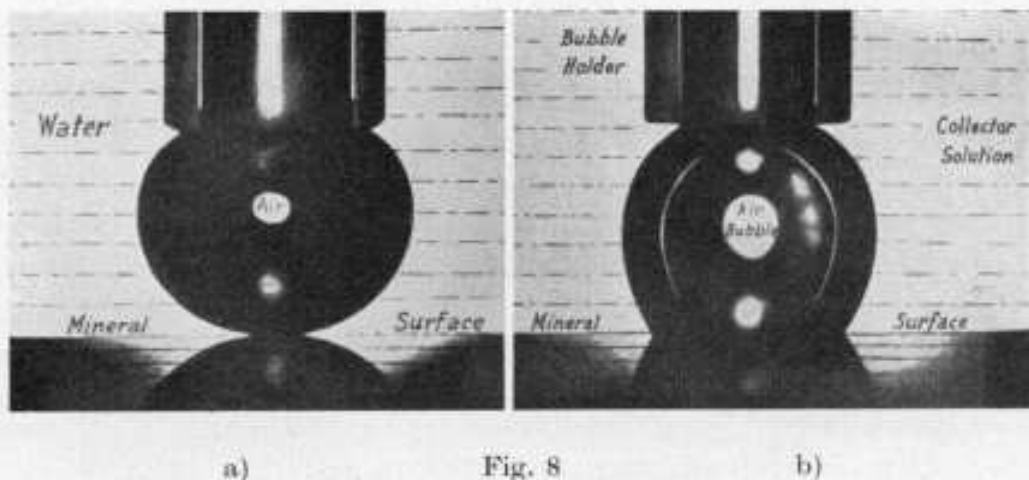
a) An air bubble does not stick to a clean, polished galena surface. The contact angle equals 0°

b) An air bubble adheres to galena immersed in a solution of ethyl xanthate. The contact angle equals 60°. (After Wark)

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Wark has shown that an air bubble brought into contact with a clean mineral surface normally does not attach to it. If a suitable collector is added, however, the selected mineral will acquire a water-repellent and air-avid surface, as shown in figure 8.

The collectors and the frothers definitely affect the surface tension of the water and the interfacial tensions involved in flotation. Interfacial tensions are probably the most important, and *Wark* has pointed out that the amounts of chemicals used are seldom sufficient measurably to affect the surface tension of the water used.

L. J. Christmann [32], in a classic dissertation on the physio-chemistry of flotation, has explained in detail the properties and functions of the various types of reagents used in modern froth flotation. It is to be noted that since the publication of his disser-tation nothing has been discovered in the art of flotation that his postulates do not explain. The dissertation can be summarized in the following points:

1. A hypothesis of froth structure has been given.
2. Every solid particle may be termed a macromolecule.
3. Whether such a molecule can be floated depends upon whether it is wetted by, or will repel, water (i. e. whether it is hydro-phile or hydrophobe).
4. All particles possessing an ionic lattice are easily wetted by water.
5. Most minerals and gangues possess a layer or atomic lattice, but oxide minerals and gangues adsorb water into the lattice and are readily wetted by water.
6. Sulphide minerals possess a lattice which does not adsorb water. When such a lattice (macromolecule) adsorbs ions or is oxidized, it forms a macroion which is hydrophile.
7. Promoters have been classified as:
   a) Organic acidic compounds or their alkali metal salts.
   b) Organic compounds with atoms possessing "lone pairs" of electrons.
8. The promoters give negative ions which react with the macro-ionic minerals to give new non-polar macromolecules.

9. The (R) of the promoter attaches itself to the (R) of the frother to give a mineralised bubble.
10. The depressants react with positive macroions to form complex negative macroions which do not react with the promoters.

A comprehensive paper has been published by *R. S. Dean* and *P. M. Ambrose* [33], in which many observations are explained in terms of a general theory of the use of wetting agents, according to which there is a balance between two influences: a) a tendency to clean and wet the mineral particles, and b) a tendency to preserve or form a water-repellent coating on them and thereby favour attachment of air bubbles. The agents are typically compounds in which one end of the molecule is particularly hydrophilic, and the ions formed in water are still polar and become oriented at oil-water and air-water interfaces.

A. F. Taggart and *N. Arbiter* [34] have correlated contact-angle measurements and solubility data with the mechanism of collection of minerals by amine-type collectors; they proceed to interpret practical flotation results on the basis of the theory that a precipitate of amine salt containing the acid ion of the mineral is formed at the solid surface.

A study by *Rogers* and *Sutherland* [35], of the Commonwealth Council for Scientific and Industrial Research, Melbourne, of the theory of the flotation process states that the earlier theories, such as those due to *Taggart, Dean,* and *Wark,* are oversimplified and too generalized, and that each example of flotation needs to be considered as a separate problem.

5. Application of flotation to gold ores

One of the most striking differences between the ores of gold and those of all other metals lies in the extremely small proportion which the desired material bears to the worthless gangue by which it is accompanied. Occasionally, hand specimens in vein stuff are found containing several per cent of the precious metal, but these are of exceptional occurrence, and have no economic importance.

The greater part of the vein gold now being produced is derived from ores containing only about one part of gold in 100,000, or even less.

There has been a rapid increase in the application of flotation to gold ores since the change from acid to alkaline circuits and from oils to synthetic organic chemical reagents. The advent of chemical promoters, such as xanthates and dithiophosphates, has greatly helped in the development of the flotation of gold ores.

In ores where the gold is associated with such minerals as pyrite or other sulphides of iron, arsenopyrite, stibnite, or cupreous sulphides, two methods of attack are available, viz.:

a) Cyaniding, which may necessitate grinding to slime in order to release the gold, and

b) Flotation with the recovery of a concentrate containing most of the sulphide minerals, which are re-ground and then treated with cyanide.

If the association of the gold with the pyrite is extremely intimate, the ore may indeed be refractory to cyanidation. Up to the introduction of flotation the best means for treating such ores involved a roasting operation followed by cyanidation, which is relatively expensive. This can now be avoided by flotation of the sulphides, at much lower cost. In cases where roasting or re-grinding is necessary, flotation usually reduces the amount of material to be roasted or re-ground. Moreover, flotation has a particular merit as an auxiliary process for the removal of cyanicides that might be found with the ore.

Floating auriferous sulphides is, in general, not widely different from floating the same sulphides without gold, and these ores may be regarded as sulphide gold ores even though the gold does not occur as sulphide.

According to *J. Traube* [36], flotation cost of such ores is lower than that necessitated by direct cyanidation, and is but a fraction of the cost of roasting followed by cyanidation.

Direct cyanidation is dear because of the fine grinding necessary and the cyanide consumption, which is large compared with the reagent consumption in flotation.

Examples of practice:

Lake View and Star Mine. The gold is here present as free gold, as tellurides, and in association with pyrite. The combined concentrate from different flotation processes is dried, roasted and cyanided. A total extraction of 90 per cent of the precious metals is obtained.

Wiluna Gold Mines, West Australia. The ore is a complex sulphide containing pyrite, arsenopyrite, some telluride and copper, silver and gold. The treatment is practically the same as that at the Lake View and Star. 92 per cent of the gold is recovered by flotation.

McIntyre Porcupine. Fine grinding and total cyanidation have given place to flotation, re-grinding the concentrate to 325 mesh, and then cyaniding.

Golden Cycle Mill, Colorado. Telluride and sulphides are floated in water, and 70 to 80 per cent of the gold is recovered. The concentrate is roasted with crude ore and subsequently cyanided.

Dealulkruzii, Roumania. The recovery of gold was formerly effected by amalgamation and hearth roasting. The introduction of flotation has raised the recovery from 55 to 90 per cent. The flotation concentrates represent only one-thirtieth of the weight of the raw ore.

Brad, Roumania. 92 per cent of the values are recovered by flotation, and the concentrate is treated with cyanide solutions without previous roasting.

Rand Gold Mines. For ore assaying 0.145 oz./ton, the pH value had to be above 7.2 in order to get both a good froth and a good recovery. Re-grinding of the pyritic concentrate to 325 mesh and continuous air agitation are indicated as necessary.

IV. Cyanidation

The cyanide process will always be indissolubly associated with the names of its discoverers, *MacArthur* and *Forrest.* The solvent action of potassium cyanide on metallic gold has long been known, but it was believed that the use of an electric current in conjunction

was needed to quicken the action, which was otherwise too slow to be of any practical value. Until the surprising results of the process were made public, metallurgists were disposed to regard the use of cyanide as a solvent for gold in ores as chimerical, on account of the instability of potassium cyanide and the slowness of its action on gold, combined with its high cost and poisonous character.

The process consists essentially in attacking gold ores with dilute solutions containing less than one per cent of alkaline cyanide, a slight excess of caustic soda or lime being added to ores rendered acid by the oxidation of pyrite, and then precipitating the precious metals with zinc.

The process was originally applied to the residue of gold ore after amalgamation, but was later used in many instances for the direct treatment of ore or concentrates.

It is necessary to use dilute solutions, which have little action on pyrite and many other base minerals that might be found in the ore, but readily dissolve gold and silver.

1. Action of potassium cyanide on gold

In general, when potassium cyanide is named, it may be taken to include sodium cyanide and mixtures of the two salts. Also, in the equations given, potassium and sodium are interchangeable except from the point of view of ionic dissociation.

Wright, of Birmingham, discovered in 1840 that metallic gold is soluble in potassium cyanide when a current of electricity is passing through. *Bagration*, in 1843 [37], studied the action of cyanides on plates of gold in the absence of currents of electricity, and noted that they were slowly dissolved. *Faraday*, in 1857 [38], pointed out that gold-leaf is dissolved by a dilute solution of the salt, and also showed that if the gold floats on the surface of the liquid, so that one side of the leaf is in contact with the air while the other is bathed by the solvent, the action is much more rapid than if the metal is completely submerged. *White* [39], confirming this, found that gold-leaf floating in 6—10 per cent cyanide solution

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is dissolved much more rapidly than submerged pieces. *Elsner* [40] had previously, in 1846, furnished some evidence that the presence of oxygen is required for the solution of the gold. *W. A. Dixon* [41] found, in 1877, that although cyanide by itself was slow in dissolving gold, its action was hastened by the addition of alkaline oxidising agents. He also mentions calcium hypochlorite, potassium ferrocyanide and manganese dioxide as hastening the action.

On evaporation of the solution, colourless octahedral crystals of auropotassium cyanide, $KAu(CN)_2$, are formed, which may be viewed as being a double cyanide, produced as follows:

$$4 \, Au + 8 \, KCN + O_2 + 2 \, H_2O = 4 \, KAu(CN)_2 + 4 \, KOH$$

This equation is often referred to as "Elsner's equation", but *Elsner* himself does not give it.

According to the equation given above, 130 parts by weight of KCN in the presence of 8 parts of oxygen suffice for the solution of 197 parts of gold. This has been proved by *J. S. Maclaurin* [42] to be the case in all carefully-conducted experiments.

According to *G. Bodländer* [43], the chemical action in the dissolution of the gold is as follows:

$$2 \, Au + 4 \, KCN + 2 \, H_2O + O_2 = 2 \, KAu(CN)_2 + 2 \, KOH + H_2O_2$$

and subsequently

$$4 \, KCN + H_2O_2 + 2 \, Au = 2 \, KAu(CN)_2 + 2 \, KOH$$
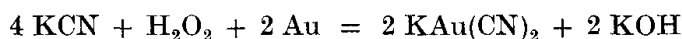
the ultimate result being as in the "Elsner" equation mentioned above.

Evidence has been adduced that a substance is formed reacting like H_2O_2, but the action represented by the equation given by *Bodländer* may be limited to an insignificant part of the whole mass.

Janin [44] suggested that hydrogen is liberated, thus:

$$2 \, Au + 4 \, KCN + 2 \, H_2O = 2 \, KAu(CN)_2 + 2 \, KOH + H_2$$
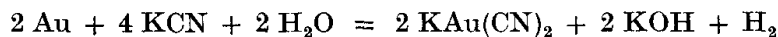
Ramsay [45] also contends, from the change in potential of gold in cyanide solutions, that hydrogen is evolved when gold is dissolved in cyanide. The weight of evidence, however, is against this view.

Barsky, Swainson and *Hedley* [46] have compared the free energies of the equations said to represent the solution of gold in

cyanide, viz., *Elsner's*, *Bodländer's* and *Janin's*, with the respective equilibrium constants. They have found that the Elsner equation best fits the case and that the reaction will proceed to completion in the presence of oxygen—i. e. until the metal or the cyanide has been removed. *Elsner's* equation is considered not strictly correct, however, as it does not show the intermediate formation of hydrogen peroxide. *Bodländer's* equation, from the same reasoning, is found to be possible and, it is suggested, may partially represent the state of affairs.

It has been suggested by *Christy* [47], and *Julian* and *Smart* [48], that the dissolution of gold in cyanide depends primarily on the electromotive force exerted. *H. F. Julian* [49] attached great weight to the formation of local voltaic circuits. *Christy* [50] contends that the rate of dissolution of gold in cyanide depends on two factors, the electromotive force of gold in cyanide solutions, and the number of hydroxyl ions in the solution. *Michailenko* [51], however, denies the latter contention, and states that OH ions do not accelerate the velocity of solution, but that an excess of them has a retarding influence.

Allen [52] observes: "Pure gold is doubtless insoluble in a pure solution of cyanide in oxygen-free water, but perfect purity and absolute uniformity are unattainable under commercial conditions. A minute degree of contamination is sufficient to cause galvanic action, stimulating dissolution. Gold in cyanide solution in a glass vessel is only slightly attacked, but dissolution is active if the metal is in contact with iron. The action diminishes, however unless the iron surface remains exposed, or polarisation is prevented, and oxygen is effective for this purpose. *Elsner's* equation ignores galvanic action and subsidiary chemical changes that may have an important influence on commercial extraction (for example, pyrite is electronegative to gold, as *Julian* demonstrated)".

2. Influence of oxygen in cyanidation

It is clear that oxygen, or some other substance acting similarly, is necessary to assist potassium cyanide in dissolving gold. The oxygen unites with the hydrogen and reduces the loss of cyanide as

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hydrocyanic acid gas. It was formerly supposed that any oxygen coming in contact with the solution would be at once taken up by the cyanide, forming cyanates, but it is now well known that oxygen and cyanide remain side by side without rapid union between them taking place. Cyanates are without action on metallic gold. The most noticeable effect of extra aeration of the solution is a substantial reduction in cyanide consumption [53]. In cyanide solution oxygen is dissolved to the extent of about 8 mg. per litre. The solubility is practically unaffected by increasing the cyanide up to as much as 2 per cent sodium cyanide [54]. *Wallace* [55] draws attention to the drop in extraction due apparently to loss of dissolved oxygen in mill solutions. In a vacuum no gold dissolves in cyanide solution.

The rate of dissolution of gold in cyanide solutions is proportional to the amount of oxygen dissolved in the solution or, in other words, to its partial pressure. Hence, to attain the greatest rate of dissolution the solutions must be fully aerated. To supply the oxygen, various oxidising methods have been tried, such as the passage of a current of air through the solution and the addition of various oxidisers.

In practice, aeration of cyanide solutions has been attempted by the use of agitators, Pachuca tanks, turbo mixers and "super agitators".

3. Effect of temperature

Bagration [56] states that solution of metallic gold in potassium cyanide is facilitated by a rising temperature. *Julian* and *Smart* [57] observed that up to 85° C. the solubility of gold in 0.25 per cent potassium cyanide solution increases, but after that declines.

In practice, the solutions are rarely heated. It is generally believed that an increase of temperature is accompanied by an increase in the consumption of the cyanide.

4. Cyanidation of flotation concentrate

During the past several years many mining companies have installed flotation plants for the treatment of gold ores. In most cases the choice of flotation was dictated by the character of the

ore, which was refractory to cyanidation owing to the presence of cyanicides or to the intimacy of association of the gold with sulphides. *Wiluna Gold Mines* in West Australia, *Beattie Gold Mines* in Canada, and many of the gold properties in South Africa, are examples of this. Some of the larger companies made a careful comparison of the capital and operating costs of direct cyanidation versus flotation followed by cyanidation of the concentrates, and found that the latter process yielded higher net cash returns. This was the case with *McIntyre Porcupine Gold Mines Ltd.* when the new plant was built in 1930.

Although flotation has proved to be a highly versatile process in the treatment of gold ores, it is in reality only an intermediate step in the ultimate reduction of the ore to bullion. Whereas cyanidation is a complete process, producing bullion which can be shipped directly to the refinery, flotation produces concentrates which must be subjected to further treatment in order to obtain the gold in the form of bullion. Two processes are available for the production of bullion from concentrates, viz: smelting and cyanidation. Smelting is usually applied when the concentrate is very rich.

The concentrate is a fraction of the original ore from which it came, and will contain not only gold but also many sulphides and cyanicides. Special treatment of the concentrate may be necessary, which treatment would otherwise have to be applied at considerable cost and probably not so efficiently to the whole bulk of the material. Some gold concentrates are definitely not amenable to cyanidation without a preliminary treatment such as selective flotation of cyanicides, or roasting. Other concentrates yield their gold content to cyanide after extensive grinding, oxidation, or very long periods of agitation.

No standard method for the treatment of concentrates has yet been, or can be, decided on, as it must vary with the nature of the concentrate and the mode of occurrence of gold in it. Hence, the practice differs at almost every mill.

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C. Experimental part

I. Mineralogical examination of the ore

The ore dealt with in this work is from one of the gold mines in the Eastern Desert of Egypt. The main constituent of the ore is quartz. The following is a mineralogical examination of a representative sample of the ore [58].

The object of this examination was to identify the mineral constituents of the ore and to determine with which minerals the gold is associated :

"A partial chemical analysis of the ore was as follows:

		%
Silica	(SiO_2)	74.45
Alumina	(Al_2O_3)	3.20
Iron	(Fe)	2.45
Lime	(CaO)	5.47
Magnesia	(MgO)	3.60
Graphite	(C)	0.14

Mineralogical examination of the sample shows the major constituents to be quartz, dolomite and feldspar. In addition the ore contains a small amount of sulphide minerals, mostly pyrite, arsenopyrite and pyrrhotite, and a little iron oxide and graphite.

Approximate mineralogical composition of the ore is as follows:

		%
Quartz	(SiO_2)	65
Feldspar	($K_2O \cdot Al_2O_3 \cdot 6\ SiO_2$)	15
Dolomite	($MgCO_3 \cdot CaCO_3$)	17
Total sulphides	(FeS_2, Fe_nS_{n+1}, FeAsS)	0.5
Iron oxides	(Fe_2O_3 etc.)	2 ·

The sulphide minerals are mostly very finely dispersed through the gangue constituents, and very fine grinding would be necessary to effect their complete liberation from the gangue minerals.

No free gold was seen in the sample, even in extremely finely ground ore. The sulphide minerals were found to contain gold, and

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it is considered that most, if not all, of the gold is in association with these sulphide minerals.

Very little graphite was seen in the ore, and what little is present is very finely dispersed through the gangue minerals, probably mostly with the quartz".

II. Ore treatment method (flotation)

According to the earlier discussion about the relation of the manner of gold occurrence to metallurgy, the existence of gold in this ore—so intimately associated with the sulphide minerals, in addition to the presence of some graphite—suggests the general lines of treatment to be briefly as follows: "Collective flotation of the sulphide minerals followed by cyanidation of the concentrate".

Straight cyanidation of the ore gives erratic and unsatisfactory results, as will be shown later on. The whole of the ore may be roasted to burn off the graphite, but such an operation is an expensive one. It is less costly to float off the gold and refractory minerals as a low-bulk concentrate which may be roasted, if necessary, and then cyanided. Flotation may be used as an auxiliary process for the removal of graphite and similar cyanicides by depressing them before the flotation of gold is proceeded with.

1. Practical adaptation of flotation tests

The ore consists of whitish and greyish lumps of quartz. The whole ore-bulk was divided into two equal parts. One part was prepared for further tests as it was, and the other part was sorted, by hand-picking, into the whitish and the greyish lumps. A series of flotation tests was carried out on each of the three batches separately. The gold content in the three batches was different, the greyish being the richest and the whitish the poorest.

The practical adaptation of the froth-flotation tests involves a sequence of steps. These various steps may be listed as follows:

a) Liberation of the values by crushing and grinding.

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b) Dilution of the ground pulp with water to the desired degree of most efficient flotation.

c) Conditioning of the pulp with reagents prior to flotation.

d) Flotation for separation of desired minerals from worthless gangue.

Liberation of the values.—Before the minerals can be separated by froth flotation they must, of course, be liberated from the gangue, so that each particle is composed as nearly as possible of valuable or worthless material only.

Dilution of the ground pulp.—For effective flotation, it is necessary that the ground pulp be sufficiently dilute to give the minerals ready mobility. This step is performed in the flotation machine itself.

Conditioning of the ore pulp prior to flotation.—Although many of the reactions between reagents and minerals in the flotation process are practically instantaneous, there are instances where appreciable time is required to bring about the desired effects. In such cases the pulp must be "conditioned". This step is also carried out in the flotation machine itself.

Flotation operation.—In the actual flotation operation, the separation of the minerals is accomplished in specially designed machines. A considerable variety of such flotation machines are in use. Choice of flotation machine is dependent on many factors in addition to metallurgical and mechanical efficiency. However, the modern trend is towards machines of the straight mechanical type capable of providing intense agitation and a large volume of finely dispersed air in the pulp. According to *Rose*, the mechanical types are probably best suited to gold ores, as they give a better conditioning effect in the cells themselves owing to the violent agitation which can be obtained.

2. *Factors considered in flotation tests*

The batch flotation tests are conducted along the lines dictated by experience with other ores. According to *Gaudin* [59], once a fairly successful test has been obtained, the results can be improved

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by "cut-and-try methods" assisted by experience. Small increases or decreases in the quantities of reagents, changes in the pH, in the grinding conditions, in the place of addition of reagents, in the time of contact between reagents and ore, in the extent of aeration during flotation, etc., are tried to get substantial improvements in metallurgical results. The replacement of one reagent by a different reagent having slightly different properties is also tried.

These laboratory investigations are conducted with the prime objective of obtaining the utmost in metallurgical efficiency. Once ideal metallurgy is attained, further test work could be conducted for the simplification and cheapening of the process for commercial application.

The factors to be considered in the following investigations are:

> The size of the particles
> The reagents to be used
> The type of machine
> The ratio of water to solids
> The amount of agitation and aeration required
> The temperature
> The pH value of the circuit
> The nature of water used
> The time of flotation

Particle size. Grinding should free all the valuable mineral from the gangue. For flotation of sulphides, the floatable particles must not exceed some maximum size ranging from minus 48 to minus 65 mesh. The ore is first broken to about $^1/_2$-inch size in a jaw crusher. Portions of the crushed product are taken for the individual tests and reduced to the necessary fineness by grinding for varying times in a laboratory steel ball mill.

Reagents. For successful flotation, a suitable chemical "weather" must be established and maintained. This is obtained through the addition of reagents. Which reagents to use, and in what quantity, is ascertained by testing and comparison with practice. According to *Gaudin*, no hard and fast rule can be given as to the quantity of reagent to use. He goes on to say that the variations are very great from one ore to another, and appreciable from day to day or even

Fig. 9 a)

Glass side 500-gram Denver
laboratory flotation machine
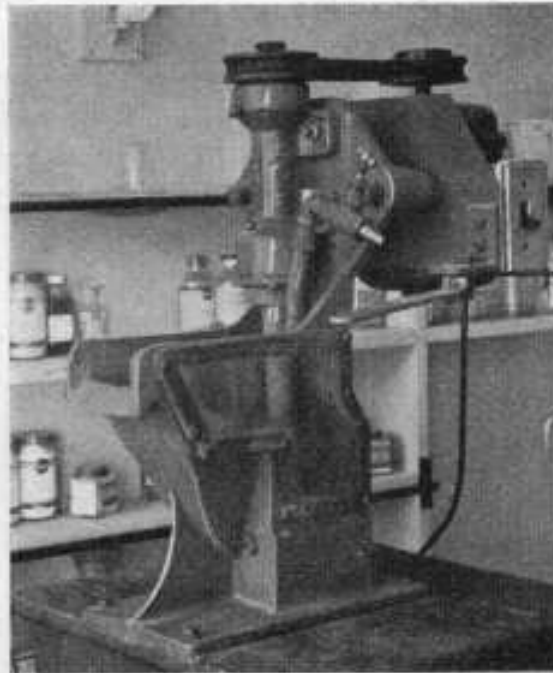
Fig. 9 b)

Section of a Denver flotation
machine showing three zones

Concentration zone

Separation zone

Agitation zone

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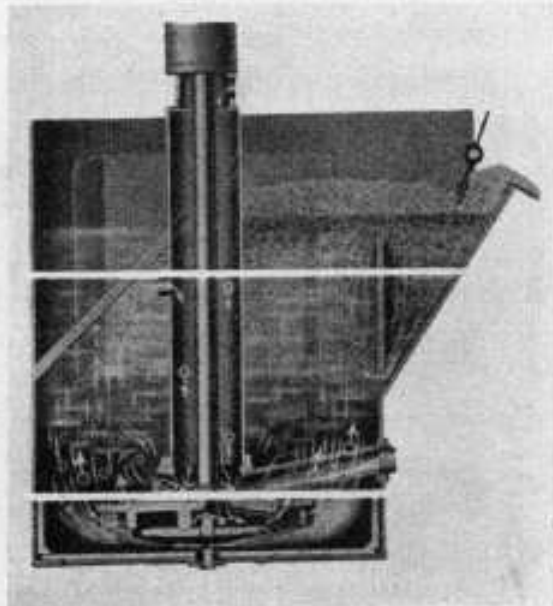
hour to hour on any one ore. Liquid reagents are introduced in the flotation cell drop by drop, by means of an eye-dropper. Water-soluble collectors are made up as 5% solution and added with a pipette. Addition agents such as soda ash and copper sulphate are added in the solid form. It is claimed by the makers of the laboratory machine that it may be found that a laboratory machine requires more reagents than will be used in practice.

Type of machine. The machine used is the 500-gram laboratory Denver "Sub-A" cell. This machine is of the subaeration type which can handle a batch of ore equal to 500 grams based on a pulp density of approximately 4 parts of water to one part of solids by weight. The design of the laboratory cell follows the same principles as those of the commercial sizes. The makers claim that it will be, therefore, possible to duplicate test results in actual practice with the larger Denver Sub-A cell. In practice, flotation is not carried out in one large cell, but in a series of cells, thus dividing the time necessary for flotation by the number of cells used.

The Denver Sub-A principle consists of three actions that take place inside the cell. These actions are grouped into three zones:

a) Agitation zone. In this zone the pulp is thoroughly mixed and aerated with both reagents and air. As the pulp cascades over the impeller it is thrown outwards by the centrifugal action of the impeller. This action also causes a positive suction of air down the stand pipe. Bubbles are thus created and the pulp is intimately diffused with air bubbles. During agitation the necessary amount of air is drawn down the stand pipe, the amount being controlled by the valve at the top. Outside air under low pressure may be added if necessary.

b) Separation zone. The pulp, thoroughly conditioned and aerated, is passed directly through the separating zone. Once the bubbles pick up their load of minerals and start toward the surface, agitation must cease or the bubbles will drop their load. This is assured through the use of a baffle vane. Mineral-laden bubbles separate from worthless gangue and pass upward to form froth.

c) Concentration zone. As the mineral-laden bubbles move to the

pulp level, they are carried forward to the overflow lip by the crowding action of succeeding bubbles, and the balance of the material is returned into the agitating chamber for retreatment.

Pulp density. For the separating stage in flotation, the pulp must be dilute enough to permit particle rearrangement to proceed freely. It is a practical fact that no pulp thicker than 35 to 40 per cent solids by weight can be used. For the preparatory or conditioning stage the thickest kind of pulp is advantageous, as it results in an economy of most reagents. The limit is set by the appearance of plastic properties, i. e., of a situation in which proper mixing is impossible. There is, of course, no metallurgical objection to having the pulp more dilute than 15 to 20 per cent solids during the separating stage; in fact somewhat better results are obtained in many instances with more dilute pulps. There is, however, an economic objection to overdilute pulps; such pulps require greater quantities of reagents, more water, more energy, and more machinery for each ton of ore.

Amount of agitation and aeration. The amount of agitation can be varied by varying the speed of the machine, which is done by moving the shifting lever. The amount of aeration can be controlled by the valve at the top of the air stand pipe; also, outside air under low pressure may be added if necessary.

Temperature. These tests are conducted at the temperature naturally assumed by the pulp, frequently in the range of 12° to 20° C.

p_H *value.* The p_H is determined on the tailing water for each flotation test by using the universal indicator paper (p_H 1—10).

Nature of the water. Tap water is used in these tests. In some of the tests a 3% NaCl solution is used in order to investigate the possibility of using sea water in the treatment process in actual practice. The saline solution used is not of exactly similar composition to sea water available at the mines, but the results may be sufficiently informative.

Time of flotation. In order to afford every particle an opportunity to stick to gas bubbles, it is necessary to prolong the floating operation for a certain length of time.

3. Graphite in ore

While the ore is hard and abrasive, it produces only a small amount of primary slimes on grinding, and settles rapidly without the use of any flocculating agents. The use of a slight flocculent gives a clear overflow. Settlement in saline solution is particularly rapid, although the graphite remains floating on the surface of the pulp.

Graphite is one of the mineral constituents of this gold ore, and the cyanidation of such ores is usually accompanied by erratic results. The gold is dissolved by the cyanide solutions, but unfortunately carbon possesses the property of adsorbing a gold compound from cyanide solutions, with the result that much of the gold may be discarded with the tailings. The adsorptive properties of the different varieties of carbonaceous minerals appear to vary considerably, and while some graphitic ores may be cyanided with little ill effect, there are others where much of the gold is adsorbed. Evidence is in favour of the fact that the ore dealt with in this work is of the latter type, as will be shown later on.

In fact, the carbon content of this ore is so small that even the complete elimination of this material does not appreciably affect the ratio of concentration, but the ore on floating gives large carbon-coated bubbles and a black froth, and the graphite rejection improves the character of the froth.

If some of these carbon-coated bubbles are examined under the microscope, the graphite particles are seen attaching themselves to the outer walls of the bubbles as seen in figure 10.

If some pure graphite powder is shaken vigorously with water and allowed to stand, it does not settle. Most of the particles float on the surface of the pulp and some remain in suspension. If the floating layer is examined with a strong hand lens, it is found to be composed of graphite particles with air bubbles entrapped around them. If such a pulp is agitated for five minutes with a depressing agent, such as starch or one of the 600 series reagents (primarily derivatives of lignin) developed by the *American Cyanamid Company*, and then allowed to settle, all the graphite sinks to the bottom of the pulp, forming a homogeneous sediment with no air bubbles
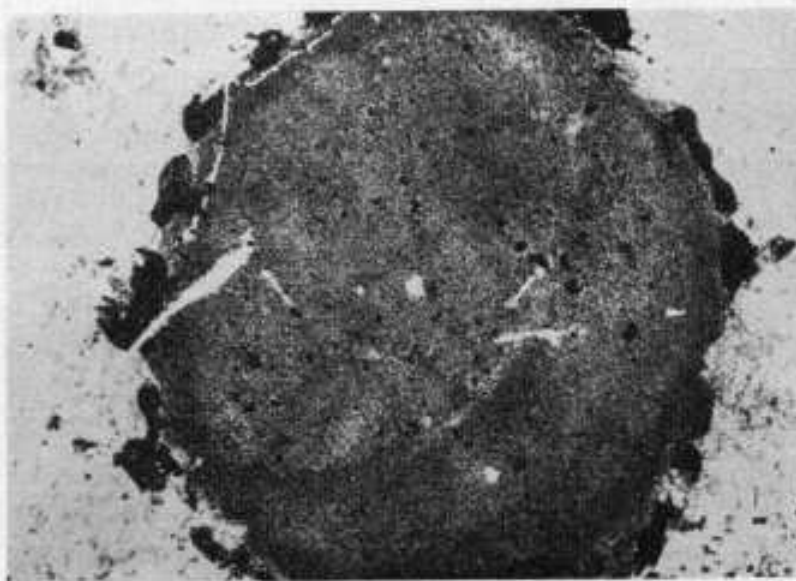
Fig. 10. Carbon-coated bubble. (Actual photomicrograph)

in between. If the same treatment is applied to a sample of the ground ore, no carbon-coated bubbles are formed on flotation and the froth is lighter in colour. This is due to the fact that the graphite becomes completely wetted and is depressed in the pulp.

In the tests where depressants are used before the flotation of gold is proceeded with, no carbon analysis is made for every test. Good depression is indicated from observations of the froth.

The following table summarizes the results of some of the tests where graphite is depressed by using members of the 600 series reagents. Depressing agents are added and agitated with the pulp for ten minutes in a 33.3% solids pulp. Other reagents are then added and the gold is floated off. The time of float is 10 minutes.

The determination of the graphite is carried out by decomposing the sulphide minerals in the sample by heating with a 50% nitric acid solution, followed by washing and drying the sample. The carbon is then estimated by burning it into CO_2, and determining the amount of CO_2 formed.

Table 2. Depression of graphite

Depressant	Grams/m.ton	ratio of concentration	% rejection of carbon
R 610	480	5,14/1	61.98
R 633	480	5.56/1	62.51
R 637	480	4.53/1	56.7
R 637	480	10.00/1	64.5
R 645	480	5.29/1	60.5
R 645	480	6.55/1	68.9

Action of depressants on graphite:

The term "wetting" is applied to solid surfaces, and means exactly what it says, that is, that the solid is brought into direct contact with the liquid. Most investigators agree that the degree to which a solid body is wetted by a liquid may best be measured by the contact angle at the liquid-solid-air line. As an example we may take the case of a droplet of liquid resting on a solid plate surrounded by air. The contact angle, in such a case, is the angle between the liquid-gas interface and the liquid-solid interface, that is to say the angle is measured within the liquid. The angle of contact is 0° when the solid is completely wet and in this case the liquid attracts the solid as much as it attracts itself. If the contact angle is positive the solid is incompletely wet. The measurement of contact angle is by direct observation of the angle of a droplet of the liquid in contact with a plate of the solid. The microscope is used in the actual measurement of the angle.

In the case of graphite, a small droplet of water is placed on the face of a clean, well-polished plate of graphite and the droplet is viewed from the side through a microscope fitted with an arrangment in the eye piece for the measurment of the contact angle. The contact angle is measured several times with different droplets, and ranges between 61° and 63°. Then similar droplets of starch solution and of solutions of reagents 610, 633, 637, 645 are allowed separately to rest on the surface of a fresh, similar graphite plate and viewed through the microscope. The contact angle is observed to decrease gradually and the droplets to spread on the graphite

plate, and after ten to fifteen minutes the droplets are completely spread and the contact angle becomes 0°.

Another interesting point noticed during the measurment of the contact angle is that there appears to be a more or less permanent difference in contact angle depending upon whether the graphite and the reagent in question have or have not been in contact prior to the measurment. In other words, the work of adhesion is less for a dry solid than for one which has been previously wetted, and this points to the possibility that some of the liquid is adsorbed during the first contact of solid and liquid, thus changing the character of the solid surface.

4. Gold flotation

a) Metallurgical calculations

Throughout this work all assays are referred to the short ton of 2000 lbs., and all screen analysis are in terms of the British Institution of Mining and Metallurgy Screen Series.

Ratio of concentration.—The ratio between weight of feed and weight of concentrate is a measure of the concentration in weight that has been effected; this ratio is known as the ratio of concentration.

Recovery.—From the relative weights of the concentrate and tailing, and their assay, it is possible to figure what fraction of the gold contained in the feed is recovered in the concentrate. This fraction, when expressed as a percentage, is known as the recovery.

The simplest method of calculating the recovery involves a knowledge of the weight and assay of the concentrate:

Let (A) be the weight of the concentrate and (a) its assay, (B) and (b) the corresponding quantities for the tailing, and (F) and (f) the corresponding quantities for the feed.

By definition, the recovery (R) in the concentrate is:

$$R = \frac{Aa}{Aa + Bb} = \frac{Aa}{Ff}.$$

Recovery may be calculated from the metal contained in the feed (Ff) or from that contained in the products (Aa + Bb). Theoretically the two methods should give the same answer; practically they do not. This discrepancy arises from the character of analytical results, and from operating losses.

b) Assay for gold

The method used in this work for the assay of gold is the crucible or pot fusion method. The principle of this method is as follows:

The very finely ground and sifted ore, or the concentrate, is mixed with lead oxide, charcoal, and a suitable slag-forming material. The crucible charge usually consists of the ore mixed with:

aa) Litharge or red lead, which oxidizes the metalloids present in the ore and supplies the metallic lead which collects the gold.

bb) A reducing agent which will react with the red lead to form metallic lead and will reduce certain basic oxides that might be present in the ore (e. g. Fe_2O_3) to a state in which they will form fusible slags.

cc) A flux or fluxes which will combine with the gangue to form slag which is fluid at reasonable temperatures.

The following charge was found to be most suitable for this ore:

Ore	1 Assay ton (29.166 g.)
Read lead	40 g.
Charcoal	3.5 g.
Sodium carbonate	30 g.
Borax	20 g.

The charge is weighed out, mixed, and transferred to a fire-clay crucible. The crucible is placed in a moderately hot muffle furnace, and the charge allowed to frit for 10—15 minutes, and the temperature is then raised to 1000° C. When the charge is in a state of quiet fusion it is poured into a warm, dry mould. When cool, the lead button is cleaned of slag by hammering. The button should weigh 25—30 g. If it is too small, the quantity of reducing agent should be increased. The lead button obtained is cupelled in the

muffle furnace at 1,100°C. The resulting bullion is flattened by hammering and then "parted" by being dropped first into hot nitric acid of specific gravity 1.2—1.3, when the silver dissolves, leaving a brown skeleton of gold. This is washed, dried, annealed to remove brittleness, and weighed. If the proportion of silver present is too small, parting is not complete. The amount of silver in the bullion is made about four times that of the gold. As the silver content of the ore is not required, the excess silver may be added to the charge before fusion.

For convenience it is customary to weigh out the ore in "assay-ton" units. Ore is usually weighed in avoirdupois tons (2000 lbs.) and gold in troy ounces. The assay report usually gives the ounces troy contained in a ton avoirdupois. Since there are 29,166 troy ounces in an avoirdupois ton, the assay ton is made equal to 29.166 g. Then, if one assay ton of ore is used, each milligram of gold obtained will represent one ounce per ton. For weighing out the ore an inexpensive balance is used (pulp balance). For weighing the gold button, a button balance sensitive to 0.01 mg. should be used.

Polarographic determination of gold in ores: The fire assay method previously mentioned is the practical method used for the determination of gold in ores. According to *Herman* [60], gold in ores can be determined polarographically, and he recommends the following procedure:

"The solution of gold obtained from ores is treated with excess potassium iodide to reduce the gold to the aurous state, and the liberated iodine is removed with excess sodium sulphite. The solution is then made about 0.1 N with respect to potassium cyanide and 2 N with respect to potassium hydroxide, and, after filtering, the polarogram of the aurous cyanide complex is recorded".

Some samples of the ore and concentrate are analysed polarographically, but the results, when compared with those of the reliable fire assay method, are found to be unsatisfactory. It is probable that this procedure has not been given enough study in this work, and continuation of this study to include other samples and other varieties and types of gold ores, if they can be obtained, might greatly assist in achieving more satisfactory results.

c) Flotation reagents

Reagents used in the flotation tests are always designated in grams per metric ton. The following is a list of the chief reagents used, their description, and the method of feeding.

Table 3. Reagents used in flotation tests

Reagent	Description	Method of feeding
Aerofloat 15	Frother and promoter, 15% phosphorus pentasulphide in cresylic acid	Undiluted liquid
Aerofloat 25	Frother and promoter, 25% phosphorus pentasulphide in cresylic acid	Undiluted liquid
Aerofloat 208	A non-frothing water-soluble dry aerofloat	5% solution
Copper sulphate	Activating agent, blue crystalline solid $CuSO_4 \cdot 5\ H_2O$	Solid
Pine oil Yarmor F	A standard high grade pine oil of uniform consistency, steam distilled, frother	Undiluted liquid
Reagent 301	Promoter. A xanthate of a higher petroleum alcohol	5% solution
The "600 Series" reagents	Depressing agents, derivatives of lignin	2—3% solution
Soda ash	Regulating agent, white powder. Na_2CO_3	Dry
Starch	Depressing agent, white granular solid	2—5% solution
Xanthate Z-6	Promoter, potassium pentasol amyl xanthate	5% solution
Wood creosote No I	Frother and promoter, black, slightly viscous liquid	Undiluted liquid

d) Flotation tests

The following table summarizes the flotation tests carried out on the three batches of ore previously mentioned. The table includes a description of the procedure, the different reagents used, and the

results attained. Miscellaneous data for individual tests follow. Tests are arranged in the order in which they were actually conducted.

Tests 1—23 were conducted on the greyish quartz, tests 24—35 on the whitish quartz, and tests 36—54 on the mixed batch which is supposed to represent the whole bulk of the ore.

The gold values differ from one batch to the other, the greyish quartz being relatively the richest and the whitish quartz the poorest. The gold values vary in every single batch. In the greyish quartz values vary from 0.152—0.228 oz./ton, in the whitish quartz from 0.0505—0.0965 oz./ton, and in the mixed batch from 0.11—0.15 oz./ton. This variation in values is due to the patchy nature of the gold ore. Moreover, flotation tests in this work were conducted in small series on separate portions of the ore taken from the batch and prepared by crushing and grinding to the required fineness; this to ensure that the tests were always conducted on freshly ground portions.

Table 4. Flotation tests

Test no.	1	2	3	4	5	6
Feed: Au oz./ton	0.152	0.152	0.152	0.228	0.190	0.170
Grind: %— 40 mesh	—	—	—	—	—	—
— 50 ,,	100	100	100	—	—	—
—150 ,,	61.5	61.5	61.5	100	100	100
—250 ,,	—	—	—	86.7	68.92	68.92
Concentrate: Au oz./ton I	0.61	1.396	2.405	1.318	0.49	1.458
II	—	—	—	—	—	—
Recovery: % Au I	56.8	56.1	56.6	52.0	21.1	66.1
total	—	—	—	—	—	—
Ratio of conc.: I	7.2	16.7	28.5	11.1	12.1	12.9
total	—	—	—	—	—	—
Conditioning % solids	33.3	33.3	33.3	33.3	33.3	33.3
Flotation % solids	20	20	20	20	20	25
Tap water or % NaCl sol.	W	W	3%	W	W	W
Time (total) min.	15	10	10	10	10	6
pH	7—8	8—9	8—9	8—9	8—9	8—9
Promoter, Frother: g./ton						
Reagent 301	75	75	75	75	—	83
K-ethyl-xanthate	—	—	—	—	75	—
Xanthate Z-6	—	—	—	—	—	—
Aerofloat 208	75	75	75	75	—	83
Aerofloat 25	—	—	—	—	—	—
Aerofloat 15	310	310	124	124	—	105
Pine oil Yarmor F	—	—	—	—	95	—
Wood creosote	—	—	—	—	—	—
Depressing agents: g./ton						
Reagent 637	480	480	480	480	—	500
Reagent 645	—	—	—	—	500	—
Starch	—	—	—	—	—	—
Activating agents: g./ton						
Copper sulphate	—	—	—	—	2500	—
Regulating agents: g./ton						
Sodium carbonate	—	1000	1000	1000	1500	1666

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Table 4 (cont.)

Test no.	7	8	9	10	11	12
Feed: Au oz./ton	0.170	0.196	0.224	0.224	0.199	0.199
Grind: %— 40 mesh	—	100	—	—	—	—
— 50 ,,	—	—	—	—	—	—
—150 ,,	100	—	100	100	100	100
—250 ,,	68.92	29.2	66.8	66.8	66.8	66.8
Concentrate: Au oz./ton I	2.612	3.057	1.116	1.524	0.758	0.993
II	—	0.26	—	—	—	—
Recovery: % Au I	62.4	59.5	63.6	52.5	79.0	63.4
total	—	69.7	—	—	—	—
Ratio of conc.: I	24.6	22.9	7.8	12.9	4.8	7.9
total	—	8.3	—	—	—	—
Conditioning % solids	33.3	33.3	22	22	22	22
Flotation % solids	20	30	22	22	22	22
Tap water or % NaCl sol.	W	W	W	W	W	5%
Time (total) min.	6	8	10	10	12	25
pH	8—9	8—9	8—9	8—9	7—8	7—8
Promoter, Frother: g./ton						
Reagent 301	83	133	150	150	146	146
K-ethyl-xanthate	—	—	—	—	—	—
Xanthate Z-6	—	—	—	—	—	—
Aerofloat 208	—	133	73	73	146	146
Aerofloat 25	—	—	—	—	—	—
Aerofloat 15	105	210	210	210	56	112
Pine oil Yarmour F	—	—	—	—	46	92
Wood creosote	—	—	—	—	—	—
Depressing agents: g./ton						
Reagent 637	500	480	500	500	—	500
Reagent 645	—	—	—	—	—	—
Starch	—	—	—	—	455	—
Activating agents: g./ton						
Copper sulphate	—	—	—	1818	—	—
Regulating agents: g./ton						
Sodium carbonate	1666	1666	1500	1500	—	—

Table 4 (cont.)

Test no.	13	14	15	16	17	18
Feed: Au oz./ton	0.199	0.199	0.167	0.178	0.178	0.178
Grind: %— 40 mesh	—	—	100	100	100	100
— 50 ,,	—	—	—	—	—	—
—150 ,,	100	100	—	—	—	—
—250 ,,	66.8	66.8	29.2	29.2	29.2	29.2
Concentrate: Au oz./ton I	1.720	1.524	2.468	1.434	1.064	1.439
II	0.390	—	—	0.368	0.429	0.504
Recovery: % Au I	65.6	39.6	74.5	71.7	60.3	61.2
total	76.8	—	—	86.8	74.0	75.7
Ratio of conc.: I	13.1	20.3	19.9	11.3	9.9	13.1
total	7.5	—	—	6.8	6.4	8.0
Conditioning % solids	22	35	35	40	40	30
Flotation % solids	22	35	35	40	40	30
Tap water or % NaCl sol.	5%	3%	3%	3%	W	W
Time (total) min.	15	20	20	20	20	20
PH	7—8	8—9	8—9	8—9	7—8	7—8
Promoter, Frother: g./ton						
Reagent 301	219	114	114	150	150	150
K-ethyl-xanthate	—	—	—	—	—	—
Xanthate Z-6	—	—	—	—	—	—
Aerofloat 208	219	114	114	150	150	150
Aerofloat 25	—	82	82	144	144	190
Aerofloat 15	168	—	—	—	—	—
Pine oil Yarmour F	138	29	29	102	102	135
Wood creosote	—	—	—	—	—	—
Depressing agents: g./ton						
Reagent 637	500	—	—	—	—	—
Reagent 645	—	—	—	—	—	—
Starch	—	—	—	100	—	—
Activating agents: g./ton						
Copper sulphate	—	—	—	—	—	—
Regulating agents: g./ton						
Sodium carbonate	—	2970	2970	3000	—	—

Table 4 (cont.)

Test no.	19	20	21	22	23	24
Feed: Au oz./ton	0.178	0.178	0.175	0.175	0.175	0.0505
Grind: %— 40 mesh	100	100	100	100	100	—
— 50 ,,	—	—	—	—	—	100
—150 ,,	—	—	—	—	—	—
—250 ,,	29.2	29.2	29.2	29.2	29.2	54.6
Concentrate: Au oz./ton I	0.893	1.547	1.488	2.14	1.93	0.596
II	0.251	0.308	0.109	0.242	0.2	—
Recovery: Au % I	53.8	64.7	88.7	78.7	85.8	97.3
total	65.9	77.0	95.3	86.1	94.0	—
Ratio of conc.: I	9.4	13.4	9.6	15.6	12.9	12.1
total	5.2	6.9	4.7	8.5	6.7	—
Conditioning % solids	30	25	40	20	20	20
Flotation % solids	30	25	40	20	20	20
Tap water or % NaCl sol.	W	W	3%	3%	3%	3%
Time (total) min.	20	15	20	20	20	10
pH	7—8	7—8	8—9	8—9	8—9	8—9
Promoter, Frother: g./ton						
Reagent 301	150	100	150	125	175	100
K-ethyl-xanthate	—	—	—	—	—	—
Xanthate Z-6	—	—	—	—	—	—
Aerofloat 208	150	100	150	125	175	100
Aerofloat 25	190	—	—	—	—	—
Aerofloat 15	—	100	62	124	186	62
Pine oil Yarmour F	135	80	102	200	200	—
Wood creosote	—	—	—	—	—	—
Depressing agents: g./ton						
Reagent 637	—	—	—	—	—	—
Reagent 645	666	—	—	—	—	—
Starch	—	—	—	—	—	—
Activating agents: g./ton						
Copper sulphate	—	—	—	—	—	—
Regulating agents: g./ton						
Sodium carbonate	—	—	3000	3000	3000	3000

Table 4 (cont.)

Test no.	25	26	27	28	29	30
Feed: Au oz./ton	0.0505	0.0505	0.0965	0.0965	0.088	0.077
Grind: %— 40 mesh	—	—	—	—	—	—
— 50 ,,	100	100	100	100	100	100
—150 ,,	—	—	—	—	—	—
—250 ,,	54.6	54.6	76.6	76.6	76	76
Concentrate: Au oz./ton I	0.445	0.74	0.82	1.505	1.377	1.457
II	—	—	—	—	—	—
Recovery: Au % I	98.5	96.7	76.8	96.8	75.7	98.0
total	—	—	—	—	—	—
Ratio of conc.: I	8.9	15.1	11.0	16.1	20.7	19.3
total	—	—	—	—	—	—
Conditioning % solids	20	33	20	33	33	33
Flotation % solids	20	20	20	20	20	20
Tap water or % NaCl sol.	W	W	W	W	W	W
Time (total) min.	10	10	10	10	10	10
pH	7—8	7—8	7—8	7—8	7—8	7—8
Promoter, Frother: g./ton						
Reagent 301	100	100	100	100	100	100
K-ethyl-xanthate	—	—	—	—	—	—
Xanthate Z-6	—	—	—	—	—	—
Aerofloat 208	100	100	100	100	—	100
Aerofloat 25	—	—	—	—	—	—
Aerofloat 15	62	62	62	62	62	62
Pine oil Yarmour F	—	—	—	—	—	—
Wood creosote	—	—	—	—	—	—
Depressing agents: g./ton						
Reagent 637	—	—	—	—	—	—
Reagent 645	—	—	—	—	—	—
Starch	—	400	—	400	400	400
Activating agents: g./ton						
Copper sulphate	—	—	—	—	—	—
Regulating agents: g./ton						
Sodium carbonate	—	—	—	—	—	—

Table 4 (cont.)

Test no.	31	32	33	34	35	36
Feed: Au oz./ton	0.086	0.086	0.086	0.080	0.086	0.1128
Grind: %— 40 mesh	—	—	—	—	—	—
— 50 ,,	100	100	100	100	100	100
—150 ,,	—	—	—	—	—	—
—250 ,,	76	76	76	88	88	64.5
Concentrate: Au oz./ton I	1.23	1.26	1.078	1.482	1.21	1.898
II	—	—	—	—	—	—
Recovery: Au % I	92.6	92.6	82.7	82.7	98.0	67.6
total	—	—	—	—	—	—
Ratio of conc.: I	15.4	15.7	15.1	22.3	14.3	24.9
total	—	—	—	—	—	—
Conditioning % solids	33	33	33	33	33	33
Flotation % solids	20	20	20	20	20	20
Tap water or % NaCl sol.	W	W	W	W	W	W
Time (total) min.	10	10	10	10	10	5
pH	7—8	7—8	7—8	7—8	7—8	7—8
Promoter, Frother: g./ton						
Reagent 301	100	75	75	75	75	75
K-ethyl-xanthate	—	—	—	—	—	—
Xanthate Z-6	—	—	—	—	—	—
Aerofloat 208	100	75	75	75	75	75
Aerofloat 25	—	—	—	—	—	—
Aerofloat 15	62	62	62	62	62	62
Pine oil Yarmour F	—	—	—	—	—	—
Wood creosote	—	—	—	—	—	—
Depressing agents: g./ton						
Reagent 637	—	—	—	—	—	—
Reagent 645	—	—	—	—	—	—
Starch	400	400	400	400	400	400
Activating agents: g./ton						
Copper sulphate	—	—	—	—	—	—
Regulating agents: g./ton						
Sodium carbonate	—	—	—	—	—	—

Table 4 (cont.)

Test no.	37	38	39	40	41	42
Feed: Au oz./ton	0.1128	0.1128	0.1128	0.1128	0.1128	0.1125
Grind: %— 40 mesh	—	—	—	—	—	—
— 50 ,,	100	100	100	100	100	100
—150 ,, ·	—	—	—	—	—	—
—250 ,,	64.5	63	63	63	73	88
Concentrate: Au oz./ton I	1.476	1.7315	1.365	1.19	1.055	0.605
II	—	—	—	—	—	—
Recovery: Au % I	65.4	75.2	72.1	81.4	84.5	76.7
total	—	—	—	—	—	—
Ratio of conc.: I	20.0	20.0	17.1	12.9	11.1	7.1
total	—	—	—	—	—	—
Conditioning % solids	33	20	20	20	20	20
Flotation % solids	20	20	20	20	20	20
Tap water or % NaCl sol.	W	W	W	W	W	W
Time (total) min.	10	10	10	10	10	10
p_H	7—8	7—8	7—8	7—8	7—8	7—8
Promoter, Frother: g./ton						
Reagent 301	75	75	75	125	75	75
K-ethyl-xanthate	—	—	—	—	—	—
Xanthate Z-6	—	—	—	—	—	—
Aerofloat 208	75	75	75	125	75	75
Aerofloat 25	—	—	—	—	—	—
Aerofloat 15	62	62	62	124	124	124
Pine oil Yarmour F	—	—	51	—	—	—
Wood creosote	—	—	—	—	—	—
Depressing agents: g./ton						
Reagent 637	—	—	—	—	—	—
Reagent 645	—	—	—	—	—	—
Starch	400	—	—	—	—	—
Activating agents: g./ton						
Copper sulphate	—	—	—	—	—	—
Regulating agents: g./ton						
Sodium carbonate	—	—	—	—	—	—

Table 4 (cont.)

Test no.	43	44	45	46	47	48
Feed: Au oz./ton	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125
Grind: %— 40 mesh	—	—	—	—	—	—
— 50 ,,	100	100	100	100	100	100
—150 ,,	—	—	—	—	—	—
—250 ,,	95	95	63	63	63	63
Concentrate: Au oz./ton I	0.83	0.755	1.262	1.003	0.938	1.238
II	—	—	0.13	0.11	—	0.1
Recovery: Au % I	67.7	59.0	77.6	82.6	85.5	82.3
total	—	—	86.6	89.8	—	91.8
Ratio of conc.: I	10.9	11.4	14.4	10.7	9.7	15.5
total	—	—	6.7	6.3	—	5.5
Conditioning % solids	20	20	20	30	20	20
Flotation % solids	20	20	20	30	20	20
Tap water or % NaCl sol.	W	W	W	W	W	W
Time (total) min.	10	10	20	20	10	20
PH	7—8	7—8	7—8	7—8	7—8	7—8
Promoter, Frother: g./ton						
Reagent 301	150	150	200	250	—	—
K-ethyl-xanthate	—	—	—	—	—	—
Xanthate Z-6	—	—	—	—	150	250
Aerofloat 208	75	75	200	250	50	—
Aerofloat 25	—	—	—	—	144	288
Aerofloat 15	124	124	186	186	—	—
Pine oil Yarmour F	—	—	—	—	—	—
Wood creosote	—	—	—	—	—	—
Depressing agents: g./ton						
Reagent 637	—	—	—	—	—	—
Reagent 645	—	—	—	—	—	—
Starch	—	400	—	—	—	—
Activating agents: g./ton						
Copper sulphate	—	—	—	—	—	—
Regulating agents: g./ton						
Sodium carbonate	—	—	—	—	—	—

Table 4 (cont.)

Test no.	49	50	51	52	53	54
Feed: Au oz./ton	0.12	0.136	0.136	0.15	0.13	0.11
Grind: %— 40 mesh	—	—	—	—	—	—
— 50 ,,	—	100	100	100	100	100
—150 ,,	100	—	—	—	—	—
—250 ,,	—	84	84	52	84	84
Concentrate: Au oz./ton I	1.015	0.82	1.07	1.205	1.415	1.366
II	0.145	0.18	0.08	0.135	—	—
Recovery: Au % I	80.1	74.2	79.1	64.9	83.6	96.7
total	84.0	85.4	85.1	72.6	—	—
Ratio of conc.: I	10.5	8.1	9.9	12.4	13.0	12.8
total	7.8	4.8	5.0	6.0	—	—
Conditioning % solids	20	20	20	20	33	33
Flotation % solids	20	20	20	20	20	20
Tap water or % NaCl sol.	W	W	W	W	3%	3%
Time (total) min.	20	15	15	15	20	20
pH	7—8	7—8	8—9	7—8	7—8	7—8
Promoter, Frother: g./ton						
Reagent 301	—	200	200	200	—	—
K-ethyl-xanthate	—	—	—	—	—	—
Xanthate Z-6	250	—	—	—	300	400
Aerofloat 208	—	200	200	200	—	—
Aerofloat 25	144	—	—	—	—	—
Aerofloat 15	—	186	186	186	248	124
Pine oil Yarmour F	—	153	153	153	—	—
Wood creosote	—	—	—	—	—	100
Depressing agents: g./ton						
Reagent 637	—	—	—	—	—	—
Reagent 645	—	—	—	—	—	—
Starch	—	—	—	—	—	—
Activating agents: g./ton						
Copper sulphate	—	—	—	—	—	—
Regulating agents: g./ton						
Sodium carbonate	—	—	3000	—	—	—

The following table no. 5 contains miscellaneous data and remarks complementary to table no. 4.

Table 5. Miscellaneous data of flotation tests

Test no.	Remarks
1	Depressing agent agitated with the pulp for ten minutes. Other reagents then added and agitated with the pulp for five minutes. Valve on standpipe completely opened. Froth very watery and loaded with gangue particles. Colour of froth light.
2	Sodium carbonate is added to the pulp before other reagents. Other reagents are then added as in test no. 1. Air valve on standpipe adjusted so as to allow the froth just to overflow the lip. Froth less watery and less loaded with gangue.
3	Valve on standpipe adjusted to allow minimum amount of aeration.
4	Aeration as in tests no. 2–3.
5	Aeration as in tests no. 2–4.
6	Aeration as in tests no. 2–5, at the end air valve completely opened.
7	Aeration as in test no. 6.
8	Half the amounts of the promoters conditioned with the pulp and, after 4 minutes' flotation, the filtrate readded to pulp. Rest of reagents added and float another 4 minutes.
9	Air valve opened gradually during flotation. Froth more copious than in the previous tests.
10	Conditions as in test no. 9.
11	Half the amounts of promoters conditioned with the pulp. Rest added during flotation. Froth carries much gangue.
12	Half the amounts of promoters and frothers conditioned with the pulp. After 10 minutes' flotation the rest is added and flotation continued.
13	After 10 minutes the filtrate of the concentrate is readded to the original pulp and flotation continued for another 5 minutes. Application of outside air under low pressure.
14	No application of outside air.
15	Air valve gradually opened. Froth watery but does not carry much gangue.
16	Application of outside air. After 10 minutes filtrate readded to pulp and flotation continued. Reagents added at intervals.
17–18	No application of outside air.
20	Application of outside air.
21	Reagents added at intervals. After 10 minutes flotation filtrate readded to pulp and flotation continued. Violent aeration and agitation.

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Table 5 (cont.)

Test no.   Remarks

22   Reagents added at intervals, test carried out in two steps with violent agitation and aeration.

23   Reagents added at intervals. Violent agitation and aeration in all tests up to no. 54.

24   Pulp conditioned 7 minutes with half the promoters and the whole amount of Aerofloat 15. Rest of the reagents added at intervals.

26   After conditioning with starch, pulp is diluted to 20% solids and conditioned with other reagents as in previous tests.

27   Presence of more slimes, due to finer grinding.

29   Test repeated giving same recovery.

30   The average of 6 successively repeated tests is 93% recovery.

31   Time of conditioning reduced to 5 minutes, instead of 10 minutes.

33   Conditioning the pulp with the promoters at a thicker pulp. Rest of conditions as in previous test. Lower recovery. Test repeated giving the same result.

34   Reduction in amount of aeration to get higher ratio of concentration.

35   More aeration than in previous test. Other conditions the same.

36   Pulp conditioned with starch, then diluted and conditioned with promoters. Violent agitation and aeration. Carbon-coated bubbles disappear on application of starch, and froth improves.

37   Same conditions as in test no. 36, time of float doubled.

40   Addition of reagents at intervals. Froth is stronger and does not break quickly.

41   Froth is strong.

45   Flotation in two steps. Addition of reagents at intervals.

46   Flotation in two steps. Addition of reagents at intervals.

47   Flotation is carried out for 5 minutes and filtrate readded to original pulp, then another 5 minutes' flotation.

48–51   Flotation in two steps.

52   Test repeated several times on same grind with increase in amounts of reagents and aeration, giving always unsatisfactory recovery.

54   Average of 4 identical tests giving almost the same satisfactory recovery.

5. Discussion of the factors considered in flotation tests

With reference to the list of factors on page 48, we may now discuss these factors in the light of the results obtained from the previously-mentioned investigations:

a) Particle size: Grinding should free as much as possible of the mineral in the ore. In treating the greyish quartz, best results were achieved at a relatively coarse grinding, i. e. at minus 40 mesh with 29.2% minus 250 mesh. No tests at a coarser grinding with exactly the same conditions were conducted, but good recovery, under different conditions, was always favoured by relatively coarse grinding. Finer grinding, with all other conditions being the same, gave lower recovery with the same ratio of concentration (see tests no. 14 and 15), and even when the ratio of concentration is lower, i. e. when more of the middling product comes with the concentrate, the recovery is still lower (see tests no. 2 and 4).

A typical decrease in recovery due to finer grinding is demonstrated in tests no. 41—44. The ore treated in each of the tests no. 42—44 is a portion of the ground ore dealt with in the preceding test, reground finer. All the tests were conducted under exactly the same conditions.

In the whitish batch of quartz which produces more slimes, finer grinding at minus 50 mesh, with 76% minus 250 mesh (test no. 27) gave much lower recovery than relatively coarser grinding at minus 50 mesh, with 54% minus 250 mesh (tests no. 24, 25, 26). An important factor in getting good recovery at finer grinding in this batch is the depression of slimes (compare tests no. 27 and 28). It is known that in the separation of sulphides the maximum size of the particles rarely exceeds 35 mesh; separation becomes less efficient at bigger sizes due to insufficient liberation of values. Flotation separations normally are much less efficient on sizes below 600 mesh, and recovery is greatly reduced if particles are finer than 5 to 10 microns. Therefore, both over-grinding and coarse grinding should be avoided. The lower limits of size amenable to flotation vary with the type of mineral to be floated, as well as the nature of the gangue constituents. There must be an optimum size of particle to attain good recovery. In tests conducted on the whitish ore we may be in the optimum range of particle size. Lower recovery is caused by excessively finer grinding.

Lower recovery obtained at excessively fine grinding may also be due to the contamination of the surfaces of the mineral in the presence of slimes [18], as the use of starch as a slime depressant

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(test no. 44) to improve the recovery had no effect in this respect.

All the results obtained during these investigations concerning the sizes of the particles agree with the known rules and laws of flotation.

b) Reagents used:

Promoters and frothers: According to *Gaudin*, sulphide flotation is applied to gold ores in which the value of the base metals associated as sulphides with gold is small. Such ores may be regarded as sulphide gold even though the gold does not occur as sulphide. The gold in such ores may occur as free particles or as particles occluded in sulphides (see fig. 5, p. 18). From the mineralogical examination of the ore (p. 45—46) it is clear that no free gold was seen in the sample, even in extremely finely-ground ore. The sulphides were found to contain gold and it is considered that most, if not all, of the gold is in association with these sulphide minerals. All evidence, therefore, was in favour of chosing xanthates and aerofloats as promoters in these investigations. Aerofloats used were Aerofloat 15 and Aerofloat 25, both containing cresylic acid. They are promoters and frothers, although Aerofloat 25 has less frothing power. As frothers, pine oil and wood creosote were used in the same tests.

The froth was almost always watery, composed of spherical bubbles and high in pulp content. It is known that such is the case when treating ores in which the mineral content that is to be floated is low, as in gold ores. Application of pine oil improves the nature of the froth (compare tests no. 38 and 39), but it does not improve the recovery. Increase in amounts of Aerofloat 15 and other promoters improves the froth and gives better recovery (test no. 40), the improvement in froth being due apparently to the increase in amount of Aerofloat 15 (compare tests 40 and 41).

Another reagent used to improve the nature of the froth is wood creosote no. I (see test no. 54). Wood creosote is usually used in cases where the total mineral to be floated is small as compared to the gangue.

Reagent 301 in combination with Aerofloat 208 effected good results with the greyish and whitish quartz. Tests no. 12 and 13,

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22 and 23 show an improved recovery with increase in amounts of promoters applied under similar conditions. Not every increase in amounts of promoters improves the recovery; improvement of recovery depends on other factors such as the application of other reagents such as activating agents, depressing agents, regulating agents, agitation, aeration etc. An example of this is demonstrated by tests no. 7 and 9 where, under nearly the same other conditions, the amounts of promoters were substantially increased with no effect on the recovery.

In test no. 5, potassium ethyl xanthate was applied as the main promoter instead of reagent 301 used in test no. 4. All other conditions were nearly the same. A considerably lower recovery was effected in test no. 5. This accords with experience in the flotation of sulphides, that the xanthates with the ethyl and propyl groups are less powerful than those with groups of greater molecular weight.

A lower recovery due to finer grinding and in spite of the use of the same amount of promoters is an old experience in flotation work. This fact is fairly demonstrated in tests no. 41—44.

Tests no. 31 and 32 conducted on the whitish quartz gave the same good recovery and ratio of concentration under the same conditions, the only difference being in the amounts of promoters used. More promoter in test no. 31 did not give higher recovery.

Xanthate Z-6 was used successfully in combination with Aerofloat 15. In test no. 54, it was used in combination with Aerofloat 15 and wood creosote. Very satisfactory recovery with a relatively higher ratio of concentration was achieved.

From all these different tests it is seen that no hard and fast rule can be given as to the quantity of reagent to use. The variations are great from one batch to another. Each condition in every single flotation test plays its part in effecting successful flotation. It is seen that for the greyish batch a quantity of 460—730 g./ton of promoters and frothers was necessary to effect a recovery of over 90%, while for the whitish ore a quantity of 260 g./ton was sufficient. *Gaudin* [61] states that the variations in quantity of reagent are very great from one ore to another, and appreciable from day to day or even hour to hour on any one ore.

Depressing agents: The object of depressing the graphite in the pulp by depressing agents was to obtain a concentrate from which a high percentage of the graphite is rejected. As will be shown later, the flotation concentrate produced without the use of any depressing agents is amenable to good cyanidation, in spite of the fact that the concentrate, being a portion of the original ore, may contain graphite in a more concentrated form.

In tests where depressants, such as the 600 series reagents or starch, were used, the depressants behaved variably. In some of the tests (see tests no. 18 and 19) reagent 637 seems to depress the sulphides as well as the graphite. The action of starch seems to be more favourable, as can be seen in tests no. 11 and 16 in comparison with tests no. 1—10, 12, 13, and 19.

In tests conducted on the whitish ore, where grinding produces more slimes, larger quantities of starch could be used without effecting the flotation of gold. Very satisfactory recoveries were attainable with a higher ratio of concentration. Lack of starch caused lower recoveries (test no. 27). Starch has another important effect in that it caused quicker filtration of the concentrate owing to its action on the slimes. It seems that starch is a more suitable depressing agent than reagent 637, as it raises the ratio of concentration considerably, but its use should be controlled carefully as it may depress the sulphides when very little slime is present (compare tests no. 36 and 37 with test no. 38).

Similar results in regard to the effect of starch in depressing slimes are reported by *Leaver* and *Woolf* [62].

Good recovery and relatively high ratio of concentration were attained by treating the mixed batch without using any depressing agent (test no. 54).

Activating agents: Experience gained from flotation of metallic sulphides indicates that flotation of sulphides can be activated by the use of copper sulphate. It is said that this action is due to an exchange between the surface of the sulphide mineral and many heavy metal ions, and copper sulphide is known to be one of the best floating minerals with the xanthates. Actually, no noteworthy improvement was obtained in tests where copper sulphate was used as an activator (see tests no. 5 and 10).

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A ,similar effect of the use of copper sulphate is stated by *Lange* [63]. *Leaver* and *Woolf* [64], working on a synthetic gold sample composed of definite, known constituents, obtained a lower recovery than when no copper sulphate was used.

Regulating agents: Gaudin [65] states that, in the flotation of pyritic gold ores, the pulp is best maintained at a p_H near 7 to 8. The natural p_H of the pulp in the investigations carried out on the three batches of ore is between 7 and 8. Sodium carbonate was applied in some tests to see the effect of changing the p_H of the pulp. Sodium carbonate, in this respect, is a better regulating agent than lime, because lime tends to depress free gold, if there is any, in the pulp. Sodium carbonate was mostly used in tests where a depressing agent was applied. Its effect on the recovery where no depressing agents were used is not very appreciable, and the lack of soda ash in tests no. 11 and 13, where p_H is between 7 and 8, does not lead to lower recoveries, though the ratio of concentration is lower—apparently owing to the fact that sodium carbonate, when used, increases the difference in floatability between the valuable mineral and the depressed gangue.

In tests dealing with the white batch very good recoveries and higher ratios of concentration were achieved by applying starch without any alkalinity regulators.

Fairly good results, also, were obtained by floating the original batch of ore *without using any depressing or regulating agent* (test no. 54).

c) Type of machine used: As there was only one type of machine available for these investigations, namely the 500-g Denver laboratory flotation machine, there was no opportunity for comparison between different types. At all events, with this machine quite satisfactory results were obtained. A description of the machine is on page 50—51.

d) Ratio of water to solids: Tests were conducted at a grinding fineness of minus 40, minus 50 and minus 150 mesh, with a percentage of minus 250 mesh which varies from 29 to 95. Such a fineness of grind suggested flotation in a relatively thin pulp. Most of the tests were conducted at 20 per cent solids pulp, and

very good recoveries were obtained. Similar good recovery was achieved at a thicker pulp of 40 per cent solids with the same fineness of grind but with a lower ratio of concentration (compare tests no. 21 and 23). *Gaudin* states that the probability of contact between air bubbles and ore particles increases with higher pulp density. This point was not given enough study in this work, but it is known that, for the separating stage in flotation, the pulp must be sufficiently dilute to permit particle rearrangement to proceed freely. Practically speaking, no pulp thicker than 35 to 40 per cent solids by weight can be used, even with heavy sulphide ores. For every ore there is a critical ratio which yields good flotation results. In this connection we may say that a coarse grind requires less moisture than if the ore is ground finer.

It was noticed that quick conditioning of the pulp with promoters and frothers, without the application of any depressing agent, and at a pulp thicker than that used for the flotation operation itself causes the disappearance of the large carbon-coated bubbles which cover the surface of the pulp, thus improving the character of the froth. This may be due to the fact that the graphite, which is, in this ore, a natural floater, is depressed to a certain extent by very rapid agitation in a thicker pulp. Needless to say, some of the graphite came out with the concentrate when flotation was started, but it was noticed that the first portion of the concentrate that came out was lighter in colour than in cases where slow conditioning at a more dilute pulp was practised.

The most suitable pulp for conditioning with the promoters was variable. While in the white batch it was more favourable to condition at a dilute pulp of 20 per cent solids than in a thicker pulp (compare tests no. 32 and 33), in the greyish batch conditioning was always at the same pulp density at which the flotation was conducted, and conditioning at a thicker pulp when dealing with the original ore yielded more favourable results. This variability may be due to a slight variation in the nature of the ore.

e) Amount of aeration and agitation: Experience gained from the first series of tests on the greyish quartz showed that violent agitation and aeration were essential to a good recovery. Up to

test no. 20 variations in the amount of agitation and aeration were tried by increasing or decreasing the speed of the motor and by controlling the amount of air entering the pulp by means of the valve at the top of the air standpipe. From test no. 20 onwards the motor was always run at maximum speed, which allowed for 2400—2600 r. p. m. Outside air under low pressure was always introduced into the pulp. This action produced a fairly deep froth which contained not only the free mineral, but also the middling product; this was found most essential to produce by flotation alone a low grade tailing that could be discarded to waste. A disadvantage of violent aeration was a lower ratio of concentration.

f) Temperature: All flotation tests were conducted at the temperature naturally assumed by the pulp, frequently in the range of 12°—20° C. It is a known fact that a rise in temperature increases the efficiency of the process and gives cleaner concentrates. In some instances the pulp is heated to accelerate slow action. An example is the heating of pulp for sphalerite activation. *Huber Panu* [66] claims that in floating zinc blende with xanthates the velocity of floating increases with a rise in temperature up to 40° C., but that beyond this temperature it decreases.

In this work no heating of pulp was practised.

g) p_H value: This point was discussed in connection with the regulating agents.

h) Nature of the water used in flotation: Pure scientific research on flotation should always be conducted in a distilled-water pulp. For industrial purposes this is impossible, because in such research, apart from the scientific side, the conditions under which the laboratory investigations are conducted must be comparable to those obtaining in actual practice. All flotation tests in this work were conducted in a tap-water pulp. Some of the tests were conducted in sodium chloride solution. Good recoveries were obtained in both cases. The use of sodium chloride improved the ratio of concentration, yielding a cleaner and richer concentrate (compare tests no. 2 and 3). It is known that salt flocculates fine, suspended

solids, and its action here may of course be related to this property. Another effect of using saline solutions was a shorter time of filtration. The application of salt solutions in some of the tests was, as mentioned before, to investigate the possibility of using sea water in the treatment process in actual practice. The results obtained in these tests may be sufficiently informative.

j) Time of flotation: In actual practice, flotation is not carried out in one cell; it is customary to use 8 to 12 cells in series. In our laboratory investigations one cell only was used, and each floating operation was prolonged for a certain length of time, i. e., the pulp was kept in the same machine throughout the time of flotation. This floating time was varied from 5 to 20 minutes with a view to determining an optimum time for a good recovery. Sometimes flotation was carried out in two steps; e. g., if we wanted to float for 20 minutes, the filtrate from flotation concentrate after 10 minutes was readded to the original pulp and flotation continued for another 10 minutes. In every flotation test there are many variables, such as the proportion of the mineral in the ore, the pulp density, fineness of grinding and the degree of agitation and aeration secured in the machine. Owing to these many variables it was very difficult to achieve substantial uniformity in the floating time required.

In the whitish batch of ore, which is the poorest, 10 minutes was sufficient to get a recovery as high as 98.5 per cent (see test no. 25).

In both the greyish and the original batches, which are relatively richer in gold content, a longer time of flotation was necessary to obtain a satisfactory recovery higher than 90 per cent. In both these batches, floating for 20 minutes was necessary to achieve such a recovery (tests no. 21, 23, 48 and 19). In these tests, flotation in one step gives a higher ratio of concentration than when it is carried out in two steps (compare test 54 with tests no. 21, 23 and 48). From this it would seem that the content of the to-be-floated mineral plays a relatively large part in determining the flotation time needed for a good recovery.

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6. *Abstract and conclusions*

If one appreciates how ore deposits were formed and how vast is the number of variables in each ore, one can easily realise how the laboratory investigation of the flotation of ores is different from the flotation of pure substances.

In floating ores there are so many variables that it would be necessary to conduct a vast number of flotation tests in order to establish an optimum range for each of the factors considered in the operation. The investigations conducted in this work were assisted by carrying out the tests along the lines dictated by experience with ores of similar composition from all over the world, thus avoiding the necessity of making such a vast number of tests as might have exhausted the limited quantity of ore in our possession without achieving the aim of the work.

In addition to gaining some knowledge of the flotation of low-grade pyritic ores, the objective of the work seems to have been attained:

a) It proved possible to obtain good recovery by flotation at a moderate fineness of grind, leaving a very low grade tailing that can be discarded to waste.

b) Excessively fine grinding should be avoided, as lower extraction is made when the ore is over-ground.

c) Xanthates with groups of greater molecular weight, such as pentasol amyl xanthate, are more powerful collectors than those with ethyl groups, such as potassium ethyl xanthate. Amounts of reagents applied vary according to the nature of the ore and the conditions of the test.

d) In cases where primary slimes are produced from grinding, the application of starch depresses these slimes, thus improving the ratio of concentration. The amounts of starch added should be regulated very carefully for every case as starch causes some depression of the gold-bearing sulphides by producing an extremely brittle froth.

f) Violent aeration and agitation are most essential to good recovery by flotation. This action produces a deep froth column

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which carries, in addition to the gold-bearing minerals, most of the middling product. A disadvantage of this action is a low ratio of concentration.

g) Good results are obtained if flotation is conducted in saline solution. Salt is known to flocculate fine, suspended solids, and it thus improves the ratio of concentration.

h) Good recovery is achieved at the natural p_H of the pulp, which is between 7 and 8. There is no need to use an alkalinity regulator.

III. Cyanidation

1. General procedure

The cyanidation tests carried out in this work are made by bottle agitation with the allowance of sufficient aeration. Solutions used vary in strength from 0.1—0.3 per cent potassium cyanide per ton of solution at a ratio of 3 of solution to 1 of ore. The cyanide used is the "Cassel" brand sodium cyanide, which contains 96 to 98 per cent of NaCN and approximately 0.15 per cent of Na_2S. The solution of the salt in water is slightly alkaline. The alkalinity is due to small quantities of sodium hydroxide and sodium carbonate, in addition to which there are small amounts of sodium cyanamide, sodium silicate and sodium cyanate. The alkalinity of the solution is not objectionable; on the contrary it favours the dissolution of gold, as the maintenance of a slight degree of protective alkalinity in the cyanide solutions is desirable in order to conserve cyanide. Experience obtained in testing ores from all over the world indicates that the least trouble will be experienced if the solutions are kept just faintly alkaline or even neutral.

The following is a brief description of the general procedure used for conducting the cyanidation tests:

Grinding. When concentrate regrinding is required, a sample of the fresh, dry concentrate is transferred to a porcelain ball mill. Sufficient barren cyanide solution is added to the mill to give the desired grinding dilution. A pulp containing 65% solids is employed. At the end of the grinding period, the pulp is washed from the mill with cyanide solution and the test is continued.

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Aeration. Aeration in these tests is effected by violently aerating the cyanide solutions before they are used in the tests. This is done by placing the solution in a flask fitted with a tube that opens at the bottom and is connected at the top to a supply of air under low pressure.

Agitation. The ore or concentrate to be cyanided is placed in a wide-mouthed two-litre glass flask. The pulp is made 25% solids with the cyanide solution. The flask is stoppered with a cork through which two glass tubes are inserted to allow entrance of air during agitation. The flask is then placed on a shaker. The agitation effected by the shaker usually gives satisfactory aeration during cyaniding.

Filtration. Filtration is effected by means of the usual Buchner funnel and flask apparatus. The cyanide pulp is poured directly from the agitating bottle into the funnel. When the pregnant solution is completely filtered, a sample is titrated for free cyanide. After the removal of the pregnant solution from the filter flask, the residue in the funnel is given several short washes with water; each wash is allowed to drain by vacuum through the residue.

Sampling. The filtered and washed cyanidation residue is dried and sampled for gold assay in the usual manner previously mentioned. The filter paper used in the Buchner filter is dried with the sample and burned. This product is added to the dried residue in order to avoid the possible removal of fine free gold which might be entrapped in the pores of the paper.

Roasting. If the concentrate is to be roasted prior to cyanidation, an electrically-heated muffle furnace is suitable for this purpose. The furnace must be accurately controlled and should be equipped with a reliable temperature indicator. The sample of concentrate is placed in a clay roasting dish and inserted in the muffle. The important variables to establish are the maximum temperature and optimum time of roasting.

2. Cyanide solutions

During agitation, the cyanide solutions are changed every 24 hours by filtering and addition of fresh solutions; this is done to avoid fouling of solutions. The filtrate is titrated for free cyanide, and thus the consumption of cyanide can be estimated. In the early days of the cyanide process potassium cyanide was used. Nowadays, cyanide is manufactured in the more concentrated form of sodium cyanide. However, it is still customary to work in terms of potassium cyanide, and where this is done it must be remembered that, in order to make a solution containing a given equivalent amount of potassium cyanide, it is only necessary to use three quarters of the weight of "Cassel" brand sodium cyanide. To express this in another way, 3 parts of "Cassel" brand sodium cyanide will be as effective as 4 parts of potassium cyanide, therefore 100 parts of "Cassel" cyanide will be as effective as 130 parts of potassium cyanide—hence the familiar expression "130 per cent KCN".

Example: If it is desired to make up a solution containing 0.1 per cent KCN, for each ton of water it would be necessary to add 2 lb. 4 oz. of potassium cyanide, or $2^1/_4 \times {}^3/_4 = 1$ lb. 11 oz. of "Cassel" cyanide.

Testing cyanide solutions. All the required solutions must be made up with distilled water, as ordinary water contains dissolved salts which may react with some of the chemicals and so falsify the subsequent determinations.

The ordinary method of estimating the amount of free cyanide present in a liquid is by titration with a standard solution of silver nitrate. Silver cyanide is formed, and redissolves in the excess of potassium cyanide until one-half of the latter has been decomposed.

The equations are as follows:

$$AgNO_3 + KCN = AgCN + KNO_3$$
$$AgCN + KCN = KAg(CN)_2$$
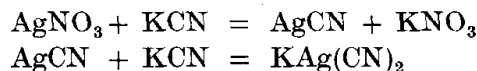
When one-half of the KCN present has been converted into AgCN, an additional drop of $AgNO_3$ solution causes the formation of a permanent white precipitate of AgCN. The amount of silver

solution added is then read off and the percentage of cyanide calculated. The equation of the end reaction is:

$$KAg(CN)_2 + AgNO_3 = 2\ AgCN + KNO_3$$
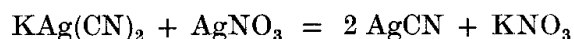
A few drops of 10 per cent solution of potassium iodide are added to make the end reaction sharper and to prevent inaccuracy through the presence of some substances in which silver cyanide is soluble.

The silver nitrate solution is prepared, as suggested by *A. Johnston* [67], by dissolving 13.046 grammes of crystallised silver nitrate in distilled water and making up to one litre. This strength is so chosen that, if 10 c. c. of the cyanide solution is taken for the test, then each cubic centimetre of silver nitrate solution required is equivalent to 0.1 per cent KCN.

The potassium iodide is prepared by dissolving 10 grammes of the salt in 100 c. c. of water, and is then transferred to a dropping bottle.

3. Cyanidation tests

Table 6. Cyanidation tests on the original gold ore and the flotation concentrates

Test no.	1	2	3	4
Sample	ore	ore	ore	ore
Grind: %— 50 mesh	100	100	100	100
%—250 mesh	29.2	66.8	54.6	90.0
Regrind	—	—	—	—
Roast, maximum temp. °C	—	—	—	—
Roast, time min.	—	—	—	—
Assay of ore: Au oz./ton	0.178	0.194	0.0505	0.102
Solution strength used % KCN	0.13	0.13	0.13	0.295
Time of agitation: hours	24	24	24	96
Cyanide consumption: NaCN lbs/ton of ore	1.02	2.52	1.5	14.4
Assay of tailing: Au oz./ton	0.106	0.101	0.0285	0.032
Recovery of gold %	40.5	47.9	43.5	67.8

Table 6 (cont.)

Test no.	5	6	7	8
Sample	conc.	conc.	conc.	conc.
Grind of ore: %— 50 mesh	100	100	100	100
%—250 mesh	52.0	52.0	52.0	52.0
Regrind	—	—	—	dry
Roast, maximum temp. °C	—	700	—	—
Roast, time min.	—	120	—	—
Assay of ore: Au oz./ton	0.9	0.96	0.9	0.905
Solution strength used % KCN	0.32	0.32	0.32	0.3
Time of agitation: hours	24	24	24	24
Cyanide consumption: NaCN lbs/ton of ore	5	1.5	8.6	14
Assay of tailing: Au oz./ton	0.325	0.4	0.295	0.305
Recovery of gold %	63.9	58.3	67.2	66.3

Table 6 (cont.)

Test no.	9	10	11	12
Sample	conc.	conc.	conc.	conc.
Grind of ore: %— 50 mesh	100	100	100	100
%—250 mesh	52	52	61	58
Regrind	dry	dry	dry	—
Roast, maximum temp. °C	—	500	—	—
Roast, time min.	—	90	—	—
Assay of ore: Au oz./ton	0.905	0.96	0.56	1.325
Solution strength used % KCN	0.3	0.3	0.3	0.29
Time of agitation: hours	48	48	48	96
Cyanide consumption: NaCN lbs/ton of ore	12	8	24	14
Assay of tailing: Au oz./ton	0.247	0.205	0.04	0.119
Recovery of gold %	72.7	78.6	92.8	91.1

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Table 6 (cont.)

Test no.	13	14	15	16
Sample	conc.	conc.	conc.	conc.
Grind of ore: %— 50 mesh	100	100	100	100
%—250 mesh	58	51	51	51
Regrind	wet	—	wet	—
Roast, maximum temp. °C	—	500	—	500
Roast, time min.	—	100	—	120
Assay of ore: Au oz./ton	1.325	1.078	0.86	0.92
Solution strength used % KCN	0.29	0.29	0.295	0.3
Time of agitation: hours	48	96	90	72
Cyanide consumption: NaCN lbs/ton of ore	9.6	14.5	62	7.6
Assay of tailing: Au oz./ton	0.107	0.0475	0.237	0.095
Recovery of gold %	91.9	95.6	72.4	90.6

The following table contains miscellaneous data concerning the cyanidation tests in table 6.

Table 7. Complementary remarks concerning the cyanidation tests

Test no.	Remarks
1	Grey quartz from the first batch. At the beginning of agitation some graphite floats on the surface of the pulp, but disappears after a few hours' agitation.
2	Portion of the same batch as in test no. 1.
3	No graphite appears on the surface of the pulp.
4	A portion of the mixed batch which is supposed to represent the ore bulk.
5	All concentrates used in the following tests are from the mixed batch of quartz, i. e. the original ore.
7	Change of solution to avoid fouling. Air bubbled under low pressure through the pulp during the test. No roasting.
8	Concentrate reground dry to 100%—250 mesh. Air bubbled through pulp with change of solution. Bubbling air during agitation causing high cyanide consumption.

Table 7 (cont.)

Test no.	Remarks
9	No air bubbled through pulp. Pregnant solution after filtration is turbid and filtration is slow.
10	Pregnant solution after filtration is quite clear, and filtration is rapid.
11	Concentrate used in this test is from another series of flotation operations. The concentrate is reground dry in a ball mill before cyanidation.
12	Concentrate from another series of flotation operations.
13	Concentrate reground wet in a 65% solids pulp in 3% NaCN using a porcelain ball mill with porcelain balls.
14	Concentrate from another series of flotation operations.
15	Concentrate reground wet in a porcelain ball mill using steel balls and in a 65% solids pulp in 3% NaCN.
16	Same concentrate without regrinding.

All tests: The pregnant solutions, whether from the treatment of the ore or the concentrate, were always alkaline towards litmus. There was no need for any addition of lime to maintain the protective alkalinity in the cyanide solutions.

4. Discussion of the different factors in cyanidation tests

From the results obtained in the cyanidation tests we can easily see how the concentrate is more amenable to cyanidation than the ore from which it came. If we compare test no. 3 with test no. 5 we see how the ore gave a recovery of 43.5 per cent at a fineness of grind of minus 50 mesh, 54.6% minus 250 mesh, while the flotation concentrate resulting from nearly the same fineness of grind yielded an extraction of 63.9% under the same conditions. In this case the cyanide consumption for the concentrate is 3.6 times that for the ore, but at the same time the gold content in the concentrate is 16.4 times that in the ore.

If we increase the time of agitation for both ore and concentrate to 96 hours, the recovery for the concentrate rises to 91.1 per cent in test no. 12 without any special treatment being applied, while

for the ore it reaches only 67.8 per cent in test no. 4 in spite of regrinding finer. In both tests, 4 and 12, a very interesting point is the cyanide consumption, which is the same in both cases (about 14 lbs. NaCN per ton), whereas the gold content in the concentrate is 13 times that in the ore and recovery is much higher.

Graphite: The presence of a certain amount of graphite, 0.14 per cent in the original bulk, was thought to be the cause of unsatisfactory cyanidation of the ore, and trials were therefore made during the early flotation tests to depress as much as possible of this graphite in the pulp by using specific depressing agents such as the 600 series reagents. Later on it was realised that the graphite was not responsible for low recovery in cyaniding the original ore. All the concentrates used in the cyanidation tests originated from flotation operations in which no depressants were used. In such concentrates graphite is undoubtedly found in a more concentrated form than in the original ore, and in spite of this fact very satisfactory recoveries were attained, while no satisfactory recovery could be achieved by cyaniding the ore. It is most probable that this is due to the presence of some cyanicides in the ore, apart from the graphite; such cyanicides do not come out with the concentrate and remain with the flotation tailing. Flotation in this respect served a double purpose, firstly that of concentrating the gold in the ore, and secondly that of separating the harmful cyanicides from the flotation concentrate, such cyanicides being discarded to waste weith the flotation tailing.

Effect of grinding: If we compare tests no. 1 and 2 we can see that finer grinding gave better recovery, with all other conditions being the same. Fine grinding is necessary in such an ore where the sulphide minerals are mostly very finely dispersed through the gangue constituents (see mineralogical examination of the ore p. 45—46), but overgrinding should be avoided in cyanidation treatment as well as in flotation operations. Finer grinding causes higher consumption of cyanide for nearly the same recovery and with the other conditions identical (compare tests no. 7 and 8).

Overgrinding had a bad effect on recovery, as can be seen in test 15, where the concentrate was reground wet in a porcelain ball mill using steel balls.

Time of agitation: There is a certain time of agitation necessary to obtain a satisfactory recovery. This length of time depends on the fineness of grind, assuming all other conditions to be the same. If we compare tests no. 12 and 13, we see that in test no. 12, a period of 96 hours was necessary to achieve a recovery of 91.1 per cent, while in test no. 13, where a portion of the same concentrate was reground wet, a recovery of 91.9 per cent in half this time was attained.

Consumption of cyanide: It has already been mentioned that the consumption of cyanide increases with finer grinding, but the cyanide consumption is also a function of the time of agitation and of the amount of gold recovered. If we compare tests no. 12 and 13 we see that consumption in test no. 13, where regrinding is applied, is less than in test no. 12, where no regrinding is applied, but we must not forget that the time of agitation in test no. 12 is double that of test no. 13.

An interesting point in test no. 15 is the extraordinarily high cyanide consumption, 62 lbs/ton of ore, for a recovery of 72.5 per cent, as compared with 14.5 lbs. for a 95.6 per cent recovery in test no. 14 and 9.6 lbs. for a 91.9 per cent recovery in test no. 13.

Concentrate in test no. 15 was reground wet in a ball mill using steel balls, in test no. 14 no regrinding was effected, and in test no. 13 the concentrate was reground using porcelain balls instead of steel balls. If we compare the facts in these three tests, 13, 14 and 15, we see that in test 14 the time of agitation was 96 hours with no regrinding, in test 13 the time of agitation was 48 hours after regrinding with light porcelain balls, and in test 15 agitation for 90 hours and regrinding with steel balls. The extraordinary cyanide consumption in test 15 must be due to some other factor in addition to the very fine grinding. It is most probable that it is also due to the ill effects of iron from the steel balls. Such adverse effects are noticeable in actual practice.

Roasting: In cases where roasting of the concentrate was applied, it was noticed that there was a maximum temperature for a good recovery. If we compare tests no. 5 and 6 we see that roasting at 700° C. for 120 minutes caused a drop in recovery from 63.9 per cent in test 5 to 58.3 per cent in test 6, using the same concentrate

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under the same other conditions. Apparently, such a temperature caused the fusion of some of the gangue present in the concentrate, thereby causing the gold minerals to become encased in slag. A maximum temperature of 500° C. and time of roast of from 90 to 120 minutes gave a good roast that yielded good recovery, as shown in tests no. 14 and 16. Under similar conditions, a roasted concentrate gave a higher recovery with less cyanide consumption (compare tests no. 9 and 10).

From this, we see that roasting shortens the agitation time and lowers the cyanide consumption.

5. *Abstract and conclusions*

a) The ore is not amenable to good cyanidation, most probably owing to the presence of some unknown cyanicides which do not come out with the flotation concentrate.

b) A fairly good recovery by cyanidation of the flotation concentrate was attained without the application of any special treatment such as washing the concentrate from flotation reagents, roasting and regrinding. In such cases adequate time of agitation (96 hours) is necessary.

c) Regrinding the concentrate shortens the time necessary for cyanidation, but at the same time causes higher cyanide consumption. Overgrinding should be avoided as it causes lower recovery. Regrinding with steel balls has an adverse effect on the cyanide consumption.

d) Roasting the concentrate has a favourable effect on cyanidation. It shortens the time of agitation necessary, in addition to reducing cyanide consumption. Temperature of roast should not exceed 500° C. for 90—120 minutes.

D. Summary

1. A historical survey of gold metallurgy is given, with a brief review of the development of gold recovery since the Ancient Egyptians started the earliest mining and metallurgical operations (4000 B. C.) and including the most modern combinations of processes used at the present time.

2. No two ores in nature are exactly alike. Consequently, each ore must be regarded as a problem to be solved. The gold ore dealt with in this work is from the Barramia district, in the Egyptian Eastern Desert, half way between Edfu (Nile Valley) and the Red Sea. The ore represents a certain type of gold ores, namely the low-grade pyritic gold ores. It contains a certain amount of graphite. It is refractory to straight cyanidation, and it was thought that this was due to the presence of graphite. The object of this work was to conduct laboratory investigations on this ore in the light of modern scientific and metallurgical principles.

3. A mineralogical examination of the ore showed the major constituents of the same to be quartz, dolomite and feldspar. In addition, the ore contains a small amount of sulphide minerals, mostly pyrite, arsenopyrite and pyrrhotite, and a little iron oxide and graphite. No free gold was seen in the ore and the sulphide minerals were found to contain gold, therefore it is considered that most, if not all, of the gold is in association with these sulphide minerals. This mode of occurrence of gold suggested the general line of treatment which was pursued in our investigations.

4. A part of the ore was sorted by hand-picking into the whitish and greyish quartz; a series of flotation tests were conducted

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on each, in addition to tests conducted on the original ore. In each case, satisfactory recovery by flotation proved to be attainable.

5. With regard to the theoretical aspects of flotation, nothing really new was found. Placing such investigations on a very scientific basis is not an easy undertaking when one appreciates how such ore deposits were formed in nature. Facts about the different factors to be considered in flotation operations were confirmed, such as the different types of reagents used, the size of particle amenable to flotation, the pulp density, the amount of aeration and agitation, and the p_H of the pulp.

6. Some studies on the depression of graphite were made with reference to the action of depressing agents, by measuring the contact angle.

7. It was proved that the ore is refractory to straight cyanidation owing to the presence of some other cyanicides, and not to the presence of graphite.

8. Cyanidation of flotation concentrates yielded a good recovery without the application of any special treatment. Investigations were conducted into the influence of regrinding and roasting on the time of agitation and on the cyanide consumption.

9. Finally, it can be said that it was possible to combine the basic features of flotation and cyanidation and thereby to achieve results unattainable by either process alone. Maximum recovery attained by flotation was 98.5 per cent, and by cyanidation of the flotation concentrate 95.6 per cent.

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Ancient Egyptian Illustrations

Fig. 1. Clay Crucible found at El Argar (page 13). British Museum.

Fig. 2. An Egyptian Metallurgist (page 13). From *Garland* and *Bannister*, "Ancient Egyptian Metallurgy" (Charles Griffen & Co.).

Fig. 3. Egyptian Goldsmiths' Workshop in the Pyramid Age (page 15). From *Partington, J. R.*, "Everyday Chemistry" (Macmillan & Co., Ltd.).

Fig. 4. Metal Workers' Workshop in Egypt (page 15). From *Breasted, J. H.*, "A History of Egypt", 2nd edition (Charles Scribner's Sons, New York; Hodder & Stoughton, Ldt., London).

Curriculum vitae

I was born on May 11th 1912 in Cairo, where I completed my primary education in 1923 and my secondary education in 1928. I joined the Faculty of Science of Fouad 1st University and graduated in 1935 with the degree of B. Sc. In the same year I obtained an appointment as chemist at the Suez Government Petroleum Refinery, after which I was transferred to the Mineralogical Research Laboratory, Department of Mines and Quarries. In 1936 I was moved to the Government Sukkary Gold Mines, where I worked for ten years, at the end of which, in 1945, I was chief chemist and mill superintendent. I was then selected as a lecturer at the Institute of Chemical Technology, Farouk 1st University. In October 1949 I joined the Swiss Federal Institute of Technology on a mission from the Egyptian Government to study for a Doctorate in Inorganic Technology under the supervision of Prof. Dr. A. Guyer.

Zurich, July 1951.