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Cyanide

NOTES ON GOLD EXTRACTION

BY MEANS OF

Cyanide of Potassium (MacArthur-Forrest Patents),

AS CARRIED OUT ON THE

WITWATERSRAND GOLD FIELDS,

TRANSVAAL, SOUTH AFRICA.

Written by W. R. Feldtmann, Managing Chemist to the African Gold Recovery Company, Limited.

Illustrations by Ernest W. Hutt.

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NOTES ON GOLD EXTRACTION

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CYANIDE OF POTASSIUM (MACARTHUR-FORREST PATENTS),

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From the monthly statistical returns of the Witwatersrand Chamber of Mines, and from other sources, most people interested in the progress of these Fields have obtained at least a very good notion of the importance to the mining industry of the part played by the Cyanide process of gold extraction.

The following table shows the increase in returns from Milling and from Cyanide treatment respectively, for the last five years ;—

Output of Gold in the Witwatersrand District, by Mills and Cyanide.

	1889.	1890.		1891.		1892.		1893.		1894.	
	Mills.	Mills.	Cyanide.	Mills.	Cyanide.	Mills.	Cyanide.	Mills.	Cyanide.	Mills.	Cyanide.
January	...	35006	...	52595	610	72589	11971	91236	17138	106263	43551
February	22457	36887	...	48532	1547	75752	10897	76889	16363	107829	44041
March	27919	37779	...	51348	1601	81771	11473	91471	20002	115882	49490
April	27028	38696	...	54726	1645	82062	13500	91966	20086	...	...
May	35028	38836	...	53612	1061	86217	13219	92906	24004	...	...
June	30877	37419	...	54263	1600	87752	15500	96708	26199	...	...
July	31091	39226	230	52750	2174	85084	10195	98278	27891	...	...
August	30519	42807	56	55524	3546	85928	16392	101773	34296	...	...
September... ...	34143	45486	...	61993	3609	91148	16702	96122	33463	...	...
October	32214	45248	...	69028	3765	94836	17771	101726	34956	...	...
November... ...	33721	46782	...	68020	5373	89098	17695	101825	36815	...	...
December... ...	39050	50351	...	71981	8331	73184	17973	107060	39297	...	...
Total... ...	344047	494523	286	694372	34862	1005421	178688	1147960	330510		

From the time of the introduction of the process in 1890, previous to which an occasional heavy thunder shower, which carried a few thousand tons of tailings down the creeks, was looked upon as a perfect godsend, to the present, when deposits of 3 dwt. tailings are either being worked or else hugged as a valuable asset, the increase in output from cyanide plants has been marvellously rapid and steady.

As is shown by the above table, "Cyanide gold" for the month of March, 1894, reached the very respectable figure of 49,490 ounces, or 29·9 per cent. of the total Witwatersrand return for the month, the proportion due to cyanide of the total output for 1893 being about 22 per cent. The value of the "cyanide" gold can safely be put down at an average of £3 per ounce. That the steady increase in "cyanide" output has been accompanied by, and is even in some measure due to, the many improvements of more or less value which have been effected, and the practical experience which has been gained in the working of the process, goes without saying. This although no improvements effected have been of such vital importance that the process as first introduced to the public by Mr. John S. MacArthur and Dr. Forrest can be said to have been in the slightest degree altered in principle.

This short pamphlet is written with a view to endeavouring to describe succinctly the process as it is worked on the Rand Gold Fields, with the appliances and by methods suggested by local experience. At the same time the writer will endeavour, where possible, to give such hints and suggestions as may be useful to laymen and practical metallurgists interested in the process in this and other parts of the world.

The subject will be dealt with under Sections:—

1. Appliances.
2. Practical conduct of the process.
3. General notes.
4. Notes on the chemistry of the process.

I.—APPLIANCES.

The numerous (about 50) plants, of an aggregate treating capacity of some 230,000 to 240,000 tons per month, at present (April, 1894) in use for the treatment of tailings and concentrates on these Fields are of great variety, both in size and design, the general principle in all of them being, of course, the same. The objects which they have to fulfil consist briefly in:—

1. Lixiviation of the ore, tailings, or concentrates with weak solution of cyanide of potassium for the purpose of dissolving the gold.
2. Treatment of the solution to precipitate the dissolved gold in the metallic form.

In all plants we have the three main features:—

1. Leaching or filter vats.
2. Launders, for precipitation, *i.e.*, "zinc boxes."
3. Storage vats for solutions.

The important differences between one plant and another may be said to consist in the material, size, and shape of the vats, the methods for charging and discharging the tailings, and the relative positions to each other of leaching vats, storage vats, and zinc boxes.

As to vat material, the choice lies between brick and cement, concrete, and timber. For a permanent plant, or one which will have several years to run, either brick or concrete is probably preferable to timber; although, as far as is possible to judge, contact with cyanide solutions does not appear to shorten the life of timber.

As a brick or concrete tank may be considered the water-tight lining to a hole in the ground, which is the actual vat, the first essential in deciding to use one of these materials is to find suitable ground to be excavated for a site. Any ground which is moderately firm and free from springs may be considered suitable. In case, as is usual, such brick or concrete tanks be constructed with their tops just level with the ground, the question of facilities for discharging is an important one. At the Langlaagte Estate and Block B Companies' plants, where each circular brick vat is of a capacity of about 400 tons, *i.e.*, about 40 feet diameter and 10 feet deep, the discharging is very effectively and economically carried out by means of travelling cranes, which lower the bodies of empty trucks into the vats, there to be filled by Kaffirs; and, bringing them out full, place them on their carriages, to be wheeled away to the dump. A sketch, showing the idea, is given on Plate I, Fig. 1. At the Crown Reef Company's new plant the square filter vats are constructed of brick and cement, and measure 36 feet x 40 feet x 10 feet, and they are provided with doors through which the discharging trucks can be run into the vats. (Plate I, Fig. 2).

In most cases, however, and especially for smaller plants than the above-named, timber seems to be in favour, more particularly for the filter vats.

The sizes and shapes of filter vats have been many and various. The present tendency is to restrict the number of vats, but make them of a size limited only by the nature of the material available, and in consequence of this large size the shape selected is the circular, as being strongest. Wooden filter vats have been constructed on these Fields as large as 42 feet diameter and 14 feet deep, at the Simmer and Jack Company, and 40 feet diameter x 7 feet deep at the Durban Roodepoort. It is evident that there is economy in constructing a few large vats as against a larger number of smaller ones of the same aggregate capacity. It is claimed, moreover, for deep filter vats that they give a better extraction, and not without a certain show of reason, as will be explained under Section II.

The larger the filter vats the more difficulty does the discharging present, unless special appliances be adopted. When vats were regularly constructed of a maximum diameter of perhaps 20 feet, with a depth of 5 feet, it was no great matter to throw the sand over the side into trucks. This more particularly if the vats were square or oblong in shape. The adoption of special discharge facilities, such as bottom or side discharge for wooden vats or the travelling crane for buried brick vats, obviates the main drawback to large tanks. Sketches are given on Plates II and III of bottom discharge, Plate II, Fig 2, being sketch of one patented by Mr. Charles Butters, and Plate III Fig. 1, of one designed by Mr. W. F. Irvine. The side discharge door for wooden vats, Plate II, Fig. 1, is a modification of a design made by the writer. Another method of side discharge, designed by Mr. W. F. Irvine, is in use at the Crown Reef Company's fine cyanide works, where the large brick and cement vats are provided with doors which allow of the ingress of discharging trucks. The door frames are bolted to the cement walls, and the plate iron doors are drawn tight against these by means of an ingenious arrangement of sliding lugs, bolts, and nuts. At the Barrett Company's plant, near Barberton, a system is in use of discharging through a bottom discharge door into a launder, whence a copious stream of water carries the residues into the creek below.

Details of the false bottoms of the leaching vats are shown in Plate IV. The false bottom, or filter, consists of a wooden grating covered with two layers of jute, or one layer of jute and one of cocoanut matting The method of connecting drain pipes to wooden vat bottoms is shown on the same Plate. It is essential to have the drain-pipes from leaching vats so arranged that solutions of different strengths may be drained from two separate vats simultaneously, without mixing the "weak" and the "strong."

The construction of zinc boxes is shown on Plate V. The zinc box consists practically of a wooden launder, fitted with baffle-boards, which divide the launder in such a way that solutions are forced to flow upward through the zinc shavings with which the large compartments are filled. The top division is advantageously used as a settling tank to collect any sand which may have come through the filter, or a separate settling tank may be used, particularly in plants where solutions are pumped direct from filter vats. The zinc box compartments are fitted with removable trays, consisting of wooden frames supporting wire gauze of about $\frac{1}{8}$-inch mesh. This, while it carries the zinc, allows most of the fine gold precipitate to fall through into the bottom of the box. The filter (launder) shown on the side of the zinc box need not necessarily be attached to same, but may be connected by a small wooden V launder, with the plug holes in the zinc box compartments. The zinc box sketched is large enough for each 1,000 to 1,500 tons of monthly plant capacity. But although it is generally found best to have at least two zinc boxes to a plant, it is, of course, not necessary to increase the number of them in proportion to the tonnage to be treated. The increase in precipitating capacity can just as well be attained by increasing the size, *i.e.*, the width of the boxes.

As illustrating differences in the general design of plants with regard to the relative position of the different parts, three outline sketches are given on Plates VI and VII. In No. 1 design, Plate VI, we have the leaching vats placed highest. The solution gravitates from these through the zinc boxes into storage vats, there to be made up to strength ready for pumping up to leaching vats again. The discharging of tanks is assumed to be done over the side in the sketch.

In No. 2 design the solution is either pumped direct from the leaching vat; or, running into a small sump, or an air-tight receiver (as shown on Plate VII, Fig. 1), is pumped from there into zinc boxes, and runs thence into overhead storage vats. Having been made up to strength, it is ready to run direct into the leaching vats again. The discharge system indicated is the "bottom discharge."

No. 3 design, Plate VII, Fig. 2, is a combination of the two previous ones, and is advantageously fitted with a pipe service to enable one, if desired, to run solution up through the sand in the leaching vats. As shown in the sketch, the plant is designed for side discharge; but, of course, any system of discharge may be applied to any of the three arrangements of plant. The relative advantages of these three arrangements depend partly on the nature of the site procurable, and largely on the fancy of the constructive engineer.

II.—PRACTICAL CONDUCT OF THE PROCESS.

In considering the treatment required for any given crushed ore or tailings (which latter will be comprised under the term " ore ") the following points have to be observed :—

1. Is the material, or rather the gold it contains, coarse or fine ?
2. Is the material acid or neutral ?
3. What metals with a strong affinity for cyanogen does the material contain ?

The degree of coarseness of the gold determines in a large measure the length of time required in contact with cyanide solution to effect a dissolution of the precious metal. On the Rand, generally speaking, the gold is exceedingly fine, and a period of twelve hours standing under " strong " solution, is, with tailings, usually sufficient to dissolve as much of the gold as is likely to be extracted to commercial advantage. Further treatment with " weak " solutions is chiefly in the nature of washing to remove the already dissolved gold. In the treatment of pyritic concentrates a very much longer contact with the solvent is required. This, in addition to there being in the first place more gold to dissolve, is partly on account of the gold contained in the concentrates being coarser than in the lighter tailings ; partly also, because of some of the gold being present in amalgam, which is acted on slowly ; chiefly, however, because the cyanide solution has to penetrate between the laminæ of the pyrites crystals, thence to extract the gold. As the almost infinitesimal quantity of solution which can penetrate such interstices will be strongly held by virtue of capillarity, diffusion will naturally be slow and extraction consequently retarded.

By " acidity " in the ore (which has become a familiar phrase among cyanide men) is understood the presence of the products of the partial decomposition of pyrites. These products consist chiefly of free sulphuric acid, soluble metallic salts, such as proto sulphate or per-sulphate of iron, or insoluble basic iron salts. All these substances are destructive to cyanide, forming with it compounds useless in the extraction of gold, as will be explained under Section IV. Special measures are required to obviate the otherwise ill effects of the " acidity."

Of the various concomitant metals occasionally found in gold ores, certain copper compounds, such as copper glance, are the most likely to cause trouble unless a point emphasized by the inventors of the cyanide process, *i.e.* the peculiarly selective action of *weak* solutions, be borne in mind. For instance, if we treat an ore containing much copper glance with a strong (say, 2 per cent. to 3 per cent.) solution of cyanide, the potassic cyanide may be all decomposed ; but if we apply a much weaker solution, as, *e.g.*, a 0·5 per cent. one, although the cyanide will be also decomposed the same extraction of gold will be effected. So that, though in the former case the treatment may appear commercially impracticable, the proper carrying out of the process by leaching with weak solutions will prove it to be both effective and economcal. The presence of copper in the ores may also affect, to some extent, the second operation in the process, *i.e.*, the precipitation of the gold, as will be explained under Section IV.

Some simple tests are indicated under Section IV for determining the preliminary treatment, the strength of solution, and the time required to obtain the best extractions.

To give a practical illustration of methods of treatment we will suppose two different classes of ore :—

1. A simple quartz ore free from pyrites or " acid " and giving a rapid extraction,—such as the free-milling ores on the Rand ;
2. An " acid " ore.

1. The usual course of treatment of the Rand " free-milling " tailings is as follows :—The vats are filled within an inch or two of the top with the tailings. Sfficient solution of cyanide of potassium to thoroughly saturate and cover them is run on. This generally means about one third of the dry weight of the ore. The first solution thus applied (the " strong " solution) may

be of 0·3 per cent. strength. It should always be borne in mind that the weaker the solution employed the more the particularly selective action of the chemical comes into play. In other words, as long as we use a solution just strong enough to dissolve the gold in a reasonable time any additional strength is a waste, in that compounds of iron and other substances, or even the atmosphere, will destroy a proportion of the chemical in a strong solution, when they would not act on weak solutions. After standing say for 12 hours, during which time small quantities of the solution may be drawn off occasionally to effect artificial diffusion, all this solution is drained through the zinc boxes until the tailings are dry. A weaker solution—say 0·15 per cent —and about one half in quantity of the first solution is then run on, and may be started draining within an hour or two. A further weak wash, or, if the quantity of stock solution will allow, a water wash, is then applied and drained off. After the tank is thoroughly drained dry it is ready to discharge The solutions that have drained through the zinc boxes into the storage vats, or have been pumped to the upper reservoirs, are ready for making up to strength for further use on next charge, practically all the gold having been extracted from them in their passage through the zinc shavings. This treatment applies to a class of ore which may be considered as rather an exceptionally simple one, of a kind which, on the Rand, is getting rarer as old accumulations of free-milling tailings are getting worked up and more deep level ore is being crushed. An average extraction of 80 per cent. out of 6 dwts. is nothing uncommon in dealing with this class of ore on the Rand.

2. The treatment of " acid ore," or rather tailings rendered " acid " by the partial oxidation of the crushed pyritous ore, offers more difficulty than in the case of the " free-milling". In order to overcome the cyanide-destroying qualities of the acid or iron salts present, we are obliged to have recourse to neutralization by means of an alkali or alkaline earth—caustic soda or lime—with or without a preliminary water washing to remove such soluble "cyanicides" as may be present. If the quantity of cyanicide present is large, and, by testing, a considerable proportion of it is found to be soluble in water, a preliminary water wash is generally applied. It has hitherto frequently been the custom to water wash in the same tank in which the subsequent treatment with cyanide is effected, the water wash being run to waste. In the writer's opinion this is a practice which cannot be too strongly condemned, as accounting largely for the so-called " mysterious " discrepancy between the expected yield of gold as estimated from assays made before and after the treatment, and the actual return. When a water wash charged with acid out of the ore comes in contact with residual quantities of cyanide solution lying in the bottom and adhering to the sides of the tank a certain quantity of hydrocyanic acid gas is liberated (see Section IV), which, diffusing through the whole tank, is capable of dissolving a not inconsiderable amount of gold from the ore. The worst feature of so dissolving gold is that it is not precipitated, even if passed through zinc, and is consequently run to waste with the water wash. If, in order to economise caustic soda and lime, water washing be adopted, it should be done in tanks which are reserved for the special purpose, and the ore should then be transferred to the cyanide treatment vats This means a little extra cost for handling, but will, in most cases, amount to less than the loss of gold incurred by the other method. In order to neutralise the remaining acid, a quantity of a solution of caustic soda, of which the amount necessary must be approximately determined by experimental test (see Section IV) is run on, allowed to stand for an hour or so, and then drained off into an " alkali " sump, there to be made up to strength for the next lot. Where practicable it is a good plan to mix powdered lime with the ore, as this not only saves caustic soda, but also keeps the cyanide solutions freer from suspended ferric hydrate, which, in the case of using caustic soda alone, makes them very turbid, and fouls the zinc.

The preliminary treatment having been finished, cyanide solutions are applied. The strength of the first solution may vary from 0·25 per cent. to 0·5 per cent The number and strength of subsequent solutions and washes, and the time of contact required depend entirely on the quality and nature of the ore. It may be noted that in the case of ores requiring a long treatment it is preferable, rather than allowing the charge to stand under one solution for two or three days, to draw off the " strong " solution completely, and apply a fresh lot every 24 hours.

In the case of concentrates requiring two or three weeks, or longer, under " strong " solution, it is found beneficial to draw off every two or three days, and even to turn the ore over once or twice in the course of treatment.

The average extraction from acid ores on the Rand, say from 5 or 6 dwt. stuff, may be put down at 70 per cent. Witwatersrand concentrates of 3 oz. value will give an extraction of from 90 up to 98 per cent. in three or four weeks' treatment.

With regard to the treatment in the presence of various metals with a strong affinity for cyanogen, as already pointed out, the principle to go on is to apply the weakest practicable solutions, bearing in mind that, although there is, for example, sufficient copper mineral present to decompose a strong solution of potassic cyanide, thereby making the cost of the process appear out of reason, it does not follow that a weak solution will not quite as effectually and more economically extract the gold.

The writer had an opportunity of noting the treatment of cupriferous ores at the Transvaal Company's plant, in the Lydenburg district, and it is an interesting fact, as illustrating the selective affinity which weak cyanogen compounds possess for gold, that although there was sufficient copper mineral present to decompose a solution of potassic cyanide of over 1 per cent. strength, still good results, *i.e* 70 per cent. extractions on 18 dwt. ore, were obtained with 0·5 per cent. solutions. In this instance there was reason to believe that the gold was dissolved as auric-cyanide instead of as auro-potassic-cyanide, as usually formed.

Having obtained the gold in solution, the precipitation in the zinc boxes is, as a rule, a simple matter. The only points requiring attention are to keep a sufficient stock of zinc shavings in the compartments, and so to regulate the flow of solutions as not to incur danger of fine gold precipitate being carried away. The zinc shavings are prepared by turning down zinc discs on a lathe. The discs for this purpose may be cut out of No. 15 gauge metal, and may measure from 6 to 12 inches in diameter, a hole being punched in the centre for the mandril. It is usual to put bundles of 20 such discs on a mandril. The speed of the lathe may be anywhere from 150 to 350 revolutions per minute, and the shavings are turned off by hand with an ordinary carpenter's mortice chisel.

Precipitation of the gold varies somewhat with different classes of ore treated. The completeness of the precipitation appears to depend in a measure on a slight excess of cyanide of potassium being present in the solutions. Roughly speaking, it may be said that if solutions leaving the zinc boxes assay more than 2 dwts. per ton, the precipitation is not as it should be. This may be owing to the paucity of zinc in the boxes (which should be instantly rectified), to too great a speed in the flow of the solution, or, in very exceptional cases, to insufficient cyanide in the solutions. In the case of the Lydenburg cupriferous ores above cited, this last was found to be the trouble, and was put right by making the solutions up to working strength *before* passing them through the zinc boxes. The result of so doing was that solutions which had before assayed several ounces were reduced to a few dwts. per ton. (See Section IV.) The zinc shavings in the boxes may require replenishing every day to replace amount consumed, or may run a week at a time without requiring dressing.

Having, by passage of the solutions through the zinc shavings, re-converted the gold into the metallic form, as a sludgy-looking precipitate commonly known as "slimes," the next step in the process is to get this precipitate into marketable shape. This is done by separating from the zinc, drying, roasting, and smelting. The "cleaning-up," which takes place once or twice a month, is conducted as follows :—

A sufficient amount of clean water is run through the boxes to remove the cyanide solution, which might otherwise be injurious to the workmen. The zinc shavings are taken out, being twisted and rubbed in the water to remove, as much as possible, all gold adhering to them. In some cases there is quite a thick gold plating on the zinc, which cannot well be removed by scrubbing, but as all the shavings are returned to the boxes anyway, and this plating of one month will go to form the precipitate of the next, it may be ignored. Having removed nearly all the coarse zinc, the precipitate contained in the water is allowed to settle—the addition of a little alum will considerably expedite the settling. The bulk of the clear water is then syphoned or pumped off, and the precipitate, most of which is under the trays, together with the remaining small quantity of water, is drawn off through the plug-holes into a calico or linen filter, or into a filter press. After drying sufficiently to handle with a scoop the precipitate may be further dried in iron pots, and is then ready for roasting and smelting, The object of the roasting is so to oxidise the greater portion of the zinc, which has, in the form of small chips and shavings, fallen through the zinc box trays, as to cause it to combine in the subsequent smelting with the fluxes, and leave the bullion fairly fine. Oxidation by the aid of the atmospheric air is sufficient, but a certain amount of the zinc oxide subsequently becomes reduced by the carbon of the plumbago melting pots and re-enters the bullion. A good method of roasting has been found to be the addition of a little nitre, say about 3 to 10 per cent., to the precipitate. It is best applied as a strong solution before drying the precipitate, so that it gets equally mixed with the whole mass. In the subsequent roasting the nitre not only assists by yielding up oxygen to the zinc, but to some extent also appears to flux the zinc oxide, forming zincate of potash, which is not so readily reduced as zinc oxide. In case the precipitate is very sandy—owing to tailings coming through the filters—nitre roasting is not so successful, as it tends to cake. By the addition of nitre the tendency of the precipitate to dust on stirring up in the roasting furnace is minimised, the amount of flux required in smelting is reduced, and the resulting bullion is better. In roasting the precipitate, care should be taken not to raise the temperature much above a dull red heat (to avoid partially fusing it to a pasty mass), and not to stir too violently, especially just at the commencement of the roast, or dusting and consequent loss is the result. The furnace sketched on Plate VIII, Fig. 1, will conveniently take 50 lbs. of precipitate at a time.

The precipitate having been sufficiently roasted, the next step is to mix it with suitable quantities of flux, and smelt in plumbago pots. The fluxes commonly used are bicarbonate of soda, borax, and sand. In the case of sandy precipitate, of course, further addition of sand is omitted, and it may be found advisable to add a small quantity of fluor spar. The proportions of fluxes and precipitate vary within very wide limits. For general guidance it may be stated that where much sand is present—which would give a glassy, but thick-flowing slag – the best corrective is more soda, with the addition, if necessary, of a very little powdered fluor spar. For a too basic slag (*i.e.*, a dull, lustreless one), which is too stiff, more borax will generally do good.

Examples of various fusing mixtures are given below, but it should be well understood that any one of the fluxes may have to be increased or decreased according to the amount of impurities present :—

	1. Clean ppte.	2. Very zincy.	3. Very sandy.
Precipitate - -	30 lbs.	30 lbs.	30 lbs.
Bicarbonate - -	15 ,,	15 ,,	20 ,,
Borax - -	8 ,,	12 ,,	10 ,,
Sand - -	5 ,,	5 ,,	—
Fluor Spar - -	—	—	2 lbs.

Precipitate and fluxes are well mixed and charged into the plumbago crucibles. The smelting furnaces, which may be constructed to take two or three pots at a time, should be good ones, as the heat required for this first fusion is rather in excess of the ordinary gold melting temperature. After the charges in the pots are run down, more of the mixture may be added from time to time—the whole of a charge, as given above, will go into two No. 35 crucibles—and everything being fused until perfectly liquid, the contents of the pots are poured into moulds. Conical shaped moulds are the best suited for this work. The metal settles to the bottom, and, after cooling, may be turned out and freed from the slag by breaking off the latter with a hammer. The several pieces of bullion thus obtained at one clean-up, are subsequently re-melted with borax and run together into one ingot. This re-melting should be done at as low a temperature as possible, so that the metal may solidify almost as soon as it is in the mould, otherwise liquation results, and it becomes exceedingly difficult to obtain anything like a representative sample of the bullion for assay. The slags, which generally contain a considerable amount of gold in beads, are crushed up and panned, or cradled, to obtain the metal.

III.—GENERAL NOTES.

One of the great bugbears of the Rand cyanide men has been the question of the treatment of "slimes," by which is meant the very fine, or, in the case of free milling ores, the clayey portion of the tailings. Various suggestions have been made for the treatment of these slimes, but so far the only really practicable scheme appears to be to allow them to dry thoroughly, and by screening or some mechanical method of disintegration, to reduce them to a fine powder. This powder is thoroughly mixed with sandy tailings in the proportion of 1:3 or even 1:2, and the mixture will, especially if aided by a suction pump, percolate fairly well, and give satisfactory extractions. There is no chemical difficulty in the way of obtaining good extractions from "slimes" The trouble is, that, if they go into the vats in half-dried lumps, they will absorb solution and not yield it up again. It has frequently come under the writer's notice that such lumps of slimes among tailings have left the vats rather richer in gold than they went in, owing to their having absorbed solutions charged with gold. Some of the Rand Companies are making a point of leaving the slimes altogether on one side in the treatment of tailings, and are even going to some trouble to separate them from the battery pulp as it leaves the mill. On the Rand almost any grade from 600 mesh up to any degree of fineness short of slimes seems to give good extractions by cyanide.

The solution of the slimes difficulty probably lies in the direction of dry crushing (by means of Krom rolls or other suitable machinery) and direct treatment of the powdered ore with cyanide. As long as the ore is not run away in a state of pulp the slimes cannot separate from the bulk of the sand, and as long as the sand and slimes are intimately mixed there is nothing to hinder leaching.

By way of economising labour, the experiment is being tried of running tailings direct into the cyanide leaching vats after separating the slimes by means of spitz-kasten or other contrivances. This plan is probably open to the objection that tailings run in direct have a tendency to pack so close in the vats that it becomes impossible to obtain a thorough and effective contact with cyanide solution.

A good deal has been said about an alleged " mysterious discrepancy " between calculated extractions, according to assays before and after treatment, and actual returns obtained. This discrepancy has variously been put down to absorption by the vats or the filter, loss in smelting, or to some occult phenomenon, to account for which a chemical law has yet to be discovered. In nine cases out of ten where this discrepancy occurs it is easily traceable either to incorrect measuring or weighing of the ore treated, to preliminary water washing of acid ores in the cyanide treatment vats, or to incorrect sampling of charges and residues. This last, particularly as regards the residues, is probably the commonest cause of error. It should be borne in mind that the portion of the residues in which the largest proportion of unextracted gold is contained is probably the lowest foot or so in the filter vat. By a very usual method of residue sampling, *i.e.*, by means of a sampling rod (an instrument resembling a magnified cheese scoop) one is very liable to obtain an undue proportion of the upper part of the residues, and consequently to show, by assay, a better extraction than has actually taken place A safer way to sample is to take a small sampling rod full of sand from each of the discharging trucks as it leaves the building, unite the samples so collected, and quarter them down. It is believed that the depth of the comparatively rich layer of residues at the bottom of the vats is not proportional to their depth, but is more or less constant for vats of any depth. If this be so, it would explain why better extractions may be obtained from deep vats than shallow ones, all other conditions being equal, as in a deep vat these few inches of rich residues are averaged over a larger tonnage than in a shallow one.

An important discussion is being waged on these Fields, at present, regarding the advisability, or otherwise, of removing the pyrites from tailings by concentration before treating the latter by cyanide. Some maintain that it must be more economical to treat tailings by cyanide as they leave the plates, without passing them over any form of concentrator. Others, that it is best first to collect the pyrites and subject them to separate treatment either by cyanide or chlorination. At the first blush, the direct method appears the more rational, as doing in one operation what has otherwise to be done in two or more processes. The question is, whether the gold contained in the pyrites can, in anything approaching so short a treatment as is ordinarily applied to tailings, be extracted satisfactorily. On this question, indeed, the whole problem hinges of preliminary concentration or no concentration. For each particular case the matter is probably subject for experiment. It should be mentioned, however, that gold contained in pyrites is occasionally extracted very much more quickly when the pyrites is in contact with sand than when in the form of clean concentrates. A simple way of testing whether, in a particular instance, concentration previous to cyaniding will prove advantageous, would be to treat a parcel of the tailings by cyanide and then concentrate the residues. If the concentrates from residues contain more than sufficient gold—or rather, if the *realizable* value of the gold contained in them is more than sufficient—to cover the cost of concentrating, it is obvious that preliminary concentration will in such a case be of commercial advantage.

Regarding the cost of treating tailings by the cyanide process: This is necessarily regulated by local conditions, the nature of the ore treated, and the special facilities for handling the ore. The biggest item of cost is generally for the cyanide of potassium, which probably averages on the Rand about 2s 0d. per ton of ore treated. A good deal of economy can be effected by a careful chemist in charge of a plant. He must note the nature of the ore, and know when it is necessary to use lime or caustic soda to neutralize any acid present. Further, by keeping solutions of different strengths separate, he must so regulate matters that the last "weak wash" applied to the ore (in the absence of final water washing) really is a *weak* wash. Otherwise, of course, a certain amount of cyanide is thrown out of doors with the residues.

Economy in the handling of the ore has to be provided for at the time of the erection of the plant, by a judicious selection of site—especially providing for a good dump for residues,—and efficient facilities for filling and discharging tanks. According as those facilities are favourable or otherwise, handling of the ore may vary anywhere from 9d. to 2s. per ton. Wages for white men are comparatively a small item per ton in the various cyanide works on the Rand.

It is important to have, at anyrate, one man in a works who, whether he has to do the works assays or not, possesses at least a rudimentary knowledge of chemistry For a small works treating say 2,000 tons per month, and given a convenient plant to work, a chemist and one shift man will generally be found sufficient to do all the solution work. In addition to these, at least one kafir ganger is generally employed.

It is obvious that the expense per ton will come out lower in a large plant than a small one. The average total cost of treatment on the Rand is somewhere about 4s. 0d. to 4s. 6d. per ton in large works, treating say upwards of 10,000 tons per month, whereas in small plantsit may be put down at an average of 6s. per ton.

Experiments have been and are being made in the way of obtaining a substitute for zinc precipitation. So far, no great measure of success appears to have attended these efforts, as all other proposed methods seem to be either more expensive or less effective—or both. The objections to the zinc precipitation have been stated to be the troublesome work of cleaning up and smelting the precipitate, and the cost. As a matter of fact, the clean-up is not much more troublesome, if intelligently gone about, than a mill clean-up, and the question of cost of zinc precipitation which amounts to 1½d. to 3d. per ton of ore treated, will be hard to improve on by any other method.

IV.—NOTES ON THE CHEMISTRY OF THE PROCESS.

It would be beyond the scope of this pamphlet to include an elaborate treatise on the chemistry of cyanogen compounds, or of the cyanide process. The writer's intention is merely to touch on such points as may be considered of practical interest and importance to the cyanide chemist.

The theory of the dissolution of gold by cyanide of potassium, as propounded by Elsner, is the generally accepted one. It is represented by the following equation :—

$2Au$	$+$	$4KCy$	$+$	O	$+$	H_2O	$=$	$2AuKCy_2$	$+$	$2KOH.$
Gold,		Cyanide of potassium,		Oxygen,		Water,		Auro-potassic cyanide,		Potassic hydrate.

In view, probably, of the fact that according to the above formula an atom of oxygen is required in the reaction, one or two patents have been taken out having for their object the addition of an oxidizing substance to the solutions or the ore in the filter vats. That an artificial addition of oxygen in the form of some oxygen-yielding compound is uncalled for, is evident from the results obtained without such addition. It is probable that for any ore that has as yet been encountered on the Rand, or is likely to be encountered, the oxygen present as atmospheric air dissolved by the solutions has been and will be sufficient to fulfil the requirements of the Elsner equation, accepting the same as the only possible one. Furthermore, a large amount of oxygen is occluded by the pulverised ore or tailings.

The reaction which takes place when hydrocyanic acid dissolves the gold from acid tailings (assuming that the cyanicide has destroyed all the cyanide of potassium) may be represented by the following equation :—

$$2Au + 8HCy + 3O = 2AuHCy_4 + 3H_2O.$$

Gold, Hydrocyanic acid, Oxygen, Auricyanic acid (auric hydrocyanide,) Water

It should be noted that the gold in this compound is not, or is at any rate only very imperfectly, precipitated by zinc. Even rendering the solution alkaline by addition of caustic soda or potash does not appear in such a case to cause a good precipitation. This may be owing simply to the auric compound being more stable than the aurous salt ordinarily obtained, or may be owing to the absence of free cyanide of potassium. The addition of alkali to such a solution of auricyanic acid may be assumed to form auric potassic cyanide in this way :—

$$AuHCy_4 + KOH = AuKCy_4 + H_2O.$$

Auricyanic acid, Potassic hydrate, Auric Potassic cyanide, Water.

The addition of an acid to working solutions containing free potassic cyanide and a certain amount of auro-potassic cyanide would appear—probably indirectly—to convert at least a portion of the latter into auricyanic acid. Probably the hydrocyanic acid liberated by the decomposition of the potassic cyanide combines with the auro-potassic cyanide to form auri-potassic cyanide :—

$$AuKCy_2 + 2HCy, + O = AuKCy_4 + H_2O$$

Auro potassic cyanide, Hydrocyanic acid, Oxygen, Auri potassic cyanide, Water.

which, in its turn, acted on by the mineral acid, is converted into auricyanic acid :—

$$2AuKCy_4 + H_2SO_4 = 2AuHCy_4 + K_2SO_4$$

Auri potassic cyanide, Sulphuric acid, Auricyanic acid, Potassic sulphate.

In theory the precipitation of gold from the auro potassic cyanide is as follows :

$$2Au\ KCy_2 + Zn = ZnK_2Cy_4 + 2Au.$$

Auro potassic cyanide, Zinc, Zinc potassic cyanide, Gold.

Without a doubt much of the precipation goes on according to this equation. At the same time it is also true that probably from electro-chemical reasons the precipitation is more rapid and perfect in the presence of an excess of potassic cyanide. This is particularly noticeable in cases of considerable quantities of copper being present in the working solutions. When such cupriferous working solutions are very weak in potassic cyanide, the copper may be precipitated in preference to the gold, whereas, by increasing the quantity of cyanide, the copper can be kept in solution until the precipitation of gold is complete.

In ordinary cases, the accelerating influence on precipitation of excess of potassic cyanide is probably due to generation of nascent hydrogen.

$$4KCy + Zn + 2H_2O = ZnK_2Cy_4 + 2KOH + H_2$$

Potassic cyanide, Zinc, Water, Zinc potassic cyanide, Potassic Hydrate, Hydrogen.

This nascent hydrogen steps into the place of the gold in the auro potassic cyanide :—

$$2AuKCy_2 + H_2 = 2KCy + 2HCy + Au_2$$

Auro potassic cyanide, Hydrogen, Potassic cyanide, Hydrocyanic acid, Gold.

The hydrocyanic acid thus formed recombining with any free alkali present, there is no loss of such cyanogen as was combined with the gold. From the former of these two equations it would appear that some proportion of the potassic cyanide must be consumed in the zinc boxes. As a matter of fact there is a consumption in the case of strong cyanide solutions, which, however, in the case of ordinary working, when solutions are coming off 0·2 per cent. or so, is quite inappreciable. Indeed it would appear as if a regeneration of the zinc potassic cyanide took place, the zinc possibly forming a hydrate and remaining in solution as such owing to the presence of free alkali. Given favourable conditions indeed the zinc potassic cyanide is itself capable of dissolving gold from ores, and by addition of free alkali to this salt all the cyanogen in it may be determined in the ordinary manner by means of silver nitrate solution.

It is often asked whether in view of the rather considerable amount of zinc which is dissolved in the boxes, the working solutions do not in time become very highly charged with zinc compounds. As a matter of experience it may be stated that they do no not, to any great extent, and the probable reason for this is that the small quantities of alkaline sulphides formed serve to precipitate at least a portion of the zinc as insoluble sulphide, a regeneration of potassic cyanide taking place simultaneously.

$$ZnK_2Cy_4 + K_2S = ZnS + 4KCy.$$

Zinc potassic cyanide, Potassic sulphide, Zinc sulphide, Potassic cyanide.

The presence, or rather the formation, of alkaline sulphides in the solutions is explained by the action of potassic cyanide on the iron sulphide contained in partially decomposed pyritous ores, or in markasite, etc. :—

$$6KCy + FeS = K4\ Fe\ Cy_6 + K_2S.$$

Potassic cyanide, Ferrous sulphide, Potassic ferrocyanide, Pot. sulphide.

Mr. J. S. MacArthur has even found that, in very exceptional cases, sufficient alkaline sulphide may be formed to be of hindrance to the action of the cyanide on the gold, and has discovered a remedy for this in the addition of metallic (particularly lead) salts, capable of forming insoluble sulphides. Of the reactions which take place when cyanide solutions come in contact with "acid" tailings imperfectly washed and neutralized, the chief are :—

1. Decomposition with liberation of hydrocyanic acid gas.
2. Formation of ferro and ferricyanides.

The first is caused by free acids in the ore acting on the cyanides.

$$2KCy + H_2SO_4 = 2HCy + K_2SO_4.$$

Hydrocyanic acid, Potassic sulphate.

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PAGE 10.—Second Equation should read :—

$$AuHCy_4 + KOH = AuKCy_4 + H_2O.$$

PAGE 10.—Sixth Equation should read :—

$$4KCy + Zn + 2H_2O = ZnK2Cy_4 + 2KOH + H_2$$

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equation :—

$$Fe_2(SO_4)_3 + 12KCy = 2K_3FeCy_6 + 3K_2SO_4$$

Ferric sulphate, Potassic cyanide, Potass. ferricyanide, Pot. sulphate.

The reactions of the various basic sulphates with cyanide are probably analogous to the above. All these iron salts are, however, easily rendered innocuous by application of an alkali before treatment with cyanide, the soluble iron salts becoming precipitated as hydrate, and the basic ones rapidly oxidizing after contact with alkali. It is evident that, in order to be effective in destroying iron salts, an alkali should be applied before and not together with the cyanide solutions, as these salts will destroy the cyanide as much in a strongly alkaline as in a nearly neutral solution. If free acid alone were concerned, it would be just as well to apply alkali and cyanide simultaneously.

It is also apparent that it is erroneous to assume that no waste of cyanide is going on as long as no "prussian blue" is formed. This is only formed in extreme cases, *i.e.* in solutions coming off acid. It is the result of combination of a ferrous salt with potassic ferricyanide in acid solutions. A white precipitate may be also formed by combination of ferrous salt with ferrocyanide. This white precipitate turns to prussian blue on standing.

A small amount of potassic cyanide probably is lost by oxidation, potassic cyanate and various complex organic compounds being formed.

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Auro potassic cyanide, Hydrocyanic acid, Oxygen, Auri potassic cyanide, Water.

which, in its turn, acted on by the mineral acid, is converted into auricyanic acid :—

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Auri potassic cyanide, Sulphuric acid, Auricyanic acid, Potassic sul

In theory the precipitation of gold from the auro pota

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[illegible]

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$$2KCy \quad + \quad H_2SO_4 \quad = \quad 2HCy \quad + \quad K_2SO_4.$$

Potassic cyanide, Sulphuric acid Hydrocyanic acid, Potassic sulphate.

As already stated, it is possible for the hydrocyanic acid gas thus formed to diffuse through the ore and dissolve quite appreciable amounts of gold. Hence the danger of water-washing very acid ores in the tanks which have contained cyanide.

The formation of the various iron and cyanogen compounds is very complicated. Of most frequent occurrence is undoubedly the formation of potassic ferrocyanide by action of the potassic cyanide on ferrous salts:—

$$FeSO_4 \quad + \quad 6KCy \quad = \quad K_4\,FeCy_6 \quad + \quad K_2\,SO_4$$

Ferrous sulphate, Potass. Cyanide, Potass. ferrocyanide, Potass. Sulphate.

The action of potassic cyanide on ferric sulphate may be represented by the following equation:—

$$Fe_2(SO_4)_3 \quad + \quad 12KCy \quad = \quad 2K_3FeCy_6 \quad + \quad 3K_2SO_4.$$

Ferric sulphate, Potassic cyanide, Potass. ferricyanide, Pot. sulphate.

The reactions of the various basic sulphates with cyanide are probably analogous to the above. All these iron salts are, however, easily rendered innocuous by application of an alkali before treatment with cyanide, the soluble iron salts becoming precipitated as hydrate, and the basic ones rapidly oxidizing after contact with alkali. It is evident that, in order to be effective in destroying iron salts, an alkali should be applied before and not together with the cyanide solutions, as these salts will destroy the cyanide as much in a strongly alkaline as in a nearly neutral solution. If free acid alone were concerned, it would be just as well to apply alkali and cyanide simultaneously.

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A small amount of potassic cyanide probably is lost by oxidation, potassic cyanate and various complex organic compounds being formed.

The following hints as to testing a sample of ore with a view to determining its amenability to cyanide treatment may be found of some service.

It is assumed that the total sample is crushed fine enough to pass a 30 mesh sieve. Then the customary routine is :—

1. Assay a portion of the sample.
2. Determine the amount of cyanide it will consume by shaking test For example, 200 grammes of ore are placed in a glass stoppered bottle with 100 cc's of a solution of cyanide, of 0·5 per cent. strength and shaken for 20 minutes or so. A portion of the solution is then filtered off and tested. Supposing it to be reduced in strength to 0·4 per cent, showing a consumption of 0·1 per cent. on the solution, or half as much, *i.e.* 0·05 per cent. on the ore (or 1 lb per ton), we might safely conclude that the ore will not require any preliminary treatment before leaching with cyanide. The largest consumption of cyanide takes place almost immediately after the solution comes in contact with the ore, and after 20 minutes shaking it is generally safe to assume that there will not be much further consumption.
3. If it is found that the consumption of cyanide is excessive, a third portion of the ore is tested for "cyanicide," by which is meant free acid, soluble and basic iron salts, and indeed any cyanide destroying substance which may be counteracted by alkali. A solution of caustic soda of known strength is run little by little from a burette into a weighed quantity of the ore mixed with water, the mixture being well stirred after each addition of alkali until a drop taken out on a glass rod will just turn red litmus slightly blue. A convenient quantity of ore to operate on is 200 grammes, and, using an alkali solution of 10 grammes commercial caustic soda to the litre of water, each cubic centimetre will correspond to 1-10th lb. of the same quality caustic soda required to the ton (of 2000 lbs.) of ore. If the consumption of soda be more than 3 lbs. per ton, it will generally be found advisable to water-wash the ore before giving alkaline treatment. It is easy to determine the amount of alkali which may be saved through a preliminary water-wash, by first estimating total "cyanicide," then taking another sample, water-washing first, and estimating remaining "cyanicide."

The writer would suggest that cyanide chemists make a practice, when reporting the amount of "cyanicide" in an ore, of doing so in terms of pounds and fractions of caustic soda required to neutralize a ton of the ore.

Should the consumption of cyanide in No. 2 test have proved larger than the amount of iron salts and acid present would account for, there is probably copper in the ore. The cyanide solution from test No. 2 may be conveniently examined for copper by evaporating with nitric acid, taking up with a little more nitric acid, diluting and precipitating with ammonia, when copper will be indicated by the characteristic blue coloration of the liquid.

4. Another sample of ore, or better still, some half-dozen samples, are weighed out for "extraction" tests. A suitable vessel for testing extraction is a lamp glass fitted with an india-rubber stopper with a glass tube through it, which may be closed by means of a small piece of rubber tubing and a burette clip. A filter is formed over the rubber stopper by means of a piece of sponge, some filter paper, or some asbestos fibre Plate III, fig. 2.—Into such vessels the samples are placed, say 200 grammes of each, and they must then receive whatever preliminary treatment test No. 3 has shown to be needful in the way of water and alkali washes. 100 cc's of a 0·5 per cent. solution of cyanide is then poured on. The various samples may be allowed to stand for different periods, say 1, 2 and 3 days respectively, or more if thought fit, the cyanide solutions being then drawn off and tested, and the ore, after water-washing to remove all dissolved gold being assayed again.

There is always plenty of employment for a cyanide chemist in charge of a plant. In addition to looking after the practical working of the establishment—which would include careful assays of the ore before and after treatment—there are a number of other points which demand attention. For instance, it is well to make frequent assays of the working solutions, to ascertain whether the gold is being adequately precipitated from them in their passage through the zinc boxes. The solutions may be assayed by evaporating a weighed or measured quantity in a dish made of lead foil, which, as soon as the evaporation is complete, can be rolled up and cupelled, and the resulting bead weighed. Mr. A. F. Crosse suggests estimating the gold in cyanide solutions by adding excess of nitrate of silver. This precipitates all the cyanogen present, and also brings down the gold as argentic auric cyanide. The silver salt should be added a little at a time, and the solution well shaken up after each addition. The precipitate will then cohere and settle sufficiently to enable one to see whether a further addition of silver nitrate causes further precipitation. If not, then the precipitate is at once filtered off, dried, mixed with a little flux and reduced by melting in a small fireclay crucible. The resulting silver bead is then flattened out and dissolved in nitric acid, leaving the gold to be washed, annealed, and weighed. This method has the advantage that one can operate on a fairly large sample of solution without the somewhat lengthy process of evaporation. In the case of strong cyanide solutions, Mr. Crosse recommends decomposing most of the potassic cyanide by addition of acid previous to adding the silver nitrate, to avoid the otherwise heavy consumption of the latter.

The works chemist should experiment with the residues, and find out in what portion of them the remaining gold is contained, *i.e.*, whether in the coarse, the pyritic, the quartzy, or the slimy portion, or possibly as dissolved gold contained in the moisture, and from his results he may possibly hit on some way of improving extractions. Assays of top, middle, and lower layers of residues will also be found instructive in this connection.

The testing and making up of solutions require no great amount of chemical skill, and any intelligent workman who has a fair knowledge of arithmetic can be taught to do these things. The method employed for estimating the amount of cyanide in a solution is based on the capacity of potassic cyanide to form a double cyanide with silver—added to it as argentic nitrate,—and on the fact that any silver solution which is added beyond the exact quantity which is required to convert all the potassic cyanide into argentic potassic cyanide will cause a white precipitate :—

$$2\,KCy. \quad + \quad AgNo_3 \quad = \quad AgKCy_2 \quad + \quad KNO_3$$

Potassic cyanide. Argentic nitrate, Argentic potassic cyanide, Potassic nitrate

As the combining weights of $AgNO_3$ and KCy are 170 and 65·13 respectively, it follows that 170 parts by weight of argentic nitrate may be added to 2 × 65·13 = 130·26 parts of potassic cyanide before a permanent precipitate ensues. If, therefore, we add from a burette a solution of argentic nitrate containing 17 grammes in a litre, to the solution of potassic cyanide to be tested until a faint precipitate appears, each cc of silver solution added will correspond to $\frac{13·02}{1000} = 0·013$ grammes of pure potassic cyanide. From the amount of cyanide solution operated on, the percentage contents can be calculated. It is obvious that the strength of the silver nitrate solution may be so adjusted as to save all calculation. If, for instance, it is made by dissolving 13·05 grammes of pure silver nitrate in a litre of water, and 10 cc's of the cyanide solution be taken for a test, then each cc of silver solution added will correspond to 0·1 per cent. of pure KCy in the sample tested. In testing very strong solutions—from the dissolving tank for instance,— one-tenth of the quantity of sample may be taken, by measuring 10 cc's, diluting with water to 100 cc's, and then drawing off 10 cc's for test. Of course in such a case 1 cc of standard silver solution will indicate 1 per cent. of KCy in the original sample of cyanide solution. Addition of a few drops of potassic iodide to the solution to be tested will enhance the accuracy of the test, and will moreover annul the danger of over-estimating the quantity of cyanide present consequent on the strong alkalinity of the solution.

In making up working solutions, where an almost pure chemical is used, it is sufficient to estimate the amount and strength of stock solution in storage vats, and simply add the requisite weight of cyanide of potassium to make the solution up to desired strength. The commercial cyanide, however, which, on account of its being much cheaper is commonly used, is not, as a rule, much over 80 per cent. in strength, and is moreover somewhat variable in quality. It is usual therefore, to dissolve as many cases as are likely to be required for a day or two in some 300 to 600 gallons of water, to test the strength of the solution obtained, and draw off into the storage vats the quantity required to make the stock solution up to working strength. A convenient arrangement for dissolving the cyanide is sketched on Plate VIII, Fig. 2. It consists of a small vat with filter tray (stout wire gauze, covered with jute, and mounted in a wooden frame) suspended in it. A pump is attached for the purpose of causing circulation of the solution. The insoluble impurities, chiefly carbide of iron, contained in the commercial cyanide, remain in the tray. The pumping of the solution into this tray should be kept going fast enough to keep the lumps of cyanide covered by solution, as it is found that alternate exposure of the carbide to the air and immersion in solution of cyanide, causes a certain amount of decomposition of the latter. A water-wash may be applied to remove the last trace of cyanide from the carbide before throwing the latter away. On no account should this carbide be put on the top of the sand in the filter vats, as is sometimes done. Having determined the strength and quantity of the stock solution, and the strength of the solution in the dissolving tank, the following is a simple formula for arriving at the quantity of the latter requisite to bring the former up to the desired strength:—

A.—Being desired strength of stock (in per cent.)
B.—Being present strength of stock (in per cent.)
C.—Strength of dissolving tank solution (in per cent.)
D.—Quantity in tons, lbs., gallons, litres, etc., of stock, then

$\frac{A - B}{C - A} \times D =$ quantity of dissolving tank solution to be added (in tons, lbs., gallons, litres, etc.)

For example, supposing the stock solution to consist of 25,000 gallons of 0·2 per cent. strength, and it be desired to bring this up to 0·3 per cent. by adding some 12 per cent. solution, then:—

$\frac{0{\cdot}3 - 0{\cdot}2}{12{\cdot}0 - 0{\cdot}3} \times 25{,}000 = 213{\cdot}6$ gallons of the dissolving tank solution required.

NOTES ON CYANIDE PROCESS. PLATE I.

Scale 20ft = One Inch

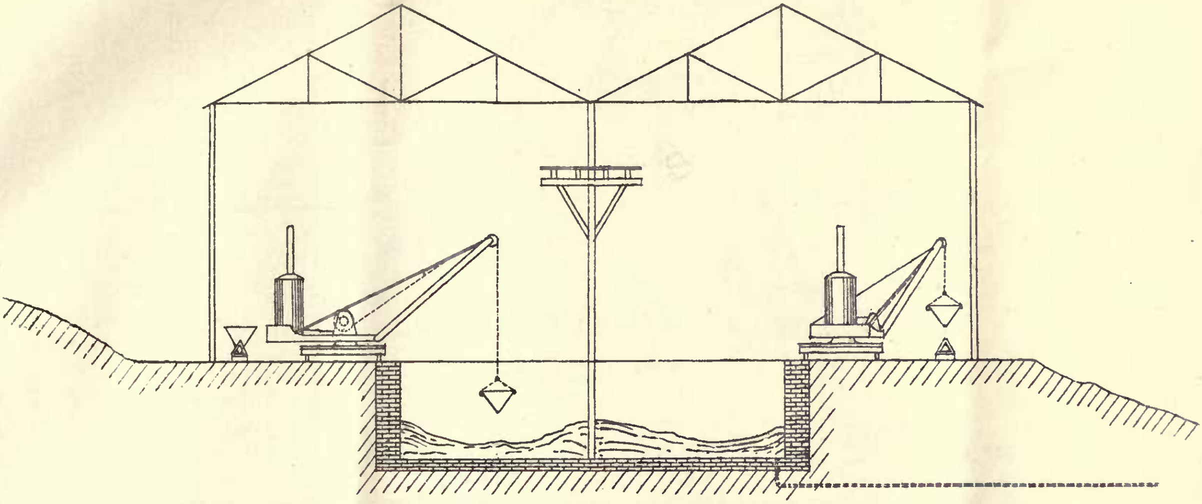

FIGURE 1.

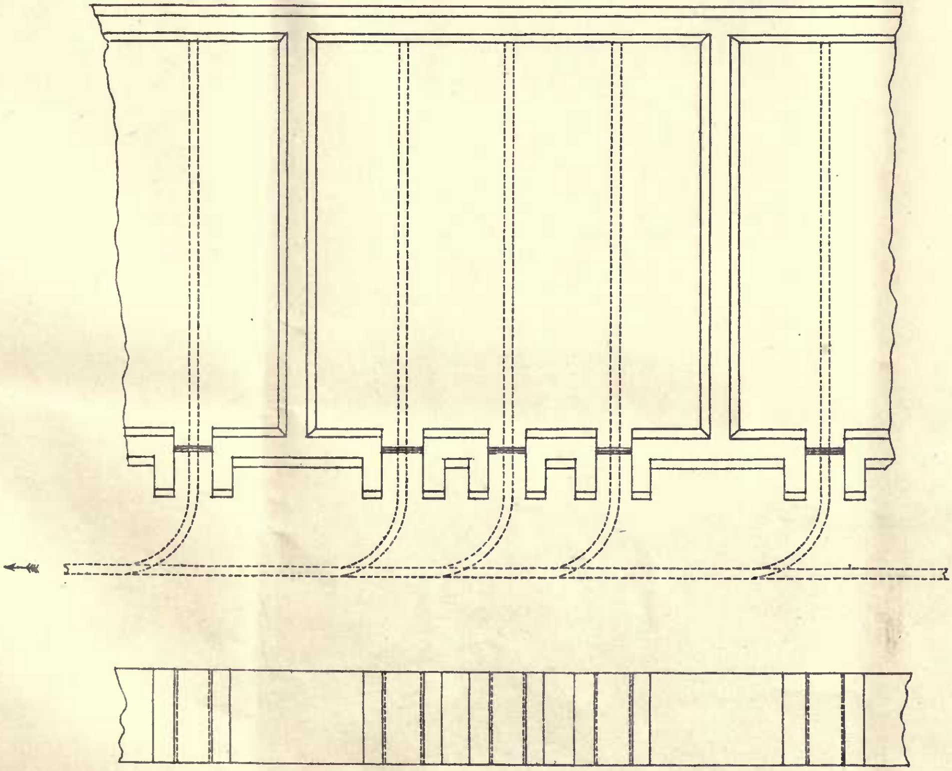

FIGURE 2

NOTES ON CYANIDE PROCESS. PLATE II.

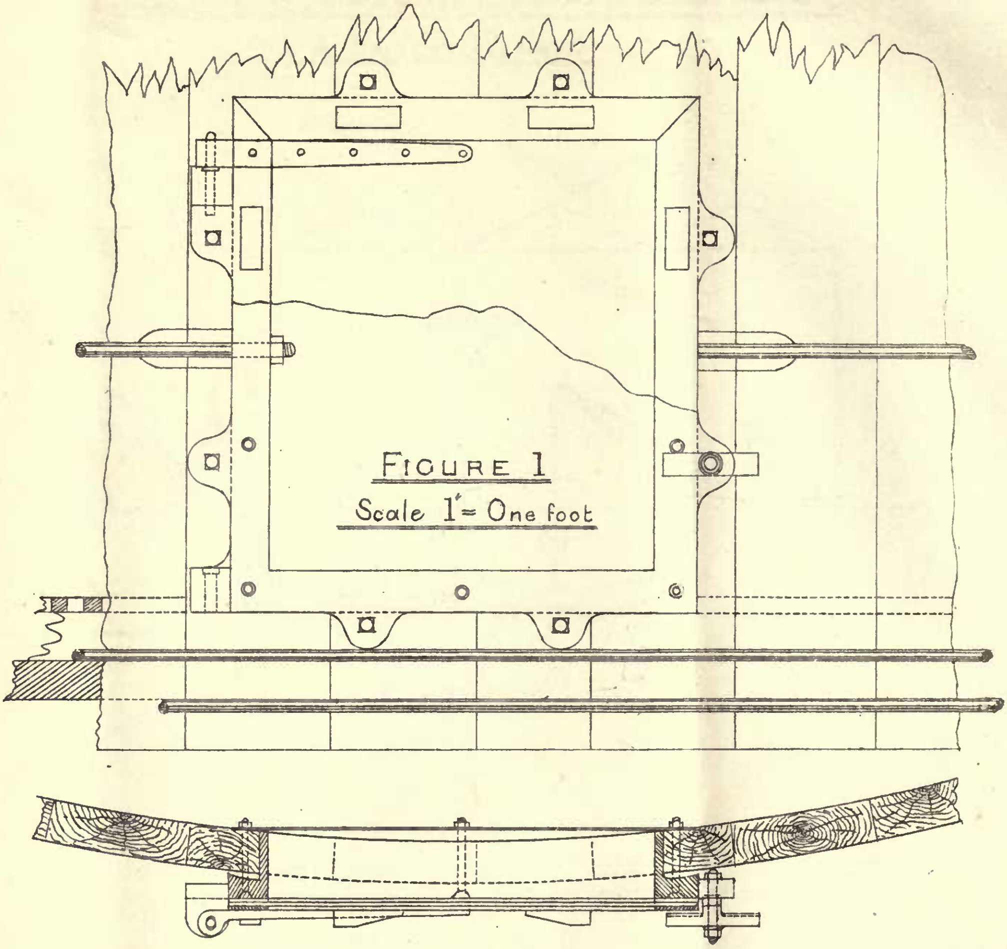

FIGURE 1

Scale 1" = One foot

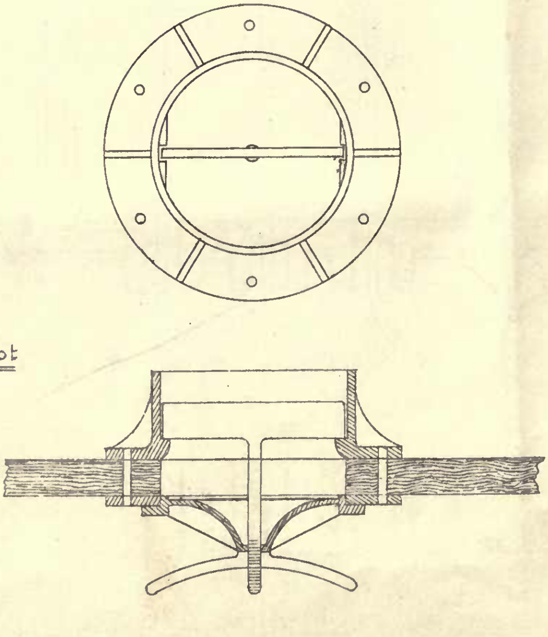

FIGURE 2

Scale $\frac{3}{4}$" = One foot

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NOTES ON CYANIDE PROCESS. PLATE III.

FIGURE 1. Scale 1" = One foot

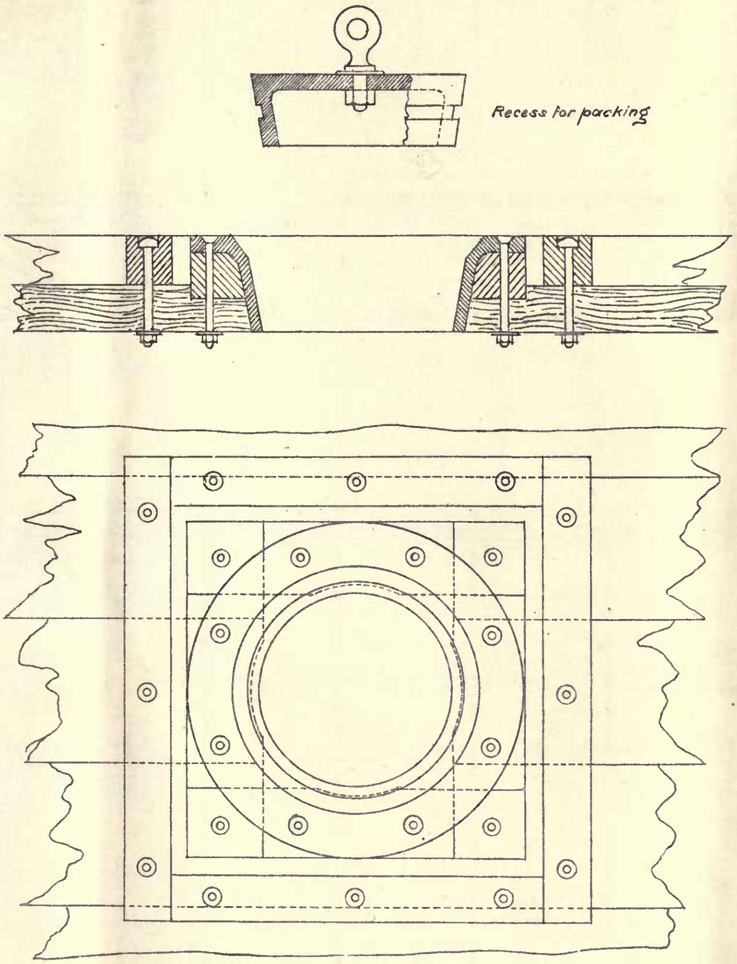
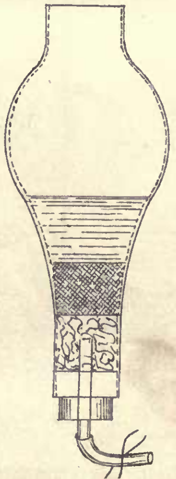
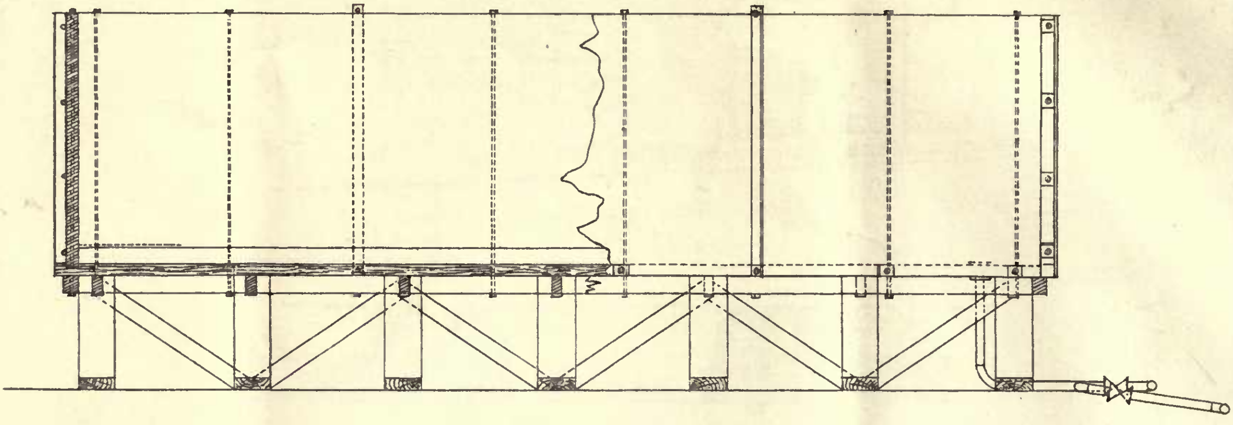
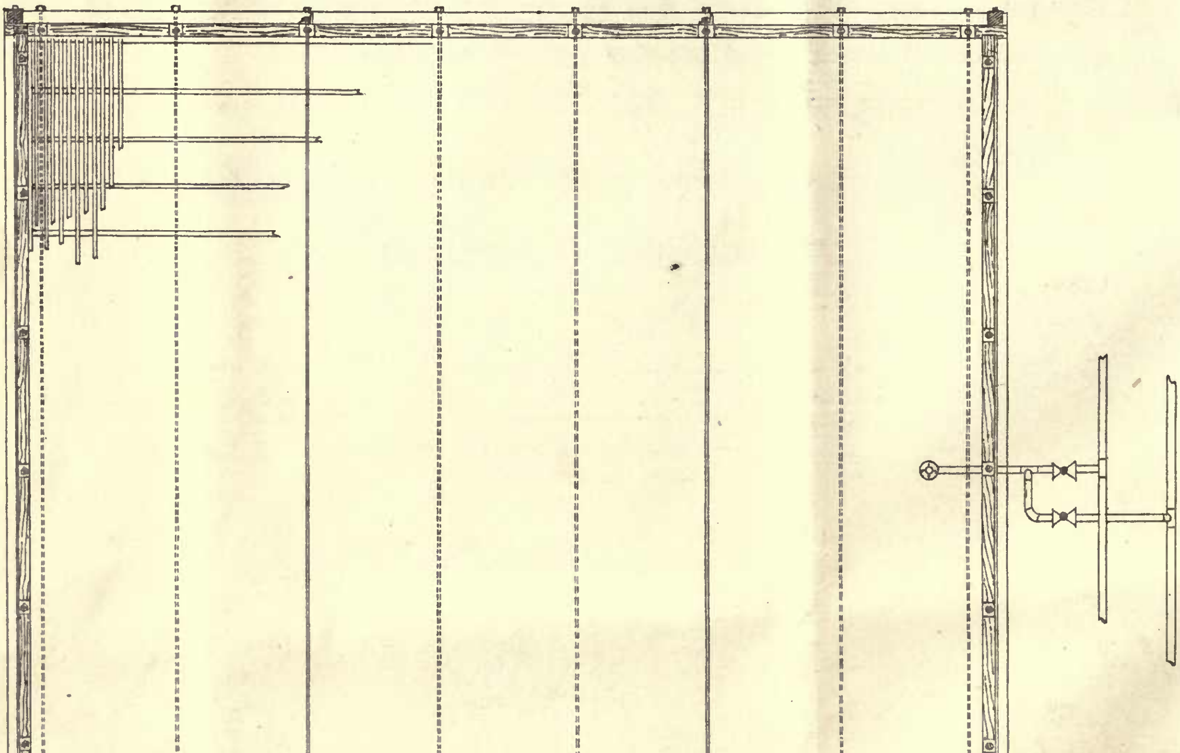

Recess for packing

FIGURE 2.

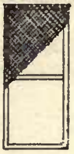


NOTES ON CYANIDE PROCESS. PLATE IV.



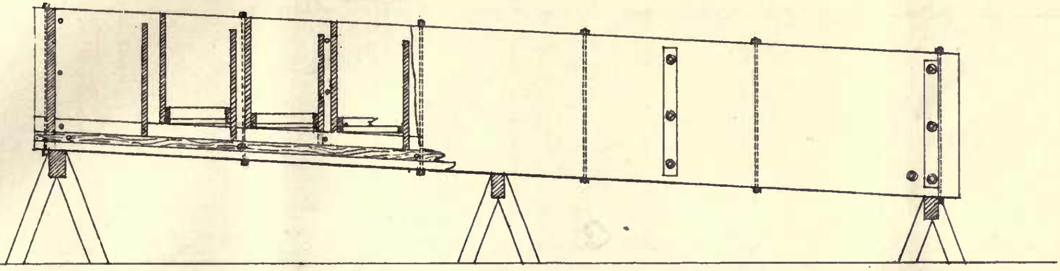


Scale $\frac{1}{4}''$ = One foot.

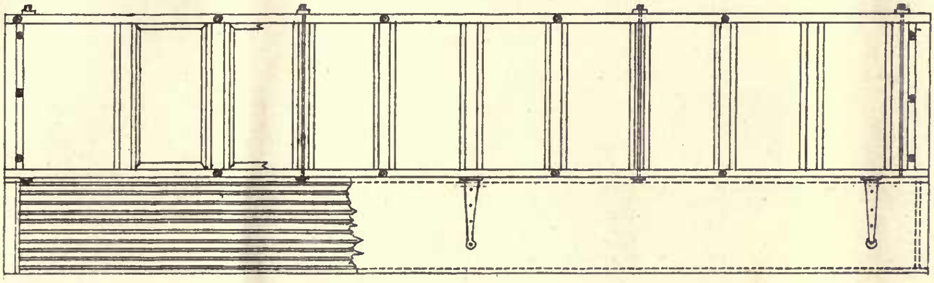


NOTES ON CYANIDE PROCESS. PLATE V.

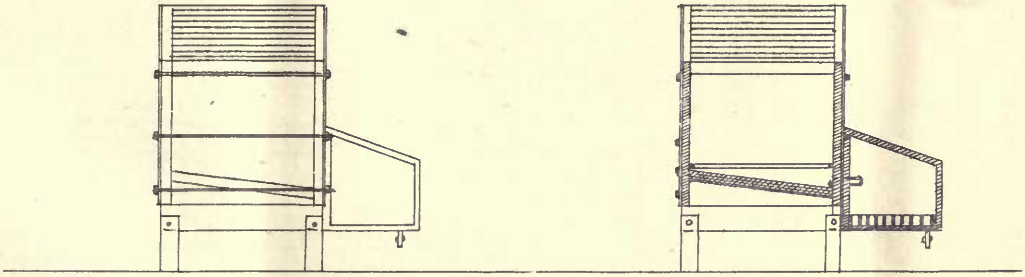
Scale $\frac{1''}{3}$ = One foot



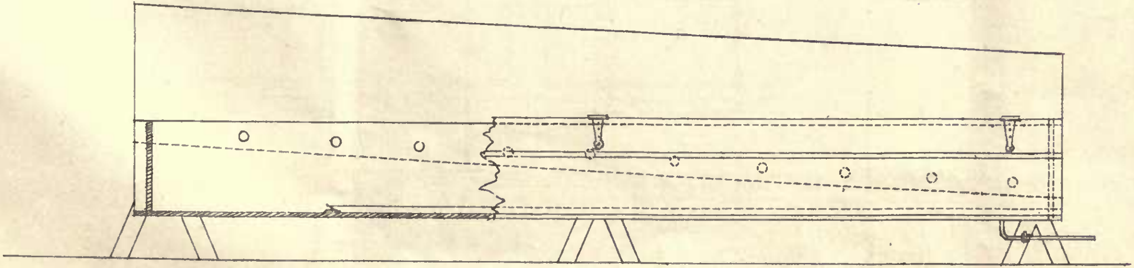
ELEVATION OF ZINC BOX WITHOUT LAUNDER.



PLAN WITH LAUNDER.

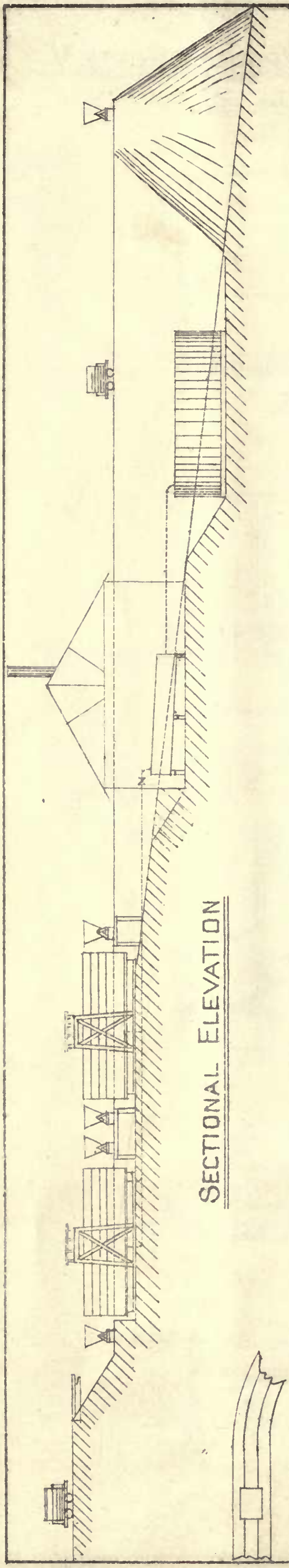


END ELEVATION AND SECTION.

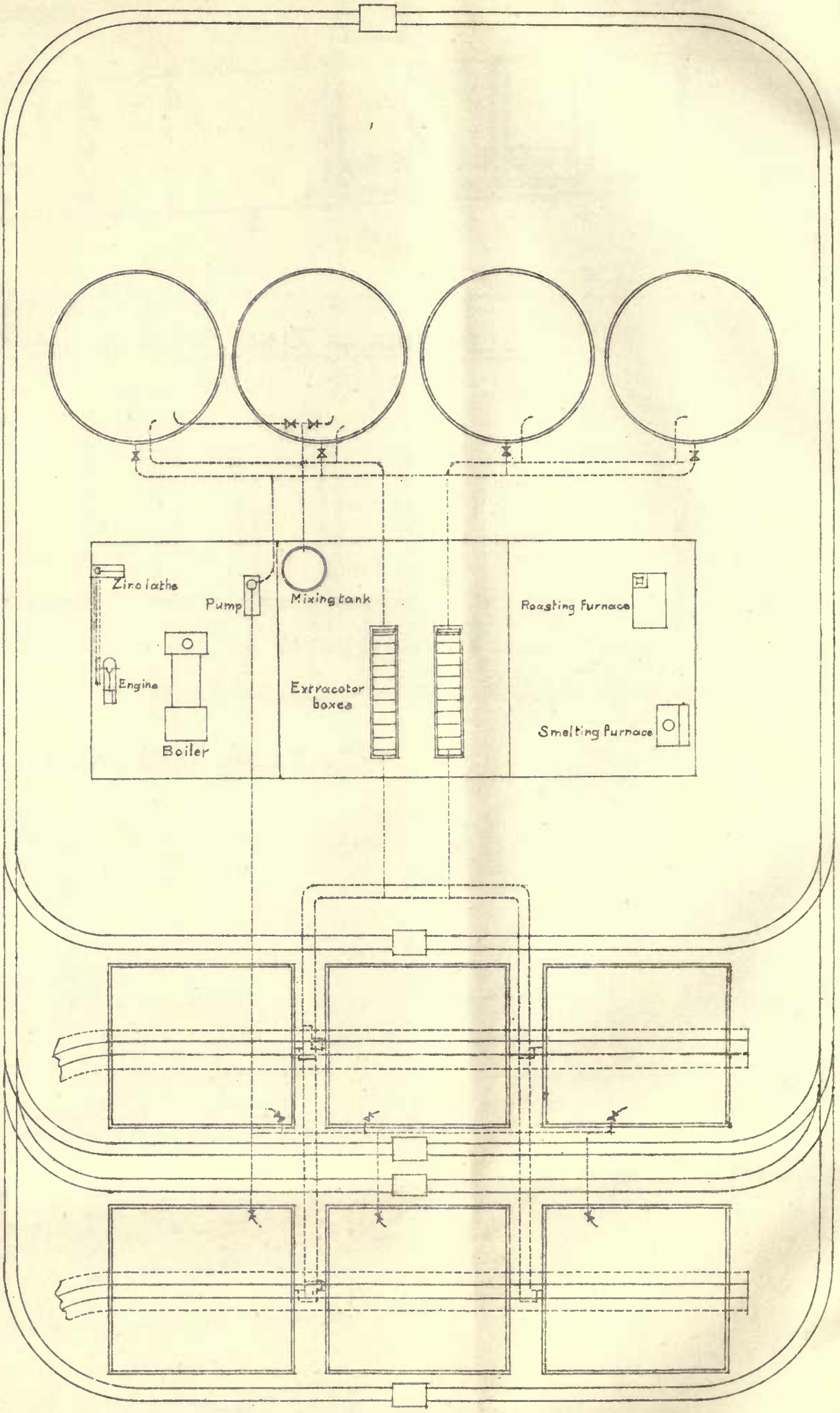


ELEVATION SHEWING LAUNDER.

NOTES ON CYANIDE PROCESS. PLATE VI



SECTIONAL ELEVATION


Zinc lathe
Pump
Mixing tank
Roasting Furnace
Engine
Boiler
Extractor boxes
Smelting Furnace

GROUND PLAN

Scale 20 ft = One inch.

Argus Coy Ltd

NOTES ON CYANIDE PROCESS. PLATE VII.

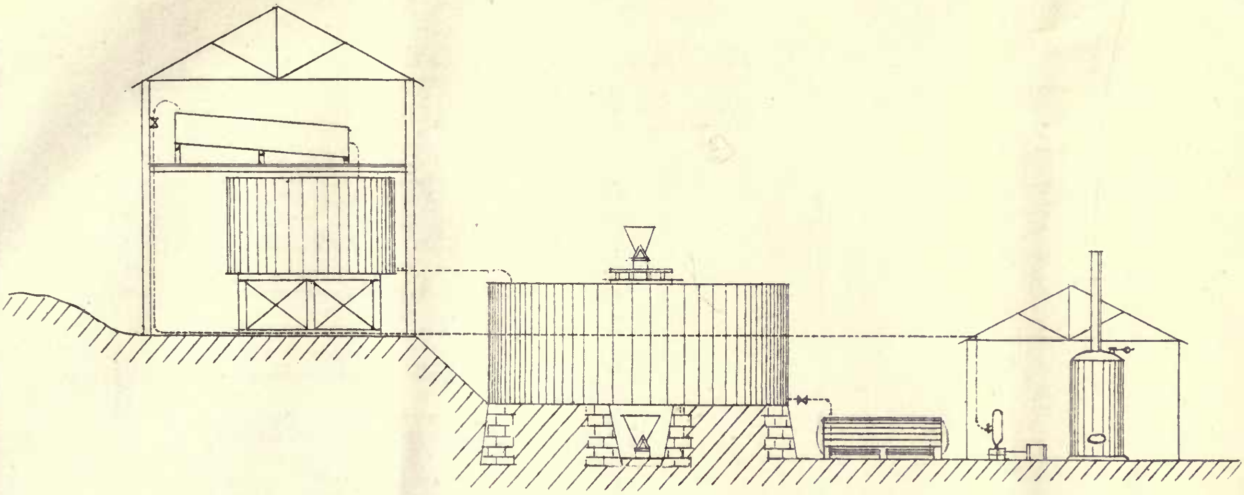


FIGURE 1

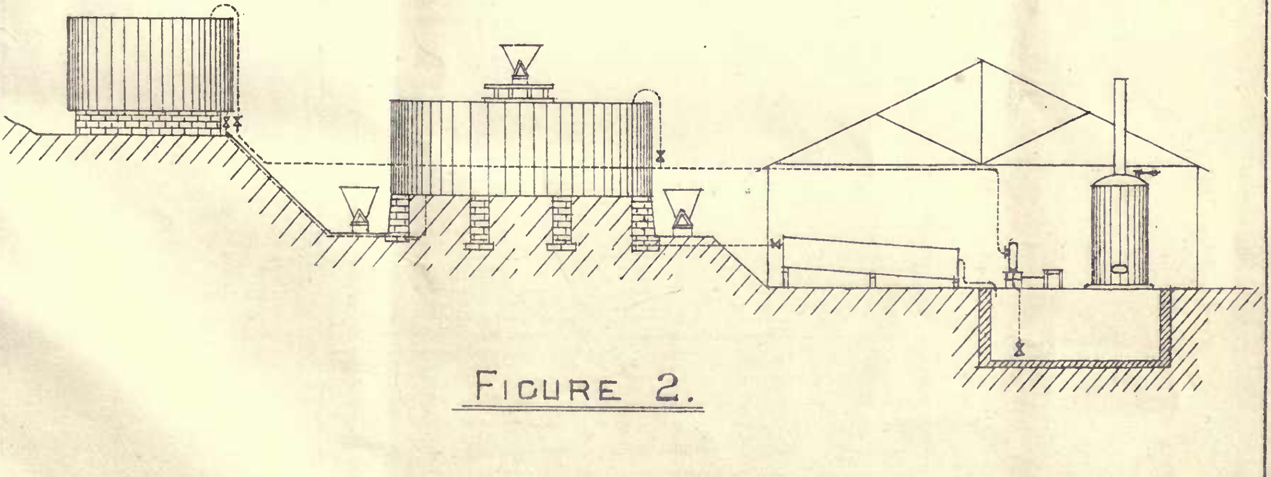


FIGURE 2.

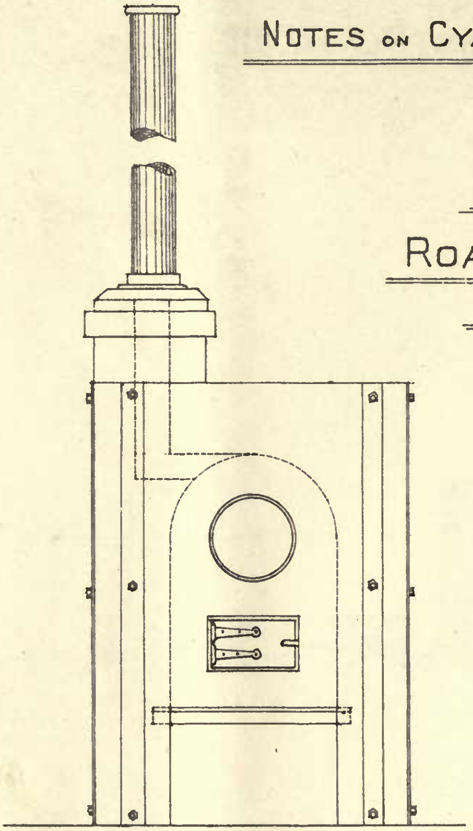
Scale 16 ft = One inch

NOTES ON CYANIDE PROCESS. PLATE VIII.

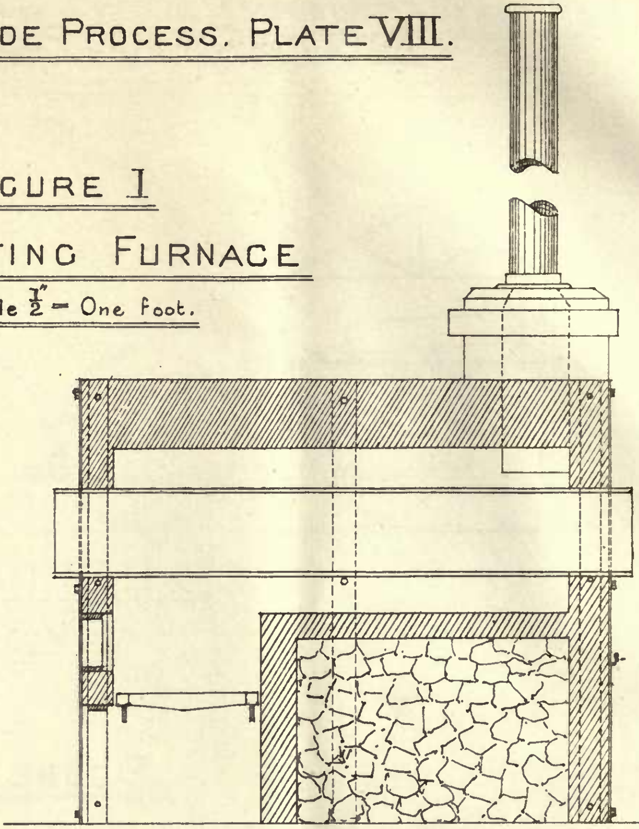
FIGURE I

ROASTING FURNACE

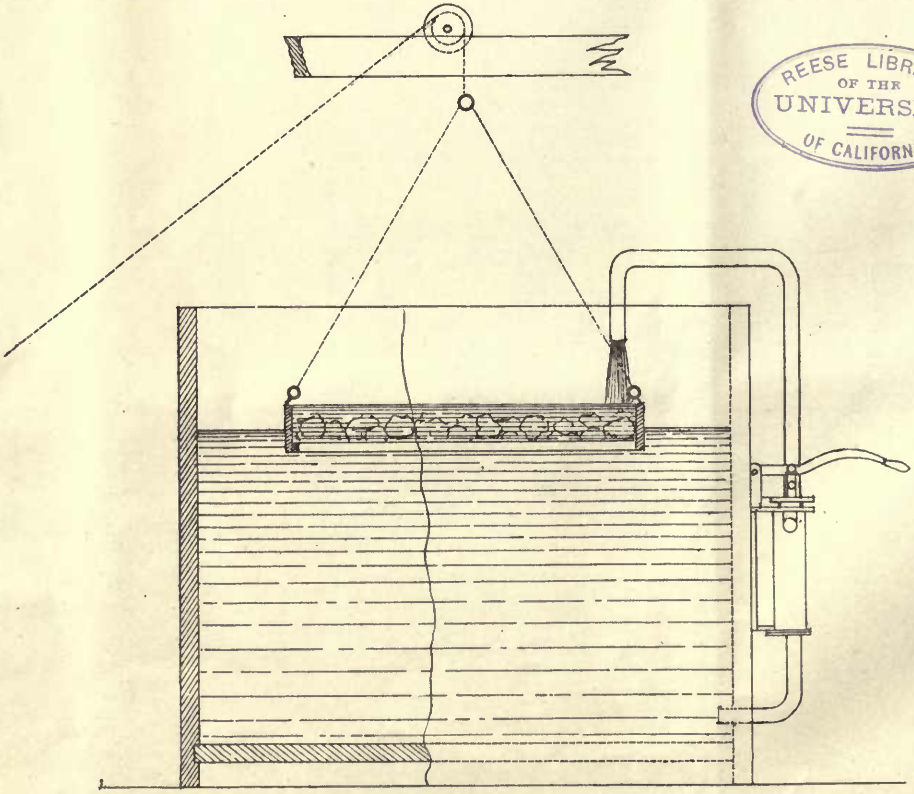
Scale $\frac{1}{2}''$ = One foot.



FRONT ELEVATION.



LONGITUDINAL SECTION.



SECTION OF MIXING TANK AND TRAY.