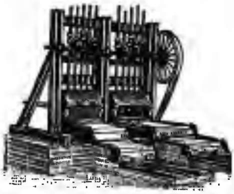
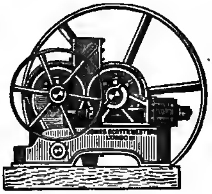
# THE METALLURGY OF GOLD

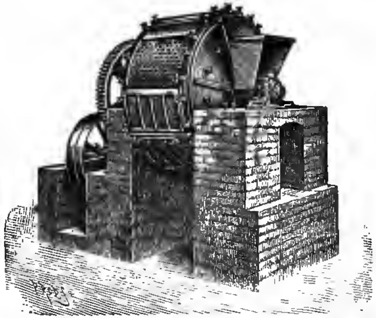
EISSLER

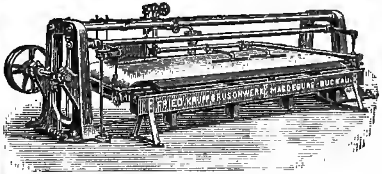


STAMP BATTERY.



KROM ROLLS.




FRIED. KRUPP GRUSONWERK MAGDEBURG-BUCKAU

# THE
# METALLURGY OF GOLD

## *BY THE SAME AUTHOR.*

Crown 8vo, 360 pp., with 150 Illustrations, price 10s. 6d. cloth.

THE METALLURGY OF SILVER: A Practical Treatise on the Amalgamation, Roasting, and Lixiviation of Silver Ores; including the Assaying, Melting, and Refining of Silver Bullion. Second Edition, Enlarged.

"A practical book for practical men . . . A convenient and useful work, containing a good deal of original matter."—*Athenæum.*

Crown 8vo, 336 pp., with about 100 Illustrations, price 10s. 6d. cloth.

A HANDBOOK ON MODERN EXPLOSIVES: a Practical Treatise on the Manufacture and Application of Dynamite, Gun-cotton, Nitro-glycerine, and other Explosive Compounds; including the Manufacture of Collodion Cotton.

"A book which should be acquired by every one who has to deal with either the use or the manufacture of the powerful explosives of modern days."—*Board of Trade Journal.*

Crown 8vo, 400 pp., with 183 Illustrations, price 12s. 6d. cloth.

THE METALLURGY OF ARGENTIFEROUS LEAD: A Practical Treatise on the Smelting of Silver-Lead Ores and the Refining of Lead Bullion. (The Illustrations include Plans and Sections of Smelting Furnaces and Plant in Europe and America.)

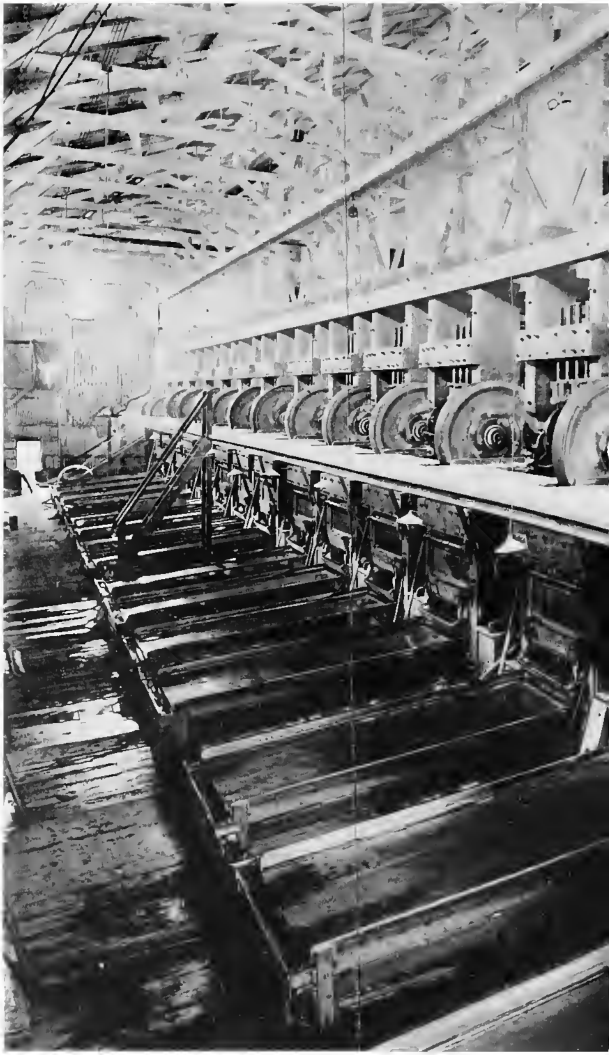
"Those who wish to obtain a thorough insight into the present state of this industry cannot do better than read this volume."—*Industries.*

Just Published. Large Crown 8vo, with Diagrams and Working Drawings, 7s. 6d. cloth. (Being substantially a reprint of Chapters XI.—XV. of the present volume).

THE CYANIDE PROCESS OF GOLD EXTRACTION; and its Practical Application on the Witwatersrand Gold Fields in South Africa.

"This book is just what was needed to acquaint mining men with the actual working of a process which is not only the most popular, but is, as a general rule, the most successful for the extraction of gold from tailings."—*Mining Journal.*

London: CROSBY LOCKWOOD & SON, 7, Stationers' Hall Court, E.C.



160-stamp

Battery ; Langlaagte Estate. [*Frontispiece.*

THE

# METALLURGY OF GOLD

## *A PRACTICAL TREATISE*

ON

## The Metallurgical Treatment of Gold-Bearing Ores

INCLUDING

THE PROCESSES OF CONCENTRATION, CHLORINATION, AND
EXTRACTION BY CYANIDE,

AND

THE ASSAYING, MELTING, AND REFINING OF GOLD

BY

M. EISSLER
MINING ENGINEER; A.I.M.E.; MEMBER OF THE INSTITUTE OF MINING AND METALLURGY
AUTHOR OF
"MODERN HIGH EXPLOSIVES," "THE METALLURGY OF SILVER," ETC., ETC.

*FOURTH EDITION, ENLARGED*

With about Two-hundred-and-fifty Illustrations and Numerous
Folding Plates

LONDON
CROSBY LOCKWOOD AND SON
7, STATIONERS' HALL COURT, LUDGATE HILL
1896

# PREFACE.

IN this volume my aim has been to present in a condensed form such an account of the several processes which are generally used for the Extraction of Gold from the Ore as shall be sufficient for the full information of investors, and others interested in gold mining operations, who may wish to gain an intelligent insight into the *modus operandi* at the gold mines. At the same time I have given full descriptions, with illustrations, of the machinery employed in those processes, in sufficient detail to make the book of substantial value to professional men and manufacturers concerned in the gold-mining industry.

The treatment of gold-bearing ores, although at one time so uncertain in its results as to be, not only a hazardous, but almost a purely speculative business, has now, through continued improvements in appliances and the adoption of more economical methods, developed into a settled industry, in which more or less profitable results may be obtained even from comparatively low grade ores.

When, forty years ago, the news was circulated throughout the world from California, that gold had been found in that territory by Mr. Marshall, in a millrace near Sutter Creek, thousands of enterprising adventurers flocked to the new Eldorado. So abundant were the glittering scales of metal which appeared in the gravel washes and in the rivers and tributary streams of the western slopes of the Sierra Nevada mountain ranges, that little or no metallurgical knowledge was required for the successful extraction of the precious gold dust. No machinery was then needed beyond a pair of stout arms, a pick, a shovel and pan—with perhaps a rocker and a few boards nailed together in a trough-like sluice; and it was with rude appliances such as these that many millions' worth of gold was taken from the Placer mines, thus laying the foundations for the Pacific empire.

The gold discoveries in Australia—which came a few years later—nearly outrivalled the rich Californian deposits, and gave a new lease of life to the Australian colonies; and to-day Great Britain looks with pride at those magnificent domains at the Antipodes, teeming with a large population and many flourishing cities, all of them owing their prosperity, in no small measure, to the discovery of gold and other metals.

From present indications the South African goldfields have also a brilliant future in store, but the discoveries in that region are too recent to afford means for the full appreciation of the possible results.

When gold was first discovered in California, the knowledge of metallurgy and mining in their several branches, among the people of the United States, was very rudimentary. But with that discovery a new era began over the vast expanse of territory extending from the Atlantic to the Pacific. The citizens of the great Republic awoke to the fact that in their mountain ranges were minerals of all kinds, representing enormous wealth, which could not be secured without extended knowledge of the requisite appliances. These were rapidly introduced in all branches of metallurgy, and in the advances thus made the treatment of gold-bearing ores has kept pace with the improvements effected in other branches of the science.

Brilliant results have been achieved in the last twenty years, and I shall rejoice if the present work assists in extending knowledge of the subject, and in further stimulating progress in invention and the development of gold mining.

LONDON,
*August*, 1888.

## PREFACE TO THE SECOND EDITION.

IN preparing a new edition—which I am glad to find so soon required—I have been at some pains to supply additional matter on those subjects a fuller treatment of which has seemed desirable in the light of my experience with the first edition. The result is the enlargement of the work by about one hundred and fifty pages and forty additional illustrations.

My best thanks are due to the public press and private correspondents for valuable suggestions for a new edition; and of these I have availed myself wherever practicable.

There has been no interference in the present edition with the main plan of the original work, although five new chapters have been added. In these (amongst other matter) will be found much additional information as to machinery employed in crushing and amalgamation, and as to mills in actual operation. Later developments of the process of Chlorination are also described, and a chapter has been added on the Smelting of gold ores and the refining and parting of gold bullion. I have also given some account of the physical features of the gold-producing districts of North-Western America, which I hope will be none the less interesting because of the element of personal experience which is there introduced. In the first chapter of the work I have inserted an account of the interesting operations of Hydraulic Mining, with which the history of many gold-producing districts is so largely concerned.

In its enlarged form I trust the work may prove increasingly acceptable and useful to those for whom it is designed.

17, BELSIZE CRESCENT,
SOUTH HAMPSTEAD, LONDON, N.W.
*April*, 1889.

# PREFACE TO THE FOURTH EDITION.

In the Third edition of this work (which was called for early in 1891), although considerable additions had been made to the Second edition, the volume was further enlarged by the addition of four new chapters and about fifty new illustrations. I was thus enabled to give particulars of various improvements in gold-milling appliances which had then been recently introduced with the view of cheapening the treatment of the ore.

The attention of inventors having been largely directed to Chlorination, and the roasting of the ores being a necessary preliminary to that process, a large part of two of the new chapters was devoted to that subject.

In the preface to the Third edition, however, I pointed out that the question still arose what should be done with the free gold ores, carrying, say, from five to seven pennyweights of very fine gold—so fine that it is impracticable to save it by amalgamation ; while in distant localities such ores would not bear the cost of treatment by chlorination; and I added that, besides the large bodies of such ores which are known to exist, there were at many mines vast accumulations of tailings, awaiting the discovery of a suitable method of utilising their hidden wealth of gold.

Since those remarks were written, the use of potassium cyanide solutions for the recovery of gold from tailings has been adopted with great success in the Witwatersrand gold-fields under the patents of Messrs. MacArthur and Forrest, notwithstanding that patents had been (since 1867) previously brought out, and the process tried in various ways in the United States, without successful results, at all events from the commercial point of view. Further steps in advance have been accomplished by the introduction—also in the Witwatersrand—of the Cyanide process as developed by Messrs. Siemens and Halske, under whose patents electricity is utilised for the precipitation of the gold from the cyanide solutions.

In the present edition, full attention has been given to these important developments; and in the five new chapters devoted to the subject those metallurgists who desire to avail themselves of the Cyanide process will (I believe) find the methods of procedure described with sufficient fulness and completeness for practical purposes. Not only the working details, but full particulars of the results actually obtained, are given, while a chapter is devoted to the Chemistry of the Cyanide Process.

That I am in a position to deal with the matter so fully is owing to the fact of my having made a lengthened stay in the Witwatersrand gold-fields —whence I have just returned—and to my having enjoyed there unique opportunities of studying the process in actual operation.

As becomes the importance of this branch of the Metallurgy of Gold, special care has been taken to secure full and sufficient diagrams in illustration of the Cyanide process and the necessary plant.

Some further additions have also been made to the present edition, including—besides a chapter on Recent Milling Operations in the Transvaal—particulars of various appliances which the energy and ingenuity of manufacturers and inventors have produced for the assistance of the gold-mining community since this work was first prepared; as well as the results of recent researches upon various questions touching the treatment of the gold and its ores; with details of working operations and results which have been collected by me at various mines which I have had occasion to visit. Some Statistical Notes which were supplied to me on those occasions, and some remarks on the Future Outlook in the South African Gold Fields, will be found in an Appendix.

I have taken the opportunity of inserting also in the present edition some additional diagrams, illustrating the construction and arrangement of Plants for Gold-mining. These illustrations—being given on a full scale, and in some cases in much detail—cannot fail to be of material service to those who have the responsibility of providing similar plants for new or existing mines.

With these several enlargements and additions, I trust the work may prove as acceptable in the future as in the past; and I shall hope, indeed, that it may prove increasingly useful to all who avail themselves of its pages.

London, 1895.

P.S.—It should be noted that the present Edition has been passed through the press during the Author's absence in Australia.

# CONTENTS.

## CHAPTER I.

### *INTRODUCTORY.*

## CHAPTER II.

### *TREATMENT OF GOLD-BEARING ORES: CRUSHING AND AMALGAMATION.*

## CHAPTER III.

### *TREATMENT OF GOLD-BEARING ORES: CRUSHING AND AMALGAMATION—continued.*

## CHAPTER IV.

### *MILLS IN OPERATION: NEW MILLING MACHINES.*

## CHAPTER V.

## *TREATMENT OF GOLD-BEARING ORES: CONCENTRATION.*

## CHAPTER VI.

## *THE METALLURGICAL TREATMENT OF REBELLIOUS GOLD AND AURIFEROUS SILVER ORES.*

## CHAPTER VII.

## *THE ROASTING OF PYRITIC ORES.*

## CHAPTER VIII.

### *THE HYDRO-METALLURGY OF AURIFEROUS ORES: CHLORINATION.*

## CHAPTER IX.

### *LATER PROCESSES OF CHLORINATION.*

## CHAPTER X.

### *THE ELECTRO-METALLURGY OF PRECIOUS METALS.*

## CHAPTER XI.

### *THE CYANIDE PROCESS FOR THE EXTRACTION OF GOLD.*

## CHAPTER XII.

### *THE SIEMENS-HALSKE CYANIDE PROCESS.*

## CHAPTER XIII.

### *ERECTION OF A CYANIDE PLANT.*

## CHAPTER XIV.

### *THE CYANIDE PROCESS IN OPERATION AT VARIOUS WORKS.*

## CHAPTER XV.

### *THE CHEMISTRY OF THE CYANIDE PROCESS.*

## CHAPTER XX.

### *PHYSICAL AND GEOLOGICAL FEATURES OF GOLD-PRODUCING COUNTRIES.*

X

## CHAPTER XXI.

### *MILLING OPERATIONS IN THE TRANSVAAL.*

## CHAPTER XXII.

### *MILL SITE AND BATTERY—RECENT APPLIANCES.*

---

## APPENDIX.

---

b

# LIST OF ILLUSTRATIONS.

# LIST OF PLATES.

# THE
# METALLURGY OF GOLD.

## CHAPTER I.

### *INTRODUCTORY.*

Section I.—Gold Mining in California.—Quartz-mining Operations—Free Milling—Failure to "Catch the Gold"—Quartz Mines at Grass Valley—Progress in Milling Appliances—Loss in Working Gold Ores—Yield in California, in Australia, in Colorado—Professor Egleston on Loss in Extraction—Total Produce of Gold in California since 1848.

Section II.—Gold as a Metal.—Its Valuable Properties—Gold in the Pure State—Distribution—Composition—Properties of Pure Gold—Gold-bearing Ores.

Section III.—Gold Washing by the Hydraulic Process.—Applied to Deposits of Gravel—How the Gold lies in such Deposits—Laying out an Hydraulic Mine—Working the Mine—Results.

### I.—Gold Mining in California.

After the exhaustion of the easily worked "placer diggings" of California, the miner turned his attention to quartz. He soon found that the appliances then in existence in other parts of the world for the treatment of gold-bearing rock did not correspond to the local conditions of the country. Improvement after improvement of those appliances was introduced, and in a few years the now perfect Californian gold mill was the result of the ingenuity of the western millmen. Prior to 1861 quartz-mining operations were merely in an experimental stage, but about that time the quartz lodes, which were the original sources from which the enormous placer deposits derived their wealth, began to be seriously mined and systematically attacked, with results that have kept California in the foremost place of the gold-producing regions of the world, and she will, no doubt, through her numerous quartz lodes, continue to occupy the position of a gold producer for many generations to come.

As a rule, quartz mining in California was at first carried on upon the basis of "free milling"—namely, taking the ore out of the mine, crushing and pulverising the same in the battery, and extracting the gold by amalgamation on copper plates. Only a very small percentage of the gold in the ore was saved in this manner, the quantity of gold which can be secured by the agency of quicksilver being only a fraction of the assay value of the whole mass, while by far the largest portion of the gold went to waste, carried away by the water which flowed from the battery and got lost in the tailings. Many failures in gold mining were the result of the inability of the millman to "catch the gold."

But it was not long before these lessons of experience brought about the introduction of new methods, and quartz mining is now assuming the character of a scientific industry; so that ores which twenty-five years ago would have been thrown into the waste heap as "base" and "rebellious," are now carefully piled up, taken to the mill, and treated with great profit.

In the year 1866, like thousands of other mining adventurers, I was engaged in "placer" mining operations in Idaho Territory, a country whose name at that time was little known even in the United States. In company with some friends I discovered a lode, which we worked at first very successfully; but as we got to a greater depth our difficulties began, owing to the increase of iron pyrites in the ores, and the decrease of the "free gold." I then undertook a journey to Grass Valley, California (which in those days meant a journey in a stage coach of nearly one thousand miles, the great trans-continental railway not being then completed), where the first quartz mines were being opened up and worked on "scientific" principles. Messrs. Watts were then the owners of the celebrated Eureka Mine, and some Cornish miners, Coleman by name, were prospecting the extension called the Idaho. The Colemans are now millionaires, and the Idaho one of the best mines in California. I was kindly permitted to investigate the works, and it was with no little interest that I studied the first processes of Californian gold extraction. These processes, in the course of eighteen years' residence in many mining districts, I have seen in development from their infancy. A Mr. Deetkin had erected the first reverberatory roasting furnaces and chlorination works for working sulphurets, and a portion of the sixth chapter of this work is devoted to an account of his operations as described by the late Professor Guido Küstel. At the Eureka mill I found the concave buddle in operation for concentrating the rich slimes; the Eureka rubber, erected by Mr. Mellville Attwood, a renowned English mining engineer; the Attwood amalgamator (also named after him), and other improvements. These appliances were subsequently described by Mr. R. W. Raymond in his mining reports to the United States Government, and his descriptions of the apparatus will be found reproduced in this book. They were the first in use, and are still in operation.

The invention of other concentrating machines and grinding pans have, since that time, enabled quartz miners to improve the yield of ore; and among others the Frue vanner, Duncan's concentrator, and Hendy's concentrator, deserve to be mentioned. The construction of the mortars has also been perfected.

We have been taught that the first condition involved in the successful extraction of gold from the matrix is to gather and save the sulphurets and other metallic combinations in which it is held, and from which the various amalgamating processes have thus far been powerless to extract the gold. As only a certain percentage of the gold in quartz mines is present in the free state, the present millman, to make gold quartz mining a success, must depart from the old beaten path of "free milling" and amalgamation. We must look to new processes and new methods to make mines pay. The lodes consisting of pure quartz and free gold are getting scarcer and scarcer every day; and although they are occasionally found in new districts, yet after a certain depth is reached this free milling character changes. This involves a systematic procedure by which "low-grade" ores can be cheaply and quickly transmuted into high-grade ores, and when such transmutation has been accomplished they are ready for scientific metallurgical treatment, by which, through the aid of chemistry, they may be converted into ingots of pure metal, the base metals with which it is combined being at the same time brought into use and made to yield their proportion of profit to the intelligent miner.

The new processes and new methods by which these important results are to be brought about mean, first, simple and cheap methods of concentration—appliances by which, when low-grade ore has been mined, it may be quickly and cheaply denuded of the large part of its useless or waste materials (its *gangue*), the *free gold* it contains caught by quicksilver and amalgamated, and its other valuable mineral ore particles concentrated and treated by chlorination or otherwise.

It is a well-known fact that in most gold-producing countries where the operations for the extraction of gold from quartz veins have been carried on, the "brown" ores, as they are called, or the true "free milling" ores, are generally comprised in that portion of a gold-bearing lode or vein which lies above the perpetual water-line proper. They are that portion of a mineral vein from which the rebellious elements have been oxidized, leached out, or eliminated by influences which have been acting through a long course of years, leaving the gold free and clean, and readily susceptible of amalgamation by and with quicksilver in any of the various methods which have been adopted for its use for that purpose. Below the perpetual water-line these gold-bearing veins generally assume the form of gold carried in sulphurets or other combination, to which the application of the amalgamating process has always proved unremunerative.

The loss in working gold ores, even with our most modern appliances, is still enormous, and even at the present day not more than one-half of the gold contained in the ore which is worked, whether by free milling or otherwise, is saved.

One of the oldest and best authorities in California (Mr. A. B. Paul) has said upon this subject: "As far as California is concerned, I am satisfied that no more than 40 per cent. of her gold is exhausted. The fact is, we are not working to save gold but to crush rock." Again he says: "Our present general system of gold mining is based upon the idea that gold is mainly coarse, while examination will show that the high percentage is in atoms finer than flour itself. In my experiments gold has been taken up so fine that in distilled water it would not precipitate in less than from five to ten minutes. Can you save gold of this kind by running water down stream? Again, can you obtain gold of this kind without minute reduction? Therein lies the secret of high assays before working and small returns after." Or, as a mining authority once remarked, "The gold is so fine in the rock that it takes a bushel of it to make an ounce."

Mr. Deetkin, of Grass Valley, in order to determine the loss of gold by the mill process, has tabulated a series of assays made of the tailings of ore of the best mills in the State, which shows the loss to have been 40 per cent. of the yield, of which the float loss was nearly 14 per cent.

In Australia, during eighteen months' milling of 85.521 tons, and daily assay of the tailings, the yield being only \$6 52c., or £1 7s. per ton, the average loss was found to be \$1 56c., or 6s. 6d. per ton, or 24 per cent. rated upon the amount saved. Add to this the amount of float gold carried away in the water, and the total loss would not fall short of 40 per cent.

The actual average yield of ores milled and smelted, calculated from Fossett's table of seven years' work in Colorado, showed the average value of the ore by assay to be \$37 97c., or £7 18s. per ton. The average value per ton saved by milling and smelting was $14 50c., or about £3; showing a loss of $23 47c., or about £4 17s. per ton in gold and silver, or more than 60 per cent. The gold caught on the copper plates is, under the most favourable circumstances, only 50 per cent. of the assay value of the ore. The gold from the blankets and in the buddle concentrator does not amount to more than 5 per cent, so that when treating the most tractable of these sulphurets, battery amalgamation does not secure more than 55 per cent. of gold.

At the Merrifield mine in Deer Creek, Nevada County, a large proportion of the ore is heavily impregnated with sulphurets, and notwithstanding the fact that the rock by fire assay showed its value to be hundreds of dollars per ton, yet after working the quartz by the old process and in the most approved manner, including the use of Frue's concentrator, it was found that the average pay of the ore was about £5 per ton, including sulphurets. They are now satisfied that nearly two-thirds of the gold in the rock was lost by being carried oft in fine particles with the sediment and water.

Professor Egleston says: "In all the methods for the extraction of the precious metals there are considerable losses. With the perfection of processes the main object is to reduce them, or else to cheapen the labour of extracting the ores. These losses are greater than is usually supposed, because as a general rule systematic assays of the tailings are not made."

It was also found that water from the mills three-fourths of a mile below them contained in suspension, on an average of twelve assays, nearly one pennyweight of gold per gallon. There were in this locality 516,000 gallons of this water flowing away in twenty-four hours, or a loss of about £70 per day. It was estimated that the annual loss of two mills working 250 days in the year was £17,000. From these and similar data the conclusion is drawn that the loss is between 40 and 50 per cent. of the total yield of the ore.

The State of California, since the discovery of gold on Sutter Creek in 1848, has produced in gold over £250,000,000; and it is estimated that more gold has been wasted in milling and hydraulic mining, and lost to mankind, by being washed down the rivers and partly carried to the ocean, than has been thus produced.

A happier state of affairs exists in the silver regions. In Nevada I have worked ores which yielded in silver from 88 to over 90 per cent. of their assay value, and it is to be hoped that in another generation inventions will have been perfected by which a higher percentage of gold may be saved.

From what has been said above it will be seen that the problem of gold milling is not the easy matter which some may imagine, but that there is a wide field open for investigation. Much has no doubt been accomplished, but there is plenty of room for improvement yet in the methods and appliances for securing the much-coveted metal.

## II.—Gold as a Metal.

From the earliest times gold has been esteemed the most valuable of all the metals. It is not only distinguished for its rareness, but it possesses many very valuable properties not common to the other metals. It is especially valued for its richness of colour, and the fact that its brightness is not liable to tarnish.

The specific gravity of gold varies between 19·258 and 20·72, and its fusing point has been determined at from 2,200° to 2,596° Fahr.

**Gold in the pure state** is very soft, like lead, and may be easily scratched by the nail. It is more malleable and ductile than any other metal, and is capable of being beaten out into leaves $\frac{1}{250000}$ inch in thickness, while one grain of it can be drawn out into a wire 167 yards long. Its tenacity, however, is inferior to that of iron, copper, platinum, and silver. Its specific gravity is inferior only to that of platinum and a few of the rarer metals, such as iridium and osmium. In its natural state gold is almost always found alloyed with other metals;

the few compounds found being *sylvanite*, or telluride of gold and silver; and *nagyagite*, or telluride of lead with gold, silver, and copper.

Gold is known to exist in very small quantities in galena, in which, according to Percy, it occurs probably in the form of sulphide.

The distribution of gold is very general throughout the world, but at the present time the largest quantities are obtained in Australia, the United States, the Ural Mountains, and in parts of Africa.

It is almost always found in a metallic state; generally in small grains or scales known as *gold dust*; sometimes in particles so minute as to be invisible; and occasionally in pieces of considerable weight, called *nuggets*.

It is doubtful whether when gold occurs in pyrites it exists in every case in minute metallic particles, or whether in some cases it may not be present in combination with sulphur. The auriferous pyrites, as Dumas remarks, contain gold scattered throughout their mass in such small quantities that it is almost impossible, even with the aid of the microscope, to ascertain in what state the precious metal exists. As roasting the auriferous ore is considered useful with a view to its subsequent amalgamation, a doubt may be entertained whether gold is really present in the metallic state, notwithstanding that this has been generally supposed to be the case. Dumas, however, thinks that the powerful electro-negative tendency of sulphide of gold affords a strong presumption in favour of the hypothesis that this metal may exist in iron and copper pyrites partly, or even entirely, in the form of double sulphide. Brongniart observes, that it is chiefly when in combination with these sulphides—as also with galena or sulphide of lead, blende or sulphide of zinc, and mispickel or arsenical pyrites (a sulphide of arsenic and iron)—that the gold becomes invisible to the eye, a circumstance which is regarded as strongly confirming the opinion expressed by Dumas.*

---

* My own opinion is that gold in pyrites is not in a state of chemical combination, but is present in the metallic state, coated by sulphur, arsenic, &c.

The other minerals with which gold is found associated are grey cobalt, lithoidal manganese, native tellurium, malachite, sulphide of silver, red silver, and sulphide of antimony.

Gold is occasionally, though rarely, found in a crystallised state, in which form it is of great value.* The most common forms are those of the cube and octahedron, or some modification of these forms. The faces are generally dull, and in most instances the edges are slightly rounded.

It is sometimes crystallised by cooling, but when such is the case it is, I believe, more by accident than design, as it is impossible to regulate the rate of cooling. I have myself frequently cast ingots of gold which, when chipping off the corners for assay purposes, I found to be in a crystallised state. I made it a rule to remelt such ingots, and found that on adding a little nitre a tough metal was obtained. From this I concluded that the crystallisation was the result of some impurity in the gold; though I never sufficiently investigated the subject to be sure on the point.

**Composition of Gold.**—Native gold is never quite pure, being almost invariably alloyed with silver, and containing frequently small proportions of copper and iron. In Siberia it is often associated with platinum, and in the Gongo Saco Mines, in Brazil, an alloy of gold and palladium of a pale yellow colour is sometimes found. In Columbia a mixture somewhat similar is obtained, the palladium being replaced by another rare metal called rhodium.

The proportion of silver, which is the principal metal found in alloy with gold, varies from 1 to 50 per cent.; and it not only differs greatly in specimens of gold obtained from different localities, but differs even in specimens from the same district.

**Properties of Pure Gold.**—Gold does not combine directly with oxygen, and therefore suffers no change on exposure to air and moisture at whatever temperature; not even if it be

---

* It becomes valuable, that is, for use as specimens—not intrinsically—gold crystals being highly prized by collectors of minerals.

kept in a state of fusion in open vessels. Neither is it attacked by mineral or simple acids, except selenic, and then only by the aid of heat.

The alkalies have ordinarily no action on gold. If, however, the gold and the alkalies together become heated during exposure to the atmosphere, an absorption of oxygen takes place, resulting in the formation of an alkaline aurate. Nitre when in a state of fusion attacks it. The persulphides, whether in the wet or the dry state, also attack it, resulting in the formation of sulphide of gold. Iodine has a weak action upon it, while bromine and chlorine at ordinary temperatures easily attack it. It is dissolved by any substance which liberates chlorine; it is dissolved therefore by hydrochloric acid, it binoxide of manganese or chromic acid be added to it.

The conductivity of pure gold for heat and electricity is little more than half that of silver. If the conductive force of silver be taken as 100, we have for gold—

| | |
|---|---|
| Conductivity for heat | 53·2 |
| Conductivity for electricity at 27° F. | 55·19 |

When in a state of fusion gold exhibits a bluish green colour. It is not sensibly volatile in the strongest heat of a blast furnace; but under the focus of a large convex lens, in the intense heat of the oxyhydrogen jet, or under the influence of a powerful electric discharge, a gold wire is dispersed in vapour; and if, in this latter case, the wire be placed just above the surface of a sheet of paper, the course of the discharge is marked by a broad, dark, purple stain, produced by the finely divided gold. If instead of the sheet of white paper a plate of polished silver be employed, it is traversed by a brightly-gilded line, which is firmly attached to its surface. When a globule of gold is placed between the terminal charcoal points of a powerful voltaic battery, it becomes almost immediately fused, and gives off abundant metallic fumes.

The contraction of gold on cooling is very great; it cannot therefore be advantageously employed for castings.

**Gold-bearing Ores.**—These consist chiefly of quartz, and in some cases they contain slate, baryta, and talc. I remember one occasion when gold was found in limestone. The precious metal is usually scattered through the rock in small particles, sometimes so minute as to be scarcely visible to the naked eye. Sulphides of iron, lead, copper, and zinc, as well as arseniurets, are frequently found with the gold. In pyrites the gold appears to exist in a metallic state, as it can be separated in some measure by grinding in the amalgamating pan.

The amalgamation of gold has been brought to a certain state of perfection by Californian millmen; and their methods of treating the gold-bearing ores have been very generally adopted in Australia and other countries. These methods are known as the "Californian stamping process," and this work is mainly concerned with an account of the several operations comprised in it, and of the machinery and appliances required in working it; but a description may be appropriately given here of the "hydraulicking" operations which form so interesting a feature in the history of gold mining.

## III.—Gold Washing by the Hydraulic Process.

By this process, or "hydraulicking," is understood the removal of large masses of gravelly soil from its original *locus* to another place of deposit by the impact of water under a very high pressure.

In California and Australia there are large deposits of this gravelly soil containing gold, and extending throughout whole mountain ridges, but not in sufficient quantity to pay, unless the washing be carried out on a gigantic scale. As a rule, these alluvial gravels, such as constitute the hydraulic mines, are not rich throughout the upper strata, and the gold is usually found concentrated in layers near the strata reposing on the country rock, designated in mining parlance "bed-rock." In some instances the bottom stratum of gravel is so rich in gold and the upper layer (which in cases reach a height of 200 to 300 feet) so poor, that it would prove unprofitable to remove the whole mountain mass, and this led to the working of the rich gravel beds by means of shafts and tunnels, whereby only the bed-rock stratum is excavated and washed, and the remainder left standing. Such mining is called "drift-mining."

In hydraulic mining, the whole depth of the deposit down to the bed-rock is washed away by means of powerful jets of water. The water disintegrates the soil, which in some cases is very hard, and during its onward journey through bed-rock tunnels and sluices submits the soil to a thorough system of comminution and trituration, which liberates the free scales and small lumps of gold. These sink by their superior specific gravity in the sluice boxes, and are collected therefrom after a run of thirty or forty days of such continuous washing.

Such a process requires a large volume of water, which has to be brought in canals or ditches, or aqueducts, from the fountain heads located in the mountains at a suitable elevation. The construction of these water conduits involves very large outlays of money and labour, as some of them extend from forty to one hundred miles in length.

The next point to be considered in hydraulicking is the removal of the large quantity of detritus resulting from the washing away of the huge masses of gravel in which the gold has to be sought—from the removal, one might say, of whole mountain ranges. But when once the proper application of the great mechanical force furnished by a large volume of water under high pressure was grasped, it was seen that the problem could only be satisfactorily solved if the auriferous gravel could be carried to some distant point and there discharged into a rapidly flowing stream. For accomplishing this object a hydraulic mine has to be provided with a tunnel or "open cut" from the nearest and most suitable ravine or river bank, so that, starting in the bed-rock, on the face of the ravine, or other selected point, it shall approach the centre of the gravel mass to be moved at a gradient of one in twenty-four to one in thirty-six. The dimensions of the tunnel are usually six feet in width and seven feet in height, sometimes wider.

The tunnels vary in length from a few hundred feet to a mile. The end of the tunnel is designed to reach beneath the under surface of the gravel in the centre or deepest part of the channel, at a point where a shaft or incline is sunk from the surface through the gravel until it intersects the tunnel. It obviously demands careful engineering to carry out works of such magnitude with the accuracy required, and, for the want of sufficient care or skill in this particular, years of costly labour and anxious expectation have been wasted in the early history of these enterprises.

It is therefore necessary in laying out an hydraulic mine operation to take precautions, by an accurate survey, (1) to ascertain the head or fall of water and its quantity; (2) to secure an outlet for the washed débris or tailings; and (3) to strike the lowest point of the channel with the bed-rock flume or tunnel. Outlet, dump, and fall have to be secured for the many millions of cubic yards of gravel displaced, and many valuable mines have had to suspend operations from their inability to get rid of the accumulated débris, which had " banked up " at the outlet of the flume (or water course) and stopped further washings.

When the tunnel has been completed and connection made with an opening in the gravel deposit, the tunnel becomes a sluice way, through the whole length of which sluice boxes are laid, to direct the stream and save the gold. For this purpose a trough of strong planks is placed in the tunnel, from three to four feet wide, with sides high enough to control the stream. The pavement is usually composed of blocks of wood from six to eight inches in thickness, cut across the grain of the wood, and so placed as to expose the end of the blocks to the wear of the current. The wooden blocks are usually alternated with sections of stone pavement, the stones set endways. In the interstices quicksilver is distributed, as much as two tons of this metal being required to charge a long sluice. The location and inclination of the sluices have an important

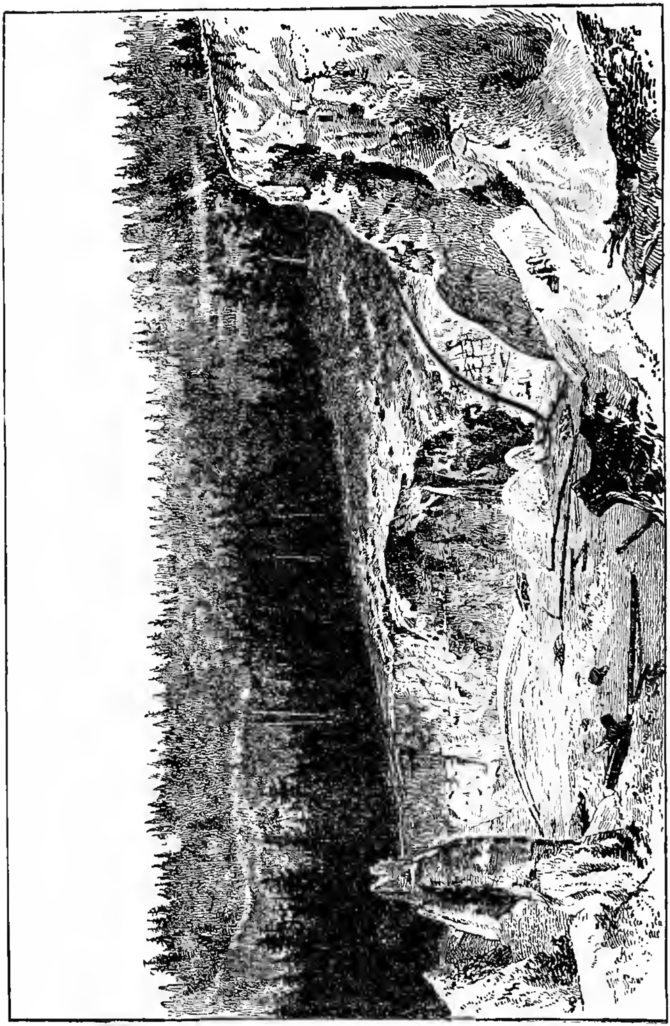


Fig. 1.—Hydraulic Mining on the Georgetown Divide, El Dorado County, California.

bearing on the results which are obtained in gravel washing and the proportion of gold saved, and proper attention has to be paid to the grade of the sluice boxes.

I will now revert to the removal of the gravelly soil by the force of the jets of water brought to bear upon it. The water is conveyed to the mining claim from the ditch or canal by means of iron pipes, the point of the supply being from 200 to 500 feet above the point of discharge. These iron pipes have to be carefully laid, and are often sustained on an incline of timbers. The water can of course be carried in pipes over depressions in the ground, and over intervening elevations, and so across valleys and mountains. The iron pipes are of great strength; they are made riveted at the joints, and measure from 12 to 20 inches in diameter. A pipe of $\frac{3}{8}$ inch wrought iron can sustain a pressure of 385 pounds per square inch.

The pipes are connected with a powerful apparatus of sheet iron provided with a universal joint, to which the outlet or "nozzle" is attached, ending in a steel ring for the delivery of the water, which varies from 4 to 8 inches in diameter, as greater or less volume of water is used. From these nozzles the streams are directed against the face of the gravel "bank" to be washed with a force comparable to that of ordnance. The volume of water employed varies, of course, with the work to be done, but it is not uncommon to see four such streams, each conveying 300 to 600 inches of water, or more, acting simultaneously on the face of the same "bank." The accompanying illustration (Fig. 1) depicts this stage of the operations. One thousand "miner's inches" of water are equal to 106,600 cubic feet of water per hour, constantly discharged under a pressure of 100 to 200 pounds to the square inch, varying with the height of the column, or pressure. Under the continuous action of this enormous mechanical force, aided by the softening influence of the water, large sections of the gravelly mass come crashing down with great violence.

The débris, speedily dissolving and disappearing under the resistless force of the torrent of water, is hurried forward in the sluices, precipitated with the whole volume of now turbid water down shafts, or dumps, prepared for the purpose, and then taken up again by the sluices, where it is brought by means of undercurrents, riffles, and other appliances in contact with the quicksilver, and thus made to give up its gold. Boulders of 100 to 200 pounds in weight are shot forward by the impetuous stream, together with masses of the harder cement, which meet, in the fall and concussion of the great boulders, the crushing agencies required to disintegrate them.

The deeper banks of gravel are usually worked in two benches. The upper is never so rich as the lower, and, being also less firm, is worked away with greater rapidity. The lower section is much more compact: the stratum on the "bed-rock," being strongly cemented by sulphuret of iron and great pressure, resists even the full force of the water stream, until it is loosened by gunpowder. For this purpose adits are driven in on the bed-rock 40 to 70 feet from the face of the bank, and a tunnel extended at right angles therefrom to some distance each side of the adit. In this tunnel a large quantity of gunpowder is placed, from one hundred to one thousand kegs, and fired as one blast by fuse or electricity. In this manner the compact conglomerate is broken up, and the water then rapidly completes the work. This disintegration of the gravel banks by means of explosives is called by miners "bank blasting."

The sluices are often made double, for the convenience of "cleaning up" one of them while the other is in action. The process of cleaning up is performed, according to the extent of the works and the richness of the material washed, at intervals of from fifteen to thirty days, and consists in removing the pavement and blocks from the bed of the sluice, and gathering the amalgam of gold and "rich dirt" collected in them, and replacing the blocks in the same way as at first. Advantage is also taken of the opportunity to reverse the position of the blocks and stones forming the pavement, where they are worn irregularly, and to substitute new ones for those worn through.

The mechanical action of the washing process on the blocks is, of course, rapid and severe, so as to require a complete renewal of them once in eight or twelve weeks. Some miners prefer a pavement of egg-shaped stones set like cobble-stone pavement, the gold being deposited in the interstices. Most of the sluices, however, are paved with rectangular wooden blocks, with or without stones as described above.

Rude as this method of saving the gold appears, experience shows that more gold is won by it than by any other method of washing yet devised; while the economical advantages it offers are incomparably greater. In fact, it would be impossible to handle so vast a body of material in any other way.

Among the noticeable features of California are the muddy rivers, which largely owe their turbid condition to the operations above described. These minor streams discharge into the Sacramento, which in its turn empties into the Bay of San Francisco, building up great bars with its burden of gravel. The damage done to farming lands in the lower portions of the Sacramento valley was, indeed, so great from the accumulation of débris, that legislative measures had to be taken to compel the hydraulic miners to impound their "tailings" and not discharge them into the rivers, with the result that many valuable mines had to stop working.

# CHAPTER II.

## *TREATMENT OF GOLD-BEARING ORES: CRUSHING AND AMALGAMATION.*

Section I.—Mechanical Preparation of Gold Ores.—Rock-breaking Machines—Blake's Patent—Reduction of the Ore by Stamps—Arrangement of a Battery described—Feeding the Batteries—Tulloch's Ore Feeder—Operation of the Stamps—The Capacity of a Stamp Mill.

Section II.—Amalgamation of the Free Gold.—Attwood Amalgamator—Hydrogen Amalgam Process—Grinding Pans—Knox Pan—Wheeler Pan—Soderling Pan—Settlers and Rubbers—Eureka Rubber—Gauthier's Shaking Table—Bazin's Amalgamator—Treatment of Amalgamated Copper Plates—Silver-plated Copper Plates.

Section III.—The Process of Amalgamation.—Operation of the Mill—Capacity and Regulation of a Stamp Mill—Professor Raymond on Crushing—Aprons, Sluices, and Blankets—Water required for Milling—Process of Amalgamation—Commencing Operations at a Mill—Amalgamation in the Battery—Treatment of Tailings—The Edison Process—Conditions of Good Result.

### I.—Mechanical Preparation of Gold Ores.

Before subjecting the ore, or the auriferous gangue, to the chemical or metallurgical process by which the gold is extracted, it has certain mechanical operations to undergo, the number and nature of which will be determined by the character of the ore itself, and its state of aggregation. Nature—unfortunately for the operations of the metallurgist—has distributed this most precious metal in the rocks in a very fine state of division, the rock being impregnated with it in the shape of dust, spangles, or small grains, which on the whole are difficult, very difficult indeed, to separate; and it is only in extremely rare cases that gold is found in nuggets which require only to be cast into ingots. Occasionally rich specimens are found in which the separation of the gold is easily effected.

A large proportion of our gold has been, and is still, obtained by the direct washing of alluvial deposits. This book, however, only treats of the extraction of gold from rocks or ores, and I begin the present chapter with an account of those operations by which the rock is artificially reduced to the state of an auriferous sand or pulverised ore.

The treatment of gold-bearing rocks comprises the following operations:—

1. Crushing it by means of rock-breakers and stamps, and amalgamating the free gold inside and outside the battery by various contrivances.

2. Concentrating the free gold lost during crushing, and amalgamating it in pans with the auriferous sulphurets; and

3. Extracting the gold contained in pyrites by roasting and chlorination.

The ore—which in California is always called "quartz," no matter what the gangue may be—is first passed through a rock-breaker, where the large lumps are reduced to pieces about the size of a walnut or even smaller, and is discharged upon a platform immediately in front of the battery of the stamping mill, or into self-feeders, from which the quartz is fed very regularly into a large oblong, rectangular, cast-iron mortar, in which five immense pestles play up and down, pulverising the quartz into very fine particles. As these pestles weigh from seven hundred to a thousand pounds apiece, in an establishment (called a mill) which works from twenty to sixty stamps, the noise is deafening, and the mill is not exactly the place for nervous persons.

After the quartz is pounded fine, we wish to get rid of the worthless waste rock, and for this purpose a stream of water flows into the mortar which dilutes the fine material, and while it is held in suspension by the water, it is splashed by the agitation caused by the stamps through an oblong slit on one of the longitudinal sides of the mortar, which is closed by a sieve. The fine particles of rock form with the water a very thin mud, which is called slime or pulp.

The question now arises, What becomes of the gold? The particles of gold which have been liberated during the process of trituration, either—by reason of their greater specific gravity—sink to the bottom of the mortar, where they come into contact with mercury and become amalgamated; or they are splashed against the sides of the mortar—which are lined inside with amalgamated copper plates—and are there "caught;" or they are splashed out of the mortar through the sieve along with the slime.

On emerging from the mortar the slimes pass over an apron of amalgamated copper plates, by which means a considerable quantity of metal is removed, as gold will adhere to these plates, and is thus prevented from passing down the sluices which immediately succeed the apron. The sluices are covered with blankets, and are set on an incline, and the gold and heavy ore particles collect between the fine hair of the blankets. These are washed every few hours in large wooden tubs, and the stuff collected on them—called "concentrations"—is treated in amalgamators, where the free gold is extracted.

The slimes, after passing the blankets, and also those from the amalgamators, discharge into the Eureka rubber, in which any particles of gold which have escaped are cleaned and brightened by rubbing, and are to some extent collected by means of amalgamated copper plates. The impoverished sands and slimes, after passing from the blankets, sluices, rubbers, &c., are discharged into concentrators, in order to collect small particles of gold enclosed in pyrites and other metallic sulphides, which would otherwise be lost. How the concentrations are treated will be subsequently described. In some mills the amalgamating apparatus is differently arranged.

Having given the rationale of the general method of dealing with gold-bearing ores, I will now enter upon a description of the various machinery and processes employed.

Before it can be submitted to the final pulverisation in the stamp mills, it is necessary that the quartz should be broken into small fragments somewhat uniform in size, and for this operation powerful rock-breaking machines are employed. Many satisfactory contrivances have latterly been introduced for economical and effective rock-breaking, but Blake's machine (which I will describe in detail) is most extensively used.

**Rock-Breakers.**—*Blake's Patent.*—This machine is shown in the illustrations (Figs. 2 and 3) in vertical section and perspective view. In Fig. 3, *a a* is a heavy frame, cast in one piece, with feet to stand upon the floor or on timbers, which

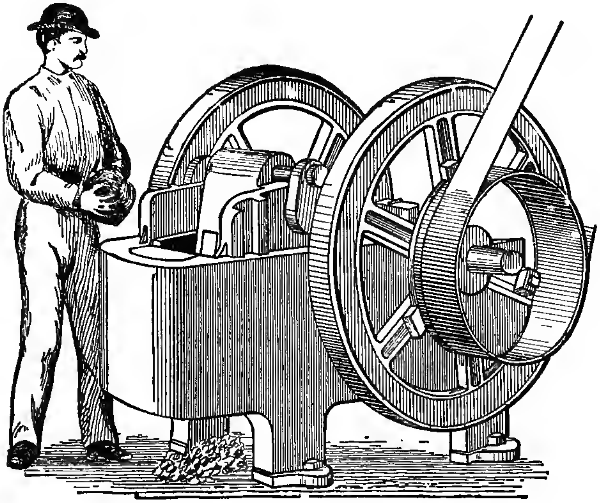


FIG. 2.—BLAKE'S ROCK-BREAKER. Perspective View.

frame supports all the other parts; *b* is the fly-wheel, one on each side, the shaft of which is formed into a crank; *c* is a pulley on the same shaft; *d* is a rod connecting the crank, *e*, with the lever, *f*: this lever has its fulcrum on the frame, *g*. A vertical piece, *h* (a front view of which is shown at B), stands upon the lever, and against its top the toggles, *i i*, have their bearings, thus forming an elbow or toggle-joint. *k* is a fixed jaw against which the stones are crushed. The jaw is held in its place by keys, *l*, which fit in recesses in the interior of the frame on each side; *m* is a movable jaw, faced with a corrugated die, *m'*. This jaw is supported by a round bar of iron, which passes freely through it and forms the pivot upon which it vibrates; *o* is a spring of india-rubber, which, being compressed by the forward movement of the jaw, aids its return. C is the lever viewed from above. E is a top view of the die, *k*, which is corrugated on one or on both sides. In the latter case it is turned after one side is worn out; *p* is a screw by which the wedge, *r*, can be screwed up, in order to bring the jaw, *m*, nearer to *k*, and thus effect a finer crushing.

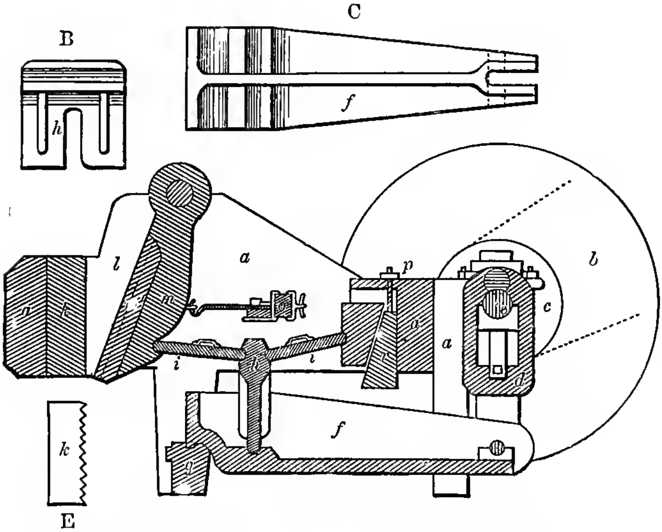


Fig. 3.—Blake's Rock-Breaker. Vertical Section.—Scale ½ in. to 1 ft.

The crank, *e*, revolves from one hundred and fifty to two hundred times per minute. The distance between the jaws may be raised to five-eighths of an inch by the screw, *p*. Other variations may be made by changing either or both of the toggles for longer or shorter ones.

**Capacity of Rock-breakers.**—A rock-breaker, of suitable dimensions, making 170 strokes of ¾ of an inch a minute, is capable in twenty-four hours of giving a preliminary crushing to 72 tons of quartz, which is the crushing capacity of a heavy 30-stamp mill. By preparing the rock for the battery, the rock-breaker increases the crushing capacity of a mill by 20 per cent.

Only the coarser quartz is passed through the breaker; the finer, which always contains a quantity of wood splinters brought from the mine, being crushed in a separate battery by itself. The splinters lessen the efficiency of the battery by clogging the screen holes, which latter require, therefore, a greater amount of attention to keep them in order.

**Reduction by Stamps.**—No machine effects the reduction of ores so well as the stamping mill. The construction of stamp mills varies very much, especially in their foundation frames, but in their main points they all agree, the work done by stamps being in principle the same as that of a pestle and mortar. There are generally five crushing stamps employed in one mortar. In California one mortar with its stamps is called a "battery." The frame of the battery is generally made of timber.

**Arrangement of a Battery: the Frame.**—For the purpose of erecting a battery a pit is sunk to the solid rock, the bottom of which is nicely levelled off for the reception of the mortar blocks, *a a*, Fig. 4. The block has a section of 3 by 5 ft., and rarely exceeds 10 ft. in length. It is set on end, and consists of two logs, 36 by 30 in. each, firmly bolted together with 1¼ in. bolts, and connected also with dowels 18 in. long, and 8 by 8 in. in section, introduced 2 ft. from each end. The horizontal section of the pit has usually a space of 24 in. all round the mortar block, filled up to the level of the mud sills with hydraulic concrete to within 4 to 5 ft. of the top of the mortar block.

After the top of the block has been carefully riveted, levelled, and planed, it is ready for the reception of the iron mortar, which is fastened to the block by means of 1¾ in. key bolts, *k k*, passing through the bottom flanges of the mortar (Fig. 4).

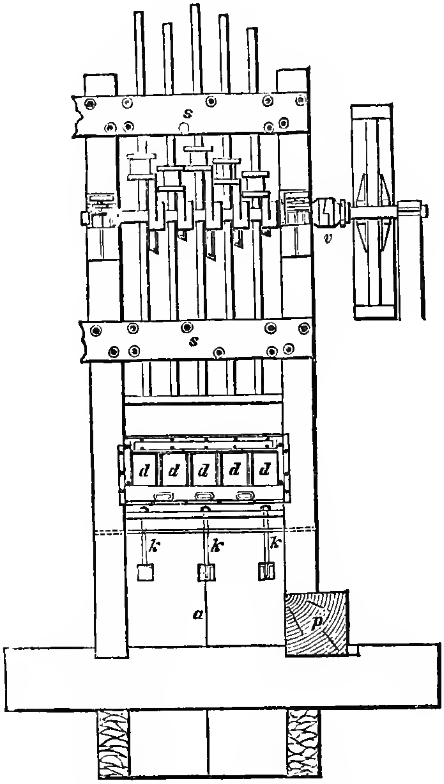


Fig. 4.—Front View of Battery.—Scale ⅙ in. to 1 ft.

The wooden battery frame which carries the cam shaft is independent of the mortar block. The woodwork is usually of sugar pine. The mud sills, *l l* (Fig. 5), 24 by 24 in., and usually three or four in number, are laid parallel with the cam shaft. The sills on each side of the mortar block are secured by iron bolts, which pass through the hydraulic concrete and have an iron anchor plate underneath. (See Fig. 5.)

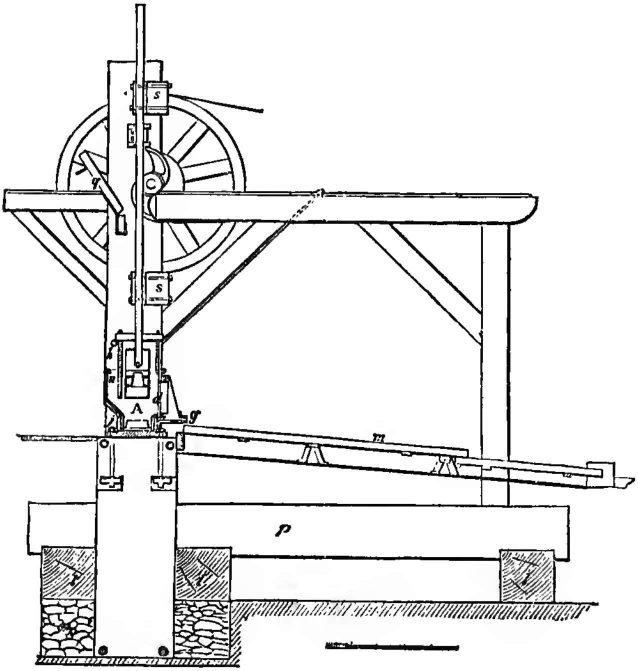


Fig. 5.—Transverse Section of Battery. Scale $\frac{1}{6}$ in. to 1 ft.

Cross sills, *p p*, 18 in. wide by 24 in. deep, and 18 ft. long, which are at right angles to the mud sills, carry the main posts, 21 by 24 in. in section, and to these posts the cam shaft is attached. The cross sills are let and keyed into the posts in the manner shown in Fig. 4.*

**The Foundation.**—When the ground is marshy or otherwise unreliable, deep excavation is not desirable, a broad base only being required for the frame foundation, as shown in Figs. 6 and 7. The surface of the ground must be dug out from 1 to 3 ft. in depth, and with as much width and length as the condition of the ground and the number of batteries may require. The illustrations show a foundation having a horizontal mortar block for two batteries. Fig. 6 is a front view and Fig. 7 a side view. Flooring underneath the timber with boards is improper, and in case the bottom appear unsafe the lower timbers must be laid close together. The bottom timbers, *a a*, are 12 in. square and 18 ft. long, and are laid for the required distance upon the carefully levelled ground. Between *a a*,

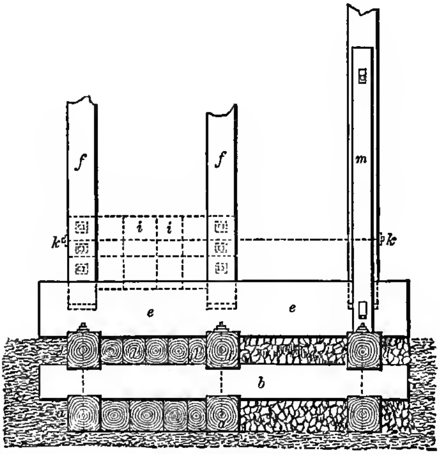


Fig. 6.—Elevation of Battery Frame. Scale $\frac{1}{6}$ in. = 1 ft.

---

* An illustration of an iron battery frame (20 stamps) will be found at p. 119.

in the centre, short pieces about 2 ft. long are laid, so as to fill up the space under the double cross timbers, *b′ b′*. There are generally as many beams, *a*, *a*, *a*, as there are uprights, *f*.

The cross timbers, *b b*, are laid upon *a*, and over *b* a row of beams, *d*, corresponding with *a a*. The mortar block, *e*, completes the horizontal frame. The latter is 20 to 30 in. square, and its end is secured by an iron band. Oak is of course the best material for these blocks, but owing to its comparative scarcity, pine blocks are more generally used. Their length naturally depends on the number and size of the batteries.

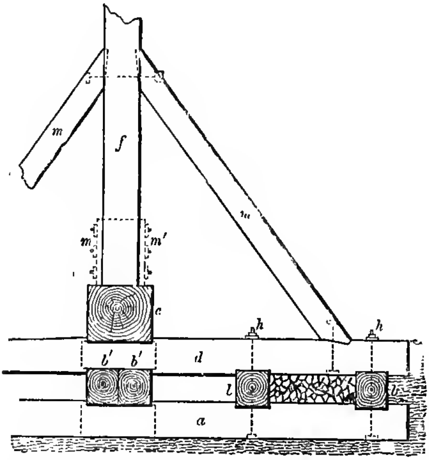


FIG. 7.—SECTION OF BATTERY FRAME. Scale $\frac{3}{16}$ in. = 1 ft.

To give greater firmness to the foundation framework the timbers are fastened together by bolts, *h*. The space between them is filled and beaten in with clay, loam, or, still better, with coarse gravel and stones, as shown in the Figs. 6, 7. When the ground becomes consolidated at a depth of four or five feet, the whole frame is set so much deeper, in which case short vertical pieces (as indicated by the dotted lines, *i*) are fitted to the mortar block and bolted together lengthwise (as shown at *k*), whilst their lower ends are let into the block. The planks, *m'*, prevent lateral displacement of the blocks, *i*.

Horizontal mortar blocks are also advantageously used on hard rocky ground, in which case, however, excavation is both difficult and expensive. In this case too the use of less timber is necessary. The ground in this case may be considered an artificial stone block, as represented in Fig. 8. The mortar block lies on the beams, *b b*, and these latter rest on the bottom pieces, *a*.

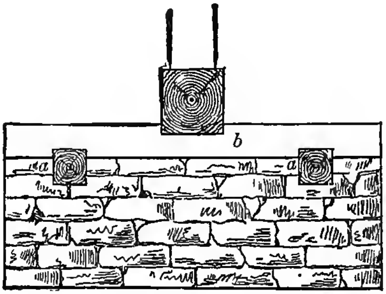


Fig. 8.—Stone Foundation for Battery. Scale $\frac{3}{16}$ in. = 1 ft.

It contributes much to the steadiness of the frame if the bottom timbers are anchored to the rocks with iron rods; but in case heavy stamps are used, a foundation such as is shown in Fig. 8 will, if the space between the timbers is filled with rocks, be found perfectly satisfactory.

Instead of wood braces iron rods are frequently used, or iron and wood together, as shown at *m n* in Fig. 9.

The advantage of iron rods is that they take up less room than wood braces, and are capable at the same time of being, if necessary, tightened. The advantage of wood braces is that they insure greater steadiness than iron rods. Fig. 10 shows the wooden frame of the battery in elevation.

**Mortars.**—Fig. 12 represents a mortar having a single discharge, and designed for copper plate lining back and front,

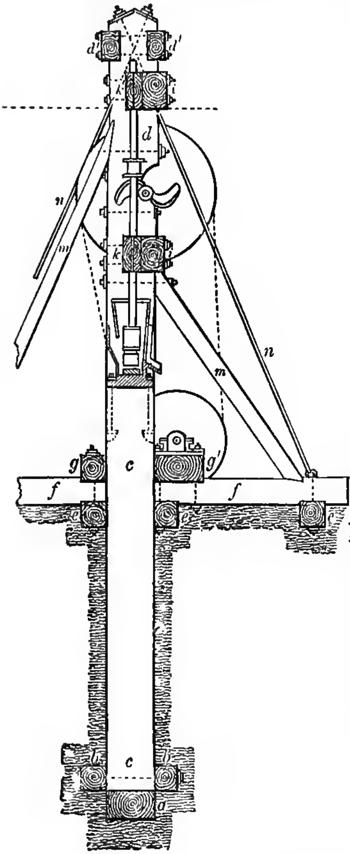


FIG. 9.—ARRANGEMENT OF STAMP STEMS.

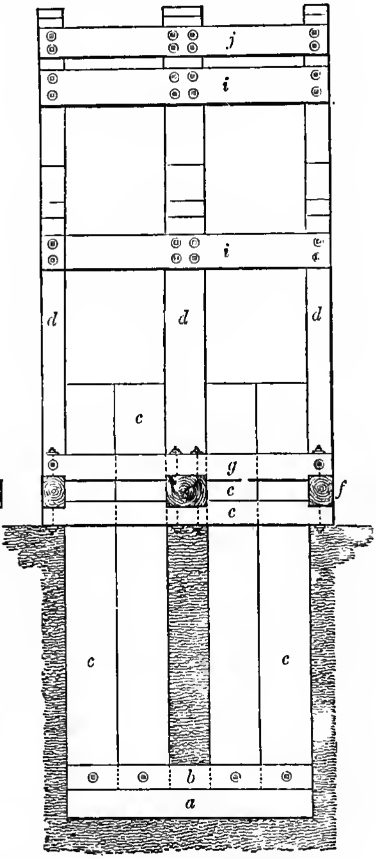


FIG. 10.—COMPLETE BATTERY FRAME.

Scale ⅛ in. = 1 ft.

the former bolted through the mortar and the latter fastened to a block under the screen frame. The die, shoe head, and a part of the stem are shown at rest in the mortar. Fig. 11

represents a mortar having a single discharge, and designed for copper plate lining in front only. It is especially adapted for crushing base ores needing concentration and fine granulation in order to separate the mineral from the rock. The discharge of the pulp is regulated by the fineness of the screens.

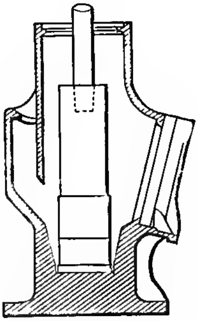


FIG. 11.—MORTAR FOR FRONT LINING.

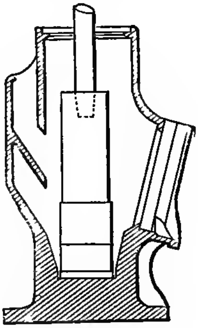


FIG. 12.—MORTAR FOR FRONT AND BACK LINING.

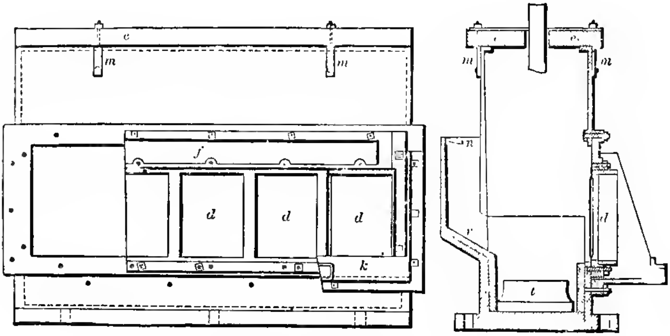


FIG. 13.—MORTAR WITH SIEVE FRAME. Scale $\frac{3}{8}$ in. = 1 ft.

The mortar (Fig. 13) consists of a single casting 3 ft. 9 in. high. The bottom is usually 3 in. thick, but sometimes more.

The walls are vertical on the outside, 1½ to 1¾ in. thick on the bottom, tapering to ½ in. on the top. The bottom flanges for fastening the mortar to the block are 2½ in. thick and 4 in. wide. In order to prevent the premature destruction of the mortar near the dies, cast-iron plates, *r*, 1 in. thick and about 24 in. high, are placed as a lining all round the inside, which can be replaced when worn out. These being wedge-shaped at the lines of contact with each other do not need any additional fastening.

At one of the sides of the mortar, opposite the discharge opening, *d*, is situated the feed slit, *n*. It is 3 in. in width and from 6 to 12 in. below the top of the mortar, and extends across the whole width of the mortar. A rim 3 in. wide is planed off round the discharge opening, *d*, for the purpose of receiving the sieve frame (Fig. 14) and the splash box (Fig. 25), both of which are of cast iron, and are bolted to it. The horizontal mortar bed receives the dies, *t*, the bottom or foot plates of which fit almost close to the sides of the plate lining of the mortar.

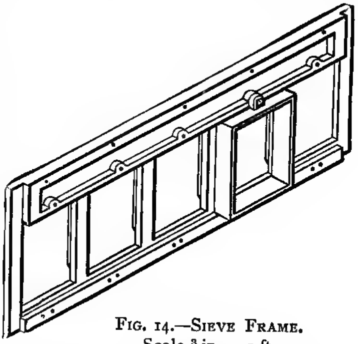


FIG. 14.—SIEVE FRAME.
Scale ⅜ in. = 1 ft.

**The Die.**—The foot plate, *a*, of the die (Fig. 15) is of a rectangular form, and its corners are cut off for the purpose of facilitating its removal in cleaning the battery. It is 1½ in. thick, and carries a cylindrical piece, *b*, 3½ in. high, the diameter of which is usually 10 in. When worn down to the foot plate it is replaced by a new die. Each battery consists of five stamps, which when new are all of a uniform weight.

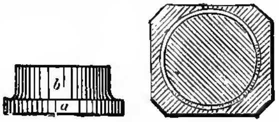


FIG. 15.—THE DIE.
Scale ⅜ in. = 1 ft.

**The Stamp.**—The stamp consists of four cylindrical pieces: the head or boss, the stem, the tappet, and the shoe.

**The Head** (Fig. 16) is of the same diameter as the shoe, and is from 15 to 20 in. long. Its lower face is provided with a conical recess, *f*, 6 in. deep, which receives the shank of the shoe, and a tapering core, *d*, to receive the foot of the stem.

To facilitate, in case of breakage, the detachment of the shoe and stem, channels, *m m*, *o o*, are provided below it for the insertion of wedges, which pass easily through the boss.

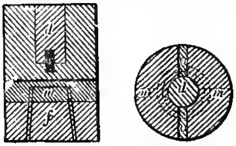


FIG. 16—THE HEAD.
Scale $\frac{5}{8}$ in. = 1 ft.

**The Stem** (*a*, Fig. 17) is of wrought iron, and varies in length from 13 to 15 ft., and in thickness from 3 to $3\frac{1}{4}$ in. It is turned true in the lathe. Both ends of the stem are tapered off, so that either may serve as a foot.

**The Tappet** (*r r*, Fig. 17) is in the main a cylinder of cast iron, from 9 to 11 in. in diameter, and 10 in. long, having a central bore. It is slipped on the stem and fastened to it by means of a wrought-iron gib, *b*, keyed up by wedges, *n n*, at right angles to the gib. The projection, *o o*, prevents the slipping of the gib, which would cause an uneven wear of the cam face. The gib is introduced into the mould previous to casting.

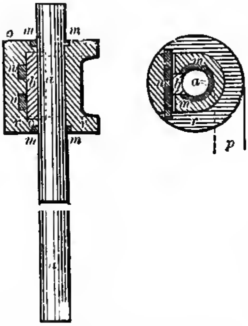


FIG. 17.—STEM AND TAPPET.
Scale $\frac{5}{8}$ in. = 1 ft.

The tappet is slipped on the stem, and wedged above and below by means of the wrought-iron gib, *b*, which is 2 in. wide, and may be as long as the tappet, or shorter. Its inner face forms a part of the circular hole which receives the stem. *n n* are the key seats, into which the gib, *b*, slightly projects, so that if a key be driven into the seat it will press the gib against the stem, and so fasten the tappet to it. The tappet may in this way be easily moved to any point which the wear of the shoes may from time to time make necessary. The tappet when worn on one side may be reversed, as both sides are alike. The use of too much grease or oil prevents the revolution of the stamp, and, as a consequence, not only does the surface of the tappet become damaged, but the advantage of a uniform wear of the shoe is destroyed.

As every part of the working face of the tappet becomes worn by the cams—with the exception of a concentric ring $\frac{1}{2}$ in. wide round the stem (the cam passing within half an inch of the stem)—an annular recess, *m m*, 1 in. deep, and concentric with the stem bore, but 1 in. longer, is bored out on the working faces to prevent the wear of the edges of the cam. Both faces of the tappet are used as working faces, and when they are worn down to the extent of an inch they are replaced by new ones.

**The Shoe** (Fig. 18) is generally made of white iron, cast wholly in sand and slowly cooled; but it is sometimes made of steel. In the Californian stamp mills the form of the shoe is generally round.

The cylindrical part or butt, *a*, is $8\frac{1}{2}$ in. long; the shank, *b*, being one-half the diameter of the butt, has the same length, and tapers upwards conically. The diameter of the butt varies from 9 to 11 in. The head is fastened to the stem by laying two strips of thin cloth crosswise over the stem head, and then driving the stem home. The tappet is then keyed fast to the stem, and the shank of the shoe, which is surrounded by small wooden wedges pointed upwards and held in position by a string, is set vertically under the shank hole of the boss. In falling, the stamp wedges the shank of the shoe firmly into the shank hole, after which two or three blows are given at the top of the stem with a sledge hammer. The battery is then allowed to play slowly, so that the stamp may be lifted six or eight times while the others are at rest. The shoe is not allowed to strike on the naked die; a piece of board or plank is therefore placed between the shoe and the die. The lining formed by the strips of wood must have sub-

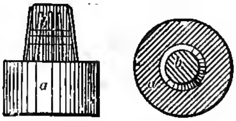


FIG. 18.—THE SHOE.
Scale $\frac{5}{8}$ in. = 1 ft.

stance enough to prevent the edge of the head coming in contact with that of the shoe. The shoe should be renewed when worn to the extent of an inch.

**Weight and Power of the Stamp.**—The weight of stamp varies from 600 to 900 lbs., and is generally between 750 and 850 lbs. The weight is 10½ lbs. per square inch of crushing surface. The shoes are placed 1 in. apart, and 1 in. from the linings of the mortar. The proportional weight of the stem, head, shoe, and tappet, respectively, to the total weight of the stamp is as 40 : 29 : 16 : 15 : 100. The weight of the die when new is usually $\frac{15}{100}$ of the weight of the stamp.

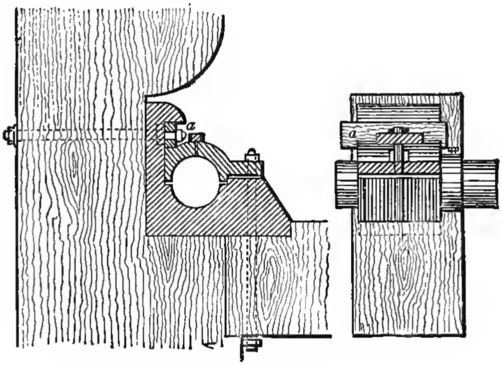


Fig. 19.—The Journal Boxes.—Scale $\frac{5}{8}$ in. = 1 ft.

The effective mechanical power per second of the stamps in a battery is found by dividing the product of the number of stamps, the weight of one stamp, the lift in feet, and the number of lifts per minute, by 60 seconds. The power applied on the shaft must be greater than the effective power, on account of the friction of the stamps in the guides, as well as of the cams and tappets and the shaft journals, and also on account of the jarring of the whole machinery, which practice has determined at from 1½ to 1¾ horse-power per stamp, according to the weight of the stamp.

The **Cam Shaft** is of wrought iron, and generally 5 in. in diameter when turned in the lathe. It is provided with two key ways, 1 by ¾ in., which for the purpose of insuring the normal position of the cams are placed at right angles to each other. The centre of the cam shaft is usually 5 in. from the centre line of the stems. The shaft is placed 9½ ft. from the mortar bed.

The weight of the stamps should be distributed on the shaft as equally as possible. It will be found an advantage to power and machinery to have one shaft for several batteries, as in this case if there be, say, thirty or forty stamps in the mill, they can be all arranged on the shaft in such a manner as to insure that no more than one stamp shall catch the tappet at the same time, in which case the shaft will bear the same weight at each moment of the revolution. If the same number ot stamps were driven by, say, six or ten separate shafts, each connected with a main shaft, this latter shaft would have to bear from four to six times as much weight at one moment as at another, depending of course on the time of starting the separate batteries.

On the other hand, there are reasons which make it inadvisable to use one shaft for several batteries. For instance, the stamps may each be easily and independently arrested in the several batteries; but should a cam get loose, all the batteries will have to be idle while the repairs are going on; while still more inconvenience would arise if one of the cams should break, as in this case all the cams in the way must be first removed. For this latter reason cams fitted directly to the shaft are not so good as those screwed or wedged on rings or collars. All things considered, a middle course would seem to be the best. Two five-stamp batteries with one cam shaft are better for the machinery than two shafts.

The position on a shaft of the cams for two five-stamp batteries is shown in Fig. 20. A is a section of the shaft; B, a front view showing perpendicular dotted lines, 1 to 10, which indicate the ten stamps of two batteries. The half-circumference of A (the cams being double) is divided into ten equal parts, commencing at 1. Through these points of division parallel longitudinal lines, 1, 2, 3 to 10, are drawn, and on the perpendicular line of each stem the dots *a*, *b*, *c*, *a′*, &c., are marked as follows, corresponding with the dots in the front view :—

| | | | | | | |
|---|---|---|---|---|---|---|
| First | line, | first | cam, | for the first | battery | = *a* |
| Second | ,, | first | ,, | ,, second | ,, | = *a′* |
| Third | ,, | fifth | ,, | ,, first | ,, | = *e* |
| Fourth | ,, | fifth | ,, | ,, second | ,, | = *e′* |
| Fifth | ,, | third | ,, | ,, first | ,, | = *c* |
| Sixth | ,, | third | ,, | ,, second | ,, | = *c′* |
| Seventh | ,, | second | ,, | ,, first | ,, | = *b* |
| Eighth | ,, | second | ,, | ,, second | ,, | = *b′* |
| Ninth | ,, | fourth | ,, | ,, first | ,, | = *d* |
| Tenth | ,, | fourth | ,, | ,, second | ,, | = *d′* |

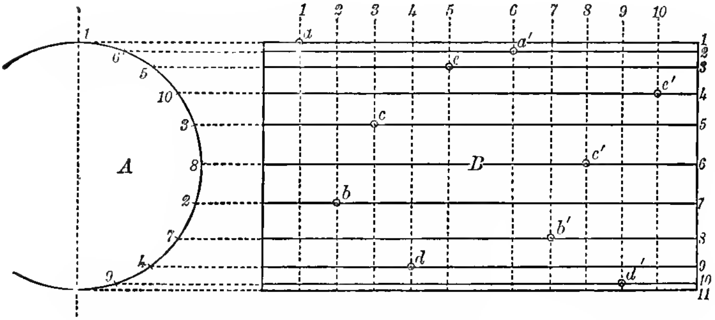


FIG. 20.—CAM SHAFT.

The points of division upon the circumference of A indicate the positions of the key-seats, which are transferred on to the cams, and seen in Fig. 21. The first key-seat on the first cam of the first battery, on the line *h*, is marked exactly in the same line on all of the ten cams. The distances from 1 to 2, then from 1 to 3, 4, and 5 (as seen in Fig. 20) are correspondingly transferred proportionately from 1 (on the line *h*) on to the second, third, fourth, and fifth cam of the first and similarly on to the following five cams of the second battery. When fastened on the shaft, and having a long common key-seat, the cams will stand as shown by the dots *a*, *b*, *c* in Fig. 20.

**The Journal Boxes** are constructed as shown in Fig. 19.

The journal cap is held down on the feed side by a wooden wedge, *a*, which is secured by a bolt passing through a slit, to allow, when necessary, of its being tightened.

The leading principle to observe in connection with lifts is, first, to distribute the weight of the stamps equally on the shaft; and, secondly, to have the next stamp on each side begin to rise before the middle stamp drops, so that the rock may be thrown on to both the right and the left dies.

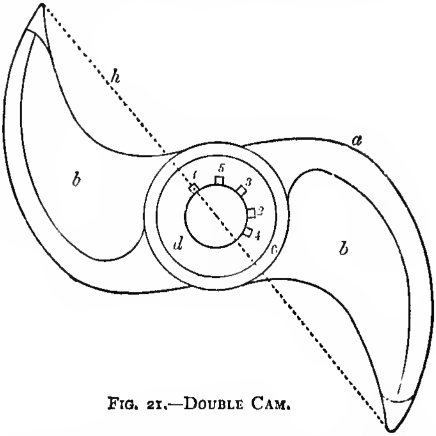


Fig. 21.—Double Cam.

**Flange Coupling.**—On the cam shaft is a bevel clutch or toothed coupling, *v*, Fig. 4, by which the cam shaft may, by means of a fork or bevel, be disengaged from the pulley shaft. It also fills the important function of preventing the breaking of the cams and the bending of the stems in case of a reverse motion of the cam shaft.

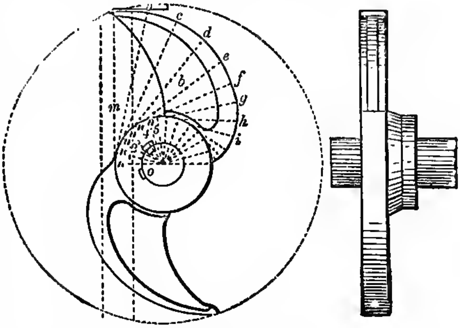


Fig. 22.—Cams and Cam Curve.—Scale $\frac{5}{8}$ in. = 1 ft.

**The Cams** (Fig. 22).—These are of cast iron, having a face of 3 in. and a depth of 1 to 2 in. They are strengthened by a rib, *b*, 1¼ in. thick and increasing in depth towards the hub, which latter has a thickness of 2½ in. The hub is always on the off side of the stem, and therefore allows the cam shaft to closely approach the centre of the stem. The stamps are lifted by means of the cams. The cam face is generally constructed in the form of an involute curve, as by this construction the same point of the tongue or tappet is, during the lift of the stamp, raised vertically and uniformly, the lift of the stamp always having a proportion to the motion of circle described by the cam. Two cams placed opposite each other (Fig. 22) are always attached to one hub, which is strengthened by a wrought-iron band.

In California the cams for revolving the stamps are double. Single cams are in less frequent use, and their purpose is to increase the number of lifts, doubling the speed of the shaft, and, of course, necessitating additional power.

The construction of the cam is shown in Fig. 22. A *n* shows the distance from the centre of the cam shaft to the centre of the stem; *n a* the length of the greatest lift, plus the distance from the centre of the cam shaft to the first front of contact of the cam with the tappet. The curve, *b a*, is changed to *b a′*; *a a′* is in this case ⅜ in. The lift varies between 8 and 11 in., and is generally 10 in., with a corresponding cam curve of 21½ in. The friction between the cam and the tappet causes the stamp to revolve, thus insuring an even wear of the shoe.

The mortar (Fig. 13) is covered with 3 in. planks, *e*, held in position by the bolts, *m m*, which are attached to the side of the mortar. They join in the centre line of the stems, and are provided with semi-cylindrical grooves to receive the stems.

**The Guides**, which are of sugar-pine, are 15 in. deep, and 10 in. broad. They are firmly bolted to the uprights which carry the cam shaft. The centre of the upper guides is 3 ft. 4 in. above, and the centre of the lower guides 4 ft. below, the centre of the cam shaft. A wood lining, consisting of two 3 in. planks, 15 in. deep, with semi-cylindrical grooves to fit and enclose the stems, is bolted on to the main guide-timbers. The complete arrangement is shown at *s s*, in Fig. 4.

**Discharge in Wet Crushing.**—The discharge of the crushed ore takes place at one of the long sides of the mortar. The discharge generally takes place either through a slit or a longitudinal opening. For coarse and fine crushing—that is, the production of fine sand, not slime—the best arrangement is to use grates or screens. The discharge or sieve frame, which is made of cast iron, is divided into five panels (Fig. 14), each measuring 11½ in. by 15 in.

**The Screen** is of nearly the same size as the panel, only exceeding it by about ½ in. all the way round, and is bolted close to the panels by means of a cast-iron frame, 3 in. deep (Fig. 23). In order to make the joint tight, a blanket binding is sewn round the edge of the screen.*

For the screens punched copper or iron plates are mostly used. Frequently brass or steel wire cloth is used, which produces a more effective discharge, as the surface of a wire cloth has more openings of the same size than the plates. The screens generally in use are made of Russian sheet iron (Fig. 24), weighing nearly a pound per square foot. The rectangular holes, which are punched by machinery, are ⅜ths of an inch long, and wide enough to allow the passage of a No. 6 sewing needle. The holes are ¾ths of an inch apart longitudinally, and their centres ⅝ths of an inch apart vertically, thus leaving a space of ¼th of an inch between them. The turned-up edge faces are on the inner side of the mortar, and when the edges are worn down the holes are closed or made smaller by pounding the edges with a mallet.

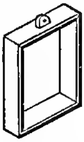


FIG. 23.—SCREEN FRAME.

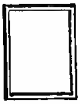


FIG. 24.—SCREEN.

Scale ⅜ in. = 1 ft.

Slot and needle punched sheet-iron screens are made of soft but tough Russian sheet iron, and the sizes most in use are Nos. 30 and 40, which means 30 or 40 holes per linear inch, and 900 or 1,600 holes per square inch. Punched Russian iron

---

[*] In the more modern mills the screen is nailed to one frame, and I find that screens 7 inches high and the length of the discharge opening answer all purposes.

screens are the best to use, as they will last, on an average, thirty days, but their area of discharge is not so great as with brass-wire screens.

The screens are generally set at a slight inclination, and a piece of canvas is hung over the screens in front of the mortar, against which the pulp splashes as it passes through the screen.

Ores which carry pyrites, and which are saved by concentration, should not be crushed too fine, as the mineralised portion of the ore is usually softer than the gangue, and therefore will be pounded so fine as to produce pyritous slimes, which concentrators will not be able to collect.

With such ores coarse stamping ought to be resorted to with a rapid discharge. It is very difficult in stamping to produce an even-grained material, as an unusually large percentage of fine is always produced even when using coarse screens such as a 30 screen, and it will be found that over 80 per cent. of the pulp will pass through a 60 screen, and a very large portion will pass through an 80, 100, or even 120 screen.

Screens are sold according to the number of needle used in their punching, which determines their number. Thus—

| | | | | | |
|---|---|---|---|---|---|
| Needle No. | 5 | corresponds to | 20 | mesh | per linear inch. |
| ,, | 6 | ,, | 25 | ,, | ,, |
| ,, | 7 | ,, | 30 | ,, | ,, |
| ,, | 8 | ,, | 35 | ,, | ,, |
| ,, | 9 | ,, | 40 | ,, | ,, |
| ,, | 10 | ,, | 50 | ,, | ,, |
| ,, | 11 | ,, | 55 | ,, | ,, |
| ,, | 12 | ,, | 60 | ,, | ,, |

In stamping, the discharge ought to be so regulated that the gold particles should not remain on the die too long, so as to be pounded too hard, as it becomes hardened and loses its porosity, and so amalgamation is prevented.

To prevent choking, wooden hammers are sometimes connected with the shaft, striking the screen at regular intervals. Without this or some other arrangement answering the purpose, the sieves must be brushed from time to time.

**The Splash-box**, which is of cast iron (Fig. 25), is provided with three discharge spouts, and is tightly bolted, with the aid of blanket packing, to the mortar.*

Sieves in an inclined position give a better discharge than those placed perpendicularly.

In some of the mills of California and Nevada the discharge through the screens takes place in dry and wet crushing at the two long sides, but it is usual for the discharge to be on one side only.

The dies are placed from 5 to 6 in. below the discharge. There is very little difference in the method of crushing.

The manner of bracing the uprights of the cam shaft is seen in Figs. 5 and 10. As the strain of the belt is in the direction of the discharge, no bracing is needed on the feed side, which gives therefore a clear working space on that side of the battery.

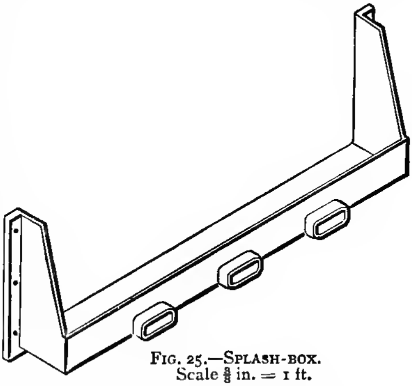


FIG. 25.—SPLASH-BOX.
Scale ⅜ in. = 1 ft.

The stamps are usually raised for repairing by means of tackle suspended on a roller strap on an overhead beam. They are kept in suspension by the props, *q* (Fig. 5).

**The Supply of Water** to the battery is effected by an iron pipe, *h* (Fig. 5), passing over the feed opening near the top of the mortar. The vertical discharge is through small apertures, which can be closed by wooden plugs. There is also a 1½ in. water pipe running along the discharge side of the battery, to allow for an increase of the quantity of water over the blanket sluices.

The quantity of water in the battery necessary to discharge

---

* The splash-box is now abandoned in most of the more modern mills.

the ore depends on the quality of the ore. Pure quartzose ore requires less water than clayey stuff. Coarse-grained ore or gold will bear more water than that in a fine condition. The quantity of floating metal and slime increases with a decrease in the quantity of water. The stamping of muddy or clayey ore sometimes requires more water than is necessary for a proper discharge, in which case water may be conveyed to the mass outside the battery. The settling of the particles of the ore is more perfect when the pulverised mass is in a sufficiently diluted condition. When crushing quartz rock in the usual way with a No. 4 screen, each stamp requires per minute from one-half to three-fourths of a cubic foot of water.

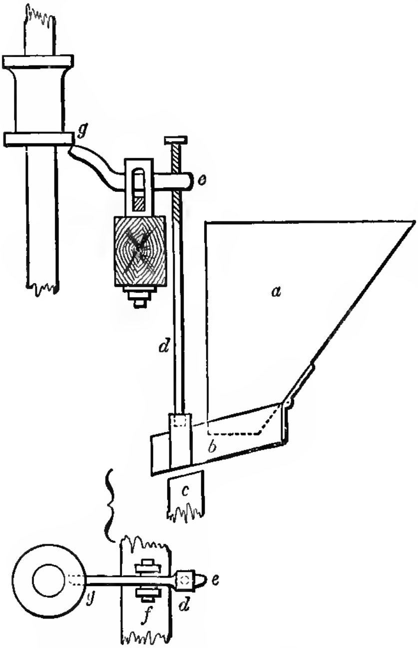


Fig. 26.—Self-Feeder.—Scale ⅜ in. = 1 ft.

**Feeding the Battery with Ore.**—This operation is performed either by hand or by some mechanical contrivance. Hand feeding is still in favour in California and Nevada, notwithstanding that a great number of mills are provided with self-feeders. The stamps always regulate the feed where self-feeders are employed, as is mostly the case.

**A Self-Feeder** is represented in Fig. 26. The feed-box, *a*, of which there are generally two to each five-stamp battery, is fixed, and has a movable shoe, *b*, resting on the support.

The rod, *d*, is connected with the lever, *e*, which rests on a half-round pin, *f*. The space below *f* serves to regulate the position of the lever. It is filled with a wooden piece, placed higher or lower according as the wear of the shoe of the stamp may require. By the screw thread, *d*, the slide of the shoe, *b*, may be altered. The depth of the box, *a*, varies according to the quantity of rock it is required to contain. Small boxes are usually preferred, notwithstanding that there appears to be an advantage in having to fill a feed-hopper only once instead of perhaps four or six times a day. Fig. 27 shows Tulloch's ore-feeder, of similar construction.

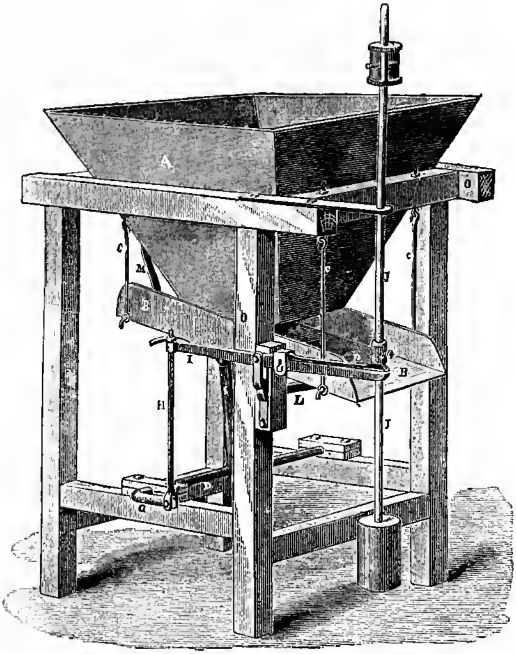


FIG. 27.—TULLOCH'S ORE-FEEDER.

**Tulloch's Ore-Feeder.**—In this machine (Fig. 27), A is the hopper into which the crushed ore from the stone-breaker is discharged; B, the shaking tray for feeding the ore into the mortar-box; C, C, C, are suspension links for carrying the tray; D G is a rocking shaft which imparts a rocking motion to the feeding tray through the arm E; the lever I P is centred at L and is connected to the shaft D G by means of the link H; by means of the rod J, the top end of which comes in contact with the tappet of the centre stamp (a hole being bored through the lower guide block of stamps for this purpose), a motion corresponding to that of the stamps is given to the lever I P; O is the framework of the machine, and M is a strong steel spring for giving a sharp recoil to the tray B, thereby insuring a proper movement of the ore over the same.

These feeders are exceedingly simple and compact, and save all hand labour in feeding the battery. They can be regulated to feed any quality or quantity of ore. If the dies are deeply covered with ore the stamp does not fall through its full height, and consequently little motion is imparted to the shaking tray; if, on the other hand, there should be no covering of ore on the dies, the tappet will travel through the full height of its fall, and the rod J will move in proportion.

Proper feeding lies between these two extremes, and therefore the rod J leading to the lever I P should be so adjusted that there shall always be a thin layer of ore on the dies, and thus avoid loss of power from unnecessarily pounding the ore.

**Recent Battery Frames and Appliances.**—In Plate I. is represented the framework of a battery erected by Messrs. Fried. Krupp Grusonwerk on the Queen Mine, near Johannesburg. Distances and dimensions being both given, the diagram will be found specially interesting and useful.

Plate II. shows a wooden-frame battery, erected by the Sandycroft Foundry, near Chester; and Plate III. a ten-head gravitation stamp battery, erected by the same firm.

Particulars of various appliances of recent introduction (1895) will be found in Chapter XXII., *post*.

## II.—Amalgamation of the Free Gold.

**The Attwood Amalgamator** (Fig. 28) consists of two hollow cylindrical troughs, *t t*, 17 in. long and 4 to 5 in. deep, and made of wood or iron. These are filled with clean quicksilver, over which the blanket washings are directed. The gold, being heavier than the quicksilver, will sink to the bottom, except that part attached to the quartz or sulphuret, which is borne to the top.

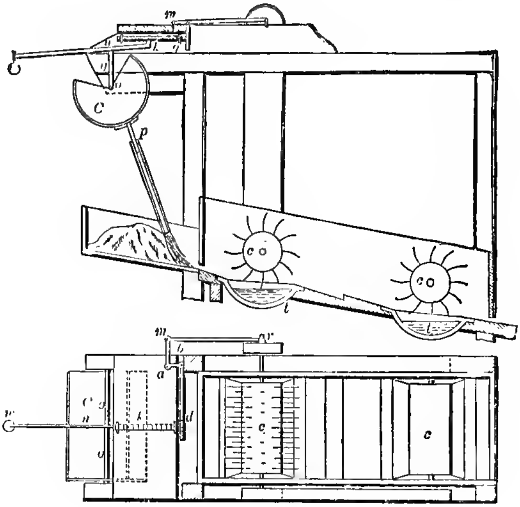


Fig. 28.—Attwood Amalgamator.

The surface skimmings are agitated by wood cylinders, *c c*, 8 in. in diameter, suspended parallel to and over the centre line of the trough, and provided with radial arms of $\frac{1}{4}$-in. round iron, whose ends are slightly curved. These arms are placed along the cylinders in twelve longitudinal rows, consisting alternately of eight and nine arms, those of each row being set opposite the spaces in the next. They are not allowed to dip into the quicksilver, but they almost touch it. The cylinders are, between the centres, 2 ft. 10 in. apart, and are 6 in. below each other. They make sixty revolutions per minute, and are driven by small belts. The arrangement for insuring a steady flow of the sand from the blanket through the amalgamator is shown in the figure. The connecting-rod, *m r*, receiving a horizontal movement from the crank, *r s*, transfers it by means of the bent lever, *m a b*, to the ratchet, *b d*, which moves, by means of the wheel, *d*, the endless screw, *k*. The endless screw, *k*, gives a rotary motion to the discharge pipe, *p*, which is attached to the water trough, C, by means of a lever, *n*, connected by the rod, *j*, with the water trough shaft. By raising the weight, *w*, suspended at the end of the lever, *n*, the female screw at the opposite end of the lever, *n*, can be placed at any point along the endless screw, and a change in the inclination of the pipe may thereby be effected. The female screw on the lever, *n*, is part of a diametrically cut nut. On reaching the point, *g*, the endless screw ceases to move the lever, *n*.

Fig. 29.—Riffle Sluices.

One amalgamator treats the blanket washings of two 5-stamp batteries. The quantity of water passing through an amalgamator is one cubic foot per minute.

The tailings from the amalgamator pass through wooden riffle sluices (Fig. 29), which are generally two in number. They are each 9 in. wide, and have an inclination of 1 in. to the foot. The riffles are 6 in. apart, and at the lower end are from $\frac{5}{8}$ in. to $\frac{3}{4}$ in. deep. They are filled with pure quicksilver and present a bright surface of from 2 to 3 in. in length by 9 in. in width. There are generally from twenty to thirty riffles to one sluice. The skimmings from the amalgamator and the quicksilver riffles, which vary from ·1 to ·5 per cent. of the crushed sands, are slowly ground and amalgamated in pans, which insures, with a minimum loss of quicksilver, a satisfactory result.

**Hydrogen-Amalgam Process** (*Molloy's Patent*).—The principle involved in this process is, that when gold is brought into absolute and maintained contact with clean or "quick mercury," the gold is absorbed by and retained in the mercury. As will be shown later on, where refractory ores have to be dealt with, they cause the mercury to "sicken"—that is, to become coated with an oxide which lies like a sheet of paper on the surface of the body of the mercury, preventing contact between the particles of gold and the clean portion of the mercury. This sick mercury also powders away—or, as it is termed, "flours"—so that the floured fluid metal is carried away and lost, leaving fresh surfaces to be attacked by the injurious ingredients in the ore, and causing a loss of mercury as well as of gold.

The object of the hydrogen-amalgam process is to save the loss of mercury and of gold by maintaining the "quickness" or "brightness" of the mercury, and by insuring a continual contact between each separate particle of the pulverized ore and the quick mercury.

The apparatus* consists of a shallow pan about one inch in depth, and 41½ inches in diameter, which contains mercury about half an inch in depth. In the centre of this pan is a porous jar, so placed and fixed that the mercury cannot enter or move it. Within this jar is a cylinder of lead and a solution of sulphate of soda. This lead cylinder, which constitutes the anode, is connected with the positive pole of a small dynamo machine, while the mercury is connected with the negative pole of the same dynamo.

When the current passes, oxygen is evolved from the surface of the lead anode, while hydrogen is evolved from the surface of the mercury. This action, which is apparent to the eye, is of course due to the decomposition of the electrolyte formed by the solution of the sulphate of soda. The mercury combines with a portion of the hydrogen, and so forms a hydrogen-amalgam, while the excess of hydrogen so formed passes away. Now, while the mercury is thus charged with hydrogen it cannot oxidize, because of the presence of an

---

* I am indebted to the Hydrogen-Amalgam Company for the information here given concerning this apparatus.

excess of hydrogen. Thus, no matter what the character of

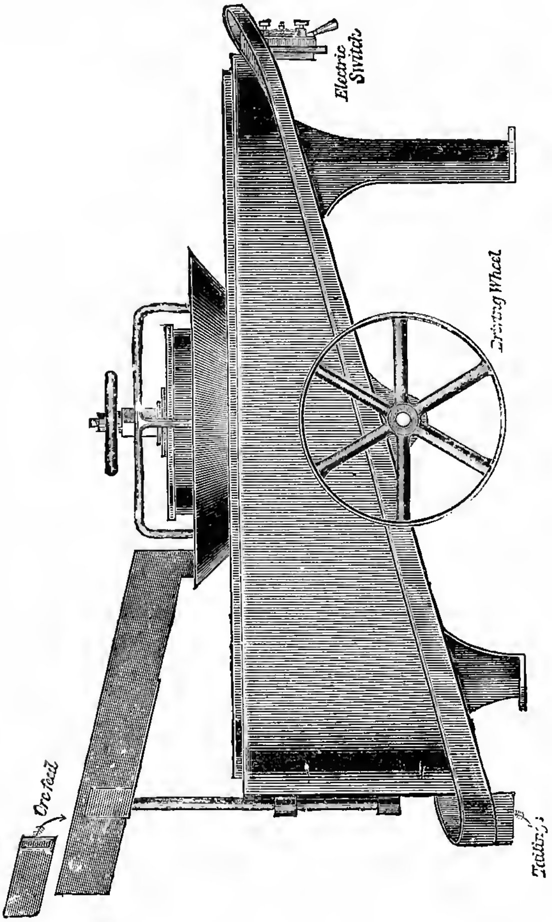


FIG. 30.—HYDROGEN AMALGAMATOR (MOLLOY'S PATENT).

the ore, the mercury, under these conditions, is always quick and therefore capable of attacking and absorbing the gold into itself.

For bringing the pulverized ore into contact with the mercury, the following method has been devised by the inventor. Floating upon the surface of the mercury is a disc forty inches in diameter, which dimension leaves a narrow outside channel all round the edge of the pan where the mercury is uncovered. The centre of the disc has a circular hole in it so as to clear the porous jar by about two inches. This central opening in the disc has a rim about two inches high which forms a hopper. The disc, as it floats on the mercury, is slowly revolved by simple mechanism. The pulverized ore, as it leaves the stamps or other crusher, flows into the hopper accompanied by a stream of water, and is then by centrifugal action carried under the revolving disc and rolled round in the mercury in ever-increasing circles until it reaches the periphery of the disc, and consequently the outward channel between the edge of the disc and that of the pan. Here, freed from the pressure of the disc, the pulverized ore floats up and over the edge of the pan and passes away, leaving behind it in the mercury every atom of amalgamable gold it previously contained. This perfect extraction is due to the rolling action which separates each particle of the ore and rolls it for some ten seconds in the bright quick mercury, which takes up in this way the *amalgamable gold.*

It is stated that the *increased* quantity of gold extracted by this process has never been less than 10 per cent., and that the whole cost of treatment amounts to only 3d. per ton for electrical and mechanical force and for labour.

Each amalgamator is said to be capable of treating from six to ten tons in twenty-four hours. The pulverized ore goes direct to the amalgamator from the battery. The maximum speed must not exceed 14 revolutions per minute, or mercury may be thrown out. It is recommended to crush very fine when using this machine—from 80 to 120 mesh per linear inch. The quantity of mercury for each amalgamator is 350 pounds.

**Pan Amalgamation.**—In many mills the heavy concentrates which accumulate on the blankets or other contrivances are submitted to amalgamation in grinding pans, of which several varieties are in general use. Two of the best known are the Knox and the Wheeler pans.

**Grinding Pans.**—(1.) *The Knox Pan.*—This pan is very much in use, and is capable of grinding, in three separate charges, the skimmings of a 30-stamp mill in twenty-four hours. Fig. 31 illustrates this pan in vertical section and plan. It is of cast iron, and is 4 ft. in diameter and 14 in. deep. The sides are ½ in. and the bottom ¾ in. thick. The pan is supported by four cast-iron legs, *l l*, bolted to the floor, and attached to the pan similar to a stove leg. On the legs are projections which support the cross-bars bearing the driving shaft. The vertical pinion-wheel can be put in or out of gear by a bevel clutch worked by lever.

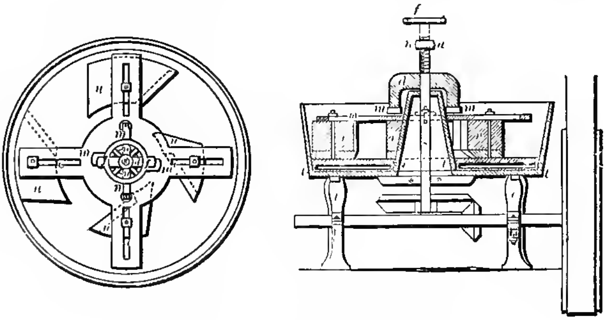


FIG. 31.—KNOX PAN.—Scale ¼ in. = 1 ft.

The pan has a false bottom, *t*, 1¼ in. thick, with a projecting vertical rim at the periphery, to form a hollow annular space underneath for the introduction of steam. There is also a radial groove in the false bottom, 1⅛ in. wide and 1 in. deep, for the accumulation of quicksilver and amalgam, which connects with the lower discharge hole situated opposite the driving shaft. The upper discharge hole is 4 in. above the lower one. The holes are closed by wooden plugs 2 in. in diameter.

The centre of the yoke, *d*, attached to the muller, *m*, is keyed to a vertical wrought-iron shaft, *s*, 2 in. in diameter, which latter is guided by a hollow cast-iron cone in the middle of the pan. The muller consists of a flat ring made of cast iron, having an inside diameter of 10 in. and being 4½ in. wide and 1 in. thick. Attached to the muller are four arms, 12 in. long and 6 in. wide, placed at right angles to each other, to which cast-iron shoes, *u*, 1½ in. thick, are bolted through the slits *c c*.

Between the muller and the shoe, a wooden shoe, *r*, having exactly the same shape as the iron one, and about 6 in. high, is introduced to prevent the unground pulp from settling on the latter, the upper face of the wooden shoe reaching above the surface of the pulp. The head of the bolt, passing through the shoe and muller arm, fits into a recess in the bottom of the iron shoe, and wears off gradually with it.

The yoke, *d*, bolted to the muller, serves the purpose of raising or lowering the latter by means of the screw, *f*, which rests on the vertical driving shaft and passes through the centre of the pan.

The muller makes from twelve to fourteen revolutions per minute, the force for which is communicated from the horizontal shaft by bevel-gearing which makes from thirty to thirty-five revolutions per minute.

**Process of Pan Amalgamation.**—This is effected by charging 300 lbs. of the skimmings or blanket washings into the pan, and adding water until the pulp just adheres to a stick dipped into it without dropping off. Then the pulp, after three hours' grinding, is heated by steam under the false bottom of the pan. At the same time is added a cupful of a mixture consisting of equal parts of saltpetre and sal-ammoniac. About 50 lbs. of mercury for every charge is also added simultaneously with this mixture.

After being amalgamated for three hours, the pulp, then very fine, is diluted with water. A few handfuls of caustic lime are added at the same time, which greatly assists the coagulation of the particles of quicksilver. The diluted pulp, which reaches within one or two inches of the top of the pan, is agitated for about twenty minutes, after which it is discharged, the muller being kept in motion, through the upper plug hole, and, subsequently, through the lower. A bucket, which is placed in front of the discharge holes, catches any quicksilver or amalgam which escapes during the discharge. When a clean up is required, the quicksilver and amalgam and the small quantity of sulphides still remaining are washed into the bucket. Quicksilver is then added and the skimmings removed,

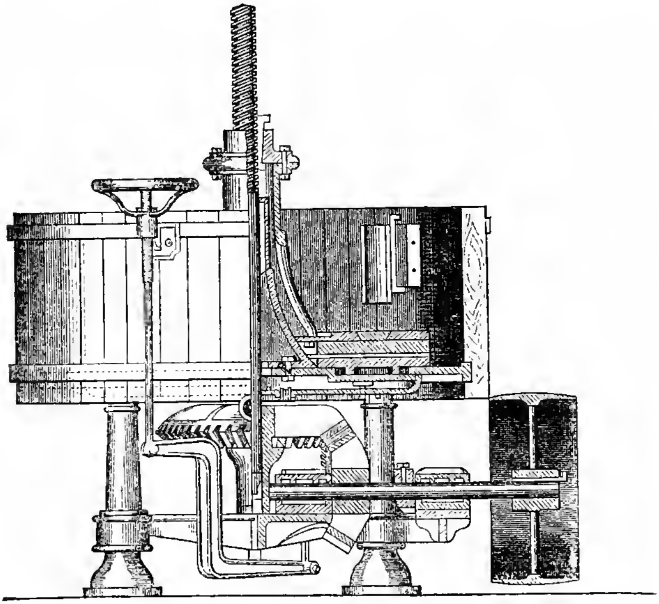


FIG. 22.—WHEELER PAN WITH WOODEN SIDES.

which go back again to be re-ground. The amalgam is worked by hand and the lumps broken up. The impurities on the surface are removed and the bright quicksilver strained through a canvas filter.

The pulp from the pan is run into a large tank to settle, and in many cases it still retains a considerable quantity of gold, amounting sometimes to a value of £25 per ton, which is added to the coarser sulphurets and treated by a process to be hereafter described.

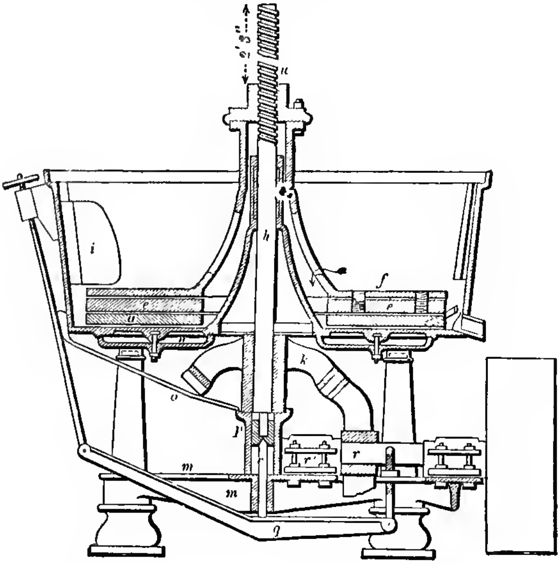


FIG. 33.—SECTION OF WHEELER PAN.—Scale ½ in. = 1 ft.

(2.) *The Wheeler Pan.*—Figs. 33 and 34 show one of these improved pans. Fig. 33 is a cross section on the line 2 of the ground view, Fig. 34, one part of which exposes the dies, and the other part the muller with the shoes, shown by dotted lines.

The pans have a cross piece, *m*, in which the box, *r'*, for the horizontal shaft, *r*, the step box, *p*, and the lever, *q*, are fixed, which vary only in shape and arrangement. The lever, if screwed up by the hand wheel, lifts, by means of the pin, *v*, the box, and with it the perpendicular shaft and muller. The upright shaft, *h*, is provided with a screw, *u*, by which the muller is screwed up so high above the rim of the pan that cleaning and changing of the dies may be conveniently performed. By this arrangement the hoisting apparatus is avoided. The guide plate, *i*, or, as it is called, the scraper, of which there are four in a pan, is short, which gives to the moving pulp sufficient direction towards the centre, and may be readily removed from the T-shaped projection of the pan; *o* is a pipe which conveys oil to the box on which the hub of the gear, *k*, runs; *n* shows the steam chamber for heating the pulp. The muller, *f*, is provided with two openings, *b b*, to receive the dovetailed projections of each shoe, as shown in Fig. 35.

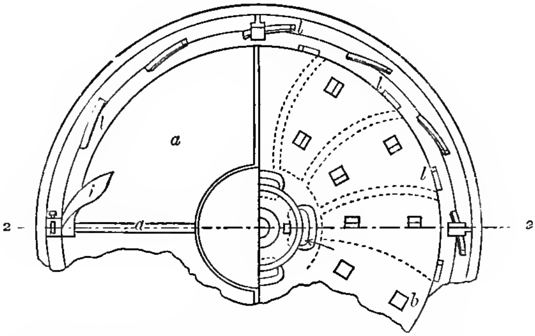


Fig. 34.—Ground View of Wheeler Pan.—Scale ½ in. = 1 ft.

There is a very simple method of fastening the shoe, *e*, to the muller. *c* is a vertical section on the line *a*; the shoe is placed under the muller, so that the projections, *b b*, fall into the openings in the muller; the space, *k*, is then filled by a wood wedge, when the shoe stands firm. The dies, *a*, Figs. 33 and 34, of which there are four to each pan, have a similar dovetailed knob, which is wedged into a recess in the bottom of the pan.

In order to impart to the stuff a rising current on the periphery of the bottom, there are on the sides of the pan inclined ledges, *l*. Smaller ledges, *l'*, for the same purpose, occur on the periphery of the muller, but these latter are inversely inclined. The space, *a'*, between the dies can be wedged with hard wood. Each pan is capable of a charge of 600 to 800 lbs. of slimes.

It is evident that a pan having a circular plane bottom must increase the grinding from the centre towards the periphery, as each point of the muller on the radial line describes a larger circle proportionately as it is farther from the centre, and must therefore wear accordingly. If the muller on this account wears sooner at the periphery, a partial suspension of the work appears to result, the points near the circumference waiting as it were until the wear towards the centre overtakes that at the periphery. It would seem, therefore, that if the muller and the dies could be constructed of a material continually increasing in hardness towards the periphery, the grinding and the wear would be rendered uniform.

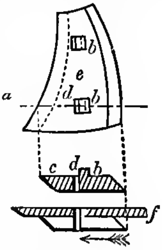


FIG. 35.—THE SHOE. Scale ½ in. = 1 ft.

It is also evident that the quantity of quartz which enters the central opening must decrease proportionately as it approaches the periphery; that is, considered in reference to the increasing grinding surface of the muller. The quartz is also being ground while on its way towards the periphery, which receives six times less quartz than the central opening (considered in reference to the larger circle), the quartz being already partially ground. It is obvious, therefore, that the wear of the muller has for this reason a tendency to decrease towards the periphery. It is necessary also to consider the radial slots between the shoes of the muller, which supply the grinding surface farthest from the centre with fresh stuff; but as the sectional size of the slots decrease by the wear of the shoes and also by their being choked with amalgam and other matter, this evil tends somewhat to rectify itself.

(3.) *The Improved Soderling Pan.* — Mr. Soderling, a Californian mining engineer, has recently introduced a form of pan which possesses novel features, and will no doubt prove of great advantage. It should prove especially valuable in the treatment of slimes and tailings which, through long exposure, have become baked together, as the disintegration of lumps can

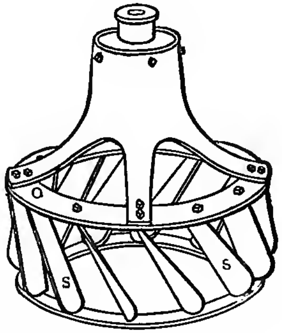
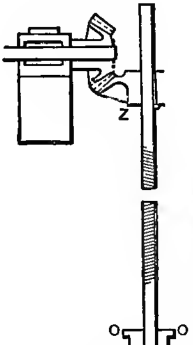


FIG. 37.—STIRRING APPARATUS.



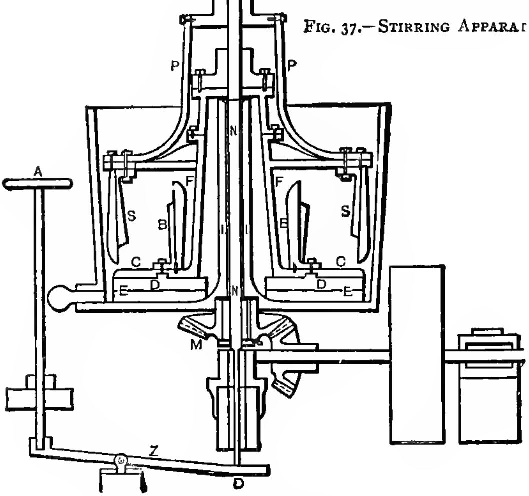


FIG 36.—SODERLING'S IMPROVED AMALGAMATING PAN.

be effected in this pan without any previous mechanical preparation. In the pans described on the previous pages, the material is caused to pass between the mullers and dies, and the two opposing surfaces grind and reduce the pulp in the presence of mercury, to save the precious metals. In this pan the material itself acts as a grinder, by the production of a counter-current. Its construction will be seen from the accompanying illustration (Fig. 36).

The pan can have a steam chamber at the bottom, and is provided with the muller (C) to which the grinding shoes (D) are attached. Dies (E) are placed in the bottom of the pan, and between them and the shoes of the muller the material is caused to pass by the centrifugal motion of the muller from the inner portion of the pan to the outer. This muller has a yoke (F) or cone extending upwards above the central cone of the pan, and is driven by means of a hollow shaft (N N), which extends up through the central cone (I I) of the pan and has a feather or key by which it is connected with the muller. This hollow shaft is driven by means of a bevelled gear and pinion M. A shaft extends up through the hollow shaft, and has a gear wheel (Z) fixed upon the opposite side from the other gear wheel, this one pinion serving to drive the two shafts in opposite directions.

Another driving arrangement, where the bevel gearings are arranged one below the other, is shown in Fig. 38.

The upper end of the solid shaft has a feather which engages with a corresponding keyway in the hub O at the upper end of the outside cone P P. This cone extends down outside the inner one F, which connects with the muller corresponding with it in shape, and is connected with a stirring apparatus which is driven by it in an opposite direction from that of the muller. This stirring apparatus S S (shown in Fig. 37) comprises rings having fitted thereto angular blades, S S, Fig. 37, thereby forming slots between the blades, these slots being angular, and having their sides curved, so that when rotating the tendency will be to carry the material from the outside of the pan through the slots and throw it toward the centre, where it will come in contact with the oppositely-moving current, which is thrown outward by the inner muller moving in the opposite direction. This action produced by the independent currents forms a circulating current above and below the mullers, and causes the harder and softer portions composing the pulp or slime to erode and disintegrate each other, and the whole mass of material will be acted upon by the chemicals employed during the process, and will be amalgamated with the mercury which is placed in the pan for the purpose.

This counter current can be increased by attaching to the muller the blades B B, which are fastened to them in the manner shown in Fig. 36. The pan, the steam bottom, dies, the muller and driving gear, are similar to those used in the ordinary grinding pans; and (as shown in Fig. 37) the extra or circumferential outer muller can be attached to any pan now in use at a small cost and be rotated from above, by a shaft, the lower end of which runs in a deep slip or guide on top of the ordinary muller, and at the upper end feathers to engage with corresponding key-ways cut in the side, the hole that fits the shaft in a gear wheel (or pulley to apply power direct by belt) in either case with a long hub, running in a boxing. The rotation of the outer or circumferential muller produces an action much greater than can be accomplished by grinding, the time being no greater than that occupied with the ordinary grinding.

The amount of power required is only that necessary to run this isolated ring, which may be made to run either faster or slower than the grinding muller by a simple change in the relative sizes of the gear wheels. By this device friction is avoided, which would be necessary if more grinding surfaces were employed. The excessive wear of the machinery is also avoided, and the action is better, inasmuch as the striking points of the opposing currents are acting all the way round the pan, and so far from the centre as to be in operation about one-third the radius of the pan from the edge, showing the impact action to increase in ratio to the cubic contents, of great importance in building large pans.

For treating tailings, pans of this construction can be utilized holding 5-ton charges, running four hours, which will give a daily capacity of 30 tons per pan, and five pans could reduce 150 tons. According to the coarseness or fineness of the tailings, grinding with the muller and the shoes will be only required for an hour or so.

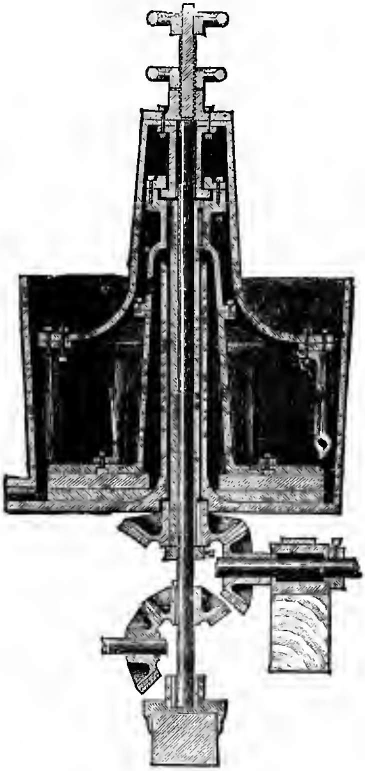


FIG. 38.—SODERLING PAN.

When it is desired to clean up, attach chain block and hoist; and the shaft being small below the hub will slide up through the muller up against the hub, 7 to 10 feet above the pan, in this way giving ample room to remove the ordinary muller and dies as usual. The lever Q, worked by hand-wheel A, is intended to raise or lower the muller. The shaft is connected by feather to the gear-wheel, which has a groove in the hub for a support to hold it in its place under the pan bottom.

**The Settler.**—After several hours' grinding in the pan, the pulp is discharged into another vessel similar to a pan, which is called a "settler" (see Fig. 39). This consists of a circular vat, in which revolves a central axis, to which iron shoes are attached. The shoes are not to come in contact with the bottom, but are faced with wooden rubbers, so as to keep the heavier parts of the pulp thoroughly stirred. Water is introduced during the operation, and the pulp can then be drawn off at plugged holes as required. An iron bowl for con-

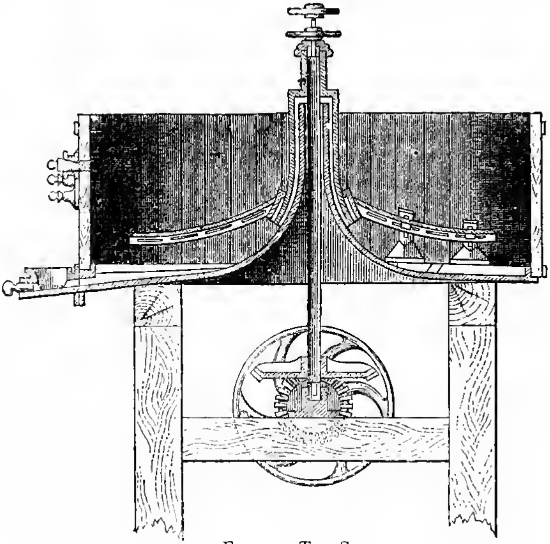


FIG. 39.—THE SETTLER.

taining the quicksilver is attached, which communicates with a radial gutter cast in the bottom of the settler.

Some mills are differently arranged, and the sands, after leaving the blankets and the amalgamators, discharge themselves into the rubbers, whereas in other mills the pulp, as soon as discharged from the copper plates, passes into the rubbers; but these machines are being rapidly replaced by better and more modern amalgamating appliances.

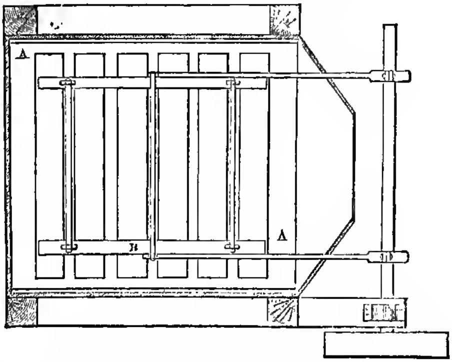


Fig. 40.—Eureka Rubber.—Scale $\frac{5}{16}$ in. = 1 ft.

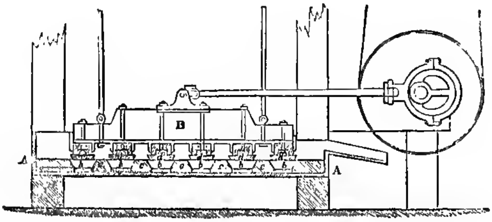


Fig. 41.—Eureka Rubber.—Scale $\frac{5}{16}$ in. = 1 ft.

**Rubbers.**—In the rubbers the particles of gold are further cleaned and brightened, while at the same time being detached from the sands. They have also an opportunity of being caught by the amalgamated copper plates of the rubber.

The Eureka Rubber (Figs. 40 and 41) consists of a rectangular cast-iron box, 7 in. deep and 4 ft. 8 in. square, which is provided with a false bottom made of cast-iron dies or plates. On these latter are cast-iron shoes affixed to a wooden frame, which have a rectilinear motion imparted to them by rods connected with an eccentric. The wooden boards for the shoes are covered with amalgamated copper plates.

In preparing the false bottom of the rubber box, the boards *ccc* (Figs. 40 and 41), which are 3½ in. wide and 1 in. thick, and of the same length as the box, are laid in at right angles to the motion of the frame, and have a space of 4 in. between them. On top of these boards are placed, broadside upwards, the plates or dies, *b b*, which are 1½ in. in thickness. Between the edges of the plates a space of 3 in. is left. The spaces, *c c*, are filled with pieces of wood, which are set up endwise, and dressed to fit the false bottom plates. They are 2½ in. long, 4 in. wide, and from 6 to 12 in. broad. For rubbing, this arrangement is found to answer better than a false bottom made of solid cast iron. The lower boards, *e e*, are introduced to give a firmer fixture to the blocks, *c c*, when the iron false bottom plates, *b*, have become partly worn. The plates are firmly secured by strips of wood, *g g*, 1 in. thick, bolted to the sides of the rubber box, and fitting into the recesses, *f f*, on both ends of the bottom plates.

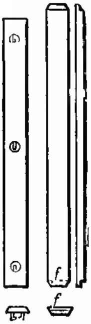


FIG. 42.
FALSE BOTTOM.

In Fig. 44 is shown the construction of the movable rubber frame. Both the iron and the wooden shoes, K K, which carry the amalgamated copper plates, fastened by iron clamps and bolts, are secured by two cast-iron frames, *l l*. Each of these frames is attached to a piece of timber, *g*, as shown in the figure. The movable rubber frame is

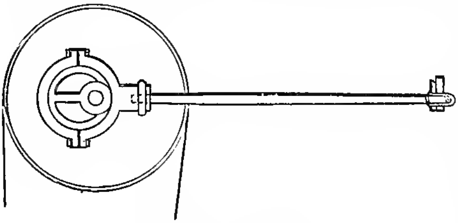


FIG. 43.—ECCENTRIC.

attached by four wrought-iron rods, *a a* (Fig. 41) to a wooden frame, *m m*, which can be raised or lowered by means of two screws, supported on the main wooden frame. The main frame, which supports the rubber box and the movable rubber frame, is constructed of pine lumber 6 in. square. The ends of the bolt, which passes through the journals, *j*, form the pins to which the eccentric rod (Fig. 41) is attached. The number of revolutions of the eccentric shaft is 55 per minute, and the length of the stroke of the rubber frame is 4 in. There is one rubber to every battery of 4 or 5 stamps.

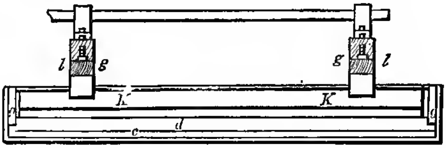


FIG. 44.—MOVABLE FRAME.

The sands after passing the rubber flow over the amalgamated copper plates. These plates are 2 ft. 6 in. wide, and are made of soft copper and heavy material; they are laid in wooden sluices, which have an inclination of ½ in. to the foot. The sides of the latter are 3 in. high, and their length is from 20 to 30 ft. Three inches per stamp is the usual width of the copper plate sluices.

**Gauthier's Shaking Table** will probably supersede entirely the apron now used in gold mills. The invention consists of a silvered copper-plated table as wide as the discharge of the mortar and 16 feet long, mounted on vibrating standards, and having a longitudinal oscillation imparted to it by means of a shaft and eccentric below. The amplitude of the vibrations is adjustable, but is usually set at 1½ inches. This table, which occupies the position of the ordinary apron, and receives the pulp from the mortar, has a slight inclination downward from the battery—much less, however, than is necessary in a fixed apron, because the shaking keeps the sand in a loose and open condition independently of the current of water. For the same reason also less water is required than with the usual arrangement.

In an experiment at the Crown Point Mine, as reported by the State Mineralogist of California, this plated table was mounted on the top of a Triumph concentrator, receiving the pulp after it had passed over the usual apron and plated sluices, and it was found that 7 per cent. of the total gold was saved on the table.

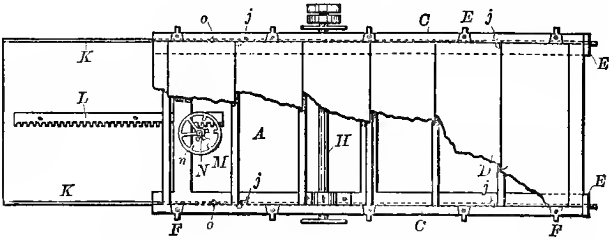


FIG. 45.—GAUTHIER'S SHAKING TABLE. Plan.

The machine is shown in the accompanying drawings, Fig. 45 being a plan, and Fig. 46 a section of the table. It is thus described by the inventor:—A is a fixed inclined bed-frame or floor, and B is the mortar of a battery at its head. C is the inclined top frame of the machine, said frame carrying the amalgamating-plates D. This top frame is supported from sills E below by means of spring-standards F. The sills rest on the floor A, and the inclination of the top frame and its plate sur-

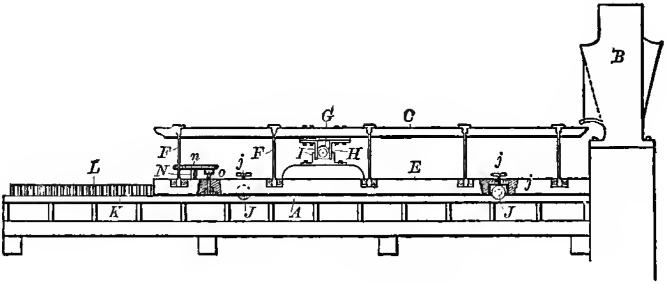


FIG. 46.—GAUTHIER'S SHAKING TABLE. Section.

face, as will be seen in Fig. 46, is a very slight one. The upper end of the top frame and plate-surface is just under the apron of the battery-mortar B and receives the ore-pulp from it. A vibratory or shaking motion is imparted to the top frame and plate-surface by means of an eccentric, G, upon a rotary drive-shaft, H, extending transversely of the machine, said eccentric playing between the jaws, I, bolted up under the top frame. This eccentric is to be driven at a high rate of speed, so that the plate-surface is moved back and forth with a longitudinal shake on its spring-standard.

Normally the machine rests positively with its sills, E, on the inclined floor; but, in order to move the machine closer to or farther from the battery-mortar, rollers or wheels, J, are provided in the sills, which wheels travel on fixed tracks, K, mounted on the floor, A. These wheels, J, are mounted in vertically-sliding bearings, *j*, which are adapted to be moved up or down by means of hand-screws, *j'*.

When the machine is in place, the wheels or rollers are withdrawn into the sills, so that said sills rest fixedly on the floor; but when the machine is to be moved the wheels are projected so as to raise the whole machine up on the tracks, thus providing for its easy motion.

The movement of the machine is effected by means of a rack, L, fixed to the floor and engaging a pinion, M, on the vertical shaft, N, mounted in a cross-bar of the machine and having a hand-wheel, *n*, on its top for rotating it. By the movement of the shaft the pinion is caused to travel in the fixed rack, thereby moving the machine on its wheels or rollers.

In order to vary and suitably regulate the inclination of the plate-surface according to the requirements of the work, the screws, O, work through the sills, E, and bear against the floor. By turning these screws the lower end of the machine may be raised or lowered to regulate the inclination.

In the ordinary amalgamating device, which receives the ore-pulp from the battery, the silver amalgamating-plates are usually placed at a considerable inclination, and a great quantity of water is used, the object being to effectually carry away the sand. This object could not otherwise be accomplished in the use of the ordinary plates, because they are stationary, and if they were set at a slight inclination and but little water were used, the sand would not separate easily. In the ordinary construction, therefore, there is a great disadvantage, because, the plates being set at a suitable inclination and there being so much water used, the quicksilver which is put on the plates does not remain, but much of it is washed down the incline with the sand, and is thus wasted. This machine obviates this disadvantage, allowing the use of a great deal more quicksilver without subjecting it to loss or waste, while at the same time the machine effectually disposes of the sand without having to use much water. To effect these results and regulate and adjust the amalgamating-surface at a very slight inclination, more than the usual quantity of quicksilver is placed on the plates.

The inclination being slight, the quicksilver has but little tendency to flow off; and by reason of the shaking motion to which the plates are subjected, there is no necessity for the use of much water, the sand being separated without it and the quicksilver remaining.

**Bazin's Centrifugal Amalgamator.**—In this apparatus the amalgamation of the gold particles is effected by the application of centrifugal action under water. The working of the apparatus is as follows:—A wrought-iron bowl, B B, Fig. 47, containing 600 to 700 pounds of quicksilver up to the level, Q Q, is mounted on the vertical axis F, which passes through the stuffing-box, S, receiving its rotary motion through the gearing, U, and the pulley, N. The bowl on top is 3 ft. in diameter, and exactly of the shape shown in the figure. It revolves inside of the sheet-iron tank, C C, which is filled with water to the line, H H, and through the rotary motion the quicksilver rises up on the sides of the bowl, B, till the velocity of the bowl is such that the quicksilver reaches within an inch of the rim of the bowl, covering its interior all over with a layer of about a inches. The velocity of the bowl must not be increased over 55 revolutions per minute, otherwise the quicksilver will

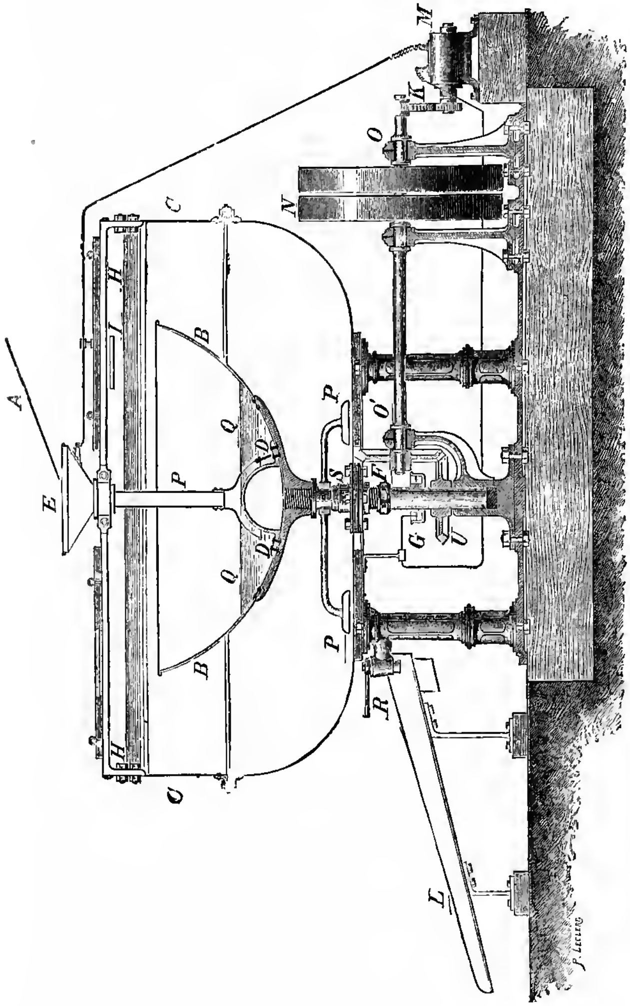


FIG. 47.—BAZIN'S AMALGAMATOR.

be projected over the rim, but by properly regulating the speed, no loss will take place.

When the bowl has reached the proper speed the pulp from the battery or pulveriser is conducted through the launder, A, to the funnel E, and passes through the pipe, P, and the distributing conducts, D D, into the bowl, whose bottom is now entirely freed from quicksilver, the latter having crept up the sides of the bowl. In the figure the quicksilver is in repose.

The pulp, on entering the inside of the bowl, through the centrifugal action imparted to it, is forced against the periphery of the bowl, and each particle of ore is whirled round in contact with the mercury for some time. In this operation the gold particles—owing to their greater density—are thrown by the centrifugal action into the quicksilver and are absorbed, while the sands overflow and fall to the bottom of the tank C, where the stirrers, P, keep the tailings agitated and discharge them through the faucet, R, into the tailing launder, L. The light quartz particles are rapidly eliminated, and any surplus water passes out through the opening, I. The whole operation forms a very interesting process.

Mr. Bazin has tried to increase the amalgamating results by electrolysing the mercury. For this purpose the axis, O O, through the gearing, K, actuates a dynamo, M, connected with the funnel, E, and with the basin, G, which is fixed concentric to the axis, F, and contains mercury, into which plunges the terminal.

The machine is capable of treating one ton an hour.

**Amalgamated Copper Plates.**—Of course, the amalgamated copper plates only catch the gold which is in a free state. The principal difficulty with which the millman has to contend is the discoloration of the surface of the plates by oxides and insoluble metallic salts, as it is necessary to keep the surface of the plates bright and free from film, which would prevent an easy and sure adhesion of the particles of gold and amalgam as the pulp passes over. A new plate is not considered in fit condition until it is capable of keeping bright without dressing for several hours. This condition is not usually attained until a firmly adhering layer of gold amalgam has attached itself to the plate, and this does not generally take place till after the loss of much gold, labour, and time.

Silver-plated and even gold-plated copper plates have been tried, but they have not met with much favour, as they have proved either ineffectual or too expensive.

In the preparation and care of amalgamated copper plates, in order that they may be put and kept in the most effective condition, and at the smallest expense, regard must be had (1) to the quality of the copper; (2) to the method of amalgamating the plate and of dressing it; (3) to the purity of the mercury and of the water; (4) to the character of the ore. The plate should be of the purest and softest copper, and free from dark, rough spots. The kind known to the trade as "braziers' copper" is the best material for selection. That which has been rolled with special care in order to make the surface smooth and hard will not be suitable, as a plate made from it would not absorb the quicksilver so well as the softer and therefore more porous copper. It is best, from the point of view of efficiency and durability, to select plates weighing not less than 3 lbs. per square foot. It would be better even if they were thicker than this. For inside plates, copper two or three times as heavy as this should be used, so that the plates may not be easily bent or torn from their position. If possible, copper which has been annealed subsequently to the last rolling should be selected. If this cannot be had, then plates annealed by exposure to heat on their under side, sufficiently to ignite sawdust laid upon the upper side, will be the next best. This may be done over a blacksmith's fire, but more conveniently over an open wood or charcoal fire. Every part of the plate should be subjected to the heat. This process will soften the plate, and make it more porous; it will therefore be capable of retaining more quicksilver and amalgam than it otherwise would. As the efficiency of copper plates is in proportion to their ability to hold quicksilver, this point should receive careful attention.

The plate is straightened by laying it on a table and using a wooden block and hammer. The hammer should not be struck directly on to the plate, but the block interposed so that the face of the copper may be compressed or drawn into shape. The plate should then be fastened to the table with iron screws —or, preferably, copper nails—long enough to clinch on the under side of the table. In either case, the heads of the screws or nails should be flush with the face of the plate. Brass screws should not be used, as the quicksilver soon penetrates the heads and makes them brittle, and thus destroys their capacity to hold. The minor inequalities of the surface of the plate may then be removed by the block and hammer.

The plate should next be scoured with wood ashes and fine sand or tailings applied with a scrubbing brush or coarse rag. In case of bad spots, the scouring should be done by means of the end of a small block of wood. The scouring must be continued until the coating or oxide is entirely removed, and the bright, metallic copper exposed. Caustic soda, concentrated lye, or sal soda, may be used instead of ashes, to remove traces of oil or grease resulting from the action of alkalies. After washing with clean water, a solution of cyanide of potassium—say, half-an-ounce of cyanide to a pint of water—should be applied with a soft brush or swab.

To amalgamate the plate, a mixture of fine sand or tailings and powdered sal ammoniac, in equal parts, with a small quantity of quicksilver sprinkled in, may be used.

This mixture is applied with the scrubbing brush, and the scrubbing continued until all parts of the plate are amalgamated. As much quicksilver as the plate will absorb should be sprinkled on to the plate during the operation, with water enough to make a thick mud of the mixture. I have used lime with the mixture, but I cannot say that the addition gives any special value to it. It is certainly unpleasant, inasmuch as ammonia is set free in great quantities. The mixture must remain on the plate for an hour or so, and then be washed off by means of clean water and a brush, which should be followed by a soft brush and cyanide solution, adding quicksilver if the plate will hold it. By continuing this treatment for three or four rounds, the plate will be found to have taken up as much quicksilver as it would ordinarily after many weeks' running. If fine gold amalgam can be spared, it is well at this stage to rub some upon the plate, using a cloth rubber wetted with a solution of sal ammoniac, in the proportion of four ounces to a pint of water. If the amalgam be heated and rubbed in a Wedgewood mortar, so as to insure the solution of the gold, it will be found to attach itself more readily to the plate.

The first scouring removes oxide and grease, and exposes the bright metal, which by annealing has been, by virtue of its porosity, rendered capable of holding a comparatively large quantity of quicksilver and amalgam. The sal ammoniac, through dissolving the oxides, assists the amalgamation by preserving a bright surface to the plate, and thus allowing the quicksilver and amalgam to penetrate the copper, filling the minutest interstices, and combining with it in atomic proportions. The result is the production of an alloy of copper, gold, and quicksilver. The air, water, and the various salts are being thus kept from acting directly upon the copper, and the formation of oxide and carbonate of copper is avoided. This is accomplished in a longer or shorter period which depends upon the mode of preparation and the richness of the ore, and also the ease with which the gold in the ore may be amalgamated. It is probable that the galvanic current arising from the contact of the two dissimilar metals plays an important part in the chemical reactions and the amalgamation of the gold, but its real action is not definitely known.

The main point in the work of preparation is to effect a quick union, in proper quantities, of the gold and the quicksilver with the copper. If this union takes place slowly, loss of gold, which is unavoidable, makes the expense much greater than would be the case if the process were assisted by a needful amount of amalgam, before running the ore over the plate. If a section of an old well-used plate be examined, the amalgam will be found to have penetrated perhaps half-way, and in some cases even completely through the copper.

After scouring the plate well with some sandstone, and then rubbing it with a solution of sulphate of copper, the copper will also be in a condition to take up the quicksilver.

Corrosive sublimate, nitrate of mercury, sulphuric acid, and common salt, muriatic acid, sodium amalgam, cyanide of potassium, and other acids and salts, have all been used in amalgamating copper; but though amalgamation may be accomplished by them in some cases quite easily, yet the plate cannot be brought by them to a good working condition so soon, even if at all, as by the method I have described.

The subsequent treatment of the plates, amalgamated in the manner described, should be varied to suit the character and quality of ore and the purity of the water. Water containing carbonic acid discolours the plate, forming an insoluble coating of carbonate of copper. Though this coating is but a very thin film, it is nevertheless sufficient to prevent the adhesion of the gold and amalgam, unless these latter be in particles large enough to break through the coating. The worst form of discoloration arises from the action of sulphates of iron and copper present in the ores, arising from the decomposition of the pyrites. The addition of lime to the water in the battery, as much as will dissolve therein, will be found to neutralise the carbonic acid and decompose the sulphates. It may be necessary to add lime to the ore, so as to furnish sufficient to react on all the sulphates.

**Silver-plated Copper Plates.**—These plates (which I have used with excellent results) are in favour with some millmen, especially when starting a new mill, as the silver amalgam will have more affinity for the gold than a copper plate simply coated with mercury; but there is nothing better than old plates, when once thoroughly "soaked" with gold.

The silver amalgam may be made as follows: Dissolve a piece of silver—coin will answer the purpose—in the smallest quantity possible of dilute nitric acid. Heat will be found to hasten the solution. Dissolve the resulting crystals of nitrate of silver in water, and pour into the vessel enough quicksilver to reduce and amalgamate the silver in the solution. This process is completed in a few hours. Wash the amalgam with clean water to remove any traces of nitrate of mercury, and then strain it to remove any surplus quicksilver. This will leave in the cloth a pasty amalgam of very pure and finely divided silver, which should be used as directed for the gold amalgam.

**Dressing the Plates.**—When the mill is in operation, the plates should be dressed every six hours, or oftener, if they are discoloured. After washing them with a stream of clear water, a solution of sal ammoniac should be applied with a small brush. The sal ammoniac should be allowed to remain on the plate for a few minutes, and the plate afterwards washed with clear water. Then, with a brush, enough of the solution of cyanide of potassium must be applied to brighten the plate. The plate should have upon it as much quicksilver as it will hold, which should not gather in drops or run off.

It is advisable not to allow the layer of gold amalgam to accumulate to too great a thickness, and it should be removed occasionally by "sweating" the plates. A good plan is to immerse the plates in boiling water until the amalgam is softened sufficiently to be easily scraped off, or to turn a jet of steam on to the plates, so as to soften the amalgam, which may then be easily scraped off. Where the pulp carries but little amalgam or quicksilver, the silver plating is soon worn out.

Where mills have occasion to re-silver their copper plates often, they have a special plant for this purpose. The operation consists in immersing the plates in tanks containing a solution of chloride of silver in cyanide of potassium, and submitting them to the electrolytic process, for which either a dynamo or batteries can be used. Such a plant is not expensive, and the power required to run the dynamo is nominal. The quantity of silver taken from the solution must be replaced by the addition of equivalent silver anodes. The cost of silver-plating is but little more than the cost of the silver. One ounce of silver per square foot of copper is the usual amount used.

Owing to the constant absorption of gold by the copper, the plates after long usage become useless, and they should then be melted into ingots.

The quicksilver should be entirely free from base metals, such as lead, zinc, tin, and copper. Of course, the presence of gold and silver is desirable.

## III.—The Process of Amalgamation.

**Operation of the Mill.**—The stamping process is not so simple as would appear at first sight. The form of mortar, the mode of exit for the stuff, the weight and rapidity of the pestles, and the quantity of water employed, must be varied to suit the manner in which the particles of gold are distributed through the mass, as well as the structure and character of the ore and of the matrix. Fine reduction is by no means always desirable, for if some kinds of stuff be reduced too fine, much of the ore contained in it will be wasted. Considerable judgment is therefore necessary in selecting the screen best suited to the material which has to be reduced.

The rock is delivered from the mine in cars, each containing, say, 13 cubic feet, and weighing 1,300 lbs., in front of the rock-breaker. At custom mills the quartz is delivered in waggons, the crushing being paid for by the load, which consists of about 35 cubic feet, and weighs, including moisture, 3,500 lbs.

As a rule heavy stamps are more effective in the quantity of ore they crush than light ones. The stamps and the lift must both bear some relation to the hardness or the softness of the ore. Other circumstances, however, have also to be taken into consideration. In California, especially, where a hard, flinty quartz occurs, there are many mills with from 800 to 900 pound stamps, and with a lift of from twelve to fifteen inches. For fine crushing a high lift light stamp (say from 600 to 700 pounds) and a deep mortar are used. For coarse crushing it is found better to have heavier stamps, less lift, a shallow mortar, and more speed.

**The capacity of a stamp mill** is calculated by multiplying the weight of the stamp with the fall in feet and with the number of drops per minute, giving as the result in force per minute exactly the number of foot pounds exerted by each stamp. Dividing by 33,000—the number of foot pounds per minute in one horse-power—we have the horse-power per stamp, from which the effective power of the whole mill may be obtained. Dividing the amount of rock crushed daily by the effective horse-power will give us the daily amount per horse-power; and this is the best measure that can be obtained for the effectiveness of the stamps.

The object of crushing being to liberate the fine particles of gold contained in the quartz, so that they may be collected by the copper plates and blankets outside the battery for amalgamation, it is desirable that the crushing be fine. In this process quick crushing does not interfere with the saving of the gold, as it is collected and amalgamated outside the battery after crushing. It is this feature which distinguishes the blanket process from that of amalgamation in the battery during crushing.

The feeding of the battery should be so regulated that there shall not be at the end of any drop more than two inches of sand between the die and the shoe. A good feeder generally knows the requirement of the battery by the clear or dull sound produced by the stamp stroke.

A battery of 20 stamps weighing 850 lbs. per stamp, with 61 drops of 10 inches per minute, crushes, without the aid of a rock-breaker, 40 tons of quartz in twenty-four hours; while a battery of 20 stamps, weighing 700 lbs per stamp, with 68 drops of 10 inches per minute, crushes 32 tons of the same rock; a No. 6 screen being used in both cases.

The power necessary to execute the work of the heavier as compared with the lighter stamps is as 850 × 61 : 700 × 68, and the work expected from them would be 35 tons for the heavier and 32 for the lighter stamps. The former, however, crush 40 tons, which is an additional quantity of over 5 tons in favour of the heavy stamps. When the rock-breaker is used, the proportional result is almost the same. The limit of weight has never been experimentally determined, though stamps weighing over 900 lbs., and having a drop of ten inches are in use. These results are on rock of the average hardness.

**The quantity of rock crushed** depends, of course, very much on the nature of the ore, the weight of stamp, the speed, the screens, &c. A 700-pound stamp will crush, at a speed of seventy-five blows per minute, using No. 4 or No. 5 screens, from one to three tons in twenty-four hours.

A slow motion of the stamp will produce more floating stuff and will crush finer than a rapid motion, because, in the case of a quick falling of the stamps, the small particles of ore are not allowed sufficient time to fall under the shoe, and are therefore carried out; while with a slow motion many grains which could pass the screen slide under the stamp, and are there pulverised finer than required. Although it is evident that a high speed will crush more stuff in a given time, yet there is a limit which the speed should never exceed. The stamp should fall on the ore with its full force, and the revolving stamp should have time to finish its motion. A quick or sudden blow is less effective, while at the same time there is the danger of a tappet being caught by a cam. The speed is excessive if it exceeds eighty blows per minute.

**Battery discharge.**—In battery amalgamation the height of the discharge—*i.e.* the vertical height of the lower edge of the screen above the die—is governed by the width of the mortar. Narrow mortars require a higher discharge than wide mortars, in order to avoid breakage of screens and to prevent scouring of the inside copper plate. It varies from four to nine inches, the mean being about six or seven inches. A uniform height of discharge should be observed. As the dies wear down the edge of the screen is lowered correspondingly, and the wooden blocks, on which the inside plates are fixed, are replaced by others of less height, thereby preserving uniformity in height of discharge.

With ores quickly crushed and readily discharged it is necessary to raise the screen—*i.e.* to increase the height of the discharge—in order to retain the pulp in the mortars sufficiently long for its proper amalgamation.

Mortars for Californian gold mills have almost invariably but a single discharge, and the following objections against double discharge mortars are instanced by the State Mineralogist of California: (1) inconvenience in the arrangement of the copper plates when adapted to double discharge; (2) the necessity of using too much battery water (especially where concentration follows); (3) the ore is not allowed time for proper amalgamation in the battery, although for certain classes of ores the double discharge mortars could be advantageously employed.

Where ores are very heavily sulphuretted, and, consequently, cannot be amalgamated in the battery, double discharge mortars may be advantageously used. Their use is especially desirable where the ores contain brittle sulphurets, which from being too long subjected to stamping are liable to be slimed.

Most ores are usually stamped faster than the screens can discharge them, and, being retained in the mortar, they are pounded too much, with the result that a great part of the product is slimed; and as the slime is generally rich in the brittle sulphurets, loss accrues.

Narrow mortars accelerate the discharge of the pulp from the battery, but very rapid discharge is not always desirable when battery amalgamation is practised. In hard, flinty ores, if the screens are brought close to the die, there is excessive breakage of screens, occasioning undue expense and loss of time in changing them. By raising the lower edge of the screens the liability of breakage may be reduced or obviated, but this increases the height of discharge, which reduces correspondingly the capacity of the battery and annuls the advantage aimed at in the use of narrow mortars.

**The rate of speed** should be regulated according to the character of the ore. There are instances where, upon an increase of speed, the yield of gold per ton fell off; and practical trials alone, therefore, should determine the rate of speed in each mill. The rapid running of the stamps, and consequent augmentation of product crushed, causes greater agitation within the battery box, and requires a larger supply of water to clear the discharge and carry away the increased amount of pulp. The excess of agitation in the battery may prevent the accumulation of gold on the interior plates, and the excess of current on the aprons may prevent the accumulation of gold there. These objections are most plausible when the gold is most finely divided in the quartz.

Professor Raymond has the following observations on this subject:* "The conditions most favourable to economical crushing must be partly sacrificed to secure efficient amalgamation. The attempt to catch the greater part of the gold on the interior plates interferes directly with the efficiency of the stamps. The success of the amalgamation at this point is in inverse proportion to the success of the crushing and discharge. There is a certain advantage gained in the force with which the pulp is dashed against the plates; but this force is liable to overdo, and thus undo, its own work, and actually remove the adhering amalgam. The same effect can be more completely secured outside of the battery."

My aim in crushing has always been to collect as much gold inside the mortar as possible, that being the best place to collect and keep it safely, even at the sacrifice of crushing less rock per day; and I have always endeavoured to get the highest percentage of assay from the rock rather than to pulverise as much rock as possible, or to make up the average daily production by an increase in the quantity crushed. The conditions for collecting the gold when once outside the mortar are not as favourable as they are inside the mortar, as the pulp is swept over a small, steep, and smooth surface of amalgamated copper plates; and even when arranged in steps or drops, the chances are that a large percentage of the gold will flow away.

In another passage, Mr. Raymond sums up his views as follows: "(1.) The stamp-mill is the most convenient and practically efficient machine for crushing quartz thus far introduced and proved by experience. It involves little waste of power in gearing; it delivers its power in the most direct and practical manner, namely, by blows, which take advantage of

---

* Official Report on Mines and Mining.

the brittleness of the rock, instead of pressure or friction, which invites the resistance of hardness; its capacities for charging and discharging are ample and easily regulated, both as to quantity and as to fineness of the product; it is subject to few and comparatively inexpensive repairs, and it can be repaired, in most cases, without complete stoppage. These and other excellent features in its construction and operation render it especially suitable for use in mining districts remote from machine shops, foundries, and centres of skilled labour.

"(2.) To obtain the best results, stamp-batteries should be built and run to secure the highest efficiency and economy in crushing only, without reference to amalgamation. The amalgamating apparatus should be adapted to the batteries, not the latter to the former. If interior plates are employed they should not be expected to catch the greater part of the gold, nor should the pulp escaping through the screens be swiftly and carelessly manipulated, when a little extra space and time devoted to it, almost without extra labour, would avoid much loss.

"(3.) The efficiency of the stamp may be described as the product of three factors—weight, fall, and speed. The efficiency of a battery of stamps involves a co-efficient—the discharge.

"(4.) When the fineness of crushing is regulated by screens, the discharge should be as large as practicable. There may be mechanical objections to continuous screens running around the whole battery; but there are, I think, no valid arguments against the double discharge, in front and rear, when the battery is properly planned with reference to it.

"(5.) Of the three factors of the efficiency of the stamp, the weight and fall determine the force of the blows at longer intervals. The blow must be heavy enough to crush the rock upon which it falls. If too heavy, it may waste power in packing the crushed rock; if too light, it may fail to crush, and so may pack. Finally, the speed should not be so great as to prevent proper clearance, or the stamp may strike a second blow upon the rock already crushed.

"(7.) The efficiency of a blow from a heavy stamp with short drop is less than that of an equal blow in foot pounds given by a lighter stamp with longer drop, because the longer drop gives greater final velocity to the stamp, and this tends to crush more and to pack less.

"(8.) The superior effectiveness of frequent blows lies in the fact that there is a limit to the amount of crushing which can be practically performed by a single impact upon a given quantity of rock distributed over a given surface. There is another practical advantage of high speed: if stamps are left, as it were, standing in the pulp between blows, the material settles around them and they 'suck' when the lift commences. A great deal of power is frequently wasted in this way, by not picking up the stamps before they become partially buried.

"(9.) But even if the efficiency of stamps were always exactly measured by the product of the three factors mentioned—that is, by the number of foot pounds delivered per minute (which is certainly not the case)—there would still be good reason for preferring rapid running. After the necessary stability and strength are secured, increased weight of machinery is an evil. If equal results can be achieved by substituting speed for weight, the change is advisable."

**The quantity of water** for the battery depends upon the quantity of sulphurets or black iron sands present in the quartz, and averages half a cubic foot per minute per stamp. It must, however, be sufficiently large to carry the crushed sands over the blankets without allowing them to permanently settle upon them—say, about 28 cubic ft. of water per cubic foot of rock crushed. For rock poor in gold the supply of water is increased, to increase the rate of working, so that the quantity amounts to 33 cubic ft. and upwards to one cubic foot of rock crushed. The foregoing proportions of 28 and 33 cubic ft. to 1 represent weights of the cubic foot of rock crushed, of about 125 and 108 lbs. respectively. In winter the battery water should be somewhat warmed so as to prevent congelation. To have less inclination of the blanket and more water, is preferable to the opposite condition.

**Apron, Sluices, and Blankets.**—On leaving the mortar, where a preliminary amalgamation on copper plates frequently takes place, the slimes pass over an apron of amalgamated copper plates, and by this means a considerable quantity of the metal is removed and prevented from passing down the blanket sluices which immediately succeed the apron. The width of the sluices for the copper plates is usually 15 in.—or 3 in. per stamp; and there ought to be two or three sets for each battery, so that during the cleaning of one set the stream can be turned on into the other set. The full width, therefore, of each copper apron per battery is 45 ins.*

The bottom edge of the lower screen holes when new is 5 in. above the dies in the mortar. The water in the battery during the crushing has a wave motion along the screens. It is of the natural temperature while passing through the battery and over the blankets; but later, when passing through the rubbers and copper plates, it becomes somewhat warmed by the influx of the hot water from the amalgamators.

The slimes after leaving the sluices (see Fig. 5, p. 25), which are covered with amalgamated copper plates, pass into sluices covered with blankets. There are three sets of these sluices for every 5-stamp battery, placed longitudinally. Each of the sets consists of two sluices made of ½-inch planed sugar pine boards, one 10½ ft. long, the other 5½ ft., with a drop of 3½ in. between them. They are from 16 to 17 in. wide, with sides of 2 in. in the clear, and have an inclination of 1 in. to 2 in. to the foot.

The upper sluice has two strong blankets, each 21 in. wide and 5¼ ft. long, the upper of which overlaps the lower about 6 in. The lower sluice has only one blanket.

The pulp flows over two of the three sets of blankets, the third being kept in reserve for use when washing either of the other blankets. The upper blankets, which catch the bulk of the gold, are washed every twenty minutes, the lower every two hours. The washing is performed in two tanks, which are used

---

* It is best to have the copper plates the full width of the mortar without any division, and when dressing or cleaning the plates to hang up the stamps.

alternately. They are made of 1½-in. planed pine boards, having a horizontal section of 3 by 4 ft., and tapering towards the bottom. They are 2½ ft. deep, and are provided with inclined shelves for the blankets, and plug holes for the discharge of water after the settling of the blanket washings. The water used in these tanks is warmed in a heater by the waste steam.

The quantity of crushed sand passing over the blanket sluices of a 5-stamp battery ranges from 8 to 12 tons in twenty-four hours. The blanket washings, consisting of gold, sulphurets, iron, and quartz sand, vary considerably in weight with the percentage of the metallic contents of the rock.

The quantity of slime passing per foot of sluice width is about 0·7 cubic ft. per minute, containing about 3·1 pounds of solid matter per cubic foot of water. The blanket concentrations amount to about 2·5 per cent. of the original crushed ore, and have a specific gravity of perhaps 3·4.

The accumulations from the blankets collected in the wash-boxes are introduced in front of the amalgamators, from which they are gradually swept into the amalgamators by a current of clean water having a temperature of from 100° to 130° Fahr., or they are treated by pan amalgamation.

**The material for the blankets** is specially manufactured for the mill trade, and is shorn on the lower side, having the nap on the upper. Its weight is eight-tenths of a pound per running yard.

**Water required for Gold Milling.**—The quantity of water required to work gold ores by the wet battery process is generally estimated as follows:—

For the boiler, 7½ gallons per horse-power per hour.
For each stamp, 72 gallons per hour.
For each pan, 120 gallons per hour.
For each settler, 60 gallons per hour.

If the water used in the battery, pans, and settlers be run into settling tanks, it can be re-used with a loss of about 25 per cent.

In making my calculations for the quantity of water required for a 30-stamp mill, I averaged the calculations as follows:—Half a cubic foot per stamp, per minute, equal to 30 cubic feet per hour; 30×24 hours makes 720 cubic feet per stamp, per day; 30 stamps×720 cubic feet makes 21,600 cubic feet water for the batteries.

Calculating the boiler for 60 horse-power, water required:—

One cubic foot per horse-power per hour = 60 × 24 = 1,440
Sixteen cubic feet per pan, per hour = 16 × 24 = 384
Eight ,, ,, settler ,, = 8 × 24 = 192
2,016
For battery ... ... ... ... ... ... 21,600
Total in cubic feet of water ... ... 23,616

Taking it that the flow per miner's inch under ordinary six-inch pressure gives 2,160 cubic feet in twenty-four hours, this would represent 11 miner's inches; but it is safer to have a flow of 12 to 16 miner's inches before venturing on the erection of a mill of such capacity, and it is better to take the maximum figure than the minimum. A stream or water supply ought never to be gauged in the wet season, but the minimum flow of the dry season should be ascertained.

**The Process of Amalgamation.**—In some mills amalgamation is performed in the battery, which is supplied with quicksilver for the purpose of taking up the gold when liberated by the crushing process. The quicksilver is usually introduced into the batteries in small quantities as may be necessary, having regard to the richness of the ore and the rapidity with which the amalgam is formed. The appearance of the latter as it issues from the battery is an indication of the quantity of quicksilver present. When the particles of amalgam are hard, dry, and dense, they indicate the absence of a sufficient quantity; while fluidity of the particles indicates the presence of a sufficient quantity, or possibly a surplus of quicksilver may be inferred. The ends of the batteries are lined with amalgamated copper plates, while another plate of the same character, about 10 or 12 in. wide, with a length the same as the inside of the battery, is so fixed in a frame that it may be introduced and secured behind the stamps in an inclined position. A similar plate, but narrower, is used on the discharge side of the battery. A portion of the amalgam, as it is formed in the battery and splashed against these plates, adheres to the amalgamated surfaces and is retained by them. The batteries and plates are cleaned at stated intervals, which vary in length in different mills.

A large portion of the total product is obtained by means of this contrivance, while the remainder is secured by other contrivances—namely, aprons or tables, covered with amalgamated copper plates—outside the battery. The inclination of the tables is usually 1 in. for every 1 ft. in length, being variously determined by different mill-men. It depends, of course, on the quantity of water used and other conditions. From these tables the material passes as before stated over blanket sluices, or rubbers, amalgamators, and in more modern mills, into the concentrators or shaking tables, or such other contrivances as the character of the ores may require, for further treatment.

**Grade or Inclination of Plates.**—This should vary on the outside plates with the amount of sulphurets in the ore, the amount of water used, and the fineness or coarseness of the gold. The grade under any circumstances must be sufficient to allow a free flow of the pulp and to prevent a deposition of the pulp on the plates. It is obvious that heavily sulphuretted and coarsely crushed ores require the maximum grade. The frame supporting the plates should be so constructed as to admit of the grade being adjusted conformably to the requirements of the ore treated.

In most mills it is customary to have the copper plate immemediately in front of the mortar wider than the succeeding ones, which are called the "sluice plates," to distinguish them from the front plates or "aprons." This system should be rejected, as on the apron we have a broad shallow stream, permitting contact of the fine particles of gold with the plates, but as the channel narrows down, the depth of water increases, and the current becomes correspondingly swifter, so that gold is apt to be swept away over the plates. Sluices and aprons, therefore, should be of the same width.

The frames of the plates, with the exception of the battery plate, which is supported on a casting bolted on to the mortar, should rest upon bearings independent of the framework of the battery, so as to avoid the jar which would otherwise ensue.

**Commencing Operations at a Mill.**—When starting a new mill it is very difficult ro obtain good results at the outset. The yield of the first crushing is very seldom satisfactory, and indeed never (one may almost say) comes up to expectation, as a fair percentage of the value of the ore is not obtained. A variable and sometimes large amount of gold must be absorbed by and remain upon the amalgamating plates of a new mill, and this item alone may considerably diminish the amount of gold obtained from the first run; but above all, the copper plates have to assume a proper working condition before they will do effectual work, and this takes several weeks, and sometimes months, as Californian experience has shown.

When starting a new mill it is always advisable, for the first twenty-four hours, to pass through the battery poor ores or waste, so as to fill up all the interstices and crevices in and around the dies and all other machinery through which the pulp has to pass, and the richer ores ought not to be crushed until the copper plates and mercury are in proper condition, so as to avoid a waste of precious metal.

**Amalgamation in the Battery** is conducted as follows:—Quicksilver is introduced through the charging slit (N, Fig. 5) from time to time, generally once in two hours, and in quantities dependent upon the richness of the ore. Care is taken to distribute it as evenly as possible under the stamps. The quantity of the quicksilver employed is, on the average, twice as much as is afterwards recovered in the amalgam, the amount lost in milling being one-twentieth of a pound, and sometimes more, per ton of rock crushed.

Finely-divided gold requires more quicksilver than coarse gold. Sulphuretted ores also require larger quicksilver charges than ores which contain no sulphurets. Ores of that character have a scouring effect on the plates, and also carry off, as loss, much quicksilver. The amalgam when discharged from the mortar must be sufficiently pasty in consistency to adhere to the plates, but not fluid enough to roll off. From one to two ounces of quicksilver are added to the battery for each ounce of free gold contained in the ore.

The quicksilver, being finely divided by the stamps, acquires the opportunity to coat or amalgamate the particles of gold. This fine division of the mercury is proved by the fact that two-thirds of the quantity charged generally escapes in the battery slimes. Through the violent motion of the battery water produced by the fall of the stamps, the particles of gold amalgam or quicksilver are carried with the pulp upon the copper plates, to which they have an opportunity to adhere. The quicksilver which escapes through the screens is thinly fluid, and contains but a small percentage of gold, while the amalgam of the plates inside the battery is either a pasty or hard layer; that which is thrown out of the screens generally contains some gold, and catches on the outside plates, where it gathers more and more gold, and grows stiffer and harder in consistency.

The movement of the pulp in the battery consists in alternate rising above and falling below the inner plates, combined with wave motions longitudinally along the battery box. Outside, the pulp streams with uniform velocity over the inclined table—a movement which does not, like that within the battery, present moments of rest favourable to the deposition of the gold. The only place where this is at all the case on the outer plates is the upper end of the apron, upon which the discharge falls from a height of several inches, the plates being arranged in several steps or drops. It is a great mistake to have one flat uniform inclined apron of copper plates in front of the battery. The more steps there are, the better for a successful amalgamation. Over the smooth plates the stream of water carries the small particles of gold along, and not being able to reach the bottom and adhere to the plates they are lost. It is therefore important to catch as much gold as possible inside the battery.

It is obvious that most of the coarse gold will settle in the mortar and remain sticking to the inner plates; while the fine gold, on the other hand, is lifted from the bottom immediately after being set free from the ore, and is held in suspension, together with the particles of amalgam, by the battery water, until it is caught on one of the inside plates, or is discharged through the sieve.

While the substances are thus floating about, it is difficult on account of the thinness of the pulp to unite the particles of gold and quicksilver; and this explains the fact that the quicksilver which escapes through the screens contains little gold.

With a copper apron having an inclination of 10° and a width of ten to twelve inches per stamp, one-fourth of a cubic foot of pulp would be passing over it every minute. The discharge moves over them in a very thin layer, which moves much more swiftly on the surface than on the bottom. Such particles of the gold and amalgam as sink with the larger and heavier pieces of ore to the bottom slide or roll slowly along, and have opportunity to adhere; but smaller particles are swept along by the more rapid surface current of the watery sheet of pulp, without being able to reach the bottom. To this latter class belong the greater part of the gold, especially of that which escapes through the sieves, and almost all the particles of quicksilver and amalgam. Moreover, the absolute quantity of gold which can come in contact with the outer plates is less than that which may touch the inner plates, by the amount retained upon the latter. For all these reasons, the inner plates must furnish the greater part of the gold obtained in the mill.

It should be added, further, that coarse gold will either not at all, or very seldom, be lifted up to the discharge level inside of the mortar, unless it remains under the stamp and becomes pounded into such a shape as not to longer withstand by its gravity the swash of the battery water, or becomes amalgamated. It may thus become reduced into fine particles and be discharged. Coarse gold, at all events, stands a good chance to remain inside the battery or to be thrown against the inner copper plates and stick there; whereas fine gold will be thrown on to the outer plates, where conditions for amalgamation are not so favourable as inside the battery.

The gold which has not been liberated during the crushing, but remains in the rock, evidently becomes a loss, as it will flow over the outside aprons to find its way to the tailing pit.

**Treatment of Tailings.**—The tailings which leave the mill are often submitted to concentration outside the mill by passing them through shallow sluices lined with canvas, to which the fine sulphurets cling, and off which they are washed periodically by means of a jet from a hose and nozzle for treatment by amalgamation in pans.

In these pans, besides quicksilver, nitrate of mercury and bluestone are used. The nitrate of mercury probably acts by precipitating a film of mercury upon the iron surface of the pan, and upon each particle of iron ground off from the shoes and dies. The mercury *in statu nascenti* probably tends, without extra energy, to attach itself to the liberated particles of gold. The bluestone, in all probability, assists in the formation of an amalgamated film on the iron, by coating it with copper; but as it is used only occasionally, and for the purpose of preventing a large loss of mercury, under certain conditions, its principal use seems to be to dissolve such particles of iron as have become amalgamated through the action of nitrate of mercury, and thus produce a powdery sort of amalgam, which, when the iron is replaced by copper from the bluestone, is converted into a soft, coherent amalgam, easily saved in the washing.

**The Edison Process.**—This process—which has been devised by the eminent American inventor whose name it bears—is designed for the treatment, with a view to amalgamation, of certain low-grade ores. It is described by the inventor, in a recent patent, as follows:—

"Heretofore, as is well known, no process has been devised by which the gold could be economically extracted from low-grade sulphide ores.

"Ordinary amalgamation is not successful, because the gold appears to be coated with an invisible envelope of material, which prevents amalgamation, and when it is attempted to obviate this difficulty by grinding the gold in amalgamating pans, excessive grinding is required and the mercury becomes floured, and very little gold is obtained. If it is attempted to roast the ore before amalgamating, the free sulphur formed in roasting will sulphurize the mercury and flour it, and thus prevent amalgamation.

"Another process is that of roasting the ore and dissolving out the gold by means of chlorine gas, but this is an expensive process, requiring a large investment in plant and highly skilled experts, and it has therefore never been successful to any great extent.

"The exact nature of the fine film upon the gold has not yet been satisfactorily determined. An Australian investigator has reported it to be sulphide of gold, but there seems to me some reason to suppose it otherwise, and I cannot state positively what it is.

"My process is based upon the fact that the coating upon the gold may be removed by the application of weak nitric acid, and that at the same time the surface of the gold may be amalgamated, if there is added to the weak acid solution a small quantity of a mercurial salt, especially nitrate of mercury.

"The process is as follows. The gold-bearing ore is first crushed in the ordinary manner to such a degree of fineness as will practically liberate the greater part of the particles of gold. Then, by water concentration, as by the use of jigging or vanning machines, the lighter matters are removed from the pulverised mass, and the sulphides are concentrated. When in this state, the sulphur of the sulphides is inactive on the nitrate of mercury solution, only the film on the gold being attacked thereby.

" A suitable quantity of the concentrated ore, several tons of it if desired, is then thrown into a suitable vat and the weak solution of nitric acid, which preferably contains also a small quantity of nitrate of mercury, is run on to it, preferably so as to just cover it. I have found that the application of the solution for about one hour is sufficient to remove the external film, and to cause all the gold particles to become fully amalgamated upon their surfaces. As much of the solution as possible is then drawn off, and the remaining mass of pulp is shovelled into a centrifugal drying machine of the well-known character employed in various other processes, such as the manufacture of sugar. The pulp being whirled around in the machine with great velocity, the centrifugal force throws the solution off the ore, leaving it entirely dry. The solution, being collected, is saved to be used over again, a small quantity of acid being added to make up for loss due to evaporation, and also if required a small quantity of nitrate of mercury.

" Instead of this there may be employed, for the purpose of removing and saving the expensive acid solution, the process of displacement by water. The pulp being placed in a vessel and water admitted above it, and the air exhausted at the bottom of the vessel, the water will, as is well known, filter through the pulp and displace the acid without mixing with it, the acid passing off through an outlet at the bottom and being preserved for the next operation.

" The ore, being removed from the centrifugal machine, or the displacing vessel, may now be amalgamated in any ordinary manner, either in an amalgamating pan or by running it over amalgamating plates, and in a few minutes the gold will combine wholly with the mercury, from whence it will afterwards be recovered in any ordinary manner.

" It is evident that the process can be carried out without the addition to the solution of the mercurial salt, the whole amalgamation being then accomplished afterward in the amalgamating pans or otherwise, but the previous surface amalgamation makes the final amalgamation more rapid and effectual, and the nitrate of mercury also assists in the removal of the film from the gold particles, and since the mercury is all retained by the gold, so that it is ultimately saved, it is better to use it.

" The strength of the solution employed depends to a great extent upon the character of the ore to be treated.

" With the ordinary concentrated sulphurets, a solution of 100 parts of water, 10 parts of ordinary nitric acid, and 5 parts of nitrate of mercury will be sufficient, but in some instances, the pulp will contain, even after concentration, matters which will reduce the nitric acid as well as the mercury, and thus in such cases a larger quantity of each of these ingredients will be required, the amount of which in each particular instance can be determined only by practice. The solution should however always be so weak as not to materially attack any portion of the pyrites.

" It will be seen that there is thus provided a commercially practicable method of working low-grade ores, by which the economical use of expensive re-agents such as nitric acid, is made possible, since the use of the centrifugal drier, or of the displacing process, allows all or practically all of the solution to be removed and saved, and thus the cost of the chemical agents required per ton of ore is reduced to an insignificant amount, and since the chemical amalgamation of the surface of the gold renders all grinding before the final amalgamation unnecessary.

" As has been stated, the nitrate of mercury helps to reduce the refractory film on the gold particles, and it is possible therefore to use this material alone without any acid; it is however much less rapid and effectual."

It would be very difficult to apply this process in distant mining localities, where nitric acid is too expensive to be used in any large quantity, as loss must occur even with a centrifugal machine; and I doubt whether the system—when applied to concentrated, *not* roasted pyrites—would extract a high percentage of the gold. The use of nitric acid would, no doubt, assist the amalgamation, as impurities would dissolve and the gold be rendered perfectly bright and made amenable to amalgamation.

One might have hoped that so distinguished an inventor, who has solved so many difficult problems in the application of electricity, would have proposed some electrolytic system for the treatment of gold ores—the object which is so generally regarded as desirable of attainment at the present time.

**Conditions of a Good Result from Amalgamation.**—These may be stated as follows:—

(1.) Very fine stamping if the gold is very fine.

(2.) A coating of gold amalgam on the copper plates. No amalgam should be removed until a hard layer of it has been formed on the plates.

(3.) The use of a proper quantity of water; too much will result in a coarse crushing, a less complete mechanical exposure of the fine gold, less contact of the gold and quicksilver, and a premature sweeping off of both. The addition of too much quicksilver, though causing the formation of more amalgam, will only lead to its being swept away by the current.

(4.) Proper temperature of the battery water. The water ought neither to be too cold nor too hot: 90° to 110° Fahr. is the best, if by artificial means it can be raised to it.

(5.) The addition of quicksilver in proper quantity. This condition should be closely studied in each mill. There is always a considerable loss of quicksilver. The quicksilver when introduced in the battery is finely divided by the stamps, and thus affords an opportunity for the amalgamation of the fine particles of gold. By the violent motion of the water in the battery, produced by the fall of the stamps, particles of gold, amalgam, and quicksilver are carried with the pulp to the copper plates, to which they adhere.

(6.) Proper height of the charge in the mortar. This should not be allowed to rise higher than about three inches below the lower edge of the inner plates. If the quartz and pulp in the battery come nearer the plates, too much stuff, which is also too coarse, is thrown upon them, thus either preventing the accumulation of amalgam or displacing it after collection.

(7.) Regular feeding.

(8.) Care in keeping the plates clean.

(9.) Care in the mill against the introduction of grease or greasy substances, and against the use of exhaust steam for heating water required in any of the amalgamating processes. In lubricating the cam shaft, journals, cams, tappets, or any other portions round the batteries, care must be taken not to drop any lubricant into the mortar.

(10.) Rejection of hydrated oxidised iron ores, silicate of magnesia, and alumina ores: they cause a frothing of the water, and coat the gold with a slime which resists amalgamation.

(11.) Avoidance of mineral waters for battery amalgamation, especially if they contain sulphur in the shape of sulphuretted hydrogen, as a coating will be formed on the gold particles which prevents amalgamation.

(12.) Care that the amalgam on the copper plates is not allowed to get too hard, as it may fail to catch the gold. If, therefore, the amalgam should get too hard, it will be well to sprinkle through chamois cloth some globules of quick-silver over the amalgam. If, however, it becomes too soft again, there is danger of "flouring" and losing it, with some of the gold.

(13.) A dilute solution of cyanide of potassium should always be kept at hand, and when yellow spots appear on the plate some of the solution should be poured over it. If this does not remove a spot, hold a lump of the cyanide over it, or rub it, which will have the desired effect.

(14.) If the ores contain soluble sulphates, arising from the decomposition of iron or copper pyrites, the addition of lime will prove beneficial while passing through the battery.

(15.) When treating gold ores containing manganese, it is necessary to clean the plates of its adhering amalgam at least once a week and give them a fresh coating of quicksilver.

(16.) It will be found that in many cases amalgamation can be promoted by discharging from the battery on to concentrators direct, which will collect all the heavy mineral particles which interfere in the ordinary amalgamating process, and submitting the concentrates to separate treatment. The overflow or tailings which pass the concentrators, if carried over copper plates, will give up the gold easily now, in case any fine particles have escaped the concentrating process, as the ore has undergone a cleaning process in the separation of the sulphuretted ores, which naturally interferes with copper-plate amalgamation. If the concentrations are submitted to treatment in grinding pans, the tailings from the settler ought to be run over slime tables to collect any escaping mineral particles of value.

(17.) When treating heavy pyritic ores, it will be found advantageous to have an iron pipe with little holes, discharging on the outside copper plates a fine, gentle shower of water, so as to assist the carrying off of the heavy sulphurets, which would cover the plates over and prevent the free gold coming in contact with them.

**Bad Condition of the Plates.**—One of the first conditions in the mill should be the keeping of the amalgamated copper plates in a bright condition, as a discoloration of the plates, indicating oxidation of the copper, is one of the causes of great losses of gold. Discoloration generally takes place when starting with fresh plates; let them be coated with silver or not, and I have thoroughly investigated the subject, and on several occasions I have found that the only remedy to apply is to keep brushing the plates with cyanide, even while the pulp was flowing over them, till I got a good coating of gold on the plates. Of course, this operation of continually brushing the plates will cause losses of gold, and it takes ten to twenty days before the plates will get into proper working condition, but when once the protecting coat of gold is formed, no further trouble will arise, and the plates will remain bright and require dressing once or twice in twenty-four hours.

I also advise not to scrape off the gold too closely when cleaning-up, but leave a thin crust of hard amalgam on the plates —it protects the copper from oxidation, and is the best gold-catcher we have, as the gold particles coming from the mortar have more affinity for the gold amalgam than for clean mercury.

Electro-plated copper plates suffer at the beginning just as much from discoloration as pure copper does, and they are preferable for the reason that they amalgamate easily, but I question very much if they catch more gold than an ordinary copper plate when once in proper working condition. The discoloration, I believe, arises mostly from the soluble salts existing in most ores arising from the decomposition products of the sulphides and arsenides. To satisfy myself on this point, I experimented once in a mill running ten stamps on surface ores carrying a large per cent. of oxides and the plates discolored, whereas the pyritic ores from another mine passed through the other ten stamps gave no trouble. I also noticed in some mills that the silvering right under the lip of the mortar scours away to the copper, showing big red patches, which had to be sprinkled over with mercury very often, and this, no doubt, is occasioned by the crystals of pyrites.

It will also be noticed that pyritic ores will cover the plates with fine particles of pyrites, and these have to be brushed off occasionally as they prevent the gold particles to come into contact with the amalgamated copper surface.

Mr. C. H. Aaron, in the *Engineering and Mining Journal* of August 10th, 1889, gives the results of some experiments with copper plates which are very interesting. He says :—

"Pieces of sheet copper were coated with quicksilver, and exposed to immersion in Spring Valley water, which supplies San Francisco.

"The plates which were amalgamated with the aid of nitric acid and mercuric chloride became tarnished in a few minutes, and, on being cleaned with solution of potassium cyanide, washed in water, and again exposed, were again promptly tarnished, and so on indefinitely. Plates amalgamated by means of potassium cyanide, not using nitric acid, resisted tarnishing during the first hour, after which they behaved in the same manner as those on which nitric acid were used.

"Tarnishing was lessened by the addition to the water of slaked lime, and was prevented by a small quantity of ferrous sulphate, also by a trace of potassium bisulphate, not at all by caustic potash. Tarnishing was permanently prevented by slowly, care being taken not to allow them to get red hot. When the proper heat is got on them the gold scale will rise in blisters; when this takes place the plates should be taken off the fire and the gold scraped off.

Another way, if a trough is available (wider than the plates), is to fill it half-full of boiling water, and when the plates are taken out, dip them in the water, when the gold will scale off. If there should be any part of the plate on which the gold has not risen, these parts should be again rubbed over with the solution and again fired. After the gold scales are collected they should be put in an earthenware dish and covered with nitric acid until all the copper is dissolved, and when this is done the gold can be smelted in the usual way, but corrosive sublimate should be put in the crucible (after the gold is melted) until there is no more blue flame given off.

I hardly think that I would recommend anybody to use this process while the mill is in operation. It is a good policy for the millman to leave gold in absorption in the plates, as plates which are well set catch the most gold. When a mine is abandoned, or a mill has no more work to do, then this method can be resorted to. In nine cases out of ten the plates get "buckled"; the copper, if heated the least degree too much, will get hard and difficult to "set" again, and I, therefore, am no advocate of the shelling system.

If the millman wants to get more gold than he can obtain by simply scraping the plates, let him turn on a little steam and heat the plates; this steaming process should always prove quite sufficient, but I only recommend this in extreme cases. I have found that it is a good policy to leave some gold amalgam on your plates when starting a fresh run.

Another method of recovering the amalgam from the plates is to pour boiling water upon the plates, or immersing them.

**Pit-Head Arrangements.**—In the annexed Plate IV. will be found drawings to scale of pit-head and ore-bin erected by Messrs. Fried. Krupp Grusonwerk at the incline shaft of the Main Reef mine, Johannesburg.

# CHAPTER III.

## *TREATMENT OF GOLD-BEARING ORES: CRUSHING AND AMALGAMATION—continued.*

CLEANING THE MILL—Cleaning the Quicksilver—Sodium Amalgam—Retorting the Amalgamated Metals—Value of the Amalgam—Workmen in the Mill and their Duties—Wear and Tear of a Mill—Cost of a Complete Mill—Cost of Milling—Working of Gold Ores in Californian Gold Mills—Statistics of Queensland Gold Mines—Mining of Gold Ores in California—Results of the Mill Process—Loss in Milling—Mr. Paul on Waste of Gold in Amalgamation—Checking the Mill Returns—Mr. Skey on the Absorption of Sulphur by Gold—Conditions of Successful Working of a Gold Mine—Amalgamation of Concentrated Sulphurets after Roasting—Iron Battery Frames—Illustrations and Specifications of Batteries—Power required for Wet Crushing Gold Mills.

**Cleaning the Mill.**—The mill is cleaned every week. All the contrivances for amalgamating are usually cleaned every week, while the battery gold is removed once every few weeks, and on that day the mill is stopped for repairs. After removing the skimmings which float on the quicksilver baths of the amalgamators, the quicksilver is scooped out of the trough into a bucket and the amalgam taken up. The amalgam forms a layer at the bottom of the trough in the Attwood amalgamator, or at the bottom of the Hydrogen amalgamator. The upper trough contains 95 per cent. of amalgam in the apparatus. In order to free the amalgam from impurities, which consist principally of sulphides of iron, copper, and lead, precipitated with the sinking gold, it is worked by hand in a bath of quicksilver. The small lumps of amalgam are broken, and the impurities floating on the quicksilver removed by means of a cloth. After thorough cleaning, the quicksilver charged with the amalgam is pressed through a strong thick piece of canvas, and the remaining amalgam formed into balls of about 2½ in. diameter, weighing about thirty-five ounces each.

The quicksilver which has been removed from the amalgam in the troughs is allowed, unless a final clean-up is desired, to flow back again into the troughs without being filtered through canvas. The quicksilver only contains a small amount of amalgam. The quantity of quicksilver in the trough of the Attwood amalgamators is about 700 lbs.

The riffles are cleaned in a similar manner to the amalgamators. The amalgam is first removed by passing a small scoop slowly and closely to the bottom, and thus allowing the quicksilver to escape at the sides. The skimmings from the surface of the riffles are added to those of the amalgamators for treatment in the pan.

The amalgam on the copper plates is removed by means of a dull chisel, which operation must be carefully performed so as not to expose the copper. When the amalgam is removed, quicksilver is sprinkled on the plates and spread over it by means of a piece of rubber belting. The surface is finally washed with clean water.

The amalgam from the copper plates is freed from impurities by rubbing it in an iron mortar, with the addition thereto of quicksilver, while a current of water kept flowing into the mortar carries off the impurities. In larger establishments, this operation is usually carried out in a small clean-up pan.

The scum, sand, sulphurets, &c., removed from the amalgam by washing, are treated in the amalgamating pan with the skimmings.

In cleaning the battery, the stamps are removed from the mortar by means of a block and tackle. The corner die, provided near the foot of the cylindrical body with a wedge-shaped recess for the introduction of the crowbar, is taken out first. All the casings are removed in the same way. Dies and casings are raised and taken out of the battery. The battery is then cleaned and the sands washed by passing through a sieve, having eighty-one to one hundred holes per square inch, into a wooden hopper, through which they are discharged into a wooden sluice box, 6 in. wide and 6 ft. long. This latter is provided with three or four riffle boards which catch the particles of gold. The sluice has an inclination of $1\frac{1}{2}$ in. to the foot. The coarse sands from the sieve, after removing the iron by hand or by a magnet, are returned to the battery, while the finer sands which passed through the sieve are treated in the amalgamating pan; but if of a poor quality these also go back to the battery. The gold taken out of the riffles is freed from sand and iron by treatment in the pan and by the magnet, and is added to the amalgam.

While the mortar is empty, any worn-out dies, casings, and shoes are replaced by new ones. Before the old shoes and dies are returned to the foundry for recasting they should be examined for gold, which is frequently found in the crevices.

**Cleaning the Quicksilver.**—One of the essential conditions after a clean-up is made is to bring the quicksilver into an active, lively condition, and if possible to maintain it in this condition during the running of the mill. It takes very little to tarnish the quicksilver, and every mill-man has his own method of cleaning it. Some pour a little water on top of it, dissolve in it lumps of cyanide of potassium, and then stir up the quicksilver. Some use concentrated lye; others dilute sulphuric or nitric acid; but if quicksilver gets very foul and dirty, I prefer distilling it in the retort, which insures success although a little troublesome and expensive in fuel.

Where electricity is obtainable for the purpose, the following method will work very effectively. Take a flat earthenware vessel, pour some sulphate of soda solution into it, and lay a long strip of lead across this vessel, in such a manner that it can be bent in the centre to plunge into the solution. Into this vessel place a semispherical non-glazed earthenware basin, large enough to fit the bottom vessel, and into this one pour the quicksilver with a little water. Put the positive wire in contact with the lead strip, which constitutes the anode, and the negative wire with the quicksilver. Evolution of hydrogen will take place, and the quicksilver will be seen to assume an active motion and agitation, with a brightening of the metal.

**Sodium Amalgam.**—This preparation—the invention of Mr. William Crookes, F.R.S.—has proved to be a very excellent ingredient in keeping the mercury active, although it has failed to accomplish the marvels which on its first introduction were anticipated from its use. Miners expected to extract the gold from iron pyrites, arseniurets, tellurides, by simply adding the sodium amalgam to the mercury in pan amalgamation, without roasting or any other chemical preparation.

Mr. Crookes gives the following directions for its prepation:—Amalgam A is a simple mixture of sodium and mercury in the proportion of 3 parts of sodium to 97 of mercury. The preparation of the mixture is as follows: Place a strong iron flask with a narrow neck nearly up to the mouth in sand on a sand bath kept at a temperature of about 300° F.; weigh out the mercury and sodium, put the former into the flask, and then add the sodium in pieces as large as a pea at a time, waiting for the action to cease before adding a fresh lump. The sodium had better be dropped in with a pair of tongs, and the hand should have a cloth over it. At each addition of the sodium a slight explosion is heard, and a bright flame will issue from the mouth of the flask. The action gets less violent as the mercury becomes richer in sodium. When the whole of the sodium has been added, pour the amalgam into a flat dish whilst still liquid, and when cold break it up and preserve in a stoppered jar. It will not require to be kept under naphtha. The amalgam forms a mass of long, needle-shaped, brilliantly metallic crystals, which interlace in every direction, but have very little cohesion. Amalgam B and C are specially prepared with an addition of zinc.

The inventor recommends the use of amalgam A in coarse powder. By sprinkling it over the wetted surface to be amalgamated, then rubbing it over with a little clean mercury, a firmly adherent and brilliant coating of mercury will be given to the metal. Not only can copper be amalgamated in this way, but the same result is produced on galvanised iron, tin plate, or lead, and less perfectly on iron and steel. In all cases it is advisable to preserve the effective surface and the amalgamating energy of the mercury on the metal plates by an occasional sprinkling of powdered amalgam A, applied from time to time as required.

I have found sodium amalgam very beneficial in pan amalgamation, as it has a tendency to keep the mercury bright and active. I never made comparative tests to ascertain whether by its use an increased yield of precious metal is obtained, but results published by eminent authorities indicate that a larger percentage has been obtained by adding the sodium amalgam to the mercury.

**The Use of the Retort.**—Amalgamation being complete, another process becomes requisite for the separation of the quicksilver and the amalgamated metals. The amalgam, which contains from 36 to 40 per cent. of gold bullion, is put in cast-iron cylinders or retorts. The cylinders are supported by brickwork on flanges, and are placed almost horizontally, having only a slight inclination toward the rear end, which gives the quicksilver, condensed on the front cover, an opportunity of flowing back to this end to be surrounded by the fire. The rim of the front plate or cover of the retort is provided with soft fine clay, and is firmly screwed to the rim of the retort by means of a clamp, so as to make it air-tight.

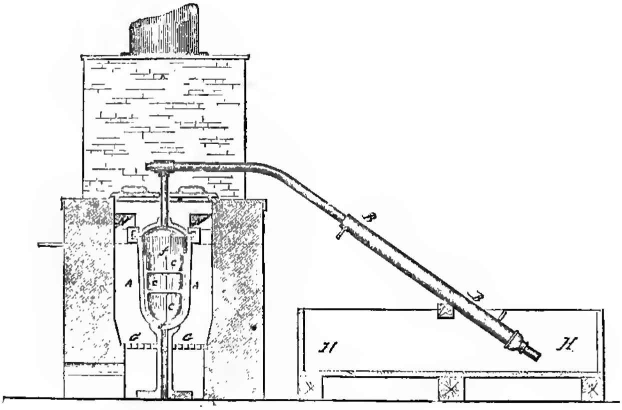


Fig. 48.—Retort for Large Mills.

Fig. 48 represents a style of retort used in large gold mills. The same contains three cups, having a tube in the centre for the escape of the quicksilver into the condenser B B. A A is the retort; *c c c* are the cups, *f f* are the central openings for the escape of the fumes; G G is the fire-grate; H H, wooden trough filled with water to collect the quicksilver; N N, flues leading into chimney.

A cylindrical horizontal retort of about 11 in. in diameter and 4 ft. 7 in. long will hold five trays, having a capacity to hold 1,350 ozs. of amalgam. The fire-grate necessary for a retort of this size measures 1 by 2 ft., with a chimney 4 by 8 in. in section. The quicksilver condenses in an iron pipe which is surrounded by cold water. The pipe is attached at the rear end near the top of the retort, and descends into a water basin. The heat, which is generated by wood in the grate under the front of the retort, passes through a return flue over the same into the chimney placed over the front of the retort. When the retort has been at a cherry-red heat for two hours, the retorting is considered complete. The time for this process, which varies with the quantity of amalgam in the retort, is from four to six hours, commencing with a cold furnace. After giving the retort time to become properly cooled, the front plate is removed. The gold bullion consists of a somewhat porous mass, and when cleaned has a bright yellow surface. It still contains a small amount of quicksilver, and a few sulphides.

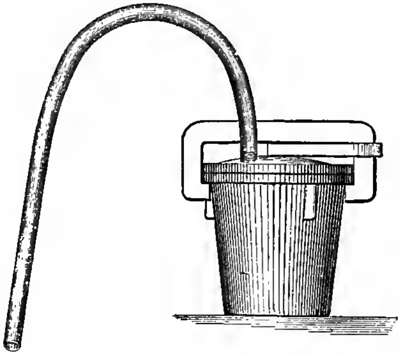


FIG. 49.—RETORT FOR SMALL MILLS.

Fig. 49 shows a small retort, such as generally used in small mills, by gold miners in general. The cover and retort are turned and ground to fit each other perfectly. The bale is of wrought iron, and takes hold under the flange on the retort; the wedge is then driven in between the cover and bale, making a secure and perfect joint.

The bullion is melted in black-lead crucibles, placed in air furnaces, usually 16 in. deep and the same in diameter, by means of charcoal. The fluxes chiefly used are carbonate of soda, borax, saltpetre, and sand. The loss in melting the bullion is 1 to 1½ per cent. The bars are from 700 to 940 fine, and contain, in addition to the usual silver, small quantities of copper, lead, and iron.*

**Value of the Amalgam.**—The amalgam from the inside plates, the linings of the battery, &c., invariably contains more gold than that from the outside plates. The value of the amalgam increases with the coarseness of the gold in the ore; finely divided and alloyed gold yields comparatively poor amalgam. At some gold mines the value of the plate amalgam averages about £1 per ounce, and the value of the battery amalgam about £1 16s. per ounce; but amalgam from ores of the same mine will sometimes vary greatly in value in different clean-ups.

**Workmen in the Mill and their Duties.**—One man is required at the rock-breaker during a 10-hour shift, and he generally has an assistant to remove the rock. One man will be required to feed each 10-stamp battery if no self-feeders are used. One man to attend to the washing of the blankets, and also to regulate the quantity of water passing over the blankets, to watch the proper discharge of the sieves, and to feed the blanket washings to the amalgamators, and to regulate the flow of water through them. There ought to be one blanket washer to every five stamps.

The attendance upon the amalgamating machinery devolves upon the amalgamator, who also prepares the screens, attends to the cleaning of worn-out shoes and dies, and various other minor duties in the retorting and melting department.

---

* See Chapter XVIII., on the Melting and Assaying of Gold.

When steam is the motor, two engineers attend to the engine and boilers of a large mill in 12-hour shifts.

There is usually a night watchman.

There is also a general manager of the works.

**Wear and Tear of a Mill.**—In a properly organised and well-managed mill accidents are of rare occurrence, although occasional breakages are unavoidable, considering the strong vibrations and jars to which all parts of the battery are subjected. Breakages are confined to the stems, the shoe shanks, and screens. The splitting of the tappet by wedging does not often occur.

The loss of quicksilver varies, but it is estimated to average 1 lb. for every 30 tons of rock crushed.

Where battery amalgamation is practised, heavily sulphuretted ores—and especially ores carrying galena and arsenical pyrites—occasion a large loss of quicksilver, and loss of gold undoubtedly accompanies the loss of quicksilver. A small part of the loss in quicksilver occurs through carelessness in handling this agent, and in retorting. Even after retorting, the retorted gold often contains some quicksilver which has not been sublimed during the operation, and which is lost in the succeeding operation of melting the gold. The loss of quicksilver generally occurs through its being "floured," and floated off with the water in very finely divided particles.

At the Keystone Mill in Amador County, California, 1,276 troy ounces of quicksilver were lost monthly in the mill. This quantity at 14·58 troy ounces for 1 lb. avoirdupois, would make 87½ lbs. at 2s. 6d. lb. is worth £10 18s. 9d. If 3,000 tons of ore were treated monthly, this would be equivalent to $\frac{42}{100}$ of an ounce per ton of ore treated.* At some mines the loss of quicksilver is as low as $\frac{18}{100}$ of an ounce per ton of ore worked.

The wear of the blankets is over a yard a month.

A 5-stamp battery requires on an average 13 sets of screens a year. A set consists of 5 sheets of from 1 to 1¼ sq. ft.

To run a 30-stamp steam mill requires from 12 to 16 in.

---

* Report of the Californian State Mineralogist.

of water (miner's measurement), *i.e.* the discharge of an aperture of 12 to 16 square in. under a 6-inch pressure measured from the centre of the aperture. The water is usually supplied by ditches at so much per inch (miner's measurement).

It takes 5 to $6\frac{1}{2}$ cords of fire-wood to furnish the steam necessary for a 30-stamp mill.

A good shoe lasts from twenty-one to forty-three days (on an average thirty-three days), and crushes 79 tons of rock. The wear is $1\frac{1}{8}$ lbs. of iron per ton of rock.

The die lasts on an average seven weeks, crushing 100 tons. The wear is $\frac{6}{10}$ lb. of iron per ton of rock.

In most mills which are remote from foundries, where transportation is an important item in the cost of shoes and dies, steel shoes and dies have replaced those of iron. Of late, chrome-steel has been introduced for shoes and dies, and has proved its superiority over most other kinds of steel used for the same purpose. Iron dies wear more evenly with steel shoes than do the steel dies, and in some mills steel shoes and iron dies are used for this reason.

The life of steel shoes and dies is about two and a half to three times the life of those of iron, while the cost is about twice as great. There are several conditions which affect the durability of shoes and dies—as, for instance, the hardness of the rock; the weight, speed, and height of the drop of the stamp; the manner of feeding the ore, etc. Iron shoes of good quality last from thirty to forty-seven days; iron dies of good quality from thirty to forty days. Old shoes wear usually down from one and a half to one inch in thickness, and weigh about twenty-five to forty pounds. Old dies usually wear down to about an inch or half an inch in thickness, and weigh from twenty to fifty pounds.

The consumption of iron or steel in shoes and dies will depend upon the coarseness of the stamping and the height of discharge. Dies wear less rapidly than the shoes, as they are protected by the thickness of the pulp, which covers them to the depth of from one or one and a half to three inches. But while the actual wear of dies is less than that of the shoes, the life of the dies is shorter than that of the shoes, owing to the fact that the shoes have several inches of greater length of wearing part than the dies. The consumption of iron (as above stated) amounts to six-tenths of a pound per ton on the die, but with ores of certain characteristics the consumption for shoes and dies amounts from one to three pounds per ton of ore crushed. To obtain the maximum crushing capacity of the battery the dies must be kept to as full a height with reference to the lower edge of the screens as may be compatible with the safety of the screens and with successful amalgamation in the battery. The dies should be maintained as much as possible all on one level, as where one die in the battery projects much above the others little or no pulp will remain upon it, and the shoe will in consequence drop upon the naked die.

The stem generally breaks square across the fibres, near the upper face of the head, and lasts, without breaking, about sixty weeks, crushing 864 tons. When the irons are new and of fine quality the breakages are less frequent, occurring perhaps but once in a hundred and twenty weeks. Rewelding, including the necessary new iron, costs on an average something over £2. The stems as well as the cams last at least ten years. The battery linings last six months, and the tappets from two to three years.

The dies of rock-breakers, weighing 250 lbs. each, last about three months. The wear is one-half of their original weight. The false pan bottoms last a year, and a set of four pan shoes three months.

**The Cost of a Complete Mill**, including engine and boiler, is usually estimated at $1,000 (or over £200) per stamp. In a large mill having 20 stamps this includes the concentrating and the chlorination works.

**The Cost of Milling.**—The full cost in a 30-stamp steam mill, the stamps weighing 850 lbs. each, with 61 drops of 10 ins., and having a crushing capacity of 72 tons per day, is

$2·04 (or about 9s.), but this does not include the expense of concentrating the tailings.

In a smaller mill, having the same expense of engineers, the milling expenses are, of course, proportionately higher.

Mr. Hague* gives the cost of crushing in a small mill as follows, for a ton of ore:—

| | s. | d. |
|---|---|---|
| Labour | 5 | 0 |
| Fuel | 4 | 0 |
| Castings and ordinary repairs | 1 | 5 |
| Quicksilver | 0 | 3 |
| Water, when purchased | 0 | 10 |
| Total | 11 | 6 |

Mr. Reichenecker* gives the cost of crushing in California per ton, as follows:—

| | s. | d. |
|---|---|---|
| Labour | 3 | 2 |
| Fuel and engineers | 7 | 0 |
| Castings and ordinary repairs | 2 | 0 |
| Quicksilver | 0 | 6 |
| Sundries | 0 | 4 |
| Total | 13 | 0 |

How cheaply ores can be mined and milled when water-power is used and timber is cheap, and large quantities of ores are treated, appears from the published reports of the Plymouth Consolidated Mining Company of California. They have two mills of 80 stamps each, making 160 in all, which crush three hundred and twenty tons a day; and the cost of production in 1887 is stated to have been as follows:—

| | s. | d. | |
|---|---|---|---|
| Mining | 9 | 5 | per ton |
| Milling | 3 | 0 | ,, |
| Saving and reducing sulphurets | 0 | 9 | ,, |
| General expenses, office, taxes, and prospecting | 0 | 9 | ,, |
| Total cost of mining and milling | 13 | 11 | ,, |

Average yield of gold per ton of ore in 1887, £1 10s. 5d. per ton.

---

* Quoted by Professor Raymond in his Report on Mines and Mining.

TABLE SHOWING ECONOMIC FEATURES IN THE WORKING OF GOLD ORES IN SOME OF THE PRINCIPAL CALIFORNIAN GOLD MILLS.*

|  | Plymouth Con., Amador County. | Loyal Lead, Amador County. | Sheep Ranch, Calaveras County. | Hite Mine, Mariposa County. | Idaho Mine, Grass Valley. | Plumas Eureka, Plumas County. | Sierra Butes, Sierra County. |
|---|---|---|---|---|---|---|---|
| Steam or water power . . . | water | water | steam | water | water | water | water |
| Number of inches water to drive mill . . . . . . . . . | — | 28 | — | 500 | 320 | — | 80 |
| Pressure of water or head in feet | 550 | 260 | — | 46 | 542 | 260 | 530 |
| Number of stamps . . . . . . | 160 | 10 | 30 | 40 | 35 | 60 | — |
| Number of horse-power if steam mill . . . . . . . . . | — | — | — | — | — | — | — |
| Weight of stamps . . . 80 | 750 | — | — | — | — | — | — |
| 80 | 1000 | 700 | 800 | 800 | 850 | 850 | 850 |
| Drop of stamps in inches . . . | 6 | 7 | 7 | $6\frac{1}{2}$ | $9\frac{1}{2}$ | $8\frac{1}{2}$ | 8 |
| Duty per stamp in 24 hours . . | 2 | $1\frac{1}{2}$ | $2\frac{1}{4}$ | $1\frac{3}{8}$ | — | $2\frac{1}{2}$ | $2\frac{7}{10}$ |
| Drop of stamps per minute . . | 90 | 85 | 85 | 80 | 72 | 80 | 85 |
| Kind of shoe and dies . . . . | iron | iron | white iron | — | — | steel | steel |
| Size and character of screens, No. | 8 | 6 | 9 | 6 | 6 | 8 | 7 |
| Inches of water used in battery . | — | 2 | 10 | 12 | 30 | — | — |
| Dimensions of apron in inches . | — | 48 × 48 | 48 × 120 | 48 × 72 | — | — | — |
| Width of sluice in inches, holding copper plates . . . . . | — | 14 | 16 | 12 | — | 15 | 15 |
| Length of sluice in feet, holding copper plates . . . . . . . | — | 12 | 100 | 14 | — | 20 | 20 |
| Percentage of gold saved in battery . . . . . . . . . . | — | 80 | 80 | 80 | — | 60 | 80 |
| Percentage of gold saved on plates . . . . . . . . . | — | 20 | 20 | 20 | — | 40 | 20 |
| Percentage of sulphurets in ore . | $1\frac{1}{2}$ | 1·66 | $\frac{1}{2}$ | 2 | $1\frac{1}{4}$ | — | $\frac{6}{10}$ |
| Value of sulphurets per ton in £ | 30 | 14 | 6 | 16 | 6 | 6 | 20 |
| Cost of milling per ton of ore . | 3s. | 6s. | 4s. | 1s. | — | 2s. | 1s.6d |
| Per cent of value extracted from pyrites . . . . . . . . . | 94 | 90 | — | — | — | — | 93 |
| Number of men employed in mill | — | 3 | 7 | 5 | 17 | 12 | — |
| Average wages in mill . . . . | — | 12s. | 12s. | 12s. | 12s. | 12s. | — |
| Cost of water per inch . . . . | — | 10d. | — | — | — | — | — |
| Wear of iron on shoes and dies in battery per ton of ore . . | 4 | — | 3 | — | — | 1·06 | 2 |
| Cords of wood per day to drive mill . . . . . . . . . . | — | — | 4 | — | — | — | — |
| Average yield per ton of ore . | 30s. | — | — | — | — | 32s. | — |
| Cost of mining and milling per ton of ore . . . . . . . . | 12s.3d | — | — | — | — | — | — |
| Loss of quicksilver per ton of ore, ounces . . . . . . . | — | $\frac{48}{100}$ | — | — | — | $2\frac{7}{10}$ | — |

---

* Compiled from the Reports (1888) of the California State Mineralogist.

SOME STATISTICAL DATA RELATING TO THE QUEENSLAND GOLD MINES FOR THE YEAR 1887.

| Gold Field. | No. of Tons Crushed. | Gross Yield in Ounces. | Average per Ton. | | | Number of Miners employed in Quartz Mines. | Average Price of Gold per Ounce. | | | Price of Crushing per Ton of Ore in Custom Mills. | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Oz. | Dwt. | Qr. | | £ | s. | d. | £ | s. | d. |
| Palmer | 1,477 | 1,981 | 1 | 6 | 20 | 127 | 3 | 17 | 0 | 0 | 18 | 6 |
| Hodgkinson | 2,688 | 2,400 | 0 | 17 | 20 | 60 | 3 | 13 | 8 | 0 | 17 | 6 |
| Mulgrave | — | — | | — | | 12 | | — | | | — | |
| Etheridge | 8,100 | 20,277 | 2 | 10 | 1 | 500 | 3 | 0 | 0 | 1 | 5 | 0 |
| Woolgar | 694 | 709 | 1 | 0 | 10 | 20 | 3 | 10 | 0 | 1 | 5 | 0 |
| Cloncurry | — | — | | — | | 20 | | — | | | — | |
| Charters Tower and Cape | 83,292 | 151,060 | 1 | 16 | 6½ | 1,960 | 3 | 6 | 0 | 0 | 13 | 6 |
| Ravenswood | 6,589 | 6,843 | 1 | 0 | 18 | 177 | 3 | 8 | 0 | 0 | 7 | 6 |
| Clermont | 8 6 | 310 | 0 | 12 | 19 | 11 | 3 | 15 | 0 | 0 | 16 | 0 |
| Rockhampton Field | 17,645 | 85,205 | 4 | 16 | 13¾ | 530 | 4 | 2 | 0 | 0 | 17 | 6 |
| Gladstone Fields | 1,067 | 2,610 | 2 | 7 | 19 | 33 | 4 | 3 | 0 | 0 | 18 | 6 |
| Gympie and Kilkivan | 90,673 | 96,939 | 1 | 1 | 9 | 1,450 | 3 | 10 | 6 | 0 | 9 | 0 |
| Gayndah Fields | 3,151 | 3,348 | 1 | 1 | 6 | 141 | 3 | 10 | 0 | | — | |
| Penningering and other small fields | 1,398 | 671 | 0 | 9 | 15 | 204 | | — | | 1 | 0 | 0 |
| Croydon | 10,951 | 31,787 | 2 | 18 | 1 | 2,000 | 2 | 13 | 6 | 1 | 10 | 0 |
| Eidsvold | 27 | 84 | 3 | 3 | 0 | 670 | 3 | 12 | 6 | | — | |
| Totals | 228,238 | 404,223 | 1 | 15 | 10 | 7,915 | | — | | | — | |

### TABLE SHOWING ECONOMIC FEATURES IN THE MINING OF GOLD ORES IN SOME OF THE PRINCIPAL CALIFORNIAN GOLD MINES.

| | Amador County. | | | Calaveras County. | | Mariposa County. | Grass Valley. | Plumas County. | Sierra County. |
|---|---|---|---|---|---|---|---|---|---|
| | Plymouth Con. Mg. Co. | New London Mine. | Loyal Lead Mine. | Sheep Ranch Mine. | Flew Gold Mg. Co. | Hite Mine. | Idaho. Mine. | Plumas Eureka. | Sierra Buttes. |
| Total length of claim in feet | 4,000 | 12,000 | 2,046 | 1,600 | — | 3,000 | — | 2,100 | 11,000 |
| General course of vein | N. & S. | N. & S. | N. 30°. W | W. 15° N. | N. 81° E. | E. & W. | — | N. & S. | E. & W. |
| Dip of vein | East | East | N. E. | East | S. W. | N. | S. | W. | N. |
| Angle of dip from horizon | 56½ | 65° | 50° | 33° | 85° | 75° | 64° | 75° | 41° |
| Number of ore shoots | 1 | 2 | 2 | — | — | — | — | 2 | — |
| Width of ore body in feet | — | 7 | 16 to 40 | 2 | 4 | 8 | — | 6 | 20 |
| Character of ore | quartz | slate and quartz | quartz | — | quartz | — | — | — | — |
| Length of ore shoot or pay chimney | 800′ | 260′ | 280′ | 1,200′ | 240 | 800 | 1,200 | 450 | — |
| Vertical depth reached in mine, feet | 1,600 | 1,045 | 200 | 1,050 | 400 | 900 | 1,790 | 1,500 | 2,200 |
| Character of hanging wall | black slate | slate | — | grey slate | mica schist | slate | — | slate | slate |
| Character of foot wall | black slate | slate | diorite | quartzite | diorite | slate | — | syenite | calcareous slate |
| Quantity of water raised in 24 hours, gallons | 18,000 | 26,000 | — | 103,000 | 144,000 | 7,000 | 160,000 | — | — |
| Cost of sinking (labour and powder) per foot of shaft | £3 15s. | — | — | £6 | £4 16s. | £1 13s. | — | — | — |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Cost of mining one ton of ore | 9s. 5d. | 9s. | — | 12s. | — | 8s. | — | 13s. | 14s. 3d. |
| Dimensions of timbered shaft, feet | — | 4 × 9 | — | — | — | — | — | — | — |
| Average feet sunk monthly in shaft | — | 60 | — | 33 | 45 | — | — | — | — |
| Average feet run monthly in each drift | — | 80 | — | — | 30 | — | — | 85 | 60 |
| Cost of timbers 16′ long × 20″ diameter | 10s. | 10s. | 6s. | — | — | — | — | 6s. | 6s. |
| Cost of cord of wood | — | 16s. | — | 16s. | 16s. | — | — | — | — |
| Cost per 1,000—5 foot lagging | — | £14 | — | — | — | — | — | — | — |
| Cost per 1,000—6 foot lagging | — | £16 | — | — | — | — | — | — | — |
| Dimensions of hoisting works, boilers | — | 54″ × 16″ | — | — | — | — | — | — | — |
| Diameter and stroke of hoisting works, engines, cylinder | — | 12″ × 16″ | — | — | 6″ × 10′<br>12″ × 14″ | — | — | — | — |
| Diameter of wire rope for hoisting works | — | $\frac{3}{4}$′ | — | — | — | — | — | — | — |
| Height of gallows frame at hoisting works | — | 40″ | — | — | 100 | — | — | — | — |
| Cords of wood burnt at hoisting works | — | $1\frac{1}{2}$ | — | — | 3 | — | — | — | — |
| Quantity of dynamite used monthly | — | 600 | 50 | — | 1,200 | 500 | — | 3,000 | 2,000 |
| Number of men employed in mine | — | 18 | 10 | 40 | 36 | 22 | 173 | — | — |
| Cost of running per foot of tunnel | — | — | — | £1 | — | £1 13s. | — | £1 | £2 10s. |
| Cost per running foot of timber | — | — | — | 6d. | 8d. | — | — | — | — |
| Character of mining pump | — | — | — | plunger | 3 jack-heads | — | — | 2 lift pumps | — |
| Diameter of pump cylinder | — | — | — | 8″ | 8″ & 5″ | — | — | 12″ | — |
| Diameter of pump column | — | — | — | 6″ | — | — | — | — | — |
| Stroke of pump | — | — | — | 6″ | — | — | — | — | — |

**Results of the Mill Process.**—The gold realised by milling on copper plates from the battery is 60 per cent.; from the amalgamators 15 per cent.; and from the concentrates 25 per cent.

The fineness of crushing is as follows. The battery sands, crushed through a No. 6 slot screen, contain on an average—(1.) Of slimes which remain suspended after a three minutes' rest in still water, 19 per cent. They contain when filtered the so-called "float gold," and consist principally of the earthy matter, mixed originally with the quartz. (2.) Of slimes passing through a sieve of 6,400 holes per square inch (No. 1 excluded), 51 per cent. (3.) Of sands passing through 1,600 holes per square inch (excluding 1 and 2), 23 per cent. (4.) Of sands not passing through 1,600 holes per square inch, 7 per cent.

The blanket washings contain 78 per cent. passing through a sieve of 1,600 holes per square inch, and 36 per cent. passing through 6,400 holes per square inch.

The "concentrates" of the blankets contain, of course, a larger amount of heavy stuff. If the pulp discharges directly on to shaking tables or vanners, the amount of concentrates will depend on the character of the ore and the amount of sulphurets it contains.

In some mills as much as 70 per cent. is saved of the gold in the rock, especially when crushing ores containing two or three ounces of gold per ton, and particularly if the gold is coarse and bright; but lower grade ores do not yield so much.

**Loss of Gold in Milling.**—The causes of loss of gold in milling (or the extraction of gold by copper-plate amalgamation) are manifold, and depend not only on the condition in which the gold exists in the rock, or on the minerals associated with it, but in many cases the earthy particles of the gangue exert a most detrimental influence in amalgamation and so cause loss.

There are talcose and aluminous ores which during crushing will be pulverized so fine that 75 per cent. of a sample will pass through a 100-mesh screen. The slimes produced during such a trituration have an effect on gold like grease, and the fine particles of gold which are held in suspension in the battery waters surrounded with such fine particles of clay and talc are bound to be lost, as they are swept over the copper plates without adhering to them. These ores are very difficult to manipulate, as no concentrator has yet been constructed which will allow of the concentration of the mineral particles in such ores. Numerous lodes, also, are mineralised with sulphides, mostly so fine as to be invisible to the naked eye; and in such rocks the gold is generally present in a fine impalpable condition, it may be as free gold combined with sulphides. It is under conditions like these that difficulties in the economical treatment of gold ores arise.

One of the points to be closely watched and studied by the mill-man must be the conditions in which the gold exists in the rock. He may find one "pay shoot" richer in coarse gold than the other, and if he uses a 30-mesh screen for one he will do well to use a 40, 50, or 60-mesh for the other, as the screens must always be in proportion to the fineness of the gold particles. Liberation of the gold must be effected before it can become amalgamable with mercury, and as long as the silica surrounds the gold entirely it will prevent the precious metal from being attacked by the mercury.

In many cases gold said to be "rusty" is the presumable cause of loss, and although I have seen rusty gold obtained from placer mining, very little rusty gold has come under my observation in quartz mines. Rustiness is caused by a coating of metallic oxide whose nature is not exactly understood.

It is advantageous to ascertain in every mill how much gold goes into the tailings, and the quantity of float carried away by the water. Of course all mill-men try to crush the largest quantity of rock in the shortest possible time, so as to get a large output of gold. This practice makes the ore too coarse to allow all the gold and silver to amalgamate, as a portion may not be released from the gangue. It would be much better to get the output by a more careful sieving of the ore, not forcing the stamp to do the work of a Blake crusher, nor sending to the mortars any ore fine enough to pass the screens. This is a matter of some importance, for it has been found with all kinds of stamps which have screens that it takes just as long to get ore which has already passed the screens out of the mortar as it does to crush and force it out. Crushing too fine is also quite as bad a practice, as it produces "float," and is likely to put the precious metals in a condition in which they will not amalgamate. Even supposing that losses due to improper working do not exist, there are still a few causes of loss which, though not in every case amounting to much, were a source of considerable loss in the early days. It has been found that holes in the castings of the stamps, pans, &c., attract the amalgam, and that it is even carried into holes deep in the interior of the piece. This was a source of profit in the early days to those who recovered the precious metals from the worn-out castings. Another loss may be incurred in cleaning the plates by taking off the amalgam too thoroughly. It is a well-known fact that new plates do not act so readily as old ones. As mentioned before, gold and silver will be attracted better by amalgam than by mercury. Too slow a current of water will keep the surface of the plates covered with a film of sand, while a too rapid current will prevent the gold being caught by the plates.

If gold is attached to a piece of the gangue rock which is relatively large, the specific gravity may be so reduced as to prevent the particles coming in contact with the mercury.

If the blankets are kept too long without washing, so that the hairs become charged, the fine particles of gold are lost.

Even if these causes of loss are avoided there are still others ; for instance, if the mercury be not kept clean, or be not made so by chemicals, the "quick," having an extremely thin film upon it, does not act upon the gold or silver. To some extent the same effect is produced when the rock is soapy, as is the case with the magnesian and aluminous rocks. If also there are too few amalgamating machines, or the sluices are too short, loss arises. A very important source of loss is the "flouring" of the mercury caused by rapid motion, or due to a too free use of chemicals, or the presence of base metals.

**Waste of Gold in Amalgamation.**—An absurd idea prevails amongst the public that gold is readily amalgamated, and that it is not necessary to be so particular in its manipulation in the mill. How erroneous is this impression may be readily shown. From tests made by Mr. H. B. Paul, of San Francisco, at some Californian gold mills, the following results * were arrived at:—*Test No.* 1. Average yield of ore in mill, $18·60; wastage after complete washing, including concentrating; silver, $3·14; gold, $10·04; total, $13·18.—*Test No.* 2. Same mill tailings 350 feet from mill: silver, $3·93; gold, 5·02; total, $8·95, showing that a percentage secreted itself in the passage down stream.—*Test No.* 3. Average yield of 150 tons, $3·50; assays of tailings carefully sampled: silver, $6·28; gold, $13·55; total, $18·83.

The unsatisfactory results were occasioned by the extreme fineness of the gold, and even the above does not show the full wastage. A large amount of gold produced is in such fine particles that it will float on water: hence its name—"float gold."

In cleaning up the slum from a water-tank for supplying the battery, where the water was used over and over again in consequence of its scarcity, it was found that this slum—that is, the settlings of the very finest particles held in suspension by the battery water after the water had remained at rest in the tailing pit, whence it was pumped into this tank—yielded, by a rough amalgamation in a tub, $33 in silver, and $56 in gold.

Another test, made from fifty pounds of tailings taken one mile below the mill, showed that they contained 55 per cent. of the average working of the mill. A test, made on 1,500 lbs. of tailings, showed the loss in mill working to be 63 per cent.

It must also be considered that a very small percentage of lead, copper, arsenic, or antimony, such as is found in most gold ores, will quickly vitiate the mercury, rendering it quite inefficient in collecting even the gold that otherwise, from gravity, might be taken up. This renders it necessary that the mercury should always be kept in a clean and active condition.

---

* Quoted by Professor Raymond in his Report on Mines and Mining.

**Checking the Mill Returns by Sampling and Assays.**—It is a rather difficult matter to get an average assay of gold ores at the mill, for not only is the gold unevenly diffused throughout the rock, but considering that only half an ounce or one ounce is generally taken as the weight for an ore assay, a very small speck of gold more or less in the assay pulp will vitiate the result. At the best, therefore, only an approximate assay of the rock as delivered at the mill can be obtained when working.

My method of sampling the ore was usually as follows: as the rock-breaker discharged the ore, I would place a cup, holding about five pounds, every fifteen minutes under the discharge till full, and this would give 20 lbs. of sample in one hour, or 480 to 500 lbs. in twenty-four hours. This ore I would spread out on the floor of a cleanly swept room, and make a square of it 12 × 12 feet; then every foot I would draw a line across with strings tied to nails driven into the floor, giving twelve imaginary divisions 1 foot wide by 12 feet long; then transversely across these lines I would draw twelve other strings one foot apart, which would then divide the ore layer into 144 one-foot squares. Each square was then carefully sampled by taking as near as possible one pound of ore out of it on the point of a shovel, giving a 144-pound sample, which in a separate grinding machine was reduced to pieces about the size of a pea. This reduced rock was again spread on a clean floor and divided into 144 one-foot squares, and from every square 4 ounces taken. The resulting 576 ounces were ground into a fine pulp, again divided on a large table into 144 squares, and a quarter of an ounce taken from each; and from the resulting lot of 36 ounces—which should be the representative sample of the twenty-four hours' crushing—the half-ounce assay was taken and tested. The quantity of rock being known which had been passed through the mill, it was easy to calculate the quantity of gold therein contained.

Now attention had to be turned to the tailings which escape from the mill through the discharge launder. Every fifteen minutes I held an iron porcelain-glazed pot under the spout till full, taking care not to let it overflow; then I set it aside, and after fifteen minutes, before taking the next sample, I decanted—by preference with a syphon—the supernatant water, taking good care not to disturb the settled part, into a large wooden tub. This operation being repeated for twenty-four hours, the settlings in the pot were dried, sampled, and assayed. This was the tailing assay. Subtract the tailing assay from the ore assay, and the balance represented the quantity and value of the gold which had been, or ought to have been, saved in the mill out of every ton. Very seldom did this prove to be correct, and if attention were now paid to the large tank containing the water poured off from the iron vessel, and this allowed to remain quiet for a day or two, then syphoned off, it would be found that the small amount of residue therein contained was rich in gold. This was the "float gold," the bugbear of mill-men and metallurgists.

**The Absorption of Sulphur by Gold.**—Mr. William Skey, analyst to the Geological Survey of New Zealand, has published an interesting paper on the absorption of sulphur by gold, and its effect in retarding amalgamation. While investigating the causes of the reported loss of gold during the process of extraction at the Thames gold-fields, he observed that much of this loss could scarcely be referred to any of those causes generally supposed operative for it. He therefore tested the actual condition of the natural surfaces of numerous specimens of Thames gold in respect to their behaviour with mercury, and examined further than had hitherto been done into its comportment with several of those substances likely to be associated with it in a natural way. He found—

(1.) That numerous samples of bright, clean-looking gold, of all degrees of fineness, refused to amalgamate on any part of their natural surfaces, though taken directly from the reef and untouched by hand.

(2.) That on such surfaces sulphur was always present.

(3.) That native gold, or gold in a pure state, readily absorbs sulphur from moist sulphuretted hydrogen or sulphide of ammonium, and absorbs it directly when administered in boiling water.

(4.) That surfaces so treated refused to amalgamate, though no apparent change could be observed in their aspect.

(5.) That gold so affected is rendered amalgamable by roasting in an open fire, unless copper is present to the extent of seven per cent., or perhaps less, while the same effect is produced by the contact of cyanide of potassium, chromic and nitric acid, and chloride of lime acidified.

(6.) That this absorption is altogether of a chemical nature.

(7.) That sulphates of iron, in presence of air and water, decompose various metallic sulphides common to auriferous reefs in such a manner as to liberate sulphuretted hydrogen.

The action of sulphuretted hydrogen upon gold, in rendering it non-amalgamable when placed in contact with mercury, was demonstrated with striking effect by actual experiments.

From these results Mr. Skey was led to suppose that a large area of the natural surfaces of native gold is covered with a thin film of an auriferous sulphide, and that the greater part of the gold which escapes amalgamation at the battery is represented by that portion of this sulphurised gold which has remained unabraded during the processes of milling or extraction from the reef; the state of the gold, rather than that of the mercury, being therefore the greatest impediment to thorough amalgamation.

In addition to these results, he communicated others relative to the effect of solutions of sulphuretted hydrogen and sulphide of ammonium upon platinum. In rendering it non-amalgamable, he believed a sulphide of the metal had formed in each case, since chromic acid rendered it again amalgamable. He further stated that this metal is so affected by ammonia or the fixed alkalies that it will not amalgamate except in presence of a mineral acid, from which he suspected that platina is capable of superficial oxidizement when in contact with alkaline substances, even at common temperatures. He found that his samples of gold were not affected by the alkalies in this manner, except in the case of one from Victoria, a singularity from which was argued the presence of palladium in this particular sample.

**Conditions of Successful Working.**—The successful working of a gold mine depends at the present day on a perfect and rapid system of "concentration." Mines which carry only £1 of free gold can be worked at a good profit if the local conditions are such as to permit of easy mining and subsequent reduction in the mill. As the attention of inventors is now specially directed to improving the concentrators, so that the lowest grade ores may by washing away the waste be converted into high grade ores, and as cheaper methods of reducing the concentrates are being developed, the field for mining operations will be almost daily enlarged.

A description of the concentrators now generally in use in different countries is reserved for a later chapter.

**Amalgamation of Concentrated Sulphurets after Roasting.**—One would have expected to find works set up where the concentrated auriferous sulphurets and arseniurets could be submitted, after an oxidising roasting, to pan amalgamation, as every experienced roaster will have observed that pyrites, after a perfect dead roast, will show pure, clean gold on horn spooning, or panning down a sample. Such gold is amalgamable gold, and will yield 95 per cent. by pan amalgamation. "Amalgamable" may not be a correct metallurgical expression, but it will, I think, convey to the reader the idea I desire to express—viz., gold which can be absorbed by quicksilver. Although, however, the desulphurising and amalgamation of gold sulphurets has for many years been extensively experimented upon in California, no plan has so far been brought into general use whereby refractory ores are satisfactorily treated, or low grade sulphurets worked with profit by pan amalgamation, the only tolerably successful method for reducing sulphurets being by the chlorine process, which is much too expensive to answer for low grade ores.

In the early days of Californian gold quartz mining, amalgamators used to grind in pans without roasting, contending that gold, being found in a metallic state, only requires grinding to a certain degree of fineness to admit of amalgamation; but this theory was not verified in practice. If the gold is in a free state in the sulphurets (a point by no means cleared up as yet) however fine the gold may be, it is coated with sulphur, iron, or some other metal, and no amount of friction will brighten it sufficiently to become amalgamable.

The best result which can be obtained by long and continued grinding of raw (unroasted) concentrated sulphurets is 40 per cent., and no doubt the greater part of the gold thus obtained is contained in the concentrates in a free state. To treat sulphurets properly, therefore, they must be completely desulphurised, and all metals in them (except the precious metals) thoroughly oxidised before the silver and gold will amalgamate in pans. If a complete oxidation of the iron and copper is not effected, on being charged into the pan and submitted to the grinding operation, they will pulverise into a fine powder, and when the quicksilver is added, it will partly flower and float on top of the water, while the other globules of quicksilver, distributed through the pulp, will become coated with a black scum, thereby losing all its affinity for the precious metals. When the ore is thoroughly oxidised, no such "sickening" of the mercury takes place, and the gold is taken up. Ores which have been roasted concentrate very easily, as everything in them, save gold and silver, loses specific gravity.

The difficulty in treating large bodies of low grade iron pyrites by amalgamation is the prohibitive cost of the roasting operation, but with the introduction of mechanical roasters, which would desulphurise the pyrites completely at a reasonable rate, I see no reason why amalgamation should not hold the field against chlorination.

**Iron Battery Frames.**—These are manufactured especially in England for exportation to countries where heavy timbers are scarce. It is better to ship to such localities an iron battery frame, which can be easily erected on the spot.

The following illustration (Fig. 50) shows an iron battery of 20 stamps.

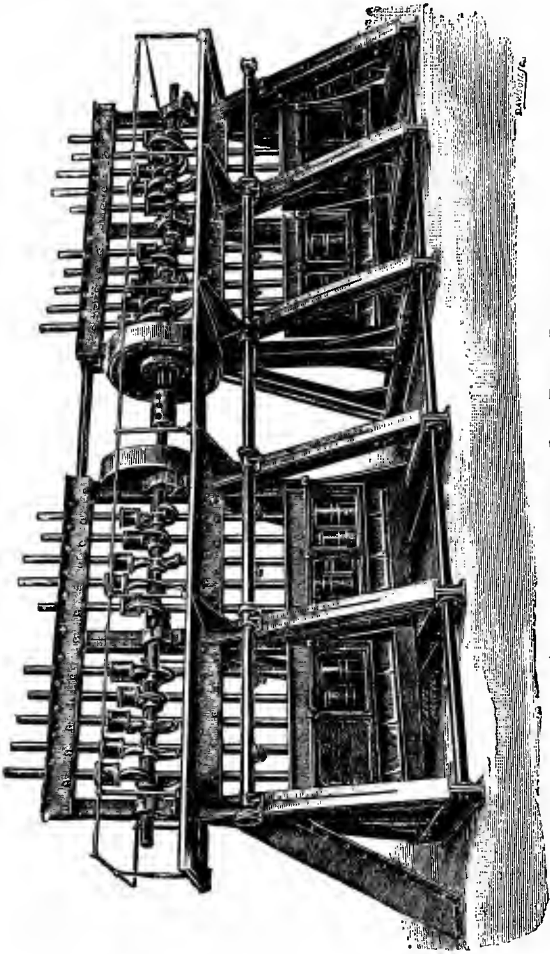


FIG. 50.—TWENTY-STAMP IRON FRAME BATTERY.

**Battery with Ten Stamps.**—Fig. 51 represents the battery when erected in its position in a wet crushing mill.

The countershaft, and also its driving pulley with belt, are seen behind the main posts. Another pulley is shown by which the stamps may be stopped or put in motion without interfering with the driving power. The pulley on the cam shaft is of wood on cast-iron flanges. This mode of construction is necessary, inasmuch as a pulley made of iron, when subjected to the rapid succession of jars caused by the falling of the stamps, would soon break. The guides for the stamp stems are made of oak, maple, or some other hard wood. They are in two parts, and are bored at proper distances for the

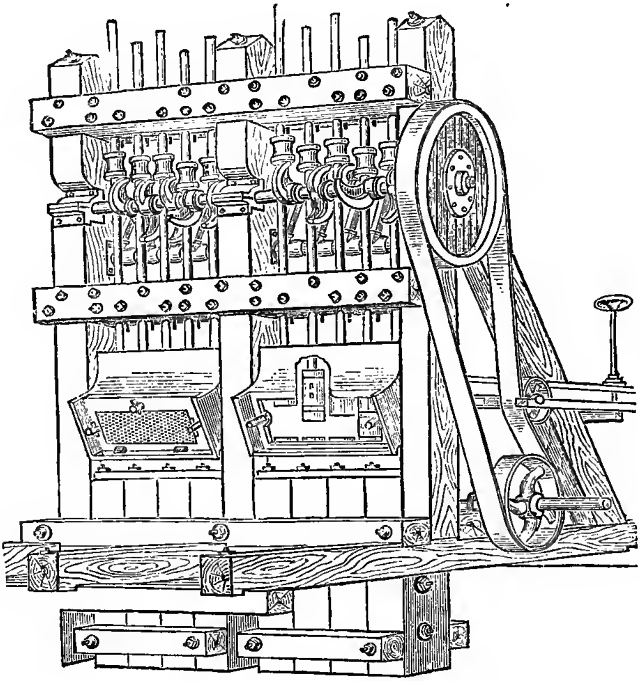


FIG. 51.—BATTERY WITH TEN STAMPS.

stamp stems. The guides, being set with keys between, can be closed together as they wear.

**General Specification for a Ten-stamp Battery.**—Two high cast-iron mortars of latest improved pattern, single (or double) discharge, planed on bottom, drilled by template, with the seats for the screen frames planed.

Two screens of hard wood fitted to the mortars.

Four wrought-iron keys for the screens.

Two screens of Russia iron or wire cloth.

Two sheets of rubber for mortar foundation, ¼ inch thick.

Ten stamp dies.

Ten stamp shoes.

Ten stamp heads or bosses, bored for stems.

Ten stamp stems of refined iron, both ends being tapered and fitted to the heads.

Ten stamp tappets, fitted with wrought-iron gib with two steel keys in each.

One cam shaft of hammered wrought iron, turned full length, key-seated and marked where cams are to be fitted.

Three cam shaft boxes, babbitted, bored, and planed on bottom and back.

Two cam shaft collars of wrought iron, with steel set-screws.

One cam shaft pulley, built up complete on double cast-iron sleeve flanges; to be thoroughly built and turned true, with the flanges keyed to the cam shaft.

One set upper hard wood guides for stems, bored for stems and bolts.

One set lower hard wood guides for stems, bored for stems and bolts.

Two jack shafts of wrought iron.

Four side boxes or brackets for jack shafts.

Ten cast-iron sockets for levers.

Ten hard wood levers for stamp holders.

## Machinery for a 10-Stamp Gold Mill.

One No. 2 Blake crusher, 10 in. by 7 in.

One grizzly or ore screen, 3 ft. by 10 ft.

Two automatic ore feeders, if such are used.

Ten stamps of 850 lbs. each in one battery, including all ironwork, wooden pulley, and hard wood guides for stamp stems.

One set of water-pipes for battery.

Two copper table plates, 54 in. by 8 ft. by $\frac{1}{8}$ in. thick, pure copper and free from flaws.

Four copper lining plates for mortars, $\frac{3}{16}$ in. thick, of same quality.

One amalgam retort and condenser.

One countershaft for stamps, with bearings and pulleys.

One countershaft for crusher, with bearings and pulleys.

All necessary belting and lace leather.

One engine, 9 in. by 14 in.; 20 h.-p.

One boiler, 40 in. by 10 ft. complete.

One feed-pump, with belt.

One heater and all pipe connections.

**Stamp Batteries.**—List of sizes, complete in detail, as per above specification.

*Weight of iron work, including wood pulley and guide.*

| | | | | | |
|---|---|---|---|---|---|
| 5 | stamps, | 450 | lbs. each | 9,500 | lbs. |
| 10 | ,, | 450 | ,, | 18,000 | ,, |
| 5 | ,, | 550 | ,, | 10,000 | ,, |
| 10 | ,, | 550 | ,, | 19,500 | ,, |
| 5 | ,, | 650 | ,, | 11,000 | ,, |
| 10 | ,, | 650 | ,, | 21,500 | ,, |
| 5 | ,, | 750 | ,, | 12,500 | ,, |
| 10 | ,, | 750 | ,, | 24,500 | ,, |
| 5 | ,, | 850 | ,, | 15,000 | ,, |
| 10 | ,, | 850 | ,, | 29,500 | ,, |
| 5 | ,, | 950 | ,, | 17,000 | ,, |
| 10 | ,, | 950 | ,, | 33,000 | ,, |

## Battery Frames.

One complete frame for 5 stamps for 550 and 650 stamps, weight 9,600 lbs.

One complete frame for 5 stamps for 750 and 850 stamps, weight 12,000 lbs.

One complete frame for 10 stamps for 550 and 650 stamps, weight 17,000 lbs.

One complete frame for 10 stamps for 750 and 850 stamps, weight 20,000 lbs.

## Power Required for Wet Crushing Gold Mills.

### *For a* 10-*Stamp Wet Crushing Gold Mill.*

| | | |
|---|---|---|
| One No. 2 Blake rock-breaker ... | 6 | horse-power. |
| Two ore feeders ... ... ... | 0 | ,, |
| Ten stamps, 750 lbs. ... ... | 12 | ,, |
| Four Frue vanner concentrators ... | 2 | ,, |
| One grinding pan, 3 ft. diameter ... | 3 | ,, |
| One settler ... ... ... | 3 | ,, |
| Friction ... ... ... ... | 4 | ,, |
| Total ... ... | 30 | ,, |

### *For a* 20-*Stamp Wet Crushing Gold Mill.*

| | | |
|---|---|---|
| One No. 2 Blake rock-breaker ... | 6 | horse-power. |
| Four ore feeders ... ... ... | 0 | ,, |
| Twenty stamps, 750 lbs. ... ... | 23 | ,, |
| Eight Frue vanner concentrators ... | 4 | ,, |
| One grinding pan, 3 ft. diameter ... | 3 | ,, |
| One settler ... ... ... | 3 | ,, |
| Friction ... ... ... ... | 7 | ,, |
| Total ... ... | 46 | ,, |

*For a 40-Stamp Wet Crushing Gold Mill.*

| | | | |
|---|---|---|---|
| 1 | Rock breaker ... ... | 12 | horse-power |
| 40 | Stamps ... ... ... | 66 | ,, ,, |
| 16 | Concentrators ... ... | 8 | ,, ,, |
| 8 | Shaking tables ... ... | 2½ | ,, ,, |
| 1 | Clean-up pan ... ... | 1½ | ,, ,, |
| 1 | Revolving barrel and batea ... | 2 | ,, ,, |
| | Total ... | 92 | ,, ,, |

**Chemicals.**—The following are the chemicals it is necessary to keep at the mill:—Fused cyanide of potassium; sal ammoniac; caustic, or unslacked lime; caustic soda or concentrated lye; and nitric acid.

**Battery Arrangements.**—In Plate V. are given, in elevation and section, detailed illustrations of an iron battery frame, as erected by the Sandycroft foundry, of Hawarden, near Chester. Plate VI. shows the structure and arrangement, with measurements, of an ore-bin and battery-house, erected by Messrs. Fried. Krupp Grusonwerk.

Missing Page

# CHAPTER IV.

## *MILLS IN OPERATION—NEW MILLING MACHINES.*

Zeile Mine and Mill—Providence Mill, near Nevada City—Father de Smet Mill—Largest Stamp Mill in the World—When Dry Crushing is Required—Huntington Milling Machine—Gates Rock Breaker—Crushing Rollers—Gates Cornish Rollers—Globe Mill.

A digression may be allowed in the present chapter, to enable me to give some particulars of certain milling establishments which may be useful to those who seek information as to the laying-out and arrangement of mills for the treatment of gold-bearing ores; and to these particulars I will append an account of some of the numerous machines which of late years have been designed with the view of superseding the stamping-mill which I have described in previous chapters.*

**Zeile Mine, in Amador County, California.**—As an example of what can be done with proper management, I give a brief account of this undertaking. The mine has been opened properly, new hoisting works have been erected, and a new 40-stamp mill put up, which crushes 100 tons of ore per day. The ore body, on the 200, 400, and 600 ft. levels, in places is very large, reaching a maximum of 50 ft. between the walls, and will average 40 ft. The rock is low-grade, and is assorted in the mine, the best being sent to the mill and the balance thrown aside in the workings under

---

* It may interest the reader to learn that in my work on the "Metallurgy of Silver" will be found an illustration, drawn to scale, of the fifty-stamp gold and silver mill erected for the Montana Company, Limited, at Marysville. The illustration shows the mill in elevation, plan, and section.

ground. It is by a careful system of working that the mine is made to pay. The rock yields only $2, or 8s., per ton in free gold, and from $5 to $6 per ton in sulphurets, which brings the yield up to a fair average.

Attached to the mine are chlorination works, which are kept running day and night, working from four to five tons of sulphurets every twenty-four hours, yielding large returns.

**Providence Mill, near Nevada City, California.**—This may be cited as an example of a model gold mill. The new mill, which was erected in 1882, works admirably. It is located, with the hoisting works, on the steep hillside above Deer Creek, a situation which gives every facility of fall, so that from the time the ore is brought to the surface and goes through the various stages of crushing, amalgamating, and chlorination, it is constantly descending from point to point with scarcely any handling, except by machinery.

The mill, which contains 40 stamps, is connected with the hoisting works by a tramway, over which the loaded cars are taken. These cars drop their loads of ore on inclined gratings, called "grizzlies," through which all the quartz and fine stuff, not exceeding two inches in size, falls into the self-feeders on the floor below, one of which is placed in connection with each battery of five stamps, and feeds the battery automatically. Such quartz as will not go through the grizzlies is thrown into rock-breakers, where it is crushed. One man is required during the day to attend the rock-breakers, two in number, and this is all the manual labour required to feed 40 stamps, as the self-feeders will do that work as well as if it was done by hand.

When the crushed quartz leaves the batteries, it passes over the usual aprons and inclined sluices, and from thence on to the Frue concentrators, sixteen of which are used in the mill to separate the sulphurets from the sands. All the work of attending the sluices and concentrators below the batteries is done by one man, so that in this large mill of 40 stamps all the ordinary manual labour required is performed by two men. The sulphurets are taken from the concentrators as often as required, and removed to a drying room adjoining, and from thence run into the chlorination works, where they are worked in the usual manner, and both the gold and the silver secured.

The power used to drive the mill is water, 100 in., brought from the north side of the creek, through a 15-in. pipe, and under 390 ft. pressure. This is discharged through a 1¾-in. nozzle upon a Collins hurdy wheel, 6 ft. in diameter, which drives the whole machinery with the greatest ease, and apparently with plenty of power to spare. There is an automatic regulator which lets on more or less water, as may be required, when less or additional power is required by the rock-breakers. The mill throughout is a marvel of simplicity, convenience, and neatness, and may be said to run itself. The driving power is inclosed in a little box placed outside in an extension of the building.

This mill, with its cheap power and the reduction of hand labour to a minimum, seems the perfection of quartz milling, and its economy is seen in the fact that the cost of milling is reduced to 55 cents (2s. 3d.) per ton, including wear and tear and keeping the works in repair. This is the cheapest milling I heard of in California. The Sierra Buttes and Plumas Eureka Companies mill their ore at the same cost, but they own their own water, while the Providence Company have to buy theirs. The latter, therefore, can claim to be doing their work at a smaller cost.

**Father de Smet Mill, Black Hills, Dakota.**—As this mill, which was designed by Mr. A. J. Bowie, involves some new and important features which have given excellent results in practical working, the particulars I am enabled to give deserve special attention from those who have large bodies of free milling ore to treat.

The mines of the Black Hills, in Dakota, being of what is termed "low grade," their successful working necessitated cheap milling, which could only be accomplished with large and economically running mills. To insure the greatest possible economy, the Father de Smet mill, which was designed and constructed under the immediate supervision of Mr. Bowie, at a cost of about £20,000, was built on a plan different, in some respects, from any previously erected. The arrangement of the building in particular was novel, and the special objects kept in view in the arrangement were four, namely: (1) the constant supervision of the batteries, tables, and sluices, and consequent safeguard against robbery; (2) free access to all parts of the machinery, and room for handling the same; (3) very large ore bin capacity, to insure steady milling; and (4) the least manipulation of the material in all stages of reduction.

Among the results attained by this construction of the mill, besides those hereafter mentioned, the following may be briefly noted. The building has been thoroughly tested, and at times there have been one and a half million pounds of ore in the bins without any resulting damage or straining of the structure. The rock breakers, which are 53 feet above the battery floor, have been run to their full capacity, with the whole mill in operation, without causing much, if any, vibration in the building, or even on the car track at the top of the main division. The carpentry of the building, in fact, is such as to insure perfect bracing of the superstructure with solidity of foundation. Eighteen persons only are required per day to run the mill to full capacity.

*The Building.*—The entire structure covers an area 140 feet long and 60 feet wide. The elevation of the main building, from the mud-sills to the apex of the roof, is 75 feet. Its greatest length is east and west. It was constructed with three distinct compartments, which for present purposes may be designated as the east division, main building, and west division. The whole structure is of timber and lumber.

The east division is 52 feet long and 60 feet wide, and contains the engine, boiler, machine and lathe rooms. The machine shop is well supplied with all the necessary implements for mill work. The main or centre division, which is 68 ft. by 60 ft., contains the batteries, rock breakers, and ore bins. The west division, 20 ft. by 60 ft., is used for the clean-up room and repair shop. It contains a small cast-iron clean-up pan, tubs for panning out amalgam, tailings, &c. There is an ample supply of water for all purposes.

*Engine, Boilers, Shafting.*—The various machinery of the mill came from San Francisco, California. The engine is a horizontal one, fitted with Meyers's cut-off, known as size No. 5. The cylinder is 20 in. diameter with 42 in. stroke; the shaft is 9 in. in diameter, and the whole weight of the engine is 25,000 pounds. The balance-wheel is 18 ft. in diameter, cast in eight segments, and weighs 14,360 pounds. The engine is of sufficient capacity to drive the present eighty stamps and twenty additional, should the latter be required. There are two boilers 54 in. in diameter, and 16 ft. long. They are tubular boilers, provided with a 12-ft. steam drum and all the necessary accessories. To supply the boilers, and for other purposes, there is a No. 5 Knowles steam pump.

The main shaft and driving pulleys are situated in the centre of the east end of the main building. The pulleys (96 in. in diameter), by means of six-ply rubber belts, drive 54 in. pulleys on the line shafting; and eight other pulleys, also 54 in. in diameter, on this line of shafting are belted to 72-in. pulleys on the cam shafts. The cam shaft belts are provided with tightener pulleys 15 in. by 16 in. in size. There are two lines of line shafting, one on each side of the main building, placed behind the batteries, and directly on the battery sills, the head ends being supported on pillow blocks. The machinery and line shafting are so placed as to be readily accessible.

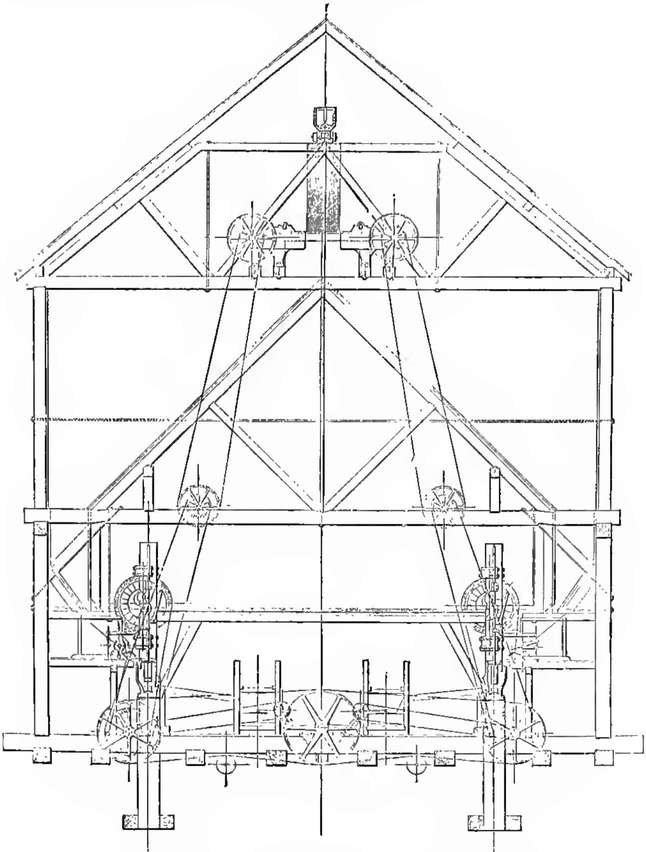
*The Batteries.*—The batteries, sixteen in number, containing five stamps each, are arranged in two lines, eight batteries on each side of the mill. They discharge to the centre of the main building. They are of the usual style, built independently of the building, and braced, one line against the other, with 12 in. × 18 in. timber. Each stamp weighs 758 pounds. The tappets are set to drop from 7 to 9½ in., 85 drops per minute being the calculated speed. The cam shafts are 5 in. in diameter, 14 ft. 6 in. in length, each weighing 860 pounds.

*Mortars, Shoes, and Dies.*—The mortars used here are of the gold mill pattern, with lining for sides and ends. The frame and lining weigh about 4,900 pounds each. They are set on the blocks and bolted to them, the tops of the blocks being previously covered with blankets. In the bottom of the mortar is a longitudinal groove, through the central axis, made for the purpose of holding the die to its place, each die being cast with a corresponding lug. As the dies wear down, it was intended to introduce a false bottom, or heavy casting 3 in. thick, filling the bottom of the mortar, and provided with a lug for the groove in the mortar, and a groove in the top for the lug on the die. The object in view was to economise iron by wearing the dies as thin as possible without lowering too much the level of their surface. The screens used are No. 6, punched, size 14 × 52 in.

Experience has since shown that, with the frequent introduction of dies, the sides of the grooves chip, and the dies then require wedging. It has been further demonstrated at the mill that, when old dies have been worked down on the false bottoms, the batteries by no means crush as much rock as with new shoes and dies, though the line of discharge is kept relatively the same. A very important question in milling low grade gold quartz is here presented, where economy requires that the largest possible quantity should be treated at the smallest possible cost in the shortest time, with due reference to the percentage of gold.

The conclusions so far arrived at are: (1) that the square bottom is the best form of die, and (2) that there is no economy in using dies after their surfaces have been irregularly worn to any great extent.

In milling gold quartz, almost the principal expense is the consumption of shoes and dies, and any method by which a saving can be effected here is most desirable. If the above conclusions are correct, it is economy to cast shoes and dies with the smallest depth practicable. This necessitates at least monthly replacing, but it insures the largest amount of work per stamp with the smallest consumption of iron. In addition, it is estimated that the quantity crushed with new shoes and dies is so much in excess of that crushed with those which are much worn as to well repay frequent renewal.



NOTE.—*This section shows the mill with the Eureka rubbers as originally constructed.*

FIG. 52.—FATHER DE SMET MILL. Cross Section through Main Building.
Scale $\frac{1}{16}$ in. = 1 ft.

*Feeders, Tables, and Traps.*—Immediately over the shafting on each side of the mill is erected the feeder floor. This floor is spacious and well lighted. There are sixteen of Hendy's patent self-feeders, eight on each side. They stand immediately at the back of the batteries, the lip of the feeder being on a line with the mortar feed, and the mouth of the hopper of the feeder directly under the ore bin discharge. The batteries, sixteen in number, with five stamps each, are arranged in two lines, eight batteries on each side of the mill.

In front of each battery there is an inclined table covered with amalgamated soft copper plates. The tables are so located that they are all visible from any one point on the main floor of the mill, and are thus constantly under the supervision of the amalgamators. At the lower end of each table there is an Eureka rubber, which receives the tailings, and in turn discharges them into a quicksilver trap. From the trap, the tailings pass over a small amalgamated copper plate placed below the floor, and thence run into the sluice previously described, situated in the centre of the mill. The traps are narrow wooden boxes with a centre partition which extends to within a few inches of the bottom. Above the main sluice, on a level with the floor, there is a track on which runs a small car, used to transport material as may be required.

*Rock Breakers.*—There were two rock breakers when the mill started, the ore descending on grizzlies. The monthly capacity of the mill was originally from 4,400 to 4,500 tons of quartz. Two additional rock breakers have since been added, the capacity of the mill being thus increased to 6,200 tons of quartz, and to a much greater amount—7,500 tons—when the auriferous slate also is milled, which is now being done.

It has been conclusively proved in the Black Hills that in milling low grade quartz there is great economy in large rock breaker capacity. The breakers should be set to crush fine.

**The Largest Stamp Mill in the World*** is one at the Treadwell Mine in Alaska, consisting of 12 batteries, in two rows of 60 stamps set back to back, making 120 stamps of 900 pounds each, with a crushing capacity of 360 tons per day. Over the ore bins are three 9-inch by 15-inch Blake rock

---

* Compare this now (1895) with the Langlaagte 160-stamp battery in the Rand (see *post*, p. 623).

breakers and six 5-feet by 10-feet grizzlies. In front of the batteries, and in the battery rooms, are copper-plate sluices; and in front of the battery rooms and below are the concentrator rooms. There are also the necessary cleaning-up pans and amalgamating barrels.

Each battery of 10 stamps is driven from the line shaft by an improved friction clutch pulley, so arranged that each set of 10 stamps can be started and stopped at pleasure. The stamps are fitted with phosphor-bronze bushings, and are accurately balanced, so that there is no tendency to "wobble" or wear out of the perpendicular.

The motive power is furnished by one Pelton and one Knight water-wheel, each of them 6 feet in diameter, running at a velocity of 236 revolutions per minute; the power being transmitted by twelve hemp ropes 2 inches in diameter, running over grooved pulleys 12 feet in diameter. Either of these water-wheels is of sufficient power to drive the mill.

The ore is fed into the batteries by self-feeders, and amalgamated in the mortars, and the discharged pulp runs over copper plates, and thence on to 48 Frue vanners which concentrate the sulphurets; these are then treated by chlorination.

From a statement recently published by this mining company, I gather that although the ores only pay something over 4 pennyweights per ton in gold, the monthly profits reach 20,000 dols.* This is no doubt owing to the fact that the mine has abundant water-power and the ore is mined cheaply.

Owing to the great pressure under which water can be had at many mills, a new style of water-wheel is used, having the water buckets cast on the wheel, though some have the buckets cast separately. The principle of these wheels is that the higher the head (or perpendicular fall) of the water column, the greater the power they can communicate. Those mostly in use are the Pelton and Knight wheels.

**When Dry Crushing is Required.**—In dealing with ores

---

* The mill began operations in May, 1885, and averaged the first twelve months about £20,000 a month.

which are rich enough to bear the cost of pan amalgamation, I would recommend the abandonment of the old beaten path of wet crushing, inasmuch as during wet crushing a certain amount of slimes are produced, which carry fine particles of gold in suspension, and these would float away into the tailing pits. A natural system of concentration is continually going on in wet crushing, and the pulp has to be saved in settling tanks inside the mill, to be treated in pans afterwards. The sands will settle in the tanks, but the very fine stuff which is always produced in stamping will float away, and with it some gold.

It is therefore advisable to treat such ores by dry crushing, or grinding, and then to lead the dry pulp by some automatic conveying apparatus into the pans. Add enough water to make the pulp of sufficient consistency to carry the quicksilver globules in suspension, and add the mercury after the pulp has been ground for an hour or two in the pan. The quantity of quicksilver to be added must be found by experience, and with pans holding 2,000 lbs. of ore should range from 20 to 40 lbs.

It will be found with free gold ores, if two hours are allowed for grinding and two hours for amalgamation, that 6 tons can be treated in a one-ton pan in twenty-four hours, and the increased percentage of gold saved, as compared with battery amalgamation, will pay the cost of running the pans and settlers and leave a profit. It would seem that large pans of the Soderling type should commend themselves to the mill-men, as allowing a large output; and although such a system would hardly pay on low grade ores, yet with the high grade ores—such, for instance, as are now treated in the Transvaal by simple battery amalgamation—this method should (I think) be tried, having been so efficient in the silver mining districts of the United States, where ores carrying also a large percentage of gold, are worked by this process. These ores, which in most instances are impure quartzose ores, produce in wet crushing fine slimes, carrying a large percentage of the precious metals, and as all the clayey stuff accumulates in them, they are difficult to treat by themselves in pan amalgamation. This would be the case still more if such ores as occur in the Transvaal were settled in tanks and the slimes collected in reservoirs. Dry crushing, therefore, should be adopted, so as to obtain an even material.*

The finer the ore is ground previous to pan amalgamation, the better, as with exceedingly fine stamping in the battery, or otherwise, the grinding in the pan may be done away with entirely. This would effect a large saving in power, and permit a larger output of the pan. When ore is ground so fine as to pass through an 80 or 100 mesh screen to a linear inch, no grinding in the pan should be required, unless it will help to brighten the gold particles and assist the amalgamation; but with a stamp battery such fine pulverization cannot be recommended, as it would reduce the stamping capacity.

Where ores are treated in bulk by chlorination, dry crushing should also be adopted, as there is no waste of gold; and should chlorination be so developed as to be considerably reduced in cost, so as to allow of its adoption in lieu of amalgamation, no doubt practical experience would soon demonstrate, that wet crushing should be abandoned, on account of the losses of gold caused by sliming.

It would be difficult to estimate the cost of pan amalgamation as compared with ordinary battery amalgamation in every individual case, but the records show that ores in Nevada are treated by this system at a cost (including outlay for chemicals on account of silver in the ore) of from 16s. to 20s. per ton, and most likely the cost would be the same in the Australian and African gold fields.

**The Huntington Milling Machine.**—It has been the aim of inventors for years past to produce a machine which in efficiency should equal, or even excel, the stamping mill; and although numerous attempts have been made, so far none have proved efficient enough to supersede it, at least not for wet

---

* It is certainly a satisfaction to me now (1894) to see that the suggestion here made by me six years ago is finding so many advocates at the present time, when dry crushing, with regard to the cyanide process, is being advocated by leading engineers.—M. E.

crushing as usually required in gold mills. For dry crushing, especially where the ore has to be roasted, it has been found advantageous to use rolls, and those made by Krom meet the requirements best. Amongst the machines which have been designed to supersede the stamping mill, and which have advanced beyond the experimental stage, the Huntington mill deserves to be mentioned. The special merits of this machine seem to lie in economy in expense of plant and cost of working, in the economical transportation of machinery, and its erection at the mine; and in economy of management and power. The machines are made from 3½ ft. to 5 ft. diameter, weighing, according to size, from 2½ to 5 tons; but for want of authentic data, I am not able to furnish their crushing capacity as compared with the stamping mill.

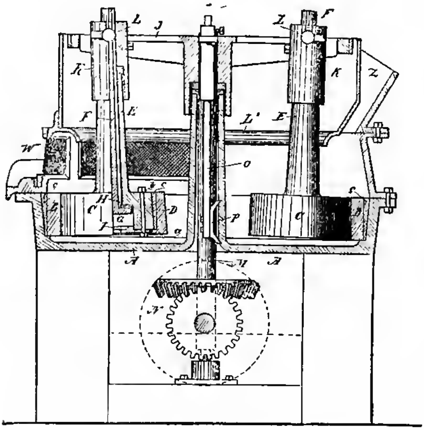


FIG. 53.—HUNTINGTON MILLING MACHINE. Section.

The Huntington mill consists of a circular iron pan, in the centre of which through a hollow cone passes a vertical axis, which imparts a rotary movement to a circular disc forming (so to say) a cover to the pan; and from the periphery of this disc are suspended four vertical shafts, with rollers turning loosely upon the lower ends of the shafts, which are so suspended that the rollers may swing inward and outward, and be caused to travel around in contact with the sides of the pan or dies by centrifugal action. A set of scrapers are also attached to the disc, which throw the ore constantly in the path of the rollers.

The details of the construction of the machine are shown in the figures, which also give a perspective view. Fig. 53 is a sectional view, Fig. 54 a plan, and Fig. 55 the scraper.

The plan shows only two of the rollers, with the mode of suspending them from the periphery of the disc, but in practice four rollers are in the machine, as can be seen in Fig. 56.

A is a circular pan, having a concentric, B, of hard steel fixed around its interior in the portion which forms the bottom pan, which has a portion of its front reserved for a discharge opening, covered with a screen as shown at W. The feed slit is at Z, into which an inch pipe discharges the water necessary for carrying off the pulp.

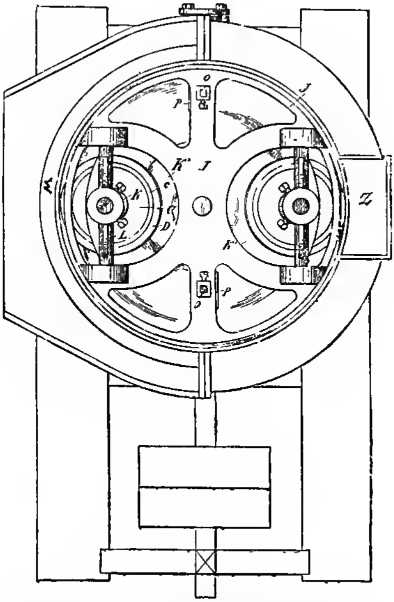


Fig. 54.—Huntington Milling Machine. Plan.

The rollers, C, which do the crushing, have exterior circular shoes, D, secured to them. These shoes roll against the interior of the die, B, so that the material is crushed between them, and the wear comes upon the shoes and dies, which can be easily replaced when worn out.

These dies are rolled steel, and are formed with a flange, *a*, around their interior near the bottom, and hook bolts, *b*, pass through the rollers, C, with the hook beneath the flange, so as to support the dies and hold them in place. Wooden wedges, *c*, are driven between the dies and the rollers, and as they swell when wet, they will hold the two firmly together.

The flanges, *a*, are held by the hook-bolts against the bottom of the rollers, and they prevent the dies from being forced up by pieces of rock which may be jammed below them. The dies around the interior of the pan are secured by wooden wedges in a similar manner.

A sleeve, E, extends upward from each of the rollers, C, and has a hole made through it to the shaft, F, by which the crushing roller is suspended, to pass down through the sleeve and through the roller. The lower end of this shaft has an enlargement or head, G, formed upon it, and a corresponding chamber is made in the bottom of the crushing roller to receive this head. Between the head and the interior of the chamber, washers, H, are placed, which serve to relieve the parts of friction.

A cap or plate, I, is fitted with rubber or other packing so as to screw upon the bottom of the crushing roller, thus making the chamber within which the head of the suspending shaft lies perfectly tight; and this chamber is then filled with oil or lubricant so that the rollers may turn easily upon the shaft, and at the same time be kept entirely free of the grit and dirt. J is the driving disc, from which the rollers are suspended.

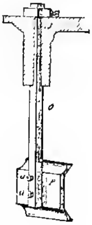


FIG. 55.
SCRAPER.

K is a sleeve, having trunnions, L, projecting to each side from it, and these trunnions turn in boxes supported by the disc, so that the trunnions stand at right angles with a radial line from the centre of the disc.

The upper end of the shaft is keyed in this sleeve, and the lower end of the sleeve has an enlarged opening which fits over the upper end of the sleeve, E, as shown. This prevents any dirt from falling in at the top of the lower sleeve and working down around the shaft, thus keeping the whole of that portion entirely clean and well lubricated.

The disc, J, is peculiarly formed, with large curved openings $K^1$, so that when the trunnions are lifted out from their journal boxes the shafts, with their sleeves and the attached rollers, may be lifted directly out through these openings without in any way disturbing any other part of the apparatus.

By this construction the rollers are allowed to swing inward and outward about the suspending trunnions, and as the disc is driven around by the central driving shaft M, to which it is keyed, and the gearing, N, below the rollers, C, will be caused to rotate about the shafts, F, rolling against the interior dies which are fixed within the pan, thus crushing all the material which may come between them until it has reached a suitable fineness.

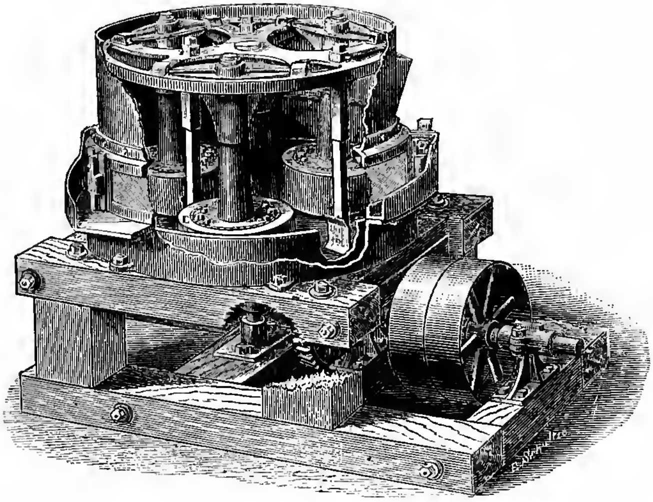


Fig. 56.—Huntington Milling Machine. Perspective View.

The pan has a flange, $L^1$, projecting inwardly a short distance above the screen-openings, and the effect of this flange is to turn the material downward and prevent the tendency to rise to the top of the centrifugal and upward action as it escapes from between the rollers and the dies. Scrapers are used in connection with these rollers. O are the vertical rods or arms to which they are secured, and which are made adjustable up and down in the driving disc, J. P are the scrapers, which are made with double edges, as shown, and have two bolt-holes through them, through which bolts, Q, pass, and by which they are firmly secured to the arms.

Whenever the lower edge of the scraper becomes worn, or for any other reason it may become necessary, it is easily removed and reversed, so that the other edge projects downwards and may be made to work.

**The Gates Rock Breaker.** — This ingenious machine works on an entirely new principle. The claim is made for it that it will do with less power more work than the Blake rock breaker heretofore described (see p. 21).

As will be seen from the appended illustrations (Figs. 57 and 58) there is a shaft G in the centre, to which a gyrating motion is imparted by the brass eccentric box, D, which is securely attached to the bevel wheel L, forming a long hub to same. The main shaft, G, is of forged steel, is supported on the chilled iron octagon step, P, and held in the centre of the shell, Q, by the top, C. Fastened to the shaft, G, is the chilled iron breaking head, F, which has two soft iron rings cast into the centre of it, one flush with the top, and the other flush with the bottom. These rings are of sufficient width to leave a space between them, the same length as the taper-planed octagon on the shaft, and the space between the rings is cored-out octagon a little larger than the taper-planed octagon on the shaft. The shaft, above and below the octagon, is turned on a taper the same length as the width of the soft rings in the head, and these tapers, as also that of the planed octagon, all taper toward the top of the shaft. The rings in the head are bored out, the upper one to fit the taper on the shaft above the octagon, and the lower one the taper below it. The head is put on over the top of the shaft, and when it has been driven down to its bearings, the octagon faces on the shaft and in the head will come opposite to each other. The space between them is run full of zinc, which keeps the head from turning on the shaft, and makes a smooth bearing against the octagon of the shaft. The rings are screwed down, thus securely fastening the head on the shaft; but it can be pulled off when required, since the whole of the shaft inside the head tapers toward the top.

There are twelve chilled-iron liners, E, placed inside the shell, Q, the space behind and between them being run up with zinc. These can be removed when required by first driving in the key liner (which has reverse bevels on its edges), by the use of a wrought iron or steel pin, through the hole in the shell, Q, at 2. There are three openings in the top, C, through which the material to be broken is thrown in all around the breaking head.

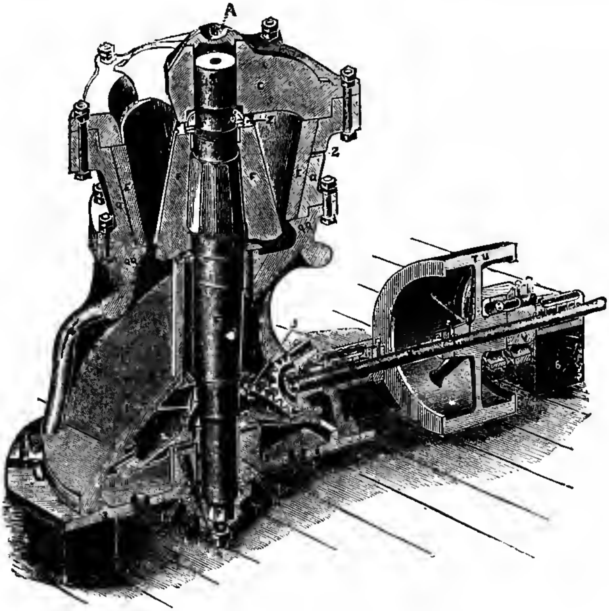


Fig. 57.—The Gates Rock Breaker. Section.

N and N represent two small square oil passages cored in the bottom plate 3, which convey the oil down to the space at Y.

The shaft, G, can be raised to regulate the size of the opening at the lower end of the chilled iron head, F, and liners E, by the screw, S.

The excentric box D, which imparts the gyrating motion to shaft, G, is babbited on the inside and outside of the thickest part, as shown by the heavy black lines, and, as all the wear is on this side, can be readily rebabbited, should it become necessary through loss of motion or throw. The outside of the excentric-box, D, is turned to fit the bore of the bottom plate, 3, in which it revolves; the inside is bored to fit the journal of the shaft, G, but is bored excentrically, enough to crowd the shaft out of the centre just sufficient to produce a fracture or breaking of the material being operated upon between the chilled iron head, F, and the liners, E.

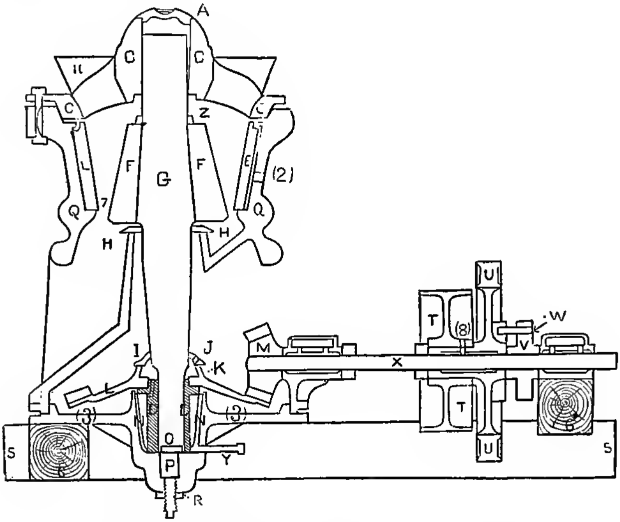


Fig. 58.—The Gates Rock Breaker. Transverse Section.

The shaft, G, is a loose fit in the excentric-box, D, *and does* *not revolve* except when there is no material between the breaking head and liners, in which case it revolves with the bevel wheel and excentric box, owing to the slight friction the excentric box causes in revolving around the journal of the shaft, but this ceases the moment any material is put between the breaking head and liners, as the resistance of the material overcomes the friction.

It will be seen that the closest point of contact between the breaking-head and liners is always at that part of the head which is exactly opposite the thickest part of the excentric box, D, and that as the excentric box is revolved around the shaft the point of contact is constantly moving before the excentric, so that when the excentric box has made a full revolution around the shaft the point of contact has moved forward to every point around the liners, E. It will also be seen that the material is broken at the point at which the head and liners are in closest contact, and when the head has been moved to the point of contact opposite, the material drops down a little to be broken again when the head has been moved around to that point again, and when broken small enough drops down on to the inclined diaphragm and slides out through the opening in the shell, Q Q. There being three openings in the top, C, through which the material is thrown in all around the breaking head, F, it will be seen that every time the excentric-box has made a full revolution, the breaking head has acted upon every particle of material in the space between the breaking head and liners, and that there is not a moment it is not breaking the material at some point.

The band-wheel, T U, is a loose fit on the shaft, X. The break-pin hub, V, is keyed fast to the shaft, X, and has a hole in it through which is passed the break-pin W, into a hole in the hub of the band-wheel. The break-pin is held in place by the set screw in the break-pin hub, V, and is of no more than sufficient strength to stand the strain necessary to break the material being acted upon, and should an accident occur (such as a piece of steel getting into the breaking surfaces), the strain would become so great upon the break-pin that it would break off, and the band-wheel would revolve on the shaft, X, while the machine would stop until the article had been removed, and a new break-pin put in. The loose collars, H and I, are to keep the dust out of the journal, and gyrate with the shaft, G.

The machine is oiled through the hole, J, in the loose collar, I, the oil finding its way down through the journals to the space, Y. By an ingenious arrangement of the oil passages, N, of which there are four in the base plate 3, the motion of the machine causes a constant circulation of the oil through the journals of the shaft and excentric-box. The old oil is drawn off through the pipe on the side of bottom plate 3, and by pouring hot water through the hole, J, in the loose collar, I, the journals are washed out.

This rock breaker ought to prove a valuable adjunct to quartz mills, as it is claimed that the work can be done with far less expense than with the other styles of breakers.

With the Blake rock breakers, there is a limit to their capacity in regard to the fineness with which they can crush. The wear being principally at the point of discharge, it is found impracticable to keep them up to any fine work. The wear of the jaw-breaker is generally on the central portion of the dies or plates, where the circular form of the Gates breaker seems to give a larger wearing surface, and owing to its peculiar movement makes it possible to reduce the ore to smaller fragments. By this means it relieves the stamps, or other pulverisers which do the fine crushing, of a large amount of work. In the Gates rock breaker, moreover, the work is continuous, while in all forms of jaw crushers, the work is intermittent, the back movement being a lost motion.

**Crushing Rollers.**—Where ores have to be ground coarse for concentrating purposes by trommels and jigs, it is advisable to use Cornish rollers. This method is to be recommended with gold ores having a large percentage of pyrites in lumps, and where the object is to concentrate the pyrites. If such ores be crushed in batteries by the wet process and stamped fine, as the pyritic portion is more friable than the quartz, a large part would be slimed and lost ; but by crushing coarse in rollers, sizing the material, and concentrating in jigs, a more economic treatment is effected. By separating the pyrites they can be submitted to a separate treatment, and the gangue, if containing free gold, stamped and amalgamated, and the fine pyrites saved by vanners.

These rollers—which were first used in Cornwall, and hence their name—consist of two cylinders which revolve against each other and are fitted on the outside with shells of steel ; these can be replaced when worn down. The manufacturers of the Gates rock-breaker produce a type of Cornish rollers which seems to be well adapted for dry coarse crushing, as it combines with the rollers a sifting arrangement. This machine—the Gates pulveriser—as shown in Fig. 59 consists of a pair

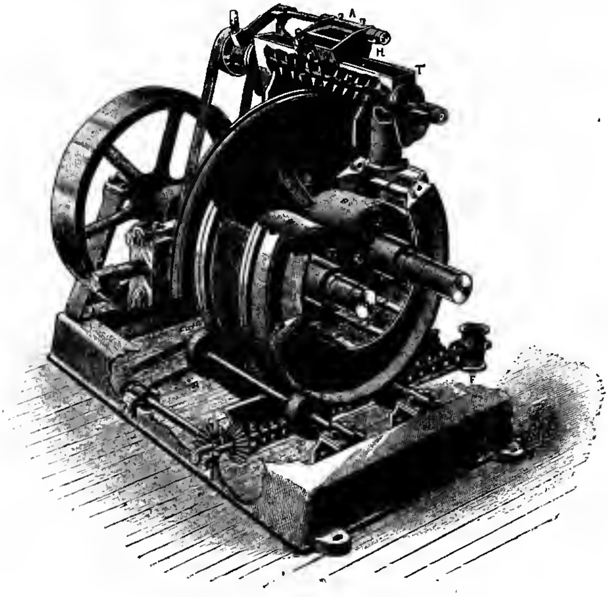


Fig. 59.—The Gates Cornish Rollers.

of Cornish rollers, surrounded by a revolving screen which has elevating buckets on its inner surface for returning the partly crushed material back to the rolls until it is crushed fine enough to pass through the screen. The rolls are 24 inches in diameter, with 16-inch faces, the wearing faces being chilled-iron shells 4 inches thick, which are held in place by bolts in the usual manner. The rolls are driven separately, but at the same speed. The battery of springs gives any desired pressure or crushing force. The revolving screen is carried on four friction rolls, R, the journals for which are outside the dust case.

When the machine is in motion the vibrating-plate, A, imparts a vibratory motion to a spout leading from the ore bin to this plate, and each time this feed-spout descends it strikes sharply upon the adjusting screw, and a portion of the material contained therein is discharged into the hopper, H, and falls into the trough, T, containing a right and left hand conveyor, which conveyor divides the material into equal portions, carrying each portion in opposite directions into the down spouts, S. By them it is discharged between the rollers, B, where it is crushed and falls through upon the wire netting of the screen, C. That portion which is sufficiently fine to pass through the screen falls through into the conveyor through E below, while the coarser particles are carried upward by the elevating buckets, D, and discharged again between the rollers to be recrushed. That portion of the material which falls into the conveyor-trough, E, is conveyed along and discharged at the side of the machine at F.

The rolls are never run close together, but about $\frac{1}{8}$ in. or more apart. As they wear a thin disc is taken out from between the boxes in which the journals to the rolls run, and thus the rolls close together, so that they are the original distance apart.

**The Globe Mill** is adapted for both wet and dry crushing, but I will confine myself here to a description of the wet crushing-mill, which is styled by the inventor the "Cyclops," and is intended for the treatment of gold rocks which are to be amalgamated and concentrated.

For this mill, a saving of 50 per cent. in steam or water power and labour is claimed in comparison with the stamping-mill, while the machinery requires only one-half the capital for its construction; the entire mill being self-contained, and consisting of few parts, which occupy a limited space. From the different types I have seen in operation, an output from 10 to 60 tons of finely pulverised material is obtained in 24 hours, when dealing with very hard quartz. The mill is illustrated in the accompanying diagrams, Plate VII.

The mechanical parts consist of a short, heavy cast-iron cylinder, having a diameter of from 3 to 7 feet, according to the capacity of the mill, and a length of from 3 to 6 feet, which cylinder rests on a cast-iron foundation. Concentric with this cylinder in its central inner portion is a circular grinding-path, made of a heavy ring of the best steel. This path is concave on the inside, forming a seat for the ball or grinder. This steel ring is securely bolted to the cylinder on the inside, and its width ranges from 4 to 8 inches.

To give the necessary rotative motion to the ball, which is made of a special metal, and is the chief operating factor of the mill, there are two steel discs fastened to the main shaft, facing one another at a certain distance, and sloping outward toward their periphery, thereby forming a U groove, and the ball has its seat between this groove and the concavity of the grinding path. The two discs, when the main shaft revolves, impart motion to the ball by simple frictional contact, as soon as the necessary impact is imparted to the ball by the velocity of the discs and the resulting centrifugal action.

The impetus resulting from such velocity, and the great weight of the ball, are factors which produce an enormous amount of work, which will be better understood when it is considered that at the ordinary speed of 300 revolutions in a 5-foot mill, the ball travels through a distance of over 4,500 feet a minute, doing continual grinding the entire distance as the ore particles are carried along the entire grinding path by the great velocity of the motion created inside the machine, the ball having exerted every second 22,500 foot pounds of work. The centrifugal force (in accordance with a well-established law) throws the ball outward toward the periphery, and the quartz particles are ground between the ball and the steel path, and when once the ball is in motion very little frictional contact between the two discs and ball is required to keep the same in motion, which accounts for the small wear on the discs, which are made of the best steel.

A stream of water is fed into the mill, which is continuously agitated by the moving ball and dashes the pulp out through the screens fastened on both ends of the cylinder, and these are covered with sheet-iron plates 1½ inch distance from the screens, having an open space near the bottom from where the pulp discharges into troughs cast on to the frame and from there flows over amalgamated copper plates.

The mill can also be arranged for fitting copper plates inside, so as to amalgamate in the cylinder.

The ore after passing through an ordinary rock-breaker is fed by means of a hopper into an automatic feeding apparatus placed on the top of the mill.

A mill capable of crushing 60 tons of hard quartz in twenty-four hours, discharging through 40-mesh screen, requires 15-horse power, and weighs 12½ tons, doing the work of a 30-stamp battery, which requires 45-horse power to drive it.

**Krupp Grusonwerk Dry Grinding Ball Mill.**—One of the best machines for the dry crushing of mineralized ores is the Gruson patent ball mill (made by the firm of Fried. Krupp Grusonwerk, Magdeburg), which is in use in many parts of the world. It consists of a cylindrical drum made of the very hardest iron and steel, with grinding plates furnished with holes, an outer ring of perforated sheet iron, over which a set of sieve frames delivers the crushed material through an outlet in the dust casing on to the floor below. The mill is perfectly self-contained. Inside of it very hard chrome steel balls exercise on rotation of the mill a pulverising and triturating action on the ore contained in the mill, and no other dry pulverizer has given such good results in actual practice.

Missing Page

The great advantages of the mill are—that comparatively large-sized pieces of ore, of the size of a double-fist, can be fed into it; that it requires little or no supervision, and very small horse-power; and that the wear and tear of the mill, if got direct from the makers (as everything depends upon the good material employed in its manufacture), is relatively very small for the work done by it. The internal parts of the mill are always easily got at.

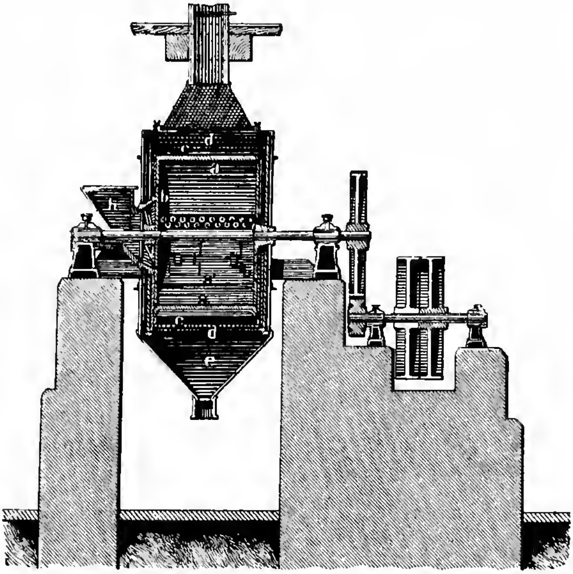


FIG. 60.—KRUPP GRUSONWERK DRY GRINDING MILL.

The mill is largely in use in connection with the cyanide process of gold recovery, as it crushes very regularly to any mesh of sieve required, and does not make too many slimes.

In Figs. 60, 61, the mill is shown mounted on a wooden staging, which is easily put up and taken down again if a removal to other parts of the mine is contemplated; and in Plate VIII. it is shown in several aspects and in full detail.

The mill consists of a rotary drum composed of exchangeable hard steel plates, "a" and "b," and containing a great number of hard steel balls of various sizes. The drum makes, according to the size of the mill, from 20 to 45 revolutions per minute, and requiring for this rotation only a small expenditure of power. During the rotation the balls, moving freely up and down in the drum, beat and grind the material until it is sufficiently reduced. It then falls through the holes provided in the plates, "a," on to a perforated sheet steel cylinder, "c," surrounding the drum, and having much smaller holes than the drum plates. By passing these smaller holes it falls at last on the cylindrical sieve, "d," surrounding at a certain distance the perforated sheet steel cylinder, and consisting of battery screening of any number of mesh desired. Passing this screening, the material leaves the mill through the discharge funnel as a finished product. Those parts of the material which have left a drum through the holes of the plate, "a," but are still too coarse to pass the perforated sheet steel cylinder or the external sieving, are conducted during the rotation by means of the sheet-iron scoops, "f," to the channels, "g," through which they fall back into the interior of the drum to be again subjected to the action of the steel balls. It is a construction as simple as it is ingenious.

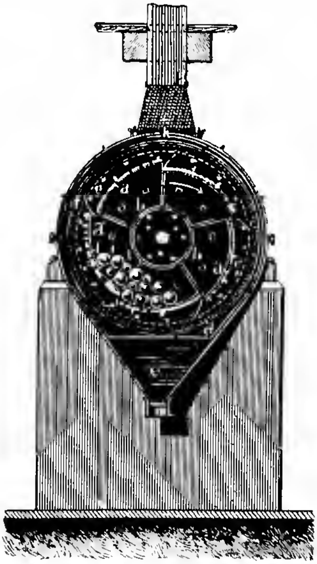


FIG. 61.—KRUPP GRUSONWERK DRY GRINDING MILL.

Very practical also is the way in which the material to be ground is introduced into the drum. When put into the funnel, "h," it is taken up by the nave disc of the drum, the

spokes of which are formed similarly to a ship's screw, so that in their rotation together with the mill drum they act as a screw conveyor transporting the stuff right into the drum, and at the same time preventing the steel balls from darting from the drum into the charging funnel, "h." The interior of the drum is easily accessible through a manhole.

**Krom Rolls.**—This machine, of American origin, which was formerly used principally for reducing silver ores to be treated by lixiviation processes, has become one of the accepted means of fine-crushing gold quartz. It will crush either wet or dry, but has been principally used hitherto as a dry crusher.

Its economic principle is that of gradual reduction and simultaneous separation, the object being to reduce the whole of the ore as nearly as possible to the required size, without much of it being too large or too small, and consequently with little fine dust, which makes slimes. The system lends itself to this end (*a*) by subdividing the work of reduction, and after each crushing separating that part of the ore which is now fine enough; and (*b*) by the "cracking" action of the rolls on the pieces of ore, which causes the ore to break up along its natural lines of cleavage, and as both rolls run at the same speed, there is no surface friction to grind up the ore into dust.

Fig. 62 shows the Krom rolls in section, and from this their general construction will be readily understood. The tyres are made of wrought steel exceedingly hard, and if the automatic feed is properly attended to, they will wear quite evenly for several months. After several thousand tons of ore have been put through, they should be trued up, for which a special emery grinder is supplied to get on to the frame of the rolls. We understand that the life of a pair of tyres is about 15,000 tons of ore.

Plate IX. shows a Krom roll plant to crush about 40 tons per day to 30-mesh, arranged for a mill site where the ground is level. As in most crushing plants, a hill-side should be chosen if possible.

The plant consists of the following :—(1) Blake crusher—

coarse; revolving screen. (2) Blake crusher—fine; finishing screen—say 30-mesh; Krom rolls; return elevator to finishing screen; and the necessary elevators and conveyors to suit the mill site.

It is customary, also, when crushing dry, to have an automatic dust-collector, which, by collecting the dust made in the several crushings and screenings, not only keeps the mill-house and the bearings of the machinery free from dust, but in some ores the collected dust—which in quantity is usually from $\frac{3}{4}$ to $1\frac{1}{2}$ per cent. of the ore crushed—is found to be rich in gold.

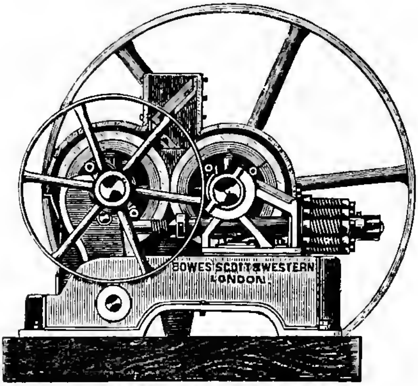


Fig. 62.—Krom Roll.

The power required to drive such a plant is about 30 effective horse-power. After the ore is crushed as above described by the roller plant, it can of course be treated in any of the usual ways, viz., by mercury, cyanide, or chlorine.

If it is crushed dry and is to be amalgamated on tables, it is made into a pulp by a special mixer, with copper plates inside, and a lip delivering it on to the tables exactly as from a stamp battery. For cyanide or chloride extraction it is best taken dry to the filter-vats, as it will pack in them much more readily and evenly than if wet.

There can be no doubt that the system of gradual reduction is a sound one, and the resulting crushed ore is much more even in quality than when reduced by stamp or frictional crushers.

# CHAPTER V.

## *TREATMENT OF GOLD-BEARING ORES: CONCENTRATION.*

Concentration explained—When Concentration precedes Amalgamation—Operations in Concentration—Spitzkasten: Rittenger's Pointed Box—The Trichter Apparatus—Concentration in Sluices—Further Concentration in Rockers and Buddles—Green's Jigger—The Dolly Tub—The Round Buddle—Collom's Buddle—The Concave Buddle—The Tossing Tub—Hendy's Concentrator—The Frue Concentrator—Its Operation described—Concentration of the Sulphurets—Results of Concentration—Colonel Taylor's Experience.

In every ore there are two separate and distinct portions—the worthless and the valuable. That the valuable portion is the smaller is obvious, and in treating ores on a large scale it is an economical question how to remove the worthless portion by cheap mechanical appliances.

The previous chapters on milling ores will have shown that the whole process of amalgamating is simply a concentration of the "free gold" aided and assisted by mercury; but when gold is associated with other ingredients, concentration by means of mercury cannot be effected, and mechanical appliances are used whereby the ore particles are separated from the worthless stuff; and after the separation the valuable material is subjected to metallurgical treatment. A science has sprung up in this branch of metallurgy, called concentration, or ore-dressing. Not alone has inventive skill brought this department to such perfection as to effect the separation of the ore from the gangue, but in case of complex ores—which, for instance, carry copper pyrites, blende, and galena intimately associated together—they can be separated cleanly enough one from the other and made to yield in separate departments afterwards the copper, zinc, and lead.

The conditions which are most favourable for concentration are those where the valuable portion of the ore possesses a much greater specific gravity than the gangue accompanying it, so that in a moving body of water the heavier portion will settle and the lighter be carried away.

The concentration of fine sands and slimes—a process upon which, as I have already remarked (p. 98), depends the successful working of a gold mine—is generally performed on wooden, canvas, or metal inclined planes. The principle on which the separation of assorted grains on an inclined plane is based consists in the resistance offered by the grains, by sliding or rolling friction, to the impulse of the water on their surface.

The sorting of the sands is best performed by their free fall in moving water; and by sorting in moving or standing water different sizes of different density (that is, equal-falling grains) are obtained. Equal-falling grains are such as sink with equal speed, or which fall the same distance in the same time. Such grains do not permit of separation under water. It is, therefore, important to convey a thin layer of diluted stuff on the table. In this case the water will not strike all points of each grain with equal force, as it would in a deeper stream; but the larger, lighter particles will suffer a stronger impact of water on their higher points than the smaller ore grains, for the obvious reason that in a thin water stratum immediately on the table-plane the water has less speed, on account of adhesion, than in the upper layer. Consequently, of the equal-falling grains, with a certain medium speed of the water—depending on the inclination of the table—the larger gangue particles will be washed off, while the smaller ore grains still remain on the plane. Most tables on which the concentration is effected receive an inclination of from 6 to 8 degrees.

Although sorting is a condition of proper separation, still a perfect concentration cannot be expected, not only because of the impossibility of ever obtaining a uniform size, but also on account of difference in shape, which tends to modify the influence of specific gravity. For this reason the best concentrating apparatus will yield a medium quality stuff between the rich and the waste portions which, while not good enough to warrant extraction of the metal, will yet be too rich to throw away. The treatment of this stuff may be profitable in one place but not in another—like California or Nevada, for instance—unless it is effected in a simple way by machinery. This is now done by small elevating wheels, 4 or 5 ft. in diameter, by which this medium stuff is conveyed back to the table, so that by this arrangement only two sorts come from the concentrator—namely, the rich portion and the worthless tailings. But, notwithstanding these contrivances, it is in many cases advisable to regulate the concentration for only two educts, and to dispense with the medium stuff by turning it either into the rich portion or into the tailings, as desired.

The influence of specific gravity decreases with the increase of the fineness of the ore particles, so that in the condition of the finest slime the influence of specific gravity is very small. Hence arises the difficulty of concentrating such stuff.

Sometimes concentration of gold ores should precede amalgamation: for instance, (1) when the ore is poor; (2) when the amalgamation of middle-class ore is performed in grinding pans; (3) when the ore contains auriferous sulphurets.*

On the other hand there are cases when concentration should follow amalgamation: as (1) when ore rich in free gold is worked in grinding pans or other amalgamators; (2) when middle-class ore is treated by amalgamation without grinding.

The sands, therefore, which flow from the amalgamated copper plates are subjected to concentration to collect the free gold with the auriferous sulphurets they contain.

**Concentration** consists of the following operations:—

(1.) Sizing the sands by means of pointed boxes (Spitzkasten).

(2.) Concentrating the sands in the pointed boxes in sluices, having a self-raising gate or riffles.

---

* See my "Metallurgy of Silver," *art.* Milling at the Montana Mine.

(3.) Subjecting the sands in the sluice to a further concentration in rockers, buddles, Hendy's concentrators, &c.

(4.) Giving the buddle concentrates a more perfect cleaning in the tossing tub.

(5.) In more modern mills, the pulp from the battery discharges directly on to vanners, or concentrating tables, for concentrating the escaping gold and sulphurets, or they discharge first into sizers and from these on to vanners.

(6.) In some establishments where the battery pulp is subjected to pan amalgamation, the tailings from the settlers are discharged on to vanners, and to concentrate the resulting finely-ground slimes it is advisable to discharge from the vanners on to round buddles, which prove effective for treatment of very finely-ground material.

**Spitzkasten.—Rittenger's Funnel or Pointed Box.**—In this apparatus the funnel boxes are rectangular pyramids, with the base upwards. They were designed to replace the "labyrinths" or settling pits—the oldest classifying arrangement—for assorting sands for concentration directly from the battery. The battery sands are too fine for sorting by sieves.

The stuff flows from the battery through several of these boxes—each different in size, and each delivering a different sized grain—directly to the concentrators. In this way the sands may be separated into different degrees of fineness at very little expense, in a simple apparatus, and without the help of hands.

These boxes have also the advantage of getting rid of incurring surplus water, which would interfere with concentration. The sands are always obtained from the boxes in the necessary state of dilution for the tables. The principle is the same as with the labyrinth. The first box is narrow, which causes the water to flow swiftly, thus allowing only the coarser grains to sink, while the balance is carried into the next wider box, where the same quantity of water, spreading, assumes a slower motion, so that the next finer sand is separated, and so on with the other boxes.

Figs. 63 and 64 represent a pointed box; Fig. 63 is a vertical longitudinal section, Fig. 65 a vertical cross section. The stuff flows from the battery, or from a sifting apparatus, at *a*, into the box. The sinking grains concentrate at the point *o*, and flow out through the ascending conduit, *o*′, into the trough, *p*. The finer stuff which could not resist the current of the water is carried over the spout, *b*, into the next larger box. The conduit, *o*′, ascends in order to counterbalance the water pressure inside the box.

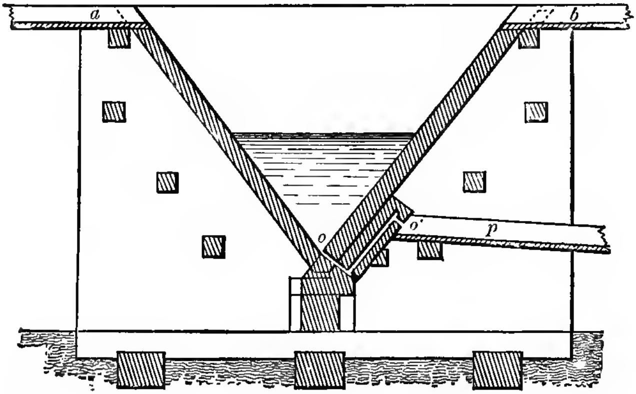


FIG. 63.—SPITZKASTEN. Vertical Longitudinal Section. Scale ½ in. = 1 ft.

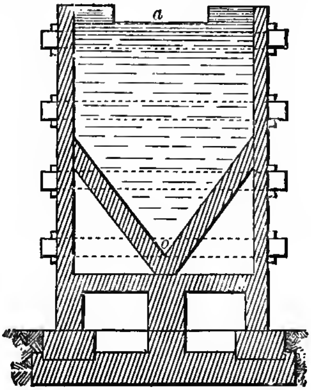


FIG. 64.—SPITZKASTEN. Vertical Cross Section. Scale ½ in. = 1 ft.

To obtain all the advantages of this apparatus theoretical and practical knowledge must be taken account of in construction. The width of the separate boxes is important. It depends on the quantity of stuff which is intended to enter the box in a second, and on the degree of the coarseness and density of the grains in it. According to experience, the first box by which the coarsest part of coarse ore is to be separated should receive one-tenth of a foot width to each cubic foot of stuff flowing per second. Each of the next following three boxes receive twice the width of the preceding.

If, therefore, such an arrangement be intended to separate 10 cubic ft. of coarse ore per second, the first box should receive 10 × 1·10 = 1 ft. width, and the whole four boxes as follows :—

1, 2, 4, 8 feet width, and
3, 6, 9, 12 ,, length.

The depth is given by the inclination, and measured from a horizontal line is fifty degrees. This inclination is given to the sides of the width. The conduit, *o'*, can be formed also by a pipe, *i*, Fig. 65, which stands vertical on the last boxes, containing the fine sands. The pipe should be shorter with coarser sands.

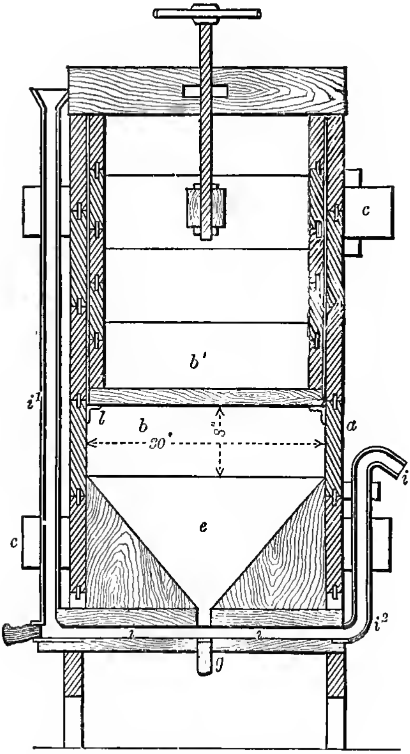


Fig. 65.—Spitz Lute. Scale ½ in. = 1 ft.

Should it be necessary to obtain a thicker stuff for the concentrator, there must be a valve at *o*, opening at short intervals. The connecting troughs between two boxes (Fig. 66) widen towards the larger boxes, and must be sufficiently inclined to insure that no sand can deposit in them. The coarser the sand, the more inclination. The boxes are not always close together, in which case the conveying troughs are narrow, and discharge in a distributing board of the next box.

The conveying troughs have a square section. It is calculated that five square inches will answer for each cubic foot of stuff per minute. If, for instance, seven cubic feet of diluted stuff flows through the trough per minute, it must have 7 × 5 = 35 square inches or about 6 by 6 in the clear.

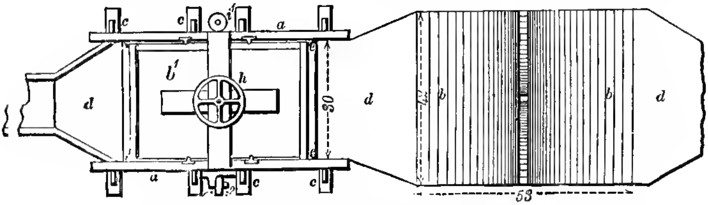


FIG. 66.—SPITZ LUTE. Scale ¼ in. = 1 ft.

The inclination of the conveying trough must be at least, in each 6 ft.—

| | |
|---|---|
| For coarse sands | 1 to 1½ inch. |
| ,, middle fine | ½ ,, ¾ ,, |
| ,, fine | ¼ ,, ½ ,, |
| ,, slime | ⅛ ,, ¼ ,, |

This inclination is for galena ore containing not over 5 per cent. of galena. The outlet of the pipe *o'* (Fig. 63) is regulated by mouthpieces, of which there must be several of different sizes. Once these are regulated, there is no other work to be done, except to watch the regular flow, as some accident may cause the conduit to become choked. To obviate such mishaps the larger boxes are provided with a rod in the centre, on the lower end of which, just above the opening, are two small rings, so that if the sand or pulp accumulate by neglect of the watchman this rod is turned until the stuff becomes loose enough to force itself through.

The percentage received from boxes of the dimensions described is as follows :—

| | |
|---|---|
| From the first box . . | 40 per cent. of sand. |
| ,, ,, second ,, . . | 28 ,, ,, |
| ,, ,, third ,, . . | 18 ,, ,, |
| ,, ,, fourth ,, . . | 10 ,, ,, |

So that the loss is not more than from 4 to 6 per cent. It sometimes occurs that several stamps have to stop working, in which case the supply of crushed stuff is diminished. It is then necessary to replace this deficiency of stuff by water. But if the supply of water depends on circumstances that cannot be regulated, or if the deficiency cannot be counterbalanced by water, then other means must keep the fluid in the normal condition, and for this purpose each box is provided with a movable longitudinal board. It is sufficient to make this board from 12 to 18 in. wide. In proportion as the supply of water is less, the dividing board is moved towards one side. If, for instance, the supply should decrease to half the normal quantity, the board is placed in the middle of the box, and all the stuff must enter on one side. The active portion of the water reaches only a few inches below the surface, so that the dividing board need not touch the bottom.

In Schemnitz (Hungary), for crushing 20 tons in twenty-four hours, the boxes have the following dimensions *:—

| | | | |
|---|---|---|---|
| First box . . | 6 ft. long, | 2¾ ft. wide, | 4 ft. deep. |
| Second ,, . . | 9 ,, ,, | 5 ,, ,, | 6 ,, ,, |
| Third ,, . . | 12 ,, ,, | 9 ,, ,, | 8 ,, ,, |
| Fourth ,, . . | 16 ,, ,, | 15 ,, ,, | 10 ,, ,, |

| | |
|---|---|
| The first box gives . . | 40 per cent. of sand. |
| ,, second ,, ,, . . | 22 ,, ,, |
| ,, third ,, ,, . . | 20 ,, ,, |
| ,, fourth ,, ,, . . | 12 ,, ,, |

**The Trichter Apparatus** is shown in Fig. 67, which is a top view. The first two boxes, *a a*, are narrow and oblong; then follow three other boxes, *b b*, square in section, set 2 ft. 10 in. each side; next two rows of four boxes, *c c*. . . . *c*, in which the stuff divides. The depth in each is 20 in. By means of gates each box can be shut off if required.

The small boxes could be advantageously used for sizing

---

* According to Prof. G. Kustel.

the stuff or tailings of unroasted ores from the pan amalgamation if concentration were intended; but in order to obtain regular work there should be a large agitator,* with a continual discharge into funnels, which latter should be proportioned in number and size to the quantity of the tailings. A common receiving agitator is necessary, because the discharge of the tailings from the settlers is in most mills periodical. Where the pulp is excessively diluted in the settlers, the agitator, if not large enough, must be provided with a trough at top to carry off surplus water.

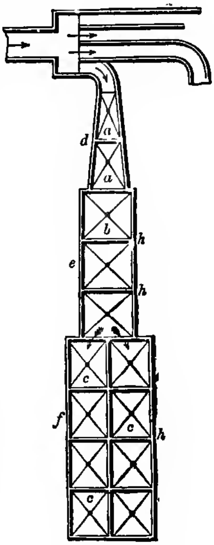


Fig. 67.—Trichter Apparatus.

**Concentration of the Pointed Box Sands in Sluices with Self-raising Riffle-gate.**—The pointed box sands flow through wooden sluice boxes of a rectangular section, provided at the lower end with a self-raising gate, acting as a riffle, in which the heavier portions of the sands, consisting of sulphurets, black sand, &c., form a deposit near the head, while the lighter particles escape over the gate. For the coarser sands the boxes have a width of 1 in.; for the finer sands they are from 1⅙ to 1½ in. wide for every stamp. They are usually 18 ft. long, and have the natural grade for the passing of the sands through them. For every two boxes or two sets of boxes there is one riffle-gate. The gate opening is generally 7 in. wide and 18 in. deep. The gate is raised to its full height once in twenty-four hours by means of a horizontal ratchet wheel, which is keyed to a vertical screw attached to the gate. Two ratchet wheels, one above the other, are attached to the screw, the teeth of one having the reversed direction of those of the other. By applying a lever arrangement on one side of

---

* See my "Metallurgy of Silver," *art.* Working Tailings at the Tombstone Mines in Arizona.

the vertical screw to one of these the gate is raised; by applying it to the other on the other side the gate is lowered. Two boxes, 15 to 16 in. wide, are filled to a depth of 18 in. by a 15-stamp battery in twenty-four hours. Two boxes or two sets of boxes are used alternately.

**Concentration in Rockers and Buddles.**—When the sluice sands are subjected to a further concentration on rockers they are discharged into a tank. The tailings of the rocker containing the finer sulphurets are treated subsequently in buddles, worked either by horse or by steam power.

**The Rocker.**—The rocker, represented isometrically in Fig. 68, consists of a wooden table of 2-in. pine plank, 20 in. wide, and 10 ft. long, supported at both ends by wooden rockers, representing a section 20 in. in width and about 3 in. in depth. The table is enclosed on the long sides and the upper end by 6-in. boards, the lower end being left open for the discharge of the tailings. It has an inclination of 1 in. to the foot. This inclination can be increased, when working coarser sands, by removing some of the supporting scantlings at the lower end. The upper end is fixed by a bolt, which, working in a slot, does not prevent the rocking motion. The floor of the table when in equilibrium is 7 in. above the support.

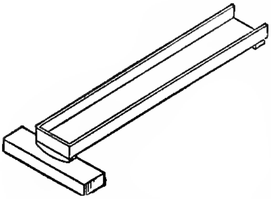


FIG. 68.—THE ROCKER.

The concentrated sluice sands are introduced at the head of the rocker in charges of from 3 to 5 shovels (the greater quantity corresponding to the sands poorest in sulphurets), and a stream of water discharged by a 1-in. pipe under a 6-in. pressure is turned on the sands by means of a rubber hose. The rocker is set in motion by the left hand of the workman, giving it about 60 strokes of 8 in. a minute. For coarser sands a greater number of strokes is required. The lighter sands gradually work down, while the sulphurets remain nearer the head. With his right hand the workman works the sulphurets up to near the head with a flat wooden shovel, which passes closely along the bottom of the rocker, while the lighter sands pass off. In some mills the rocker receives its motion by machinery. When clean the sulphurets are removed from the rocker by an iron scoop. A workman can treat 300 shovels of sand in a ten-hour shift.

Simple as this machine may appear, it does very good work, when operated with a proper quantity of water, right motion, and exact inclination.

**Green's Jigger** (Figs. 69, 70) consists of a series of sieves, which receive a reciprocating movement up and down in water by means of eccentric or revolving cranks. The space below is portioned off, so that each sieve discharges into a separate compartment. Not alone does this apparatus effect a concentration of the ore, but by making the sieves of different degrees of fineness a sizing can be effected. Fig. 69 is a plan, and Figs. 70 and 71 vertical sections, of the apparatus as fitted with three sieves, 21, 22, 23. These are fixed in a rectangular frame, 24, fitted to work up and down in a

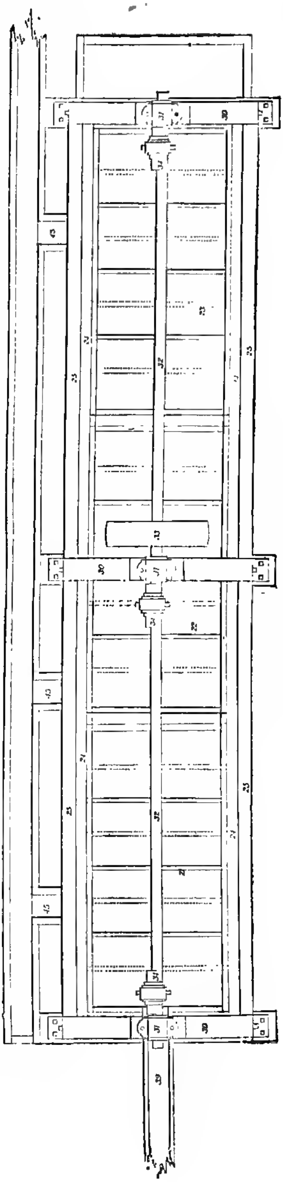


FIG. 69.—GREEN'S JIGGER. Plan.

fixed rectangular frame, 25, forming the upper part of a tank or hutch, which is supplied with clean water by a pipe, 26. The bottom of the tank is formed by three inverted pyramidal cast-iron shells, 27, 28, 29, forming separate compartments respectively under the sieves 21, 22, 23, and the whole is supported by cast-iron standards, 30.

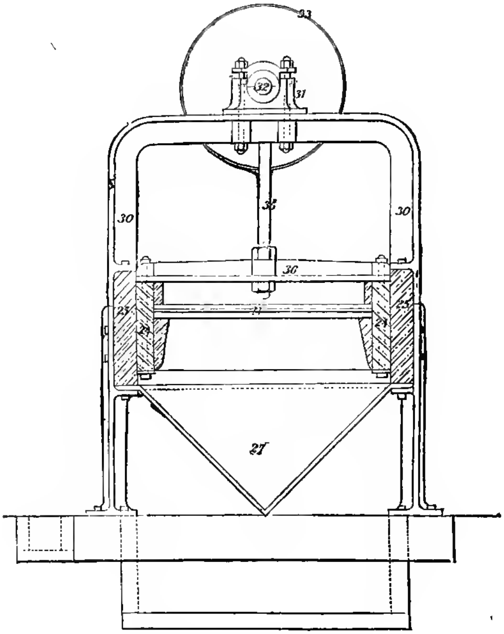


Fig. 70.—Green's Jigger. Section.

These standards are fitted with bearings, 31, for a horizontal shaft, 32, running along above the middle of the tank, and driven by a belt acting on a pulley, 33. The shaft, 32, is formed with three cranks, 34, of a small throw, to which are adapted brasses working in transverse slots formed in the heads of there connecting rods, 35, fixed to the cross bars, 36, of the

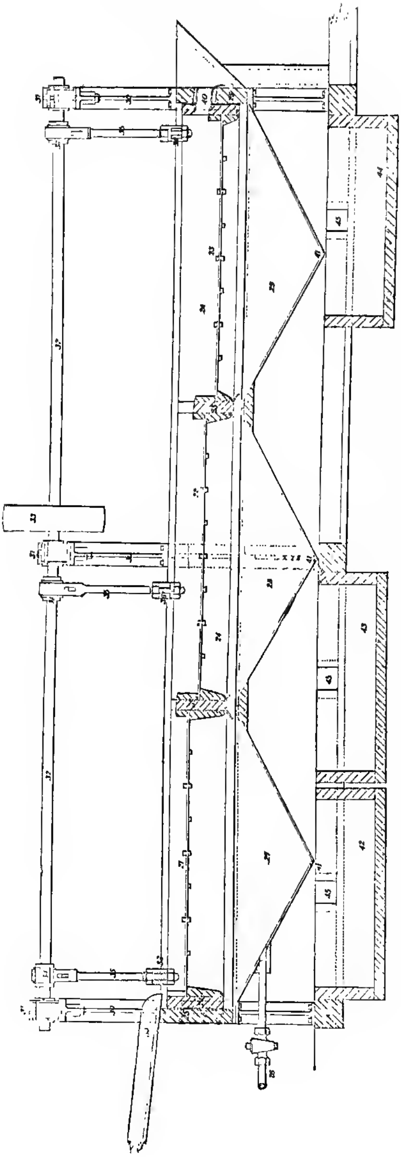


FIG. 71.—GREEN'S JIGGER. Longitudinal Section.

sieve frame, 24. The sieves, 21, 22, 23, are fixed in the frame at successively lower levels, and there are transverse division pieces, 37, 38, between the sieves, with their upper edges cut down to the proper overflow level for each sieve.

The stuff to be acted upon is fed by water from a launder, 39, upon the highest sieve, 21, and what is not separated by this sieve overflows to the middle one, 22, and some also overflows from the middle one to the third, 23. The waste finally overflows by a discharge duct, 40, at the end of the third sieve. The materials which pass through the sieves may be discharged at intervals from the compartments below, 27, 28, 29, by opening valves or sluices; or a continuous discharge may be effected into the receptacles 42, 43, and 44, these receptacles being formed with overflow ducts, 45, for the water to pass off by.

**The Dolly Tub.**—This is an apparatus used for separating ores, which can be conveniently used with the separators or classifiers, and will concentrate very fine slimes after they leave the spitzkasten or any similar apparatus. It is shown in Figs. 72 and 73 in plan and section.

It consists of a cylindrical vessel, 1, of cast-iron or wood, and has a raised centre, 2. Inside the tub revolve the paddles, 3, carried by the arms, 4, which are fastened to a vertical shaft, 5, held in bearings in a framing, 6, fixed across the top of the tub. The vertical shaft, 5, is geared by a pair of toothed bevel wheels, 7, to a horizontal shaft, 8, also carried by the framing, 6, and which has on it the fast and loose pulleys, 9, to receive a driving strap.

The part, 10, of the bottom of the tub between the centre, 2, and the sides, 1, is by preference sloped downwards from the centre or made conical, as shown. A funnel, 11, is fitted to the top of the raised centre, 2, and leads to a central discharge pipe, 12, by which the water and the light slime or waste is discharged.

When the apparatus is used in combination with "classifiers," each class of materials is fed to a different dolly tub or at a different time from the same tub. The materials, along with a regulated supply of water, are led by a launder so as to fall into the dolly tub, by preference near the side, and the whirling motion produced by the revolving paddles, 3, and the consequent centrifugal tendency cause the heavier minerals to move to the side, and they gradually descend to the deepest part of the bottom, from a point in which they are withdrawn by an aperture, 13, of a regulated size.

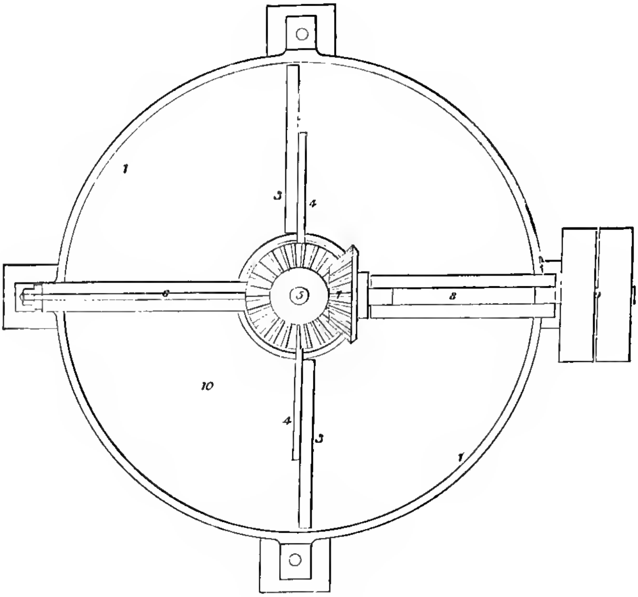


Fig. 72.—Dolly Tub. Plan.

In some cases the revolving paddles, 3, may be dispensed with, and the desired action be obtained by directing the inflowing water tangentially, or otherwise, so as to produce a suitable motion; and the action may be controlled by regulating the ingress of water, whilst the egress is of course adjusted to maintain a suitable depth of water in the vessel.

The whole of the water may be supplied along with the slimes, or a portion of it may be introduced by one or more

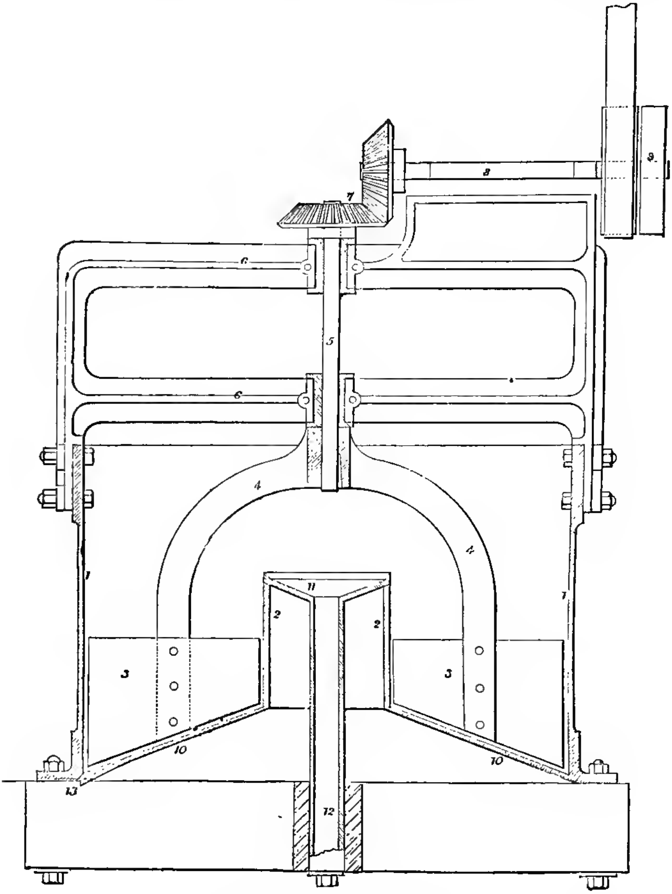


Fig. 73.—Dolly Tub. Section.

separate pipes. The discharge of the water being by the central funnel, 11, there is a continuous movement throughout the water towards that point. The circular motion, however, produces an opposite or centrifugal tendency, which, acting principally on the heavier particles, carries them to the sides of the vessel, whilst the lighter particles, being more under the influence of the inward current, are by it carried to the discharge pipe, 12. The supply and discharge to and from the apparatus may be either continuous or intermittent.

**The Round Buddle.**—This machine serves to concentrate slimes and fine sediments on a circular bottom, inclined toward the periphery. It is represented in Figs. 74 and 75. The conical bottom, *a*, is formed of wood, and is 16 ft. in diameter, or sometimes 20 ft. On this the stuff is distributed. *b* is the cone supporting the upper part of the feeding apparatus; *e*, a funnel perforated with four holes, and furnished at the top with an annular trough; *f f* are arms, carrying two brushes, balanced by the weights, *g g*; *h* is a launder for conducting the stuff into the funnel, *e*, from which it passes through the perforations, flows over the surface of the fixed cone, *b*, and from thence towards the circumference, leaving in its progress the heavier portions of its constituents, while the surface is

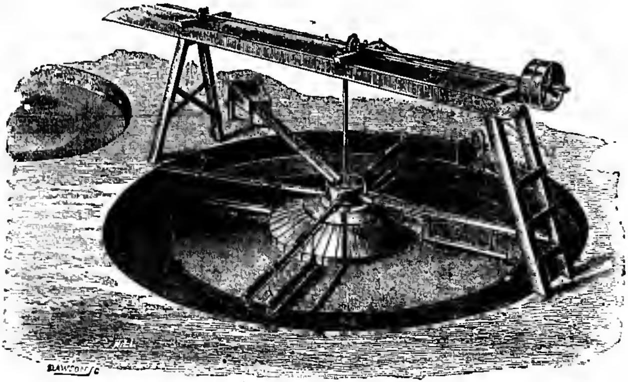


FIG. 74.—THE ROUND BUDDLE. Perspective View.

constantly swept smooth by means of the revolving brushes. By this means the particles of different densities will be found arranged in concentric circles.

The arms usually make from two and a half to four revolutions per minute, and a machine 18 ft. in diameter will work up from fifteen to twenty tons in ten hours.

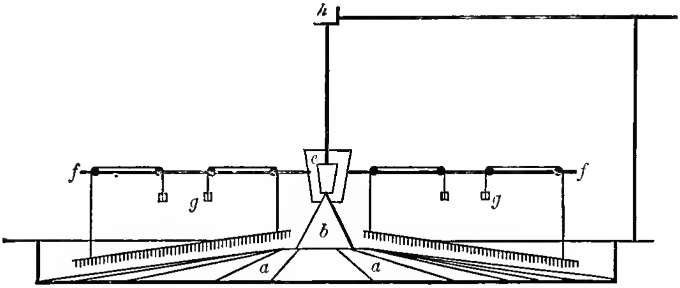


Fig. 75.—The Round Buddle. Section. Scale $\frac{3}{16}$ in. = 1 ft.

**Collom's Buddle** differs from the above by dividing the revolving table into two parts or concentric circles, one part being arranged by preference at one angle, whilst the other is arranged at a different angle. A pipe for washing off the ore is so formed as to wash off the ore from the different portions of the table at separate points, the ore from the upper portion of the table being washed off by one portion of the washing-off pipe into a launder or shoot and carried thereby into a receiver or hutch without touching the lower portion of the table, whilst the ore from the lower portion of the table will be afterwards washed by another portion of the washing-off pipe into another receiver or hutch. The water and the refuse or tailings from the general surface of the table will be washed into circular receivers or hutches. The table may be divided into two or more parts or concentric circles arranged at the same or different angles, and may be provided with several washing-off tubes, which may be arranged to operate alternately or together. The ore is fed to the apparatus by means of a launder or shoot through which a stream of water flows, and the ore is washed, dressed, or cleaned by means of a continuous flow of water from a circular tube.

This buddle is applicable to the concentration or dressing of both gold and silver ores, and consists of a circular table in which there are grooves or channels for the reception of mercury. These grooves are arranged in concentric circles, and as the pulp passes over them the mercury absorbs any amalgamable gold passing over it; while the heavier ore particles settle on the table, the remainder will be washed off by jets of water from washing-off tubes into suitable hutches or receivers. The

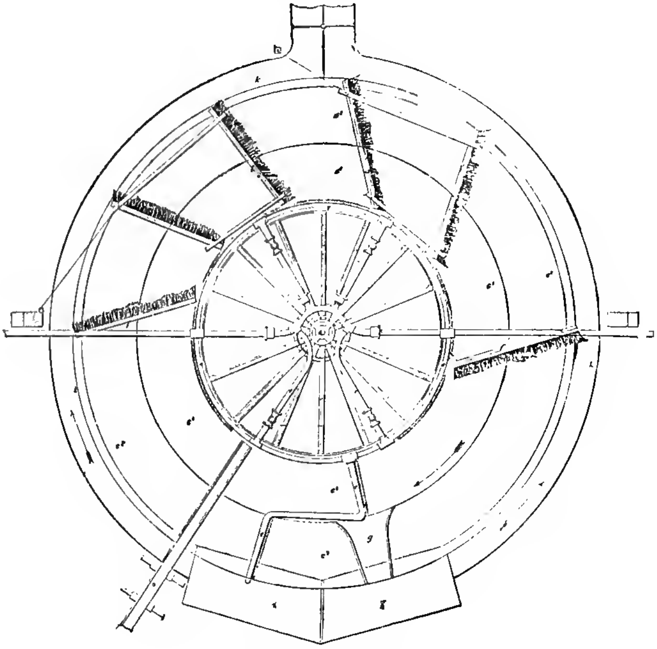


FIG. 76.—COLLOM'S BUDDLE. Plan.

water, and the refuse or tailings from the general surface of the table will be washed into circular receivers or hutches. The tables forming the different concentric circles are generally set at different angles. The tables may be arranged to incline to or from the centre.

A clearer understanding of the machine will be obtained from the accompanying illustrations.

Fig. 76 is a plan of the machine; Fig. 77 is a vertical section of the same. *a*, *a*, is a vertical axis having a bevelled wheel at its upper end gearing with a similar on an horizontal shaft, *b*, which is driven in any convenient manner, and in this way the axis, *a*, is caused to revolve at a speed of about two or three revolutions per minute. On the axis, *a*, a table is mounted; it may be conveniently about 25 ft. in diameter, and this table consists of two rings, $c^1$ and $c^2$, or it may be of three or more rings if it be desired to sort the ore or material into a greater number of qualities. *d* is a trough or launder through which the ore or material in a finely divided

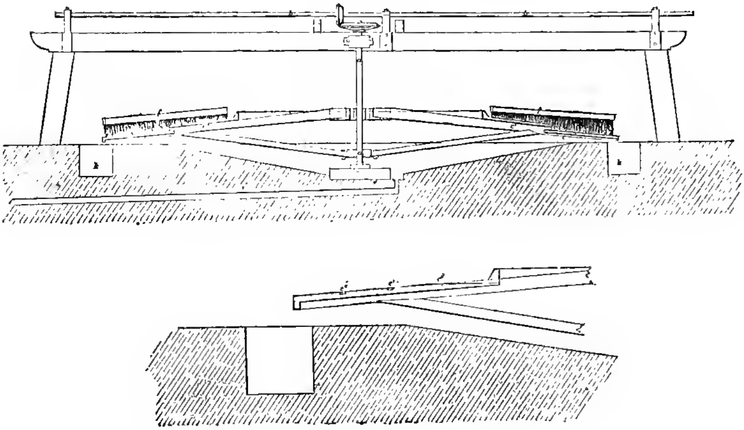


Fig. 77.—Collom's Buddle. Section.

state and mixed with water is run on to the inner side of the smaller ring, $c^1$, of the table; *e*, *e*, are perforated pipes from which jets of water issue, these pipes (it will be seen) being omitted in Fig. 77.

As the table revolves, the ore upon it is subjected to these jets, and the particles are carried a greater or less distance down the table according to their relative densities and the strength of the jets of water, which can be regulated by stop-cocks in such a manner that as the table moves round the material upon it becomes exposed to stronger and stronger jets, but yet the jets should not be so strong as to wash any considerable proportion of the more valuable material entirely off the lower ring, $c^2$, of the table; *f*, *f*, are brushes resting loosely on the table and serving to move and distribute the material. The brushes are held and kept in their places by cords or light rods; they should not bear so heavily on the table as to cause the material to accumulate against them.

In the course of the revolutions of the table, all but the heaviest and best material is washed off the upper and inner ring, $c^1$, of the table on to the lower and larger ring, $c^2$, the slope of which is by preference somewhat steeper than that of the upper ring. Here a second sorting takes place, the lighter parts being washed over the edge of the table, whilst those which are heavier remain, being able to resist the flow of the water; $e^{\times}$, $e^{\times\times}$, are perforated pipes throwing jets of water, and so arranged and adjusted as to wash all the material from the table as it passes beneath them. When the material arrives opposite to these pipes it is already sorted. The jets from the pipes $e^{\times}$ wash down the best quality into the trough or launder, *g*, which conveys it into a receiver *h*, and the jets from the pipes $e^{\times\times}$ wash down the material of second quality from the lower and outer ring $c^2$ into the receiver *i*. *k* is a channel all round the table, which leads off the water and the lighter material to settling points.

The lower portion of Fig. 77 shows a transverse section of a portion of a table, which is grooved all round for receiving mercury. The grooves, which are marked $c^{\times}$, may conveniently be three in number, and each about three inches wide by half an inch deep. The mercury with which they are filled takes up particles of gold or silver, as already stated; and when it becomes fully charged, which will only be at long intervals, it is drawn off and renewed.

**The Concave Buddle.**—The concave buddle of Paine and Stevens is shown in Fig. 78. There are generally two buddles: one for the coarse sluice concentrates, and the other for the finer. They have an exterior diameter of 18 to 20 ft., slightly inclined inwards, in the centre of which is an opening 2½ ft. in diameter. The vertical shaft is supported by the wooden block, *m*, which carries the journal box. The shaft has a rotary motion imparted by mill-gearing above, and to which several appendages are attached.

Attached to the shaft are: (1) The self-raising riffle pulley, *g*, which is raised by a rod, *p*, receiving its upward and downward motion from the endless screw, *b*, and pinion wheel; (2) the arms, *f f*, carrying the brusher; (3) the sand distributing troughs, *e e*. The clear water box, *i*, is suspended by the wheels, *v v*, on an annular flat ring. It is supplied by the stationary wooden box, *r*, and discharges the water, by the iron pipe, *k*, into the sieve boxes, *y* and *z*. The bore, *s s*, is fed by the trough, *h*, from the mixing trough, *y*, and sieve box, *z*. The vertical shaft receives its motion by the pulley, *a*, and bevel gearing, *d d*.

The first operation of the buddle consists in washing the sluice concentrates. For this purpose the sluice gate is lowered and the sands are gradually washed through the mixing and sieve box, *y* and *z*, and box, *h*, into the distributing box, *s s*, from which they are discharged by the six revolving Russia iron trough arms, *c c*, upon the annular apron. From these they flow down the inclined conical table, constituting the buddle. The arms, *ff*, carrying the poles to which the brooms are attached, revolve at the same time with the distributors, *c c*. The brooms maintain a regular and even surface to the sands. The heavy sulphates lodge near the head of the buddle, while the lighter particles move on and finally fall over the circular riffle, g, which consists of an iron pulley in the centre of the

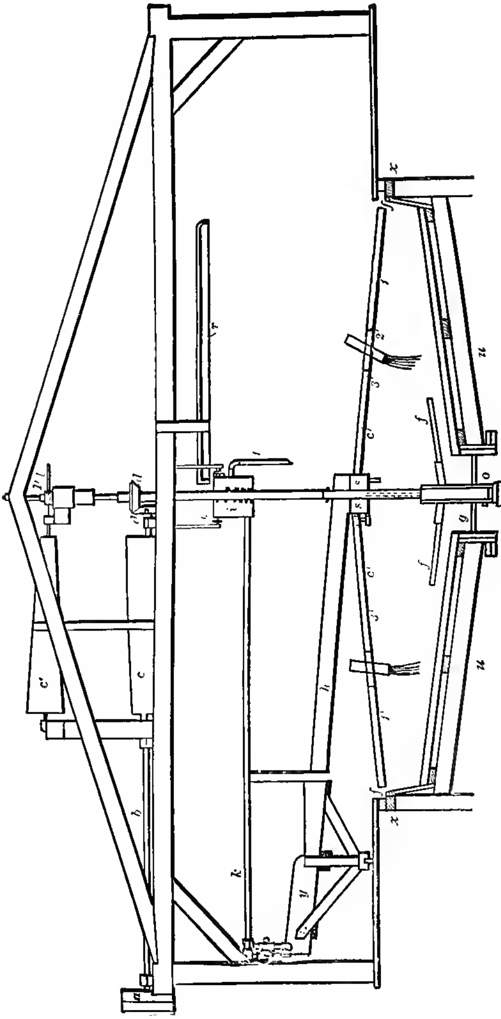


FIG. 78.—THE CONCAVE BUDDLE. Scale 1 in. = 5 ft.

buddle. This ring forms a close joint at the end of the wooden buddle floor, *u u*. The pulley, and with it the arms of the broom poles, are gradually raised as the washed sands fill the buddle.

When working the coarser ore, the time consumed to fill the buddle is six hours; for the finer sands twelve hours are required. The water required by the buddle in washing the coarse sands is 8 cubic ft. per minute, spread by the revolving distributors, which make seven revolutions a minute over a periphery of nearly 63 ft.

When the buddle is full the sands are divided into three concentric rings. The interior ring, 3½ ft. in width, is considered as waste, and is washed out. This is done by removing the end pieces, 1 1, of the distributors and turning the collars, 2 2, in the manner represented in the drawing by dotted lines. The clear water is conveyed from the stationary sluice, *r*, and washes out the central ring, leaving the distributors at the connecting collars, 2 2. The central ring is lowered at the same time. While the central ring of the sands is removed the outside ring at the periphery of the buddle, 3 ft. in width—the so-called headings—is shovelled out by the ore dresser. The middle portion of the sands, being about 2 ft. in width, is gradually washed with water directed by means of the collars, 2 2, of the distributors. The headings of these sands are added, when washed, to the first headings, and the central ring of sands is removed as we have described.

The headings from the first washing go through a second washing, called doubling. This is done by raising the central riffle ring about three inches and filling the buddle up to the rim of the ring with poor sands from the tossing tub, which are introduced into the mixing box, *y*, and washed into the buddle by the water from the pipe, *k*, and trough, *r*. When ready for doubling the buddle has an inclination of 1¾ in. per ft. for coarse sands and 1½ in. for the finer sands. All the headings are shovelled into the mixing trough, which, being supported on wheels, can be brought to any part of the working floor around the periphery of the buddle. The distributing clean-water box, *i*, with water pipe, *k*, attached, also revolves on the wheels, *u u*. The time consumed in re-washing the headings of the first washing is generally three hours.

When the operation of washing the headings is completed the sands are again divided into three concentric rings. The central portion, having a width of about 4 ft., being removed as before, the outer ring, about 18 in. in width, is ready for tossing, while the middle ring, of about 3 ft. in width, is washed down as described, the heading of it being added to those previously removed by tossing.

**The Tossing Tub.**—The tossing, or final cleaning of the sulphurets, if they are to be treated by the chlorination process, is usually performed on the buddle headings. This is done in a tub of the following description (Figs. 79, 80). The tub, having 1½ to 2 in. staves, is conical in form, tapering toward the bottom, 4 ft. in diameter, and 2½ ft. deep in the clear. Through the axis of the tub a hollow cast-iron cone, *c*, passes, reaching a few inches above the top of the tub, and fastened by a flange to the bottom. A shaft, *s*, Fig. 81, passing through this cone and resting on a journal underneath, carries the yoke, *l*, to which the horizontal flat-iron stirrers are riveted. Motion is communicated to the shaft by bevel

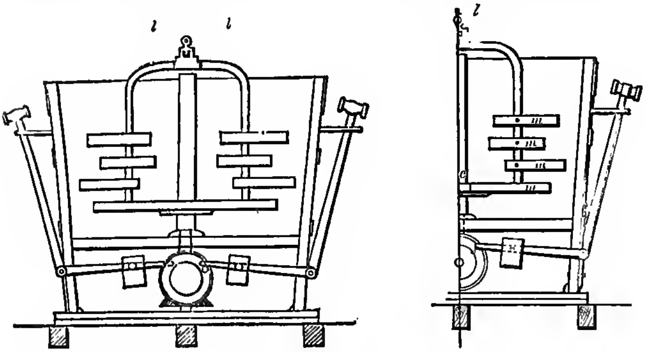


FIG. 79. THE TOSSING TUB. FIG. 80.
Scale $\frac{5}{16}$ in. = 1 foot.

gearing, *a b*. The hammers are set in motion by the pins, *r r*, attached to the vertical bevel gear, as seen in the figure. When ready for tossing, the tub is filled to nearly half its height with water, the stirrers are set in motion, making forty-eight revolutions a minute, and the ore is shovelled in near the periphery of the tub. When nearly full the yoke is lifted out by means of a rope and pulley overhead, and the sands are allowed to settle while the hammers are set in motion, making ninety-six strokes each per minute to facilitate the rapid settling of the sulphurets and sands. When the sands have settled, the water is drawn off by an iron syphon, the skimmings are removed to a depth of two inches and thrown out as waste, the remaining upper half of the sands are re-tossed, and the resulting sands above the sulphurets washed again in the buddle during doubling. The lower half of about five to six inches, consisting of sulphurets sufficiently concentrated, is delivered at the chlorination works to be further treated for gold.

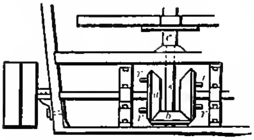


FIG. 81.—GEARING FOR TOSSING TUB.

Two buddles working the sands of a 30-stamp mill require the attendance of three men.

The riffle boxes and, to some extent, the concave buddles, lose a considerable proportion of coarse sulphurets. To minimise this loss, the tailings from the buddles and riffle sluices pass through a long string of boxes (supplied with wooden riffles of a square section of $\frac{3}{4}$ of an in., which are washed every Sunday when the mill stops) into and through a wooden box about 5 ft. wide, 10 ft. long, and 3 ft. deep. This box is open at the lower end. The heavier sulphurets form a deposit at the head, and the headings are treated by the rocker.

In concentrating by means of riffle sluices, concave buddles, and rockers for buddle tailings, the loss in sulphurets is 12·5 per cent. when treating quartz sands, which contain about ·33 per cent. of sulphur, or its equivalent in sulphurets.

**Hendy's Concentrator.**—When this concentrator is employed, it is not usually necessary to use the pointed box, nor previously to concentrate by self-raising riffle boxes. The concentrator is shown in Fig. 82, and consists of a shallow pan 5 or 6 ft. in diameter, supported by a vertical shaft in the centre, and made to oscillate by cranks on one side; these are joined by connecting rods with the periphery of the pan, and this turns upon a vertical axis through a short distance for every revolution of the crank shaft. The bottom of the pan is raised in the centre to nearly the height of the rim, in order to facilitate the movement of particles towards the circumference.

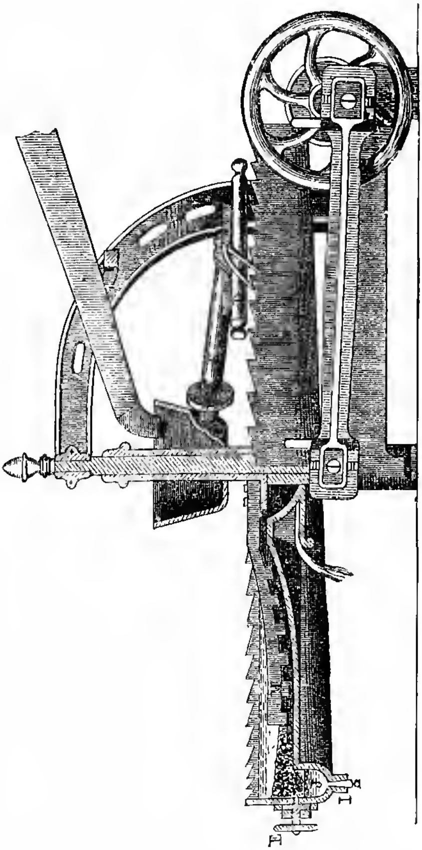


FIG. 82.—HENDY'S CONCENTRATOR.

The machine must be carefully levelled; and the tailings, direct from the blankets, are delivered by a trough to the hopper, *c*, from which they pass through the pipe, *k*, and distributor, *d*, into the pan near its outer edge; and the rotation of *d* by means of two pawls attached to it, and acted on by teeth on the rim of the pan, causes this delivery of the tailings to take place evenly at all parts of the circumference. Rake-like arms, *m*, rotate with *d*, in order to stir the compact mass of sand, &c., which settles at the bottom. The crank makes 210 revolutions per minute, and the accumulated sulphurets are discharged through the gate, *e*, while the amalgam and mercury collect in the depression, *j*. The machine concentrates 5 tons of tailings per twenty-four hours.

**Mill with Hendy's Concentrators.**—Fig. 83 shows the general arrangement of a gold mill, and from it a clear idea will be at once gathered of the several processes which intervene between the raising of the ore and the final escape of the waste sands, &c., into the tail sluices.

The ore is thrown down in front of a Blake rock-breaker, and after leaving it passes down an incline to the self-feeder, and thence to the stamps, where amalgamation often takes place. In front of the stamps is an apron of amalgamated plates, and beyond are the blanket tables where the pulp undergoes a mechanical preparation, the resulting two qualities being differently treated. The sands which pass over are received in Hendy's concentrators, and after being a second time concentrated pass into the tail sluices.

That which remains on the blankets is washed into tanks, then passed through Attwood's amalgamators and over copper riffles, and finally reaches the second concentrator. Below the concentrator in the illustration are seen a Wheeler pan and settler. The pyrites obtained on the concentrators are usually roasted and treated by chlorination.

**The Frue Ore Concentrator or Vanner.**—This machine (Fig. 84), which is of American invention, has been found so efficient in the concentration of pulp and slime that I shall describe its construction and working in full detail.

Its principle is that of a revolving blanket, having a lateral shaking motion—a motion closely resembling that given to a shovel in vanning by hand. Revolving belts of canvas, with a

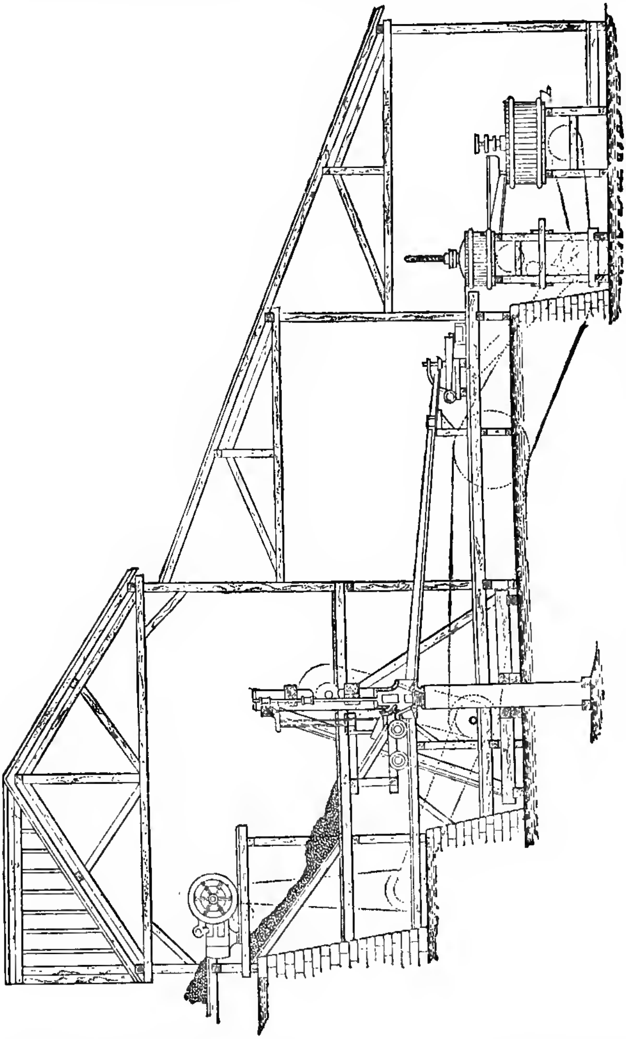


FIG. 83.—MILL WITH HENDY'S CONCENTRATORS.

lateral or end blow, have been used heretofore with pretty good results, but the side shake has proved a great improvement on the old practice, and india-rubber belts with high flanges have been introduced in place of canvas. The ore when stamped fine can be received directly from the stamps on to the vanners, which concentrate the ores to a high percentage, including those which contain galena, zinc blende, iron and copper pyrites.

In Fig. 85, A A are the main rollers that carry the belt and form the ends of the table. Each roller is 50 in. long and 13 in. in diameter; it is made of sheet-iron, galvanized, and is light and strong.

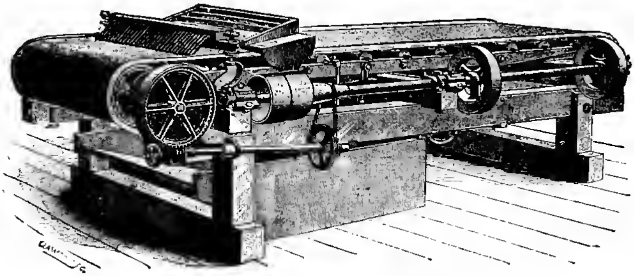


Fig. 84.—The Frue Concentrator or Vanner.

The bolts which fasten the boxes of A A to the ends of F also fasten to F the upper supports which rest on uprights, N.

The rollers, B and C, are of the same diameter, and are made in the same way as A A. The roller part of C is shorter than that of A A and B, and has also rounded edges, the upper surface of the belt with its flanges passing over it. The belt, E, passes through water underneath B, depositing its concentrations in the No. 4 box, and then, passing out of the water, the belt E passes over C, the tightener roller. B and C are hung to the shaking frame, F, by hangers, P P, which swing on the bolts fastening them to F. By means of the hand screws, B and C can be adjusted on either side, thus tightening and also controlling the belt (Figs. 86 and 87).

The boxes holding A A in place have slots and adjusting screws, so that by moving them out or in A A can be made to have a very strong influence on the belt E; and as E sometimes travels too much towards one side, this tendency can be stopped most quickly by lengthening or shortening on one side or the other of A A, remembering that the belt always travels to the near side of the pulley. The swinging of B or C also controls the belt. *c c* are bolts and washers to take up the end play of rollers, A A; the bolts pass through holes in the gudgeons of A A, Fig. 85.

D D are the small galvanized iron rollers, and their support causes the belt, E, to form the surface of the evenly inclined plane table. This moving and shaking table has a frame, F, of ash, bolted together, and having A A as its extremities. This frame is braced by five cross pieces. The bolts holding together the frame pass through the sides close to the cross pieces; the cross pieces are parallel with A A and D D, and their position can be understood by the three flat spring connections, R Q, which are bolted to three of them, one to each, underneath the frame.

The belt, E, is 4 ft. wide, 27½ ft. in entire length; being an endless belt of rubber with raised sides.

G G is the stationary frame. This is bound together by three cross timbers, which are extended on one side to support the crank shaft, H.

G G supports the whole machine. The required inclination of the table is given by elevating or depressing the lower end of G G. This is accomplished by means of wedges. The frame rests on uprights, Nos. 3 3, fastened to the sills which form the foundation of the machines in the mill.

F is supported on G G by uprights, N, four on each side. These uprights are of flat wrought iron, with cast-iron bearings above and below. Each middle bearing on F has one bolt hole; there are two of them on each side. The end bearings have two bolt holes, there being four of them, two on each side. These bolts pass through the frame, F, and also hold to the frame the bearings of A A, which work in a slot. The

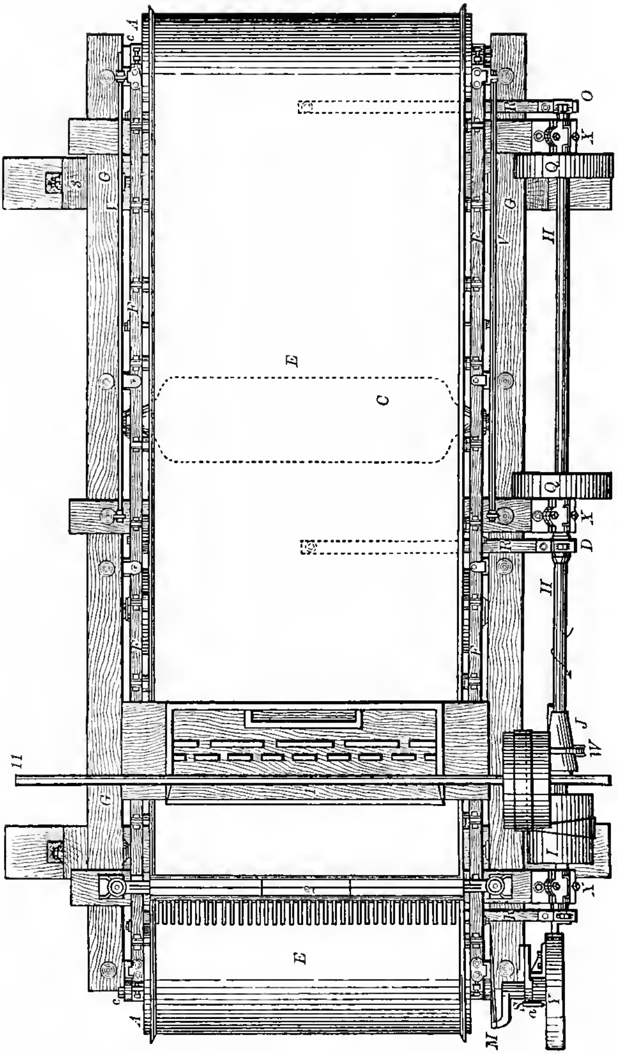


FIG. 85.—THE FRUE VANNER.

bearings of A, the upper or head roller, are higher than those of A, the foot roller; *i.e.* A is a trifle higher than the regular plane of the table, and the first small roller, D, should be raised a trifle.

The shape of the lower or bottom bearings of the uprights, N, can be understood by examining *b*, as shown in the end elevation and partly in the elevation. This lower bearing, *b*, extends across G, underneath, and is supported by a bolt passing through G. A lug on the upper side and on the outside end of *b* rests on G, and *b* hangs on the head of the bolt, and is kept stationary by the weight of *h* and its load. By striking with a hammer the face of *b* shown in the elevation, *b* is moved, changing the position of the lower bearing, and thus making N more or less vertical. By thus moving the lower supports of N, the sand corners in the belt, to be hereafter explained, are regulated.

The cross timbers binding together G G, and resting on them, are extended on one side, and on these extensions rests with its connections the main or crank shaft, H. This crank shaft has its bearings, X X X, and on them are brass cups to hold the lubricating compound; the cranks are ½ in. out of the centre, thus giving a one-inch throw. I is the driving pulley, forming with its belt the entire connection with the power. J is a cone pulley on the crank shaft, H. By shifting the small leather belt connecting J and W, the uphill travel of the main belt, E, is increased or diminished at will. The small belt connects to J the flanged pulley, W, which is on the small shaft, K, and by means of the hand wheel can be shifted on K and held in place.

The bearings of K are fastened to Y, a cast-iron shell protecting the worm, Q, and the worm gear, L; Y turns on a bearing bolted to the outside of G, and thus becomes a fulcrum for W and K. The object gained by this is that the weight of W and K (from Y) hangs on the small leather belt, preventing slipping or wear, at the same time making it positive. *a* is a screw used to relieve the small belt from the weight of K and W, taking all the strain off the small belt, and thus instantly stopping the uphill travel when desired. *m* is a hand screw by means of which the pulley can be moved, adjusting the small belt on the cone J, and thus regulating the uphill travel. K is the worm shaft, and terminates in a worm, Z, which connects with a worm gear, L. I travels in a bearing bolted to the outside of G. Z and L are protected from dirt by the shell of cast iron, Y, enveloping both (Fig. 86).

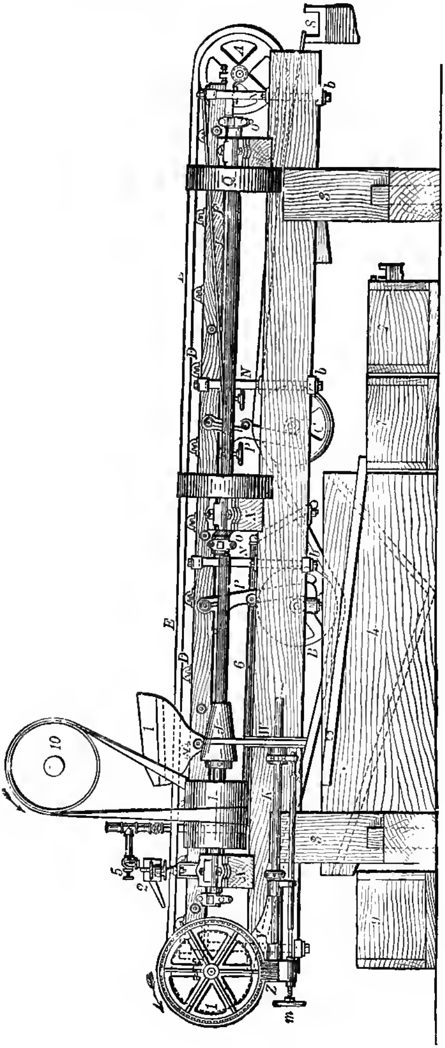


FIG. 86.—THE FRUE VANNER.

The short shaft which L revolves terminates in an arm, S, which drives a flat steel spring, M (which is a section of a circle), connected with the gudgeon of A. N N are the upright supports of the shaking table, F, carrying the belt, E. R are three flat steel spring connections bolted underneath the cross pieces of F, and attached to the cranks of the shaft, H, by brass boxes, O, O, &c., on which are cups for the lubricating compounds. These springs give the quick lateral motion, about 200 a minute. Q Q are two fly-wheels. V V are two rods passing from the middle cross timber to the lugs for the same at the foot of F. The cast-iron washers on the bolts of the cross timbers have lugs cast on them. V V pass through these lugs, and at each end are nuts on each side of the lugs. Thus V V prevent the movable frame N from sliding either up or down, and by them F is squared.

In Fig. 87 No. 2 is the clear water distributor, and consists of a wooden trough supplied with water by a pipe. The water discharges on the belt in drops from grooves 3 in. apart. Another form generally used for No. 2 is that of an iron trough, having brass spouts 1½ in. apart; by blocking every other of these holes, water jets can be made 3 in. apart. No. 1 is the ore spreader, which moves with F, and delivers the ore and water evenly on the belt. N is a copper well that fits in (and shakes with) the ore spreader at the place shown in the figure. This is used in concentrating gold ores, for saving the amalgam and quicksilver which escape from the silvered plates above, and can be taken out and emptied at any time. Into this well falls all the pulp from the battery. Its ends are lower than the wooden blocks of the spreader, so that the pulp passes over the ends of the well and is evenly distributed. For some gold ores it is desirable to use on the ore-spreader a silvered copper-plate the size of the spreader, and when this is so the wooden blocks of the spreader are fastened to a movable frame on top, so that they can be removed when the plate is cleaned up once or twice a month.

Nos. 5, 5, are the cocks to regulate the water from the pipes, Nos. 6, 6. Nos. 3, 3, are upright posts, which are firmly fastened into two sills. These posts are cut down on the inside to make square shoulders, on which rest G G. No. 4 is the concentration box, in which the water is kept at the necessary height to wash the surface of the belt as it passes through it. The overflow from No. 4 contains finely divided sulphurets in suspension; to settle these the water passes through boxes, Nos. 7, 7, 7. No. 8 is a section of the launder which carries off the tailings. No. 9 is a box into which the concentrations fall when scraped out of No. 4. Nos. 10 and 11 show the arrangement of the countershaft with the tight and loose pulleys for driving the machine; but it should be higher above the machine than represented in the drawing (Fig. 87).

In working the Frue vanner, the ore is fed with water on the belt, E, by means of the spreader, No. 1. Thus the feed is spread uniformly across the belt. A small amount of clear water is distributed by No. 2, which is a wooden trough, and contains a pipe (No. 6), or by an iron trough with brass spouts.

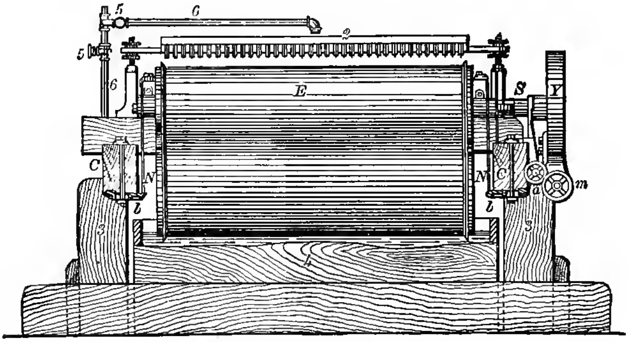


Fig. 87.—The Frue Vanner.

A depth of ⅜ to ½ in. of sand and water is constantly kept on the table. To the main shaft, H, sufficient motion is given to impart from 180 to 200 revolutions, with 1 in. throw.

The uphill travel or progressive motion varies from 2 to 12 ft. a minute according to the ore, and the inclination of the table is from 3 to 6 ins. in 12 ft., varying with the ore. This inclination can be changed at will by wedges at the foot of the machine; these wedges are under the lower end of G G, and rest on the shoulders of the uprights from the main timber of the mill.

The motion, the water used, the inclination of the table, and the uphill travel, will have to be regulated for each description of ore, but this once done no further trouble should be experienced in the manipulation.

In treating ore from the stamps, it may be found that too much water has been used by the stamps for proper treatment of the sand by the machine: in such a case there should be a box between the stamps and concentrator, from the bottom of which the sand, with the proper amount of water, may be drawn, the superfluous water passing away at the top; but as mineral matter will also pass away with it, there should be settling tanks to receive the water; the settlings can then be worked from time to time as they accumulate.

**Proper Consistency of Pulp in the Frue Vanner.**—The use of a proper quantity of water with the pulp from the stamps, and also its proper regulation, are very important. There should be formed on each side of the belt a slight corner of sand—that is to say, there should be on each side sand with less water in it than there is in the balance of the pulp on the belt. Unless this is so the corners will be sloppy, and there will be a loss. Sloppy corners are caused by using too much water with the pulp from the stamps passing on No. 1.

On the other hand, there may not be enough water with the pulp from the stamps, and the result will be too heavy sand corners. The remedy for this is to use more water in the pulp coming on No. 1.

As regards the proper amount of water to be used in the water spreader, No. 2, just enough and no more should be used to keep the ground between No. 1 and No. 2 covered, so that no points or fingers of sand shall show on the surface. The whole width of the belt between the water spreader and the ore spreader should be kept quite wet. It dry streaks or points occur, and the water as a consequence runs in streaks at the junction of the wet and dry channels, mineral will be picked up and floated away on the surface of the water, which "floating" of mineral is caused by its dryness, and not by its lightness; it has been coated with a film of air.

The proper amount of water with the pulp on No. 1 and the proper amount of water in No. 2 being fixed, the carrying over of the clean concentrations past the jets of No. 2 should be accomplished and regulated by the uphill travel only.

Frequently the sand and water on the belt will be distributed unevenly, the sand working to one side of the belt and making a heavy broad corner, while the other is sloppy. To control and remedy this, see first that there is no jar about the machine; that there are no loosely working parts; that everything is working noiselessly; and that all the parts are in line. If there is not an exact balance of the pulp on the belt, the heavy sand corner forms on one side or the other. To adjust the load and keep the sands evenly distributed on the belt, the lower bearings, *b*, of all the uprights, N, on one side of the machine, are moved forwards or backwards by slight blows of the hammer. The change of position from the vertical of N, &c., thus occasioned, affects the pulp on the belt; and by changing the position of *b*, &c., on one side or the other, the right balance or equilibrium will be obtained, and the sand and water (or pulp) will be uniformly distributed across the belt. If the heavy sand corner is on the shaft side, the bottom bearings, *b*, &c., on the opposite side must be moved out.

Again, the sand corner can be partly controlled by bending the end of the driving spring, fastened in the collar, towards the side having the thickest sands. The same effect, and even more positive, is produced by moving the crank shaft, and with it the table, the same way as the end of the driving spring is bent. The rolls underneath have also an effect on the corners, by swinging one end of each either towards one another or in the opposite direction.

The water in the concentration box is constantly agitated by the motion of the belt, and consequently the water escaping from this box carries in suspension a good amount of very finely divided sulphurets of high assay value. To save these, settling boxes, Nos. 7, 7, 7, should be used, which can be cleaned out once a month, a product being obtained which will add materially to the value obtained from the ore.

The quantity of water required for one machine will be from 1 to 1½ gals. per minute of clear water at the head, and from 1½ to 3 gals. per minute with the pulp. The boiler for a 5-stamp mill, with two concentrators, requires 1 gal. a minute; hence, in places where water is very scarce, 2 gals. per minute can supply five stamps, two Frue ore concentrators, and the boiler, by settling and pumping back.

**The Operation of the Frue Vanner.**—Recent practice has demonstrated that about 6 tons per twenty-four hours, passing through a 40-mesh screen, is as much as it is advisable to treat in a single machine. If a battery of five stamps does its proper amount of work, the quantity crushed is largely in excess of 6 tons; and for this reason the best practice is to have two Frue vanners for five stamps, if the stamps are heavy and the sulphurets are of high grade and difficult to save. Where pulp from five stamps is fed to two machines the pulp is divided, one half passing on each. The machines are generally placed in a double row on the same level, head to head, so that the attendant overlooks both rows in walking between. The concentrator floor should be so far below the level of the battery as to allow the feed launder to be above the head of the attendant. No sizing of the material is needed; the pulp passes directly from the stamps on to the copper plates (if used), and thence on to the vanners.

The belt forms the bed on which the dressing of the ore is effected, being an inclined plane 12 ft. long, and having down the two sides projecting rubber flanges, which prevent the water and sand from dropping over the sides. The arrangement of rollers permits of the belt being slowly revolved in the direction of its length and *up* the incline; thus, though the direction of the working plane remains always the same, its surface is constantly travelling. The crushed rock in a small stream of water falls near the upper end of the belt, by means of the sand distributor, No. 1, and flows down the belt towards its lower end. Now, as the inclination at which the belt is set is only from three to six inches in twelve feet, and as the stream of water is not large, and spreads also over the whole width of four feet, it is obvious that much of the crushed rock contained in the water would settle on the belt, while the water and the finer and lighter particles of sand would alone reach the foot of the table and drop over into a waste launder. In addition, we have the travelling of the belt upwards and onwards continuously.

The effect of this would naturally be to deliver all the rock which has settled on the belt over the upper end roller, *a*, and deposit the same eventually in the water tank, No. 4, below, through which the belt passes in plying around the roller, *b*. The action of the belt, then, simply amounts to this, that it forms an inclined plane, or working surface, which, by its progressive motion, will deposit in the tank, No. 4, all solid material which settles on it—that is, whatever is not carried off in suspension by the water flowing from No. 1. The belt is merely a self-discharging bed. To separate the heavier metallic minerals from the accompanying gangue or rock, it is evident that the above described action of the belt is not sufficient, for not only would the mineral be delivered in the tank below, but also a large proportion of the rock, which would certainly settle on the belt as well. A separation of the two classes has yet to be accomplished. For this purpose a second stream of water is employed. About one foot above the sand distributor—that is, just below the first small roller, D, —the water distributor is arranged, which delivers small jets of water, three inches apart, over the entire width of the belt. The revolving belt, carrying its load of settled rock and mineral, travels past the jets of water, taking with it such particles of mineral as have weight or specific gravity sufficient to resist the force of the descending water, while the lighter particles of rock are driven back by the water, and do not reach the tank, No. 4. In addition, a gentle side shake is given to the belt at right angles to the length and travel of the belt. By the introduction of this secondary motion the sand is kept in gentle agitation, uniformly distributed over the whole width of the belt, and the heavier particles of mineral, settling through the sand, cling to the belt and are carried up it, past the small jets of water, and deposited in a cleaned state within the tank for collection. Very little water is now needed to effect the separation of rock from mineral.

This machine is not adapted for the treatment of very coarse material, such as would be submitted to "jigging," but is adapted for the treatment of fine sands and slimes. Many forms of washing apparatus have been introduced for the special treatment of slimes. Some of these have been already described, such as the buddle, the inclined table, and the blanket sluice.

The important feature of the Frue vanner lies in the property which fine ore particles have of clinging to the rubber belt. The shaking motion separates the mineral from the sand as it flows slowly down the belt, as it keeps the material in gentle motion, and when once the material has touched the surface of the belt it clings and is carried up past the small streams of water at the head of the machine, and is dropped as the belt passes, in a reversed position, through the water tank.

The side shake communicated to the belt is of the utmost advantage in more ways than by the settling of the mineral from the sand; for by keeping all the material in motion the belt can be set at a slighter angle, a smaller quantity of water used, and a much greater quantity of material operated on, than would be the case if a simple belt, without lateral movement, were employed. The sand does not pack and cause the water to cut channels and to run off in small streams, but is always uniformly distributed over the whole width of the belt.

As regards the ores on which the machine will work, the only point of importance is, that there be a fair difference between the specific gravity of the mineral to be saved and that of the waste matter with it. The following minerals have been worked upon with excellent results: iron and copper pyrites, arsenical iron pyrites, zinc blende, galena, tinstone, cinnabar, native silver, carbonates of lead and copper and native copper, tellurides of gold and silver, and tailings from the amalgamating mills. "Floured" quicksilver and slimes flowing from the settling tanks have also been experimented on, and made to yield the impalpable mineral which they contain.

For running a single machine it is estimated that only one-quarter horse-power is required, and one man can attend to sixteen machines without difficulty. When six machines are used the cost of treated sands, when ready to flow on to the machine, is estimated to be about tenpence per ton.

The revolution of the belt is the agency by which the delivery of the clean material is effected. The necessity for a proper travel will be apparent, inasmuch as, supposing the belt to remain stationary, no delivery of mineral could possibly take place; while, if a quick travel be communicated, everything which fell on the belt from the sand distributor, No. 1, would be rushed past the clear water at No. 2, and collected in the tank. Between these extremes we find the desired mean: a speed which shall be sufficiently great to deliver continuously all the mineral collected by the belt, yet not so fast as to require a flood of water at No. 2 to keep back the sands. If the ore treated be poor in mineral, the upward motion of the belt should not exceed twenty inches per minute; if rich, the speed is increased accordingly, and in agreement with the inclination of the belt, being greater as the inclination increases; but usually the speed should not exceed 3½ ft. per minute.

To examine the influence of the side shake two extreme cases may be cited. In the absence of side movement, with the ordinary supply of material coming on to the belt, no separation can be effected by a reasonable stream of water at No. 2; the greater part of the rock passes over into a tank with the mineral it packs upon the belt. On the other hand, to drive the crank-shaft, H, at a rapid rate, and thus to violently agitate the belt and its load, has the effect of working everything off the foot of the table. In this matter, as with the revolution of the belt, there is clearly a desirable mean; namely, a speed at which the material on the belt is kept in gentle motion—lightly suspended in the water, and thus easily carried by it down the belt—a speed which allows and facilitates the settlement of the mineral from the rock, and so does not disturb it when once settled on the belt. The customary rate of driving this side motion varies from one hundred and eighty to two hundred revolutions of the shaft per minute—the former speed for fine, light slimes, and the latter for rough and heavy sands.

As regards the regulation of the water delivered at No. 1, the ground between No. 1 and No. 2 should be kept nicely covered with water, and the mineral brought through by regulating the uphill travel. To make the final separation of mineral from sand some little judgment is necessary. As already stated, the delivery holes in the water launder are 1½ in. apart across the whole width of the belt, and the clear mineral creeps up between these small jets of water, so that, as delivered over the head of the belt at A, the form is that of longitudinal streaks, farther or nearer apart, and of greater or less width according as the richness of the material treated is different. The primary object in the adjustment of the uphill travel is that the clean mineral shall be allowed to pass over into the tank at the same rate as it is fed on to the belt in the mixture which is to be separated.

For example, suppose that every hour 800 lbs. of mixed rock and mineral pass on to the belt, and that the mixture contains 5 per cent. of heavy mineral, say galena. Now, disregarding the small loss of mineral in the proper waste or tailings, the uphill travel must be so regulated that there shall be a steady delivery of mineral at the rate of 40 lbs. per hour. No more than this can possibly be delivered unless rocky impurities are allowed to pass and to be weighed in; and if less than this passes, there must be a continual accumulation of mineral on the belt, which will eventually produce loss in the waste.

The proper adjustment of the uphill travel may at the first appear difficult, but in reality it is very simple, in which the eye furnishes a sure guide. The gauge by which the adjustment is rendered easy is the extent of head of mineral showing at the point No. 2, where the water strikes the belt. Again, the weight of mineral as it gets strong and heavy forces it more past the water. Should the discharge of mineral exceed the quantity falling on the belt, sand or rock will be found close up to the jets of water, and by-and-by passing them in place of mineral. If the uphill travel be too slow, the mineral collects below No. 2, forming a great head extending towards No. 1, and even below, in which latter case an increased loss of mineral will assuredly take place in the waste.

When working properly, a small head is always kept below the jets of clear water, and the mineral comes over clean and regularly. A few hours' experience will instruct any one sufficiently on this fact; and having once adjusted the uphill travel, the machine will work continuously and uniformly as long as the conditions are kept constant; nothing more than this can be expected of any machine.

The machine should work almost noiselessly; if there be any jar or knock, the cause must be found and a remedy applied. These jars can be easily remedied, and are not faults in the machine, but of its setting up or adjustment. If the side shake be found to work smoothly and without a jar, the uphill travel or progressive motion can be given. The machine is now at work, and some clear water run on it from No. 2 will show whether the belt is level across; if not, it is easily levelled by the wedges at the foot before the ore feed is started.

Supposing all instructions to be followed, the machine will be working now regularly and smoothly. The belt, moving always onwards, brings all mineral up to the clear water at No. 2; and here the difference between rock and mineral becomes apparent: the clean mineral passes between the jets of water, and is deposited in the tank below; the sand works gradually down, to be replaced by other particles.

In treating slimes, as indeed with all other qualities of material, as little water as practicable should be fed on with it; a large volume of water on an inclined plane surface implies speed and force—two undesirable elements in the separation of fine mineral. From an extended experience with the machine it has been found that with a slightly increased speed of the side motion, any rough particles of rock are much more easily moved than fine mineral; that it is easy to work the coarse sand off the belt, and at the same time produce extremely slight loss even of the very finest mineral. This observation led to the working of mixtures of sizes which should properly, on the usually accepted theory, have been classified and treated separately. It is preferable to use a screen of forty holes to the lineal inch. The side motion of the belt works off the rock and never moves the very finest mineral when it has once touched the belt. Therefore the pulp flowing on the belt must not be too thick, as the particles of mineral cannot settle through it. For this reason a pretty fair current of water must be allowed to go on with the slimes, and the belt placed with very slight inclination, so that the current be not too rapid.

**Concentration of the Sulphurets.**—The success of concentration is dependent on the efficiency of the machinery employed to effect a separation of the worthless gangue from the valuable portions of the ores, and it is very difficult to lay down any rule for the guide of the mine owner in the choice of machinery for the purpose. Almost every mine having its own distinct character in regard to ores, no one machine can be adapted to the varying physical characteristics of the numerous classes of ores. But as a rule, in most ores, the gangue is harder than the mineralized portion, and the result in stamping is that the valuable ore particles are generally reduced much finer than is consistent with economical dressing. Moreover, even pulverization—a condition which is essential to perfect concentration—is difficult to obtain in practice. If ores are of a complex nature, and the precious metal is contained in minerals of varying densities and different degrees of hardness, an even pulverization becomes still more difficult.

The best method, therefore, is to effect an equal assorting or sizing of the uneven grains resulting from the stamping process before proceeding to concentration.

**Results of Concentration.**—The introduction of a proper system of concentration has in many cases proved the means of effecting large savings of the precious metal in the gold-bearing rocks, as the value of the gold carried in the pyrites often equals, if it does not exceed, the quantity present in the free state which is saved by amalgamation.

But where the gold is very fine, and the pyrites are of a brittle character, the percentage saved will be less than in the treatment of more favourable ores.

Where ores carry a large percentage of pyrites the tailings are usually richer, owing to the sliming of pyrites.

Since the introduction of good concentrating machines, the percentage of gold extracted from the ores has been raised to 75 and 85 per cent. of their original assay value, and some mills claim returns of 90 per cent. and over.

**Colonel Taylor on Concentration.**—A practical illustration of the difficulties met with in concentration has been given by an experienced Californian, Colonel J. M. Taylor, who gives the results of his observations and experiments as follows:—

"I commenced this business believing, as many other theorists have done, that all gold ores could be concentrated on coming from the battery by machinery without handling, and that gold sulphurets could be treated successfully only by the chlorine process. I continued in this belief for many years, spending time and money, and accomplishing nothing. I purchased a mine, for experimental purposes, which had the reputation of producing ore of a very refractory character. This ore assayed $30 per ton, but not more than one fourth of it was sufficiently free to admit of its being amalgamated in the battery.

"I erected a five-stamp mill, and tried various methods for concentrating the sulphurets. The best result obtained was 25 per cent., which, together with the free gold, formed only 50 per cent. of the assay value of the ore. At this juncture I abandoned everything with the word patent on it, and, going back to first principles, constructed an old-fashioned Cornish buddle, and sized the ore in two sizes, using two pointed boxes, after the plan adopted by the most improved mills in Grass Valley; all the materials held in suspension by the water were allowed to pass over the second box and go to waste. I found by concentrating the two sizes separately in the buddle that I could get about 8 per cent. more than when they were concentrated together. In this way 10 per cent. more was saved than by any other plan yet tried.

"The tailings as they came from the buddle were assayed, and found to contain 10 per cent., leaving 30 per cent. unaccounted for. A tank was then constructed 12 by 12 ft., with a partition in the centre, and the slum that ran over the second box was allowed to pass into the one and out of the other, giving it plenty of time to settle. In this way one fifth of all the ore crushed was settled in the tanks, the contents of which assayed 23 per cent., being at the rate of about $6 per ton of ore, making a saving of an assay of $48 per day with an eight-ton mill. Deducting from this 10 per cent. for loss in concentrating, 25 per cent. for working, and $1 per ton for cost of concentrating, resulted in a net profit of $25 per day, or $720 per month, to the mill. A barrel holding 60 gallons was placed under the stream of water from the tank, and when full was left twenty-four hours to settle, a little alum having been added. The top was then carefully poured off, when the sediment was found to contain about 1 per cent. of the ore, which was held in suspension by the water after it had become comparatively clear; 8 per cent. could not be accounted for. It could easily have been wasted in the battery, or more than an average might have been got in sampling the mine. This latter was hardly possible, however, as great care was taken to insure a fair sampling by drilling through the ledge in various places.

"This result did not surprise me in the least, having long been aware that a large percentage from most mills had been lost in this way. The question was how to concentrate these tailings up to a higher grade, they not being rich enough to pay for chlorinating. Various methods for accomplishing this were tried. The best result from the round buddle, using an ordinary broom for sweeping, was 50 per cent. A buddle was then constructed on a larger scale, and with much less grade than the one already in use. A piece of common mill blanket was put on the arm for sweeping, and a small stream of water turned on. This proved a success as shown by assay, twelve tons having been reduced to one ton, at a cost of 75 cents per ton. There was still a loss of ten per cent. Various tests were made in order to determine what grade of sulphuret ore would pay to concentrate. Some twenty tons of coarse tailings had accumulated from the buddle, which after testing proved to be worth $2.25 per ton. This was reduced, at a cost of $7½, to 500 lbs., which had an assay value of $30. Deducting from this 25 per cent. for loss in working, and $7½ for labour of concentrating, left a net profit of $14.

"I now became satisfied that no machine yet invented can concentrate the majority of ores to more than fifty per cent. of their assay value without their having first been sized and settled in tanks. Assuming that ore requires settling before it can be concentrated up to a high percentage, it is only a waste of time and money to attempt its concentration before settling, as the cost is the same whether it be high or low grade."

As a consequence, any machine that fails to take out more than half the value of the ore is of no practical use. All ores must be sized in three different sizes before they can be concentrated. Lead sulphurets are nine and one half times heavier than water, and five times heavier than quartz. Common iron or copper sulphurets are seven times heavier than water, and three times heavier than quartz. The coarse pulp and sulphurets capable of passing through an ordinary No. 6 mill screen are, perhaps, on an average, fifty times coarser than those found in the slum ores. They should, therefore, be concentrated separately; otherwise a current of water sufficient to carry off the coarse pulp will also carry off the sulphurets, notwithstanding the latter are from three to five times heavier than the pulp, which, being composed of quartz about fifty times more bulky than the sulphurets, exposes a corresponding surface to the action of the water.

"To obviate this trouble, we must equalise the tailings, bringing the sulphurets and the pulp to the same size, then equalise the water to correspond with the fineness of the pulp, and a current that will carry off the latter will leave the heavier sulphurets behind. No man experienced in milling will ever spend a dollar trying to concentrate in violation of these rules or natural laws. Any of the quick-motioned concentrators now in use will separate a large percentage of the coarser sulphurets from the coarse sand, but at the same time they will hold the fine sulphurets, which are of the most value, in suspension so long as the water continues in motion. The round convex buddle is the best equaliser in use, it being fed around the centre-post, which is about twelve inches in diameter. As the water recedes from the centre it spreads, and consequently decreases in force. If the current of water be strong enough to start the finest sulphurets from the head of the buddle, it will become so diminished before reaching half the distance from the centre to the circumference that the sulphurets are left behind. The concave or centre-discharging buddle is fed on the outer rim and discharged in the centre: consequently the water increases in force toward the centre; hence fine sulphurets leaving the circumference of the buddle will be carried toward the centre with the pulp by the increasing force of water.

"In the tin, lead, and copper mines of England, where concentration has been carried to a higher state of perfection than anywhere else, they have long since discarded the centre-discharging buddle, and use only the convex. Many mining superintendents contend that their ores are not rich enough to justify handling and concentrating in round buddles, but I am of opinion that all ores below permanent water level will pay to concentrate if they will pay to work at all.

"After the concentration was perfected, two hundred tons of ore were run through the mill, and further concentrated to fifteen tons, which were shown by assay to contain ninety-one per cent. of the gold found in the ore after being settled in the tanks and before it was concentrated. This second concentration costs 63½ cents per ton."

# CHAPTER VI.

## *THE METALLURGICAL TREATMENT OF REBELLIOUS GOLD AND AURIFEROUS SILVER ORES.*

WHAT IS UNDERSTOOD BY A REBELLIOUS OR REFRACTORY ORE—The Roasting of Ores—Oxidizing Roast—Dead Roast or Sweet Roast—Reverberatory Furnaces—Chemical Reactions during an Oxidizing Roast—Rebellious Ores and their Comportment when Roasted—Iron Sulphides—Copper Sulphides—Lead Sulphide—Sulphuret of Bismuth—Zinc Blende or Sulphuret of Zinc—Molybdic Sulphide—Silver Glance—Arsenical Pyrites—Sulphide of Antimony—Roasting Reactions in Reverberatory Furnaces—Influence of Gangue Rocks—Loss of Gold by Volatilization in Chloridizing Roasting—Loss of Gold at Las Minas—Professor Christy's Investigations—Proposed Condensation of Fumes by Electricity—Loss of Gold in Muffle Roasting—Mr. Wilson's Experiments—Mr. Falkenau's Experiments—Reduction of Loss of Gold in Roasting to a Minimum.

**What is understood by a Rebellious or Refractory Ore.**—In nature we but seldom find the ores of the precious metals in a condition suitable to be at once submitted to ordinary metallurgical operations, whereby the separation of the metal from the gangue is effected by amalgamation. Only those ores, in which the gold or silver is in a metallic state, or (in the case of silver) in combination with other minerals—which by direct action of certain chemicals may be decomposed, so as to set the silver free to be taken up by mercury—can be treated by the ordinary processes of amalgamation. In these cases practical experience will be sufficient to enable the operator to conduct his treatment successfully, and even with the crudest of methods, such as the Mexican Patio process, excellent results may be obtained. When, however, gold and silver are found associated with minerals "inimical" to mercury, the trouble begins; and often the best metallurgical skill is baffled in dealing with ores of this class, which are very appropriately called "rebellious" or "refractory."

Gold is generally found associated with iron pyrites or arsenical pyrites; but the actual condition in which the gold exists in these ores is as yet a matter of conjecture. Ores of this kind are usually treated by stamping in the battery, getting the free gold (or as much of it as possible) by copper-plate or other amalgamating processes, and then passing the pulverized stuff or pulp over concentrators to collect the pyrites, which are then submitted to separate chemical treatment.

Besides these pyrites the ores may contain galena or sulphide of lead, zinc blende, some copper combinations, or bismuth. Or the gold may exist as a telluride. Then, again, gold ores may be associated with true silver ores, such as ruby silver, brittle silver ores, and other ores containing with silver various proportions of antimony, arsenic, copper, &c., all of which will require special manipulation. Or we may have to deal with silver ores of the like description not containing gold, or containing it in very small quantities.

When face to face with a difficult problem, be it in dealing with gold or silver, the metallurgist has to consider, first, if it is advisable to treat the ore as it comes *en masse* from the mine, gangue and ore together; or will it be more profitable to concentrate first and treat concentrations afterwards? No general rule can be laid down, inasmuch as local circumstances and financial and other economic conditions will have to be considered.

Large bodies of gold-bearing iron pyrites exist in various parts of the world ("the mountains are full of them," one is told by Australian miners, and with depth the Transvaal mines also begin to show large bodies of pyritic ores); but often they are not rich enough to bear the heavy expense of metallurgical treatment by methods now in vogue. A cheap mode of manipulation, therefore, requiring a minimum of labour, fuel, and chemicals, is an essential condition for the utilisation of large masses of ore, which, up to the present time, have been beyond the reach of ordinary metallurgical skill.

To effectually deal with such ores we require above all a good and cheap system of roasting, and when the ore is once freed of its obnoxious substances, and the precious metal put into a condition in which it can be acted upon by quicksilver or chemicals, half the battle will be won. The question of roasting—and cheap roasting, too—will first have to be solved.

When gold or silver exists in its pure metallic condition in the ore, nothing will oppose its separation from the gangue when we bring the pulverized mineral into contact with quicksilver, and submit it to a process where sufficient contact between the precious metal and quicksilver is insured through continuous agitation, aided by some friction and heat. It is a simple process, where the affinity of mercury for certain metals is utilised, to form amalgams. These once formed, all that is required is to separate them by condensation, as mercury, being volatile, will, on heating, leave the gold and silver behind.

In spite of the great advances made in metallurgy, the fact remains that when we deal with gold ores—that is, free gold ores—it takes very little to upset the affinity between gold and mercury; and once this equilibrium is disturbed, instead of our saving the gold, it will be repulsed by the quicksilver, and carried out at the tail end of the mill by the waste or tailings to be discharged into running streams, where it is quickly lost. In my own experience, I have known cases where more gold was lost in the tailings than was saved in the mill, and although the ores were not "rebellious," they have as yet baffled the efforts of metallurgists to secure better results.

Considering that gold, although distributed in large quantities over the whole globe, occurs only sparingly in concentrated masses which will pay for extraction, so that continents have to be traversed to find remunerative workings, no effort should be spared, wherever it is found, to extract it as closely as possible.

The causes which work such havoc in the metallurgical treatment of precious metal ores are various, arising sometimes from the chemical combinations in which the gold ores occur, or from the aggregation of the ore and gangue particles. In many mines ores are found in which the gold is perfectly pure and free, having but a small proportion of base metals associated with it; and yet our present methods will fail to extract even half its value. Such ores, when pulverized, show that they contain, beside silica, various decomposed aluminous rocks, iron oxides, or manganese oxides, and these, when disintegrated, exert a deleterious mechanical influence on the amalgamation, and carry the gold over the plates into the tailings.

In by far the majority of workings, so soon as a certain depth in the mine is reached, we come into the zone of the rebellious ores—ores which contain sulphur, antimony, or arsenic in combination with other metals—and which, in the upper portion of the mine, were not found. Nearer the surface atmospheric agencies, were able to reach the ores, oxidising the baser elements, and leaving the gold in a free state.

**The Roasting of Ores.**—Before ore that is rebellious can be submitted to any treatment whereby the extraction of metal is effected it has to undergo a process of roasting. If it is roasted *after* its mechanical preparation, namely, crushing and concentrating—the operation of roasting is performed in furnaces; but if the ore is submitted to this process in the mass as it comes from the mines, the operation is carried out in heaps, stalls, and kilns.

The object to be obtained in roasting is either—(1) to convert most of the metal in the ore into oxides, or *oxidizing* roasting; (2) to convert the metals into sulphates; or (3) to convert the metals into chlorides, or *chloridizing* roasting. Each of these roasts prepares the ore for a special treatment.

**Oxidizing Roasting.**—The object of the roasting is to drive off the sulphur, arsenic, and other volatile substances with which the metals are combined — in other words, to oxidize these metals, or to produce metallic oxides in combination with acids. For this purpose the ore is exposed to heat in furnaces under access of atmospheric air. The sulphur in the ore takes fire and burns, and an oxidation of both the burning sulphur and metals is going on. The same takes place with arsenic, which is converted into arsenious acid, while the sulphur is converted into sulphurous acid, some of which combines with a portion of the oxidized metal, forming metal sulphate. Some metals which already existed in the state of an oxide in the ore may be converted into a higher oxide under the influence of heat and access of air.

If we have a complex ore, composed, say, of iron pyrites, zinc blende, copper sulphide, galena and arsenical pyrites, and these are heated in a reverberatory furnace under access of air, a chemical change of all the constituents will take place under production of iron and copper oxide, and iron and copper sulphate. Zinc oxide will partly remain and partly volatilize. A subsulphide and sulphate of lead will also be formed, while some of the arsenic will volatilize, and some combine in an oxidized state with some of the metals, to form arseniates. Should any antimony be in the ore a similar reaction will take place under formation of antimonates, showing that arsenic and antimony play during roasting the same rôle as sulphur. Now if any gold and silver be present, the gold remains in a metallic state, while the silver is partly converted into sulphate of silver, and partly remains as metallic silver. The silver also enters into combination during roasting with arsenious or antimonious acid, resulting in the production of arseniate and antimonate of silver.

Such a roast is not a complete one, except when it is desired to roast for sulphates, which, under some conditions, is desirable; the reaction, therefore, has to be carried further to arrive at a *dead* roast, or, as the Australians call it, a *sweet* roast.

**Dead or Sweet Roast.**—By this is understood the conversion of all remaining sulphides, subsulphides, and sulphates into oxides. To effect this the heat is increased, so as to decompose the sulphates, which results in driving off all the sulphuric acid as an oxide of sulphur, and metal oxides remain behind, with the exception of those which are volatile at an increased heat. If lead sulphate is in the ore in contact with some undecomposed sulphides or subsulphides, some oxygen from the sulphuric acid will go to the undecomposed sulphur of the sulphide, and it will go up the chimney as sulphurous acid; but otherwise, in the absence of sulphides, the sulphate will be inclined to bake and cause a sintering of the ore, which is very detrimental to the subsequent operations. Great care, therefore, is required in the last roasting stage, and as the silver sulphide requires a high heat for its decomposition, it is not usual to push the heat far enough to accomplish this in the presence of lead. As silver sulphate is soluble it can be recovered in the subsequent manipulations, and it is essential to carry the dead roast far enough only to decompose the iron and copper sulphates. From what has been said, it is always an advantage to operate during the first roasting stage at a *low* heat, and to increase the heat after the formation of the sulphates.

When dealing with ores containing easily fusible compounds it is not advisable to bring the ore to a high heat to effect the decomposition of the sulphates, arseniates, and antimoniates, as the ore will sinter (as stated above), and to effect their decomposition, so as to reduce the metals to oxides, an addition of carbon is made to the ore, in the shape of pulverized coke, coal, or other substances containing hydrocarbons. By shutting off the access of air, a reduction of the sulphuric into sulphurous acid is effected, which goes up the chimney; the arsenious acid is converted into arsenious suboxide, which volatilizes, while some metallic arsenic remains behind; and the reaction is accompanied by a copious evolution of carbonic oxide and carbonic acid, leaving the metals in the ores behind on the furnace hearth in the state of oxides or suboxides, which latter on the access of air and some increase of heat are reconverted into higher oxides through the absorption of oxygen from the freshly-admitted air.

The above conditions can be fulfilled by the employment of variously constructed furnaces. Those which are mostly used are the reverberatory furnaces, which are fully described in the following chapters. Here, however, I would draw attention to certain conditions to which these furnaces should respond.

The hearth surface must proportionately correspond to the fireplace surface, and the flues also should allow of the required outlet for the roasting products and fumes, and communicate with a sufficiently high chimney. The working doors should be in proportion to the hearth surface, and the bridge should be sufficiently high and wide.

During the roasting process the flame enters the surface accompanied by undecomposed atmospheric air, which is absorbed when the furnace has reached a certain degree of heat by the glowing particles of carbon and the burning gases, and the ore which is spread out on the hearth is brought to a high heat through reverberation of the flame passing over it in undulating or wave-like flowing streams. If the working doors and draughts are opened, cold air enters, and forms a layer between the ore and flame, furnishing oxygen to the non-consumed carbon of the flame and also to the incandescent volatile products of the ore, producing in the former case carbonic acid and hydrogen, resulting in increase of temperature.

The furnace should be so constructed that the vault is not too far from the hearth or too high, but of sufficient height to give oxygen to the burning ore, thus aiding their volatilization and removal. If the vault of the furnace is too high, or if the flues are too large in proportion to the other dimensions of the furnace, the process will not proceed with regularity, as too much air will enter and act coolingly on the ore, increasing the consumption of fuel and prolonging the roasting. If the vault is too low, the flame is brought too near to the ore, and the atmospheric air has no chance to give up a proper quantity of oxygen to the burning mass.

During the roasting the position of ore layers must be changed occasionally, and the surface renewed from time to time, as otherwise ores containing easily-fusible combinations would bake together like sulphide of antimony, sulphide of copper, galena, &c. To effect this the ore is stirred by rakes, whereby the surface of the ore—which is the hottest by being in direct contact with the flame—is mixed with the cooler ore lying underneath, and then the ore near the fire bridge, which is the hottest part of the furnace, is moved to the hind portion of the furnace near the flue, which is the coolest, thereby exposing the whole ore mass in rotation to an even heating, and exposing it all to the same temperature.

It has been found that stirring by hand-power is a costly operation, especially in countries where labour is dear, and this has occasioned the introduction of various furnaces in which automatic machine action is substituted for hand labour.

**Chemical Reactions during Oxidizing Roasting.** The base metal ores which are found in association with gold rocks are very various, and comprise (amongst others) iron sulphides, copper sulphides, lead sulphide, sulphuret of bismuth, sulphuret of zinc, molybdic sulphide, silver glance, arsenical pyrites, and sulphide of antimony. Of these, as the baser elements with which the metallurgist most usually has to deal in treating auriferous ores, some account may here be given.

(1) *Iron Sulphides.*—Several compounds of iron and sulphur are known, the two most important being the following:—

(*a*) *The Bisulphide of Iron* or *Iron Pyrites*, $FeS_2$, is a natural product, occurring in rocks of all ages, and evidently formed in many cases by the gradual deoxidation of ferrous sulphate by organic matter. It has a brass-yellow colour, is very hard, is not attracted by the magnet, and is not acted upon by dilute acids. When exposed to heat, sulphur is expelled, and an intermediate sulphide, $Fe_3S_4$, is produced.

(*b*) *Magnetic Pyrites*, $Fe_3S_4$, has a colour between bronze-yellow and copper-red. It is slightly attracted by the magnet. Its inferior hardness and shade of colour, as well as its magnetic quality, distinguishes it from the common iron pyrites.

As iron pyrites are generally found associated with gold, this substance is of the greatest interest to the metallurgist, and its treatment has been made the subject of special researches, which have led to the discovery of the so-called Plattner Chlorination Process.

When iron pyrites are roasted in a finely pulverized condition in a reverberatory furnace under access of air, the sulphur will commence to burn, and will furnish heat enough to keep the whole mass in an incandescent state, the burning sulphur being fed by the oxygen of the air, and converted into sulphurous acid. After a certain period it will be noticed that a swelling of the ore mass is taking place until all the sulphur has burned away, and after a little cooling, the ore appears blackish, but when cooled completely it turns red. This reddish substance is iron oxide.

As iron pyrites contain two atoms of sulphur to one of iron, the process of roasting consists in the burning away of the first atom at the beginning of the operation, and when this is accomplished the second stage of the operation commences. Plattner has shown that, during the burning of the second atom of sulphur, a certain amount of sulphuric acid beside the sulphurous acid is generated, which formation he clearly explains as the result of a secondary reaction brought about through the agency of the heated quartz and other neutral materials, which cause the sulphurous acid and the oxygen of the air to combine.

The presence of the sulphuric acid during the second period of the roasting reaction plays an important part, as the very iron which yielded the sulphur turns into an oxide, and this oxide combines with the sulphuric acid to form a sulphate, which easily decomposes in the heat of the furnace into sulphurous acid and oxygen; the latter reacts on undecomposed sulphides, converting them into sulphates, and the ferrous oxide of the salt will be converted into a ferric oxide, which is the reddish substance heretofore mentioned.

(2) *Copper Sulphides.*—There are two well-defined copper sulphides, analogous in composition to the oxides, namely, (*a*) *Cupric Sulphide*, $CuS$, which occurs native as indigo copper; and (*b*) *Cuprous Sulphide*, $Cu_2S$, which occurs native as copper-glance. It is a powerful sulphur base, uniting with the sulphides of antimony, arsenic, and bismuth, to form several natural minerals.

The several varieties of fahl ores, as tetrahedrites, consist of cuprous sulph-antimonites or cuprous sulph-arsenites, in which the copper is more or less replaced by equivalent quantities of iron, zinc, silver, and mercury. This series of ores forms by far the most important group of rebellious silver ores which we have to contend against. (*c*) *Copper Pyrites* is a cuproso-ferric sulphide, $Cu_2S$, $Fe_2S_3$. (*d*) *Purple Copper* is also a species of copper and iron sulphide, containing various proportions of the two metals.

When copper pyrites are roasted there will be formed during the first period of the roasting sulphate of iron and sulphate of copper. After continued roasting a red powder is produced, consisting of ferrous and ferric oxide, cuprous and cupric oxide, and the sulpho-salts of these oxides; but if the heat is sufficiently raised again, the sulphuric acid can be driven off and nothing but oxides will remain.

(3) *Galena*, PbS.—Although precious metal ores may occasionally occur in association of oxidized lead ores, by far the most abundant ore is the lead sulphide, commonly called galena. When crushed lead ore is roasted at a dull red heat, the sulphide becomes changed by oxidation to a sulphate. A further heating will partially decompose the sulphate with evolution of sulphurous acid and production of lead oxide in admixture with lead sulphate; but on account of the ready fusibility of lead compounds the heat must be properly regulated, or otherwise a sintering of the ore will take place.

The discovery of Mr. Russell that lead can be completely separated from a sodium hyposulphite solution, as lead carbonate, by sodium carbonate or purified soda ash, without precipitating any copper or silver, has proved a great advance in the hydro-metallurgy of precious metals, as it permits the elimination without difficulty of this obnoxious substance when treating ores in which it appears. His process refers, of course, to the separation of chloride of lead when the ore has been roasted with salt.*

---

* See my "Metallurgy of Silver,"—*art.* the Russell Process, pp. 281—294.

(4) *Sulphuret of Bismuth*, $Bi^2S^3$, is generally found in massive and in acicular crystals of a lead-grey colour. It is easily fusible in the flame of a candle. When heated in a roasting furnace sulphurous acid is developed, and it is converted into a grey powder consisting of bismutic sulphate and bismutic oxide. It is an exceedingly rare metal and only occurs in very small proportions when associated with precious metal ores.

I have made the observation that, when bismuth is found associated with auriferous pyrites, they will assay very high in gold. The first lot of bismuthic pyrites I worked from the Pioneer Mine, Idaho, contained 6,000 dollars per ton.

(5) *Zinc blende, or Sulphuret of Zinc*, ZnS, occurs native as *blende*, in regular tetrahedrons, dodecahedrons, and other monometric forms, and of various colours, from white or yellow to brown or black, according to its degree of purity. It is a valuable ore of zinc, and miners call it "Black Jack." Ores containing zinc blende are generally feared by metallurgists, and they give trouble in roasting and in the other manipulations. Such ores must be roasted at a high heat and with plenty of access of air, so that oxide and sulphate of zinc may be produced.

When zinc ores are roasted at a high temperature, with access of plenty of air, the sulphurous acid is rapidly evolved without a comparative formation of sulphuric acid, and less zinc sulphate and more zinc oxide are therefore produced, and by continued firing the zinc sulphate is converted into zinc oxide. Silver ores containing zinc blende should be always roasted at a low heat, so as to avoid loss of the precious metal by volatilization. This rule is rather contrary to what has just been stated, and it therefore will be apparent that ores containing zinc in association with precious metals will always give trouble to the operator, as high heat with zinc means volatilization of precious metal.

(6) *Silver Glance*, *Vitreous Silver*, or *Sulphuret* or *Sulphide of Silver*, $Ag_2S$.—This is a soft, grey, and somewhat malleable substance, found native in the crystallized state, and easily pro-

duced by melting together its constituents, or by precipitating a solution of silver with hydrogen sulphide. It is a strong sulphur base, and combines with the sulphides of antimony and arsenic. Examples of such compounds are found in the beautiful minerals known as dark and light red silver ores.

When sulphide of silver is submitted to an oxidizing roast, it is converted into metallic silver with evolution of sulphurous acid; but in the presence of other metallic sulphides, which possess the property of being converted into sulphates, and which decompose at an elevated temperature only in such a manner as to evolve undecomposing sulphuric acid, like copper and zinc sulphates, the metallic silver is converted again into a silver sulphate. If we have sulphide of silver only in admixture with iron pyrites, this conversion into a silver sulphate is not likely to take place, as the affinity of the evolved sulphuric acid is greater for the iron sulphides, and on these it will exert its oxidizing action.

On this reaction is based the extraction of the metallic sulphates by means of hot water, as in cupriferous silver ores both silver and copper can be converted into soluble sulphates, only the heat during roasting must not be raised to a temperature high enough to decompose the silver sulphate. It takes a higher degree of temperature to decompose the silver sulphate than the copper sulphate.

From what has been said above it will be seen that metallic silver is produced during the roasting of its sulphide and not a silver oxide, as silver has not a great affinity for oxygen, and if silver oxide is formed during the roasting operation it will be found to possess the property of giving up its oxygen at an elevated temperature, and will turn into metallic silver again.

When *light red* silver ores—a combination of silver sulphide with sulphide of arsenic—are roasted, arsenious acid, arsenious oxide, and sulphurous acid are evolved with formation of some metallic silver and arseniate of silver. When *dark red* silver ores, or *ruby silver*—a combination of silver sulphide with sulphide of antimony—are roasted, sulphurous and antimonious acid is evolved, and some metallic silver, antimoniate and sulphate of silver, are formed.

Plattner, in his "Metallurgische Röstprozesse" (Freiberg, 1856), recounts a whole series of researches which he made in order to determine the losses of gold and silver during an oxidizing roast. His muffle tests show that a loss of silver in oxidizing roasting is unavoidable. He made some fifty tests, extending over three quarter to one and a half hours, and he experienced losses ranging from 0·5 to 18 per cent. His conclusions from these results are: (1) that the loss of silver increases (*a*) with the temperature, (*b*) with the looseness or porosity of the roasting charge, (*c*) with the facility with which the air can come into contact with the silver, and (*d*) with the freedom of the silver from combination with other substances; (2) that the loss also increases with the time of roasting. Experimental researches on a large scale have shown that on some ores the loss by volatilization may amount to 20 per cent. The above roasting reactions of the different metals were worked out by Plattner, and later experimenters have only corroborated the correctness of most of his conclusions as to the chemistry of these reactions.

(7) *Arsenical Pyrites*, $FeS_2+FeAs$, is a combination of an arseniuret of iron and sulphuret of iron, and if this substance is heated, even at a dark red heat, sulphide of arsenic will be evolved; but at an increased temperature, an oxidation takes place, with evolution of arsenious and sulphurous acid. After a continued roasting, the iron is converted into an oxide, although traces of an arseniate of iron oxide may remain behind, which are difficult to decompose at a high temperature.

When arsenical pyrites or arseniurets are roasted in the presence of metallic oxides, arseniates are also formed if these metallic oxides in their nascent state are disposed to combine with arsenious or arsenic acid. Such combinations are formed in the presence of nickel and cobalt, copper, and lead oxides. Some arseniates are decomposed during roasting by sulphuric acid and converted into sulphates, if the sulphuric acid is in excess and present in a vapoury condition. In this manner an arseniate of silver can be converted into a sulphate of silver during the roasting reaction, whereby the arsenic acid sublimes and is decomposed into arsenious acid and oxygen.

When silver occurs in combination with arsenical ores, some of it will be converted into arseniate of silver, and during the roasting losses of silver will take place. Frequently silver ores contain arsenical pyrites, and when these ores assay in gold it will be found that this metal, if not present in the native state, is contained in the arsenical pyrites. The number of mines yielding auriferous silver ores, which can be treated by direct amalgamation in pans without roasting, are getting very scarce, and in most cases roasting has to be resorted to before submitting them to metallurgical treatment.

Silver ores proper are generally submitted to a chloridizing roasting, but when they carry gold this practice (as will be shown further on) may result in heavy losses. The greatest care and attention should, then, be observed in roasting, which is the most important manipulation in the treatment of rebellious ores.

When auriferous silver ores are subjected to a chloridizing roasting it will be found that a high percentage of the silver is extracted by amalgamation and that the yield of gold will in many cases be very imperfect, without reckoning any loss which may take place by volatilization, as the tailings will be very rich in gold, and show that amalgamation has not been capable of dealing with the gold after it has passed through the roasting furnace. Researches in this important subject have not as yet been sufficiently advanced to determine in what condition the gold really exists after a chloridizing roasting or to devise remedies which shall make it more amenable to quicksilver.

(8) *Antimony.*—Sb.—This important metal is found chiefly in the state of sulphide. It melts at a temperature just short of redness, and boils and volatilizes at a white heat. It is not oxidized by the air at commom temperatures; when strongly heated, it burns with a white flame, producing oxide, which is often deposited in beautiful crystals.

Antimony forms two oxides, $Sb_2O_3$ and $Sb_2O_5$, the first being a basic and the second an acid oxide. The *trioxide*, or *antimonious oxide*, $Sb_2O_3$, occurs native, though rarely, as *valentinide*, or *white antimony*, in shining white trimetric crystals; also as *senarmontite* in regular octahedrons; it is therefore dimorphous. An impure oxide can be prepared by carefully roasting the powdered sulphide in reverberatory furnaces, and raising the heat at the end of the process, so as to fuse the product: it is then known as *glass of antimony* or *vitrum-antimonii*. Antimonious oxide likewise acts as a feeble acid, forming salts, called *antimonites*, which, however, are very unstable.

The *antimonic oxide* or *pentoxide*, $Sb_2O_5$, is formed as an insoluble hydrate when strong nitric acid is made to act upon metallic antimony.

When ores containing antimony (generally in the state of a sulphide) are roasted, antimonious oxide is evolved, which is oxidized to antimonic oxide in the presence of air, but in some cases a mixture of antimonious and antimonic oxide is produced. In the presence of metallic oxides, the antimonious oxide during the roasting reaction is converted into antimonic oxide and forms antimoniate with these metals. When sulphates are present during the roasting reactions, these are often converted into antimoniates, the sulphuric acid furnishing oxygen to the antimonious oxide, and the antimonic oxide produced replacing the sulphuric acid. Even silver, when roasted in combination with antimonial ores, and the varieties of silver ores carrying antimony are numerous, will produce an antimonic oxide of silver. From these reactions it will be seen that antimony is not a desirable metal to deal with, as the antimonic oxide of silver is not soluble in water like the sulphate of silver, in case it is desired to extract this metal by simple solution.

**Roasting Reactions in Reverberatory Furnaces.**—Mr. H. M. Howe, in a valuable paper on "Copper Smelting,"* explains the roasting reactions in a reverberatory furnace as follows :—

" In roasting in reverberatories, during the intervals between the rabblings, and while the ore is lying still on the hearth, we may distinguish three horizontal zones: First, there is the upper surface, directly exposed to the atmosphere of the furnace whose oxidizing action is slightly weakened by the presence of sulphurous and sulphuric acids evolved from the underlying layers and of the products of the combustion of the fuel: second, the layers immediately underlying the surface, in which a small portion of free oxygen enters by diffusion, and in which sulphur and sulphurous acid evolved from the undermost layers oppose the oxidizing action of what free oxygen there is present: third, the undermost layers, to which little, if any, free oxygen can penetrate, and where pellet of ore is simply exposed to the action of the other pellets with which it is in contact, of volatilized sulphur and of sulphurous and sulphuric acids (anhydrides) generated by the action of sulphur on previously formed metallic oxides.

" The expulsion of arsenic and antimony as sulphides is favoured in the middle and lower zones by the presence of volatilized sulphur, mixed with sulphurous acid and at most a very limited supply of free oxygen and sulphuric acid. In the upper part of the middle layer, to which a small amount of free oxygen penetrates, we have the gently oxidizing conditions favourable to the formation of arsenious acid and trioxide of antimony. In the upper zone the stronger oxidizing conditions rather favour the formation of fixed arseniates and antimoniates, though even here part of the arsenic and antimony may volatilize and escape while passing through their intermediate volatile condition of arsenious acid and trioxide of antimony.

" Now, when we come to stir the mass up and to transfer particles, whose arsenic and antimony, while in the upper zones, had passed to the fixed acid condition, down into the middle and lower zones, it is probable that we afford these metals a

---

* "United States Geological Survey," Washington, 1885.

fresh opportunity to volatilize by reducing them to the volatile conditions of arsenious acid and trioxide of antimony through contact with volatilized sulphur and with the still unoxidized metallic sulphides of the surrounding ore pellets.

"The low oxides (protoxide of iron and suboxide of copper) of these ore pellets, and the sulphurous acid present in the middle and lower zones, also exert a reducing action on arsenic acid, with the formation of higher oxides of iron and copper and of sulphuric acid. Thus each individual atom of arsenic (and of antimony ?) may travel back and forth many times through the volatile condition, being oxidized to and perhaps past it at the upper surface, and again reduced back to if not past it as it is again drawn down below the surface, see-sawing back and forth to and from the volatile state. And every time that it arrives at this volatile condition an opportunity is offered it to volatilize and escape. This opportunity for reduction from the fixed acid state is, of course, only offered to these noxious metals as long as a considerable amount of unoxidized sulphur and iron or of low oxides of iron or copper remains to effect their deoxidation.

"But, even after the nearly complete oxidation of the sulphur and iron, similar opportunities may be, and indeed often are, offered to the arsenic and the antimony to pass once more from the fixed acid state towards the metallic state and back again, by mixing a small quantity of coke or coal dust with the roasting ore, and then excluding the air as completely as possible so as to strengthen the reducing conditions. The carbon plays a role similar to that already ascribed to the sulphides and low oxides in reducing arsenic and antimony to and perhaps through the volatile state. By prolonging the roasting after the combustion of the coal dust has been completed, we reoxidize any antimony and arsenic which may have been carried back past the volatile state, and we again pass them towards it.

"Of course, the expulsion of arsenic and antimony is favoured by the presence of a large proportion of pyrites, both because the sulphur distilled from the pyrites tends to drag them off as sulphides, and because the presence of the pyrites prolongs the roasting, and thus increases the number of times which the arsenic and antimony pass back and forth past their volatile conditions; hence it is sometimes desirable to mix pyrites with impure ores to further the expulsion of their impurities."

**Influence of the Gangue Rocks.**—Quartzose ores when roasted are not affected by the silicious gangue, except so far as they promote the formation of sulphuric acid, as explained before. Carbonate of lime or calc spar loses its carbonic acid and is converted into a sulphate or gypsum, whereas sulphate of baryta remains unaltered. From this it will be seen that lime is not a desirable component part of gold ores which have to be roasted, as it takes up a large portion of the sulphuric acid.

**Loss of Gold by Volatilization in Chloridizing Roasting.**—That loss of gold can take place to a considerable extent during chloridizing roasting seems to have been known only during the last ten years, as even so eminent an authority as Plattner does not mention the matter in his treatise on "Metallurgische Rostprozesse." He treats there at length of the volatile products formed during chloridizing roasting, but he fails to mention that loss of gold can take place during the operation. The subject was prominently brought before the metallurgical world by Mr. C. H. Aaron, who suffered a heavy money loss on a lot of pyrites, which he had purchased for treatment in his works at Melrose, California. "He was first painfully made aware of the fact of this loss by having to make good a deficiency of £600 below the yield he had guaranteed."

In his work on "Leaching Gold and Silver Ores" (1881), Mr. Aaron states that the ore was simple pyrite with no visible peculiarity. It was roasted in a three-hearth reverberatory, with 1 to 2 per cent. of salt added on account of the silver. He was first led to suspect the cause of the loss from the examination of a yellow sublimate that formed on the masonry of the furnace over the working doors, when the draught was reduced. This sublimate he "found to be very rich in gold, although there was none to be seen in it by the most careful washing. It also contained iron perchloride, and copper chloride, with lead and other substances."

He then instituted tests on the small scale with two similar samples, one with 4 per cent. of salt, the other without, the "roasting being pushed purposely to an extreme as to heat and time, and when the two tests were assayed under exactly similar conditions, that which was salted was found to contain less than half as much gold as the unsalted one."

He further adds: "I then took some light fluffy sublimate from the flue of a roasting furnace, an assay of which gave me a value of some 600 dollars per ton, chiefly gold. The quantity of this material was, however, very small, and the bulk of the matter in the dust chamber was not richer than average bulk of the ore treated, *a circumstance which indicates that the gold was actually to a great extent volatilized in some not easily condensible form.*"

Mr. Aaron further adds: "I also found that the ore sustained a loss of weight in roasting, equal to about 18 per cent., consequently the roasted ore ought to have been more than 18 per cent. richer than before roasting, which was not the case. If this is not considered sufficient proof that the gold may be volatilized in the roasting of some ores with salt, the deficiency is supplied by the fact that, as soon as I made the necessary change by reserving the salt until the nearly dead roasting of the ore was finished, not only did the roasting ore assay 20 per cent. richer than the raw, but the yield overran my guarantee, while the tailings nevertheless contained considerably more gold than before. . . . I afterwards found that a very small quantity of salt, not more than three pounds to the ton, might be mixed with the crude ore without detriment to the gold, and with decided advantage to the extraction of the silver."

**Loss of Gold in Chloridizing Roasting by Volatilization.**—Mr. C. A. Stetefeldt, an eminent American engineer, has investigated the subject of the loss by volatilization of gold during chloridizing roasting, a matter of great importance when treating auriferous silver ores. He found that the loss in gold in some Mexican ores, when roasted in a reverberatory furnace, amounted from 53 to 88 per cent.

He says that there is no doubt that the volatilization of the gold takes place with that of the copper chlorides, in case of coppery ores. The loss increased with the quantity of these chlorides formed and then volatilized. It is therefore essential during roasting to avoid the formation of a large quantity of copper chlorides. Temperature and time of roasting must also influence the result. Against this, however, he quotes an instance where 80 per cent. of the silver and from 68 to 85 per cent. of the gold were lost in chloridizing roasting with ores which did not contain copper.

Plattner speaks of the loss of gold in oxidizing roasting, but entirely neglects chloridizing roasting. G. Küstel records a loss of 8 per cent. of gold in roasting telluride ores with salt, and states that by increasing the temperature and time of roasting the loss in gold may be 20 per cent. and more. Mr. Aaron suffered heavy losses in the chloridizing roasting of gold-bearing sulphurets. In Australia, the chloridizing roasting of gold ores was abandoned on account of heavy losses.

A very large portion of the volatilized metal should be regained with proper condensation, and considerable attention is being paid now to the question of condensing fumes from roasting and smelting furnaces. Experiments on the electrical collection of metallic fumes and dust, which have been made by Mr. A. O. Walker in England, point to a successful solution of the problem.

**Loss of Gold at Las Minas.**—Mr. C. A. Stetefeldt has given the very interesting results of his investigations on the chloridizing roasting of the gold ore from Las Minas, in the State of Vera Cruz, Mexico. The ore he treated consisted mainly of 43 to 67 per cent. of magnetite, 3 to 22 per cent. of pyrite and 3.5 to 7 per cent. of chalcopyrite, the remaining minerals being quartz and garnet. The ore contained less than one ounce per ton of gold and traces only of silver. He found that the losses in a chloridizing roasting were from 42·8 to 93 per cent. of the total gold content. He further states, however, that " the copper chloride is by no means an essential element in producing this loss as shown in the following experiments, made by Mr. C. Butters with a gold ore entirely free from copper." This ore was a hard white quartz intimately mixed with about 7 per cent. of calcite and a little pyrite. It contained 5·55 oz. of silver and 0·65 oz. of gold per ton. On subjecting this ore to an oxidizing roasting a loss of 2 to 9 per cent. of silver took place, but no loss of gold. But when the same ore was subjected for one hour to a chloridizing roasting in the muffle at a cherry-red heat with 5 per cent. of salt, a loss resulted of 70 to 80 per cent. of the silver and 68 to 85 per cent. of the gold. On increasing the salt to 10 per cent. no increase of the losses took place.

Attention should be called to the claim of Mr. Stetefeldt that the loss of gold is less in the instantaneous roasting of the Stetefeldt furnace than in the more prolonged treatment in the reverberatory furnace. This claim does not appear to be unreasonable; and the indirect proof that he cites from the work of his furnace at the Lexington Mill on an auriferous silver ore tends to bear out that claim. This claim, if it could be established beyond doubt, would be of such great importance in the treatment of a large class of auriferous silver ores that must be roasted with salt on account of their silver content that it is to be hoped that he will find it possible to make sufficient direct tests on the large scale with the Stetefeldt furnace to settle the question.

**Proposed Condensation of Fumes by Electricity.**— Mr. Walker proposes to lead the fumes through dust chambers or flues, as is usual with smelting or roasting furnaces. He collects the fine particles of metals suspended in the fumes by means of a discharge of high potential electricity from metal points or edges, or other projections, situate in the flue passage chamber or other receptacle, and so placed that the current of air or gas containing the finely divided material or metallic vapour is carried or passes in close proximity to the discharging points. The discharge from the points electrifies the air or gas, and the charged air or gas then acts on the finely divided matter in it, causing it to cohere, condense, and deposit.

The separation and collection of metalliferous particles from the fume from lead-smelting furnaces is effected as follows. The flues from such furnaces are at present frequently conducted for long distances in a horizontal or inclined direction, in order that the fume may deposit a portion of the metalliferous matter which it contains. The flues may be much shorter than at present, and nevertheless a better deposit will be obtained, and the process may be applied in other receptacles or reservoirs.

At each point where the treatment is to be applied, within the flue or chamber a metal conductor is provided, which may be a rod, or combination of rods, spheres, plates, or any other convenient form of such dimensions as will be most suitable to the size of the passage or receptacle. In preference, two rods in the form of a cross—each arm of which occupies respectively about two-thirds of the height and breadth of the passage—can be used.

The surface of this conductor is studded all over with metal points or other projections—say at distances of two or three inches apart, and projecting (it may be) two or three inches from the surface of the metal conductor. This conductor is very carefully insulated in some suitable manner. It may, for example, be done by supporting it at the end of a horizontal metallic rod passing through the wall of the flue. An earthenware tube may be inserted into the wall of the flue, and the supporting rod passed through the tube, but without being in contact with it. Outside the flue the rod may be carried on glass legs, entering without contact at the neck into glass vessels, in which a small quantity of strong sulphuric acid is contained. This rod is connected with one terminal of any kind of electrical machine capable of developing high tension electricity by induction or by friction, the other terminal being connected to the outside of the flue or chamber or to earth. Or the electricity may be supplied at high potential by any other known means. The whole is so arranged that a continuous electric discharge is maintained from the points which stud the surface of the conductor within the flue or chamber into the air. Other means may be devised to insulate in like manner the sphere or conductor. Several of these arrangements may be applied along the flue at distances of a few yards.

The metalliferous particles will by this treatment be caused to agglomerate, and will be deposited in the portion of the flue beyond and near the place or places where they are submitted to the electric discharge. The deposit may be removed mechanically from the flue from time to time, or it may be removed in part by a stream of water flowing along the lower part of the flue, as is the practice at present in flues for condensing lead fume, or in any other convenient way.

**Professor Christy's Investigations**—In 1882 some investigations which were made by one of Professor Christy's pupils at chlorination works in California revealed the following facts: The ore after roasting in a double hearth reverberatory for thirty-two hours was mixed with 1 per cent. of salt, and four hours later the charge was drawn. This made a total time in the furnace of thirty-six hours. The ore lost 23 per cent. of its weight in the roasting.

The roasted ore having been carefully sampled and assayed, the result was—

| | Gold.<br>Ozs. per ton. | Silver.<br>Ozs. per ton. |
|---|---|---|
| Raw ore (average of 5 scorification assays) . | 4·536 | 19·884 |
| Roasted ore (average of 4 crucible assays) . | 2·980 | 18·583 |
| Hence $\frac{2441}{2000}$ tons raw ore contained . . . | 5·536 | 24·268 |
| And $\frac{1873}{2000}$ tons roasted ore contained . . . | 2·791 | 17·403 |
| Hence the actual net loss in roasting was . | 2·745 | 6·865 troy |
| Or in per cent. of original ore content . . | 49·58 p. c. | 28·28 p. c. |

This test shows in the most unmistakable manner what enormous losses may take place in roasting such ores with salt, unless the most minute details of the treatment are carefully looked to. At these works the upper hearth was usually kept at a red heat a good part of the time, and the temperature of roasting was undoubtedly kept generally too high.

The tailings were also sampled and assayed after they had been leached for gold and silver in the usual way with the following result:

| | Gold. Ozs. per ton. | Silver. Ozs. per ton. |
|---|---|---|
| Average of 3 crucible assays of tailings . | 1·487 | 14·233 |

The weight of the tailings after leachings, however, was not determined. If, however, we allow a loss of 10 per cent. in weight due to soluble salts, which is certainly fair to the process, we shall have as the net content of the tailings:

| | Gold. Ozs. per ton. | Silver. Ozs. per ton. |
|---|---|---|
| $\frac{1687}{2000}$ tons tailings contain net . . . . | 1·254 | 12·005 |

Or in percentage of original content there were lost in the tailings 22·65 per cent. of the gold and 49·47 per cent. of the silver.

Hence the final results of the test were:

| | Gold. | | Silver. | |
|---|---|---|---|---|
| | ozs. | per cent. | ozs. | per cent. |
| Loss by volatilization and dusting | 2·745 | or 49·58 | 6·865 | or 28·28 |
| Loss in tailings . . . . . | 1·254 | ,, 22·65 | 12·005 | ,, 49·47 |
| Actual yield by mode of treatment . . . . . | 1·537 | ,, 27·77 | 5·398 | ,, 22·25 |
| Original content raw sulphurets . | 5·536 | or 100 | 24·268 | or 100 |

A curious fact is that in spite of the long chloridizing roasting, perhaps in consequence of it, less than 23 per cent. of the silver was extracted by the hyposulphite solution, and nearly 50 per cent. was left in the tailings. The excessive losses were in part due, no doubt, to the fact that this furnace had only recently been erected at the time of this test, and the workmen had hardly got used to it. For in 1880, another of Mr. Christy's students had obtained the following results from sampling the ore from this same mine and the final tailings. The ore had been roasted in an old-fashioned, long, two-step reverberatory furnace :

| | Gold. Ozs. per ton. | Silver. Ozs. per ton. |
|---|---|---|
| Raw sulphurets . . . . | 3·27 | 10·80 |
| Final tailings . . . . . | 0·11 | 8·56 |

**Loss of Gold in Muffle Roasting.**—The result of over two hundred laboratory experiments in muffle roasting proved, that—

1. Within the limit of the salt used 4 per cent., the loss both of gold and silver increases with the amount of salt used, other things being the same.

2. It became apparent that the effect of time is to increase the loss, but the effect of an increase of temperature on the gold loss is greater than the effect of an increase of time.

3. The loss both of gold and silver is greater in all cases where the salt is added after a long oxidizing roast than where the salt is added at the start. This is the general result of all the muffle roasts that have been made. It is invariable in the case of gold, and nearly always the case with silver. The reverse is the case on the large scale where a continuous roasting takes place in the reverberatory furnace, as has been shown by the researches of Mr. Aaron.

It has also been established that the heaviest losses of gold take place during the last roasting stages, and not at the beginning of the roasting operations; and the loss of gold is undoubtedly due to the amount of chlorine which comes in contact with the gold. When the salt is added at the start, the chlorine is at first used up as fast as it forms by the sulphur, which escapes as chloride of sulphur. This fact was shown by the covers of the salted roasts, a certain amount of this substance being condensed on them. Of course, as long as the sulphur is present, it protects the gold from attack, and naturally the loss should be less. But when no salt is added till a long oxidizing roast has converted the sulphur into sulphuric acid and acid sulphates, and the salt is then added, the chlorine is then rapidly generated, and, what is more, it has a chance to act on the gold, which is now no longer protected from its action by the presence of sulphur.

**Mr. Wilson's Experiments.**—The *American Engineering and Mining Journal* (New York, April 13th and 20th, 1889), has published some results of investigations made on the subject by Mr. W. A. Wilson, M.E. The general conclusion he arrived at was that losses take place while roasting gold ores in the muffle, but not on a large scale.

In laboratory experiments of sixteen raw samples, crushed through thirty-mesh screen, by leaching, 51·1 per cent. of silver and 41·7 per cent. gold was extracted. None of the gold extracted from raw or roasted ore can be in the metallic state, for metallic gold is not attacked by the solutions of the Russell process.

Mr. Wilson gives a list of fourteen ores containing gold on which muffle-roasting tests were made, with the losses by volatilization in the muffle and the highest extraction obtainable by various leaching tests on the roasted pulp. The same shows how much greater is the loss of gold than of silver, and that no relation exists between them.

In all muffle-roasting tests made at the Ontario, the ore used was crushed through a thirty-mesh screen, the weight of ore roasted was 150 grammes, which was placed in the muffle in an earthen dish and stirred with a thick iron wire, having a piece of clay pipestem on the end, so that no metal came in contact with the roasted ore. In these roasting operations 5 to 20 per cent. of salt was used, occasionally from ½ to 4 per cent. iron matter was added, and the per cent. of silver extracted ranged from 33 to 95 per cent. The per cent. of gold extracted ranged from 34 to 96 per cent. The silver which was lost by volatilization ranged from 0 to 48 per cent. Gold lost by volatilization from 21 to 66 per cent.

**Mr. Falkenau's Experiments.**—Some muffle-roasting tests and leaching tests by the Russell process, made by Mr.

L. Falkenau, of Francisco, on Occidental ore from Virginia City, Nevada, showed that the finest part which passed through a thirty-mesh screen contained three to thirteen times as much gold per ton as any of the other grades. This accounts for the apparently large losses which sometimes took place in roasting this ore in a Stetefeldt furnace, as some of the roasted ore, except what fell in the shaft of the furnace, constituting the coarsest particles, was assayed or tested. The results obtained by muffle roasting show, that when roasting with sulphur alone for half an hour at a low heat ending with high, no loss of gold or silver took place; but there was no extraction of gold, and only 5 per cent. to 36 per cent. of silver.

With 2 per cent. of salt and no sulphur for two hours at a low heat there was a small loss of both gold and silver, and no extraction of gold and only 59 and 63 per cent. of the silver. With 5 per cent. of salt and 2 per cent. of sulphur for two-and-a-half hours at a low heat ending with high, there was some loss of gold and silver, nothing up to 50 per cent. extraction of gold, and 66 per cent. to 73 per cent. of silver. With 10 per cent. of salt and 3 per cent. of sulphur for two-and-a-half hours at a low heat ending with high, there was some loss of gold and silver, all the gold except a trace was extracted, and 90 per cent. to 94 per cent. of the silver. Under the same circumstances, but omitting the sulphur, there was the same extraction of gold, but only 76 per cent. to 81 per cent. of the silver. The addition of sulphur with 10 per cent. of salt therefore increased the extraction of silver 13 per cent. Here, as in the case of other ores, the extraction of gold and silver, after roasting in the muffle with little or no salt, was only a small fraction of the extraction from the raw ore by raw leaching, or of that from the ore roasted without salt in the Stetefeldt.

The coarseness of crushing made no difference in the amount of gold extracted, but for the various grades the average extraction of silver was affected by the fineness or coarseness of screen. In some cases the salt was added after an oxidizing roast, but the usual method was to mix it with the charge before roasting.

**Reduction of Loss of Gold in Roasting to a Minimum.**—The essential point is to carry out the roasting operation at a low heat, and the best furnace and the one most in favour is the reverberatory with long hearth—say 60 to 70 feet in length and 10 to 12 feet wide. Such a furnace will hold from six to ten tons of ore, and turn out a charge of one ton every eight hours, or three tons per day, so that the charge at the back end of the furnace remains on the hearth nearly three days before being withdrawn on the front hearth.

The ore is submitted to a continuous oxidizing roasting at a dull red heat, and when the ore arrives at the finishing hearth nearest the fireplace, the heat is continued at a low cherry-red, till all the sulphur is driven off, when the heat is lowered and the salt is added, being actively stirred in for about half an hour; when the ore is withdrawn into the cooling pit.

The results of a long series of experiments have shown (according to Mr. Christy), that the volatility of the gold in the presence of chlorine is increased by even a slight increase of temperature above a red heat, and it will be readily understood how important an influence this lowering of the temperature, when the salt is added, on reducing the gold-loss. Inasmuch as the ore remains in the furnace from forty-eight to seventy-two hours, it must also be evident that time is not the most important factor in causing the loss. Of course, if salt were present all this time the result would be probably somewhat different.

Mr. Christy states that, in all his muffle tests, he found that with a given percentage of salt the loss of gold was invariably less when the salt was mixed with the ore at the start than when it was mixed with the ore after a long oxidizing roast. The reason of this is that, when the salt is added at the start, the chlorine is partly used in volatilizing the sulphur, and so adding a given amount of salt at the start, as far as the gold-loss is concerned, is practically equivalent to adding a less amount of salt after a long oxidizing roasting. Hence, whenever a batch of ore is treated at a time, and a continuous process is not used, it is natural to expect a greater loss of gold when the salt is all added after a long oxidizing roast. But as it is well known that the reverse procedure gives the least loss of gold in continuous roasting on a large scale, as has been pointed out by Mr. Aaron, it is worth while to ascertain the reason why.

In single-hearth roasting furnaces, where the whole lot of ore is kept at the same temperature throughout, when the gold chloride has once formed and left the batch of ore, that is the last of it as far as that sample of ore is concerned; hence, the more chlorine set free after the gold becomes exposed to it, the greater the loss. Adding salt at the end, therefore, produces a greater loss of gold. Now, when the ore is treated continuously in the long hearth reverberatory furnace, it is not all kept at the same temperature, but it is red hot at one end and dark at the other; hence, excellent conditions are maintained for condensing the gold chloride set free at the hot end, by the cold ore still containing sulphur at the other. Now, if salt is all added at the start there is a continued volatilization of chloride of gold all through the furnace, and a less favourable opportunity for it to condense; while if, on the other hand, the salt is all added at the hot end at the finishing hearth, although a rapid volatilization of gold takes place there, it has a chance to condense by coming in contact with the long surface of unsalted cooler ore; some of this is yielding $SO_2$, and with the steam from the burning fuel offers excellent means for reduction of the chloride of gold right within the furnace. But the most efficient means probably is the pyrites themselves, therefore the hind portion of the furnace nearest the flue should be kept as cool as possible.*

---

* For this last valuable suggestion the metallurgical world is indebted to a communication addressed by Mr. Aaron to the State Mineralogist of California.

# CHAPTER VII.

## *THE ROASTING OF PYRITIC ORES.*

THE ROASTING OF IRON PYRITES—Heap Roasting—Kiln Roasting—Kilns with Grates—The Freiberg Pyrite Burner—English Pyrite Burner—Perret and Olivier's Shelf Furnace—Hansenclever and Helbig's Burner—Hansenclever's Improved Burner—Gerstenhöfer's Furnace—The Bousfield Kiln—Roasting Furnaces—Single Roasting Furnace—Double Furnaces—Pontgibaud Furnace—Kustel's Furnace—Mechanical Roasting Furnaces—Bruckner's Cylinder—The Ottokar-Hoffman Furnace—The Spence Furnace—The McDougall Furnace.

**The Roasting of Iron Pyrites.**—The roasting of highly sulphuretted ores is one of the difficult problems with which metallurgists have to contend. Where ores are rich, the question of economy in treatment has not to be so seriously considered, but there are large bodies of low grade ores which require economical manipulation throughout, as otherwise the cost of treatment would exceed the value of the product in precious metal.

The solution of the problem is not an easy one. Roasting in long reverberatories is the best method, but it is slow and costly. Roasting in the Stetefeldt furnace will not produce a complete desulphuration of the heavy pyritic ore, and experience with the revolving cylindrical furnaces has also shown that with heavy sulphuretted ores a combination of two furnaces, instead of one, might lead to good results, by utilising the waste heat of the first furnace to effect a partial roasting in the second. The large amount of sulphur when burning creates so great a heat in the revolving cylinders that matting of the ore will take place, and it is therefore necessary to provide a cheap and effective method to get rid of the greater portion of the sulphur before submitting the ore to the dead roast.

Results obtained at the Germania Works, in Utah, with a Bruckner cylinder,* 18 feet long, with a diameter of 7 feet, have shown that 8 tons of matte (the weight of charge) have been roasted in 48 hours to 4 to 6 per cent. of sulphur; equally favourable results have been obtained in treating pyritiferous ore containing 38 to 35 per cent. of sulphur, but no doubt the greatest care and attention is required to obtain such a result.

Some interesting experiments were conducted in the Stetefeldt furnace in the Ontario Mill,† of which the results are subjoined. The experiments were somewhat hurried and in complete, but important, and previously unpublished data were obtained. The furnace was 6 feet square and 50 feet high. A car load of Dixon ore was first treated; having the following composition: 32·5 sulphur, 25·7 iron, 16·1 lead, 15·9 silica, and 1·5 zinc, and 13·5 ounces silver to the ton, being a low grade pyritiferous galena. It was passed through the driers and stamps (having No. 30 screens) and the furnace in 5¼ hours. It screened at the battery without difficulty. The furnace was quite hot, and the scintillating burning ore as it descended the main stack of the furnace, was a beautiful spectacle. The following were the roasting results:—

1. Main stack, early part of run, side openings, 15·8 per cent. sulphur in ore.

2. Hotter furnace, central discharge open and both holes closed, 6·2 per cent. sulphur left in ore.

3. Chamber dust, 5 per cent. sulphur left in ore.

4. Side openings only, last of run, 4 per cent. sulphur in ore.

5. Faster run, return flue, last of operation, 8·3 per cent. sulphur left.

6. Mill sample of entire run which could be extracted from chambers and furnace, 7 per cent. sulphur, 17·5 lead, 15·75 ounces silver.

---

* "Ore and Matte Roasting in Utah."

† *Transactions of American Mining Engineers*, 1887.

Ten tons of iron matte of the following composition was provided for treatment: 22 sulphur, 13 lead, 6 copper, 55 iron, 59·5 ounces silver per ton. Of this amount only 3 tons were treated, owing to its obstruction of the battery screens.

**Heap Roasting.**—In the ordinary metallurgical operations in America, the roasting of matte and of iron pyrites is generally carried out in pile or heap roasting. The common practice is with pyramidal piles 24 by 18 feet at their base, 6 feet high, and fired with from 6 to 15 inches of light wood. Such a pile will contain about 80 tons of matte or pyrites, and will burn from 30 to 40 days. The material is broken to fist size, and should be handled with sluice-forks to avoid fines. A heavier layer of wood will cause a thicker zone of fused raw material, but will often result in a very excellent roast; an experimental pile burned with 6 inches of wood gave 3 inches of friable fused matte and an equally good roast.

Attention should be drawn to the means employed by the sulphuric acid manufacturers for roasting pyrites, and a solution of this difficult problem may possibly be found in that direction. One single operation will not be sufficient to free heavy sulphuretted ores, like concentrations, of all their sulphur, and produce a dead roast, which would fit the ore for either amalgamation or chlorination; but no doubt cheap mechanical roasters capable of handling large masses of ore in a short space of time are at our command which will prepare the ore for the second operation or the dead roast, by burning away a large portion of the sulphur; and in studying the matter closely, a distinction has to be made between furnaces intended for lumps, and furnaces intended for smalls or fine stuff.

It will be found necessary to keep these two kinds apart, and to employ different apparatus—or at least processes—for them, for if the broken ore is put into the burner without separating the smalls, the air channels, which ought to remain between the pieces, are soon partly stopped up with powder, and the access of air becomes irregular. Thus scars are formed, and proper work is then impossible. Apart from the coarser and finer powder obtained on breaking, a great deal of smalls is obtained through concentration. It is quite clear, that, when heap or kiln roasting of lump ore is resorted to—the roasted material is then passed through the dry crushing battery or other pulverizing machine, and then submitted to the second or dead roast, which fits it for chlorination or amalgamation.

**Kiln Roasting.**—The operation of burning of lump pyrites in kilns without grates usually takes place in such a way that the lumps of pyrites are brought to a temperature furnishing sufficient heat for maintaining the combustion without any other fuel being afterwards employed. The apparatus used for this purpose are called kilns or burners.

Kilns are shaft furnaces, with or without grates, and have fireplaces outside the shaft, or are fired directly from the inner grate. They are extensively used by manufacturers of sulphuric acid, and for a full description of the different kilns in use, the reader is referred to works specially dealing with that industry.*

Kilns with sloping sides are preferable, as shafts with perpendicular sides are found to get easily choked and so the draught is impeded. One disadvantage of these appliances is the small quantity which can be burnt in them at one time, as in metallurgical operations where lump pyrites are to be treated the object is to deal with large masses at a single operation. As occasionally the manufacture of sulphuric acid as a by-product could be economically introduced even in distant mining regions, a description of some of these kilns is given here, and their application may lead to improvements which will enable the metallurgist to utilise them on a much larger scale.

**Kilns with Grates.**—This form of kiln is an improvement admitting of a proper regulation of the draught, as with a closed ash-pit only a definite quantity of air need be admitted into the

---

* Notably to Lunge's work on "Sulphuric Acid," and to Dr. Philipp Schwarzenberg's "Die Technologie der Chemischen Producte."

pit, and in addition the air must first spread equally underneath the grate, and rise all over the area of the burner. Thus the ore is much more completely burnt. The operation of drawing out the burnt ore becomes much more regular, and offers a greater guarantee against raw ore getting into it; lastly, it does not happen so often that fused masses, or "scars," are formed in the burner. The introduction of grates led to further improvements; first a diminution of the height of the burners, which made them much handier for working, and which acted especially well with more easily fusible ores, although in some places the other extreme of too low layers of pyrites has been resorted to.

**The Freiberg Pyrite Burner.***—A transition from the metallurgical kilns to the burners of to-day is presented by the

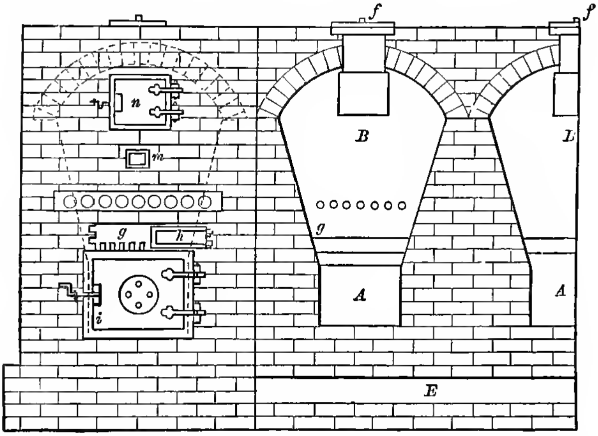


FIG. 88.—FREIBERG PYRITE BURNER. Elevation.

Freiberg Kiln for roasting coarse metal, which is shown in the accompanying illustrations. Fig. 88 shows the appliance in elevation on a scale of 1 : 50; and Fig. 89 is a section.

---

* The descriptions of this burner, of the English pyrite burner, and of Hansenclever's furnace, are taken from Dr. Lunge's work.

This burner is specially adapted for easily burning ore. B is the shaft; *f* the charging hole, furnished with a tightly-closing cover. The grate, *g*, is inclined, in order to facilitate the drawing through *h*. The smaller pieces fall through the grate into the ash-pit, A, and are emptied from time to time through the door *i*, furnished with holes for the supply of air. About 10 inches above the grate the front wall of the burner is pierced by a horizontal row of holes, *k*, in which stuffing-boxes are fixed for movable round iron bars, *l*. These can be used for loosening the ore (which they will do very incompletely); they also serve for supporting the higher lying portions of ore on withdrawing the cinders. The door, *m*, serves for observing the combustion (which, however, can only be judged of properly from the top) and for introducing a poker in case the ore must be broken up. The larger opening, *n*, likewise with a door, serves for the same purpose. The gases from C get first into the large main flue, D, which at Freiberg is upwards of 330 feet long; here they deposit dust, and especially arsenic, and then pass on to the acid chambers. The flue, E, serves for keeping the ground moisture from the burners.

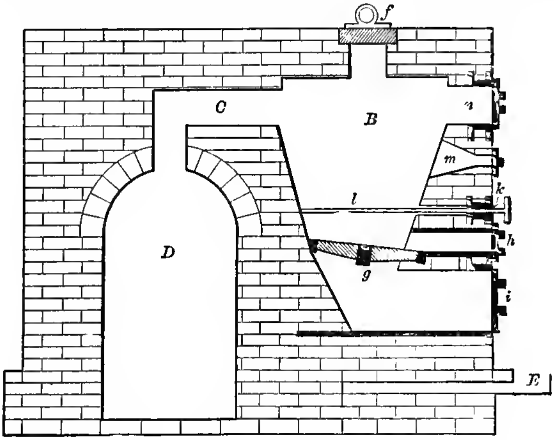


FIG. 89.—FREIBERG PYRITE BURNER. Section.

**English Pyrite Burner.**—Figs. 90 and 91 show a simple construction, which can be made with open sand castings. In Fig. 90 are two burners in front elevation, and one in section, the first burner without doors. Fig. 91 is a cross section, showing two rows back to back. *a* is the working opening, with the door, *b*, which slides in the grooved ledges, *c c*, cast to the front plate. The small door, *d*, only to be used exceptionally, is arranged in precisely the same way. The openings of the brickwork are protected by small metal plates; *e* is the movable cover of the ash-pit, provided with air holes; *f f* are the grate bearers; the front bearer, $f_1$, at the same time carries the bottom plate for the front wall, and is perforated by round holes; whilst *f f* are cut out in semicircles. The arches are sprung parallel with the working doors, and, by the draught holes, *g g*, are in connection with the gas flues, *h h*. The latter, like the burners, altogether, are cased in metal plates; they are

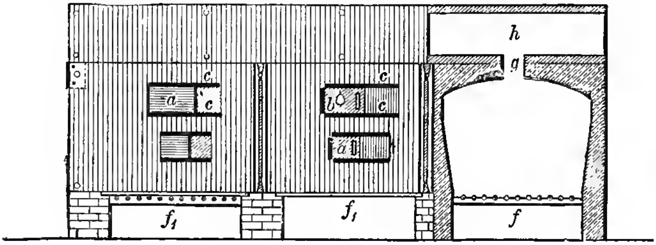


FIG 90.—ENGLISH PYRITE BURNER. Scale 1 : 65.

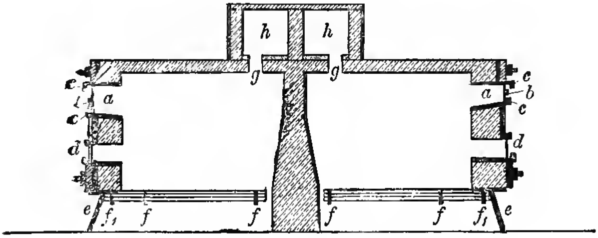


FIG. 91.—ENGLISH PYRITE BURNER. Section.

covered by fire tiles. To operate such a burner, it is necessary to lay on the grates a layer of already roasted lump ore, and on top of it the fuel is dumped and set on fire. After 12 to 24 hours the kiln becomes red hot, and the pyrites are charged. In a short time they commence to burn, and the sulphur gases are let into the lead chambers, by opening the damper leading to them, which is kept shut down during the heating up of the kiln. Care must be taken to give sufficient air during the operation; otherwise, instead of producing sulphurous acid, sublimed sulphur will be obtained in the flues and chambers.

When it is observed that the charge in the burner is nearly done, the grate-bars are turned—so as to allow the burnt ore to fall into the pit, and a fresh charge is introduced through the working door, *a*. Since the development of the sulphuric acid industry on so large a scale, further improvements have been devised for the burning of the fine ore. Combination furnaces have also been introduced, allowing the burning of the lump ore in a separate compartment, and the heat thereby obtained is utilized in the roasting of the fine ore, which is made to move gradually down a shaft over inclined shelves; or the ore is spread in thin layers on clay shelves over which the flame circulates. This arrangement is carried out in the undermentioned furnaces.

**Perret and Olivier's Shelf Furnace.**—The lump ore is burnt on a grate in the lower portion of the shaft, and the upper portion of the shaft is arranged to receive a number of fire-clay slates, placed one above the other. The fine ore is put upon them in a layer 2 inches thick, and is ignited and burnt off by the hot gases given off in the burner below.

**Hansenclever and Helbig's Burner.**—This also is a combination of a burner for lumps and one for smalls. The heat and gases from the lump burner rise in a tower-like apparatus, in which inclined shelves are placed. These shelves have an inclination of 38° to the horizontal, and the fine ore in gradually sliding over these inclines is burnt, and is gradually withdrawn as it reaches the bottom of the tower. The automatic sliding movement of the ore continually exposes fresh surfaces, which is an advantage over Perret and Olivier's burner. In this furnace smalls up to $\frac{3}{5}$ of an inch can be burnt directly without grinding.

**Hansenclever's Improved Burner.**—This furnace is shown in Figs. 92 and 93. The ore is filled into a funnel, *a*, at the top; and before getting into the muffle it must pass over a large inclined plane, heated from below by the waste fire of the blind roaster, *o*. If the ore were allowed to slide down freely on this plane, inclined at an angle of 43°, there would be formed at the end of this a heap more than 5 feet high, since finely powdered bodies, on being tipped down form a

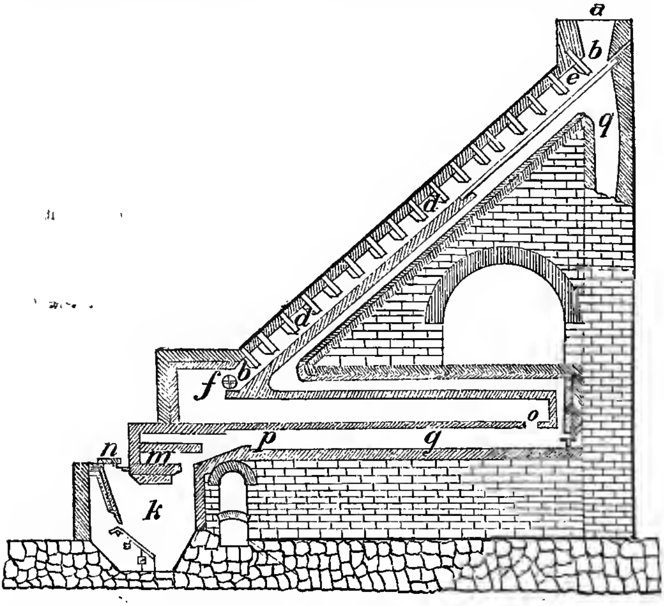


FIG. 92.—HANSENCLEVER'S IMPROVED FURNACE.

nearly constant slope of 33° ; thus any burning in the interior of the mass would be impossible. There are, therefore, at distances of 18 inches, partitions, *d d*, removed an inch or so from the inclined plane, and causing the formation of thin layers of ore along the whole slope. The partitions have a lateral opening, and are so mounted that the $SO^2$ from the blind roaster travels in the direction indicated by the arrows in Fig. 93 a long way over the fine ore, and ultimately gets at *q* through a flue into a brick chamber. Thus the gas is made richer, and the ore is gradually burnt. The fire flues below the inclined plane are readily accessible from the side, and can be easily cleaned during the working.

From the sloping plane the ore gets into the blind roaster, *o*, by means of a roller at *f*, which is hollow, so that air can circulate inside and cool it. It is moved by a small water wheel and according to the quantity of water, which is regulated by a tap, every 2 or 5 minutes throws a small quantity of dust onto the bottom of the muffle. By the movement of the roller, the ore on the inclined plane is obliged to slide backwards. Every two hours the ore collected at the bottom is spread over the bottom of the blind roaster, and gradually moved forward till it gets to the opening, *o*, and falls down to the hearth for burning it off by direct fire from *k*. The $SO_2$ given off on the bottom hearth is lost along with the fire gas, whilst the gas from the blind roaster and from the inclined plane is used for the manufacture of sulphuric acid. The temperature of the furnace is so high that antimony melts in all parts of it. It utilizes even poor blends with advantage;

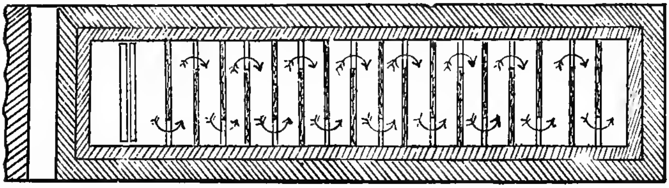


FIG. 93.—HANSENCLEVER'S IMPROVED FURNACE. Plan of Tower.

for an ore with 20 per cent. sulphur still yielded gas with 6 per cent. of $SO^2$ by volume. There were in the ore :—

| | | |
|---|---|---|
| At the end of the inclined plane, *b b*, still | 10 | o/o S. |
| At the end of the muffle at *o* ... ... | 6·4 | „ |
| Burnt off at *p* ... ... ... ... | 1·2 | „ |

The reverberatory furnace is heated by a gas generator, *k*, in order to obtain a regular heat and save fuel.

**Gerstenhöfer's Furnace** is extensively used in Swansea, as well as in Freiberg, for the roasting of pulverized matte, and the sulphurous acid is condensed for the manufacture of sulphuric acid. It gives excellent results, but is expensive to maintain. In form it is a square shaft, across which are arranged at equal distances a series of parallel horizontal fire-clay bars, having spaces between them. When fed from the top of the furnace on to these bars, the ore assumes a gradual slope, and slides from one bar to the other below. This continual dropping of the ore particles from one shelf to the other exposes them to the oxidizing influence of the air, and they arrive pretty well desulphurized at the bottom.

**The Bousfield Kiln.**—This kiln or oven is used at some copper reduction works in France. It consists of a number of separate chambers arranged in a circle round a single central furnace, which heats them either simultaneously or separately at will; an arrangement which allows the various compartments to be kept at different temperatures, which are regulated and maintained as required at each phase of the process. The form of the chambers or separate ovens has to be proportioned to the work to be performed, and may be varied according to the purpose for which the circular kiln with the central furnace is to be employed.

The advantages of this arrangement (according to the inventor, as set out in his specification) are (1) the avoiding, by reason of the central position of the furnace, the loss of heat which ordinarily takes place from the exterior; (2)

enabling the temperature of the furnace to be kept regular and constant, thereby economizing fuel, and at the same time varying at will by means of registers the temperature of the different compartments during the operations; (3) heating with a comparatively small furnace large quantities of materials; and (4) allowing the secondary products (if any) to be collected.

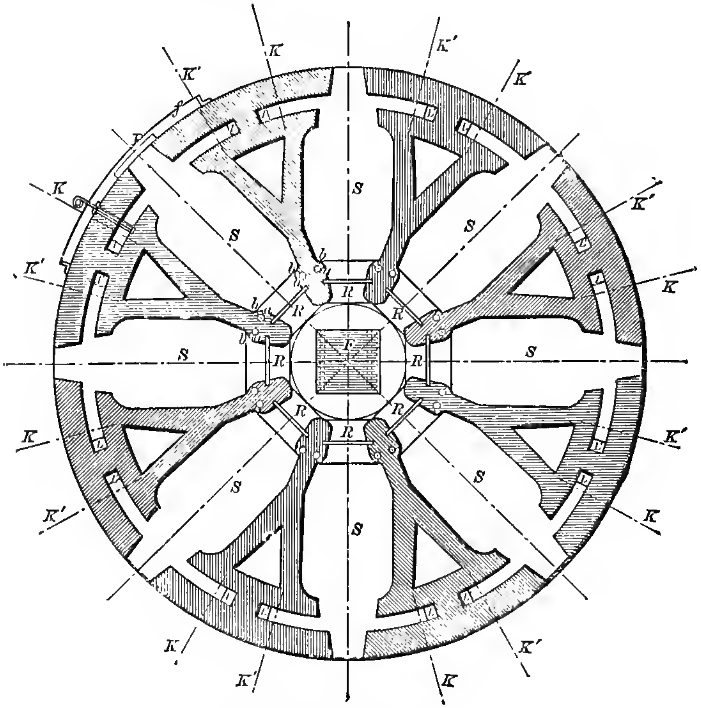


FIG. 94.—BOUSFIELD KILN. Plan.

The illustrations here given (Figs. 94, 95) show a kiln for roasting sulphurous minerals, and for transforming them to sulphates; this operation being one of those which require the most complicated construction, and for which the kiln has been more particularly designed. The operation requires that during

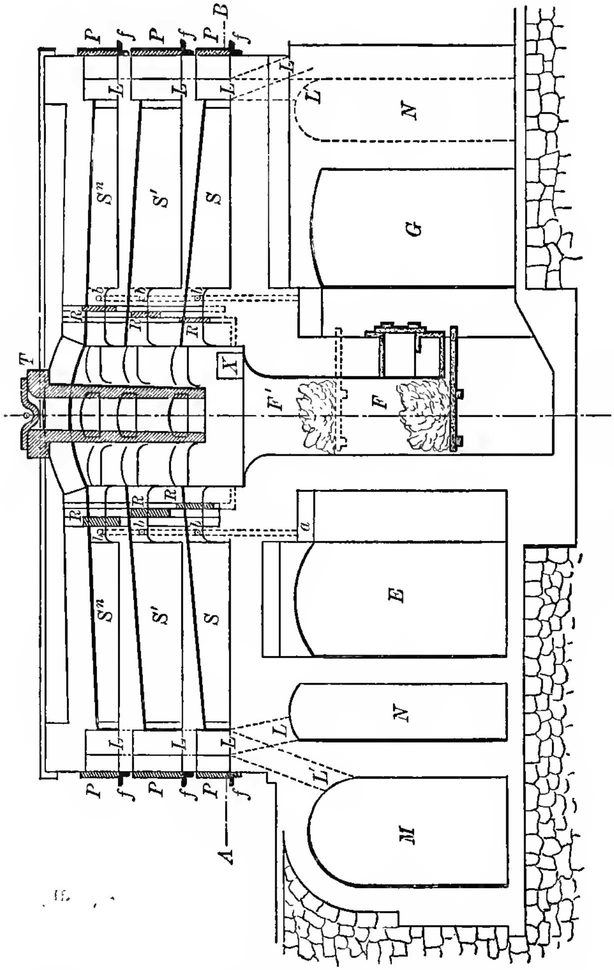


Fig. 95.—Bousfield Kiln. Section.

certain phases the temperature should be greatly moderated, and the heat introduced varied, reducing it nearly to nothing at the moment when the sulphur of the minerals inflames ; and then gradually augmenting it when, as the sulphur of the mineral is exhausted and its combustion diminishes in activity, the temperature tends to drop ; and finally at the end of the operation giving it a heat more or less intense, and more or less prolonged, to decompose one or more of the sulphates that have been formed.

The sulphurous acid disengaged during the work is partly collected in a special gallery, and is led from thence to apparatus suitable for its transformation into sulphuric acid.

In the accompanying illustrations, S, S, $S^1$, $S^1$, $S^n$, $S^n$, are circular series of eight hearths each, three series S, $S^1$, $S^n$, being placed one above the other, and making in all 24 hearths or soles to each kiln. P, P are working holes ; F, single central furnace situated below the hearths, S ; G, gallery for working the furnace, $F^1$. In case it is thought desirable, the grate can be raised as shown in $F^1$.

By means of the fire-clay hopper the fuel is fed in at the top and the furnace will be at $F^1$ ; R, R, R are fire-clay registers worked from without by means of levers working in grooves. These registers serve to regulate the heat admitted to each of the hearths during work, or even to isolate them completely during the time occupied in charging and discharging, although all the others are in direct communication with the fire. L, L, L are flues for carrying off the products of combustion ; M, M, collecting flues in direct communication with the flues, L, L, L ; K, K, K, vertical cast-iron registers one to each sole to cut off communication between the soles and flues, M, M and chimney ; $L^1$, $L^1$, $L^1$, are openings for the passage of the sulphurous acid ; N, N, gallery in communication with the openings, $L^1$, $L^1$, $L^1$, for collecting the sulphurous acids ; $K^1$, $K^1$, $K^1$, vertical registers, one to each sole, for cutting off communication between the soles and N, N, when required. A small railway allows the wagons loaded with materials to be dried to circulate in the interior.

*a*, *a*, *a* are passages for carrying the air from the stove on to the soles ; *b*, *b*, *b*, orifices for distributing the air from the passages, *a*, *a*, *a*. This distribution is regulated by valves. X, flue furnished with a register for passing the flame directly to the chimney, if necessary. *f*, *f*, *f*, circular angle iron ties, which also serve as platforms for the working doors.

It is hardly necessary to point out that the structural details of this kiln are rather complicated, and that the roasting surface of each sole is small. The idea of one central furnace is a very good one, as loss of fuel is thus avoided. Once the pyrites begin to burn on one hearth, they can be made to roast in their own heat; the communication with the fire-place is shut off, and the flame utilised in other hearths. It has occurred to me that a useful modification of such a kiln might be made by connecting the hearth, S, with hearth $S^1$ by means of a flue in the arch, and the same with $S^1$ and $S^{ll}$. In this manner eight triple hearth furnaces of considerable capacity would be produced, a better utilisation of heat would be effected, and the structure in the main very much simplified by reducing the complicated canalisation. Then when the pyrites on the lowest hearth, S, commence to burn, their products of combustion, passing over $S^1$, and then over $S^n$, would act in the same manner as in the double furnaces described on page 245; and when the ore on the hearth, S, was completely roasted the charge would be withdrawn, and the ore from $S^1$ pushed through the flue on to it, the same being followed from $S^n$ on to $S^1$, and a fresh charge dropped through a hopper on to $S^n$. In this manner the charging and discharging would also be simplified.

**The Roasting Furnaces.**—*Single Roasting Furnaces.*—Figs. 96 and 97 show the longitudinal section and plan of a roasting furnace for one ton of sulphurets at a charge; *a* is the hearth bottom, about 12 ft. square. It is made of the hardest bricks, laid edgewise, close together, forming a stratum of 4 in. in thickness. There are four working doors, *c*, *c*, which enable the roaster to reach all points of the hearth conveniently with light rakes. In the middle of the length at the bottom, near to the doors toward the chlorination vats, is a square discharge-hole, *b*, which is kept shut by a slide, *d*, during the roasting.

Below the floor is an arch through the whole width of the furnace, through which the hole, *b*, passes. An iron car on rails receives the roasted ore when discharged, and wheels it to the cooling place.

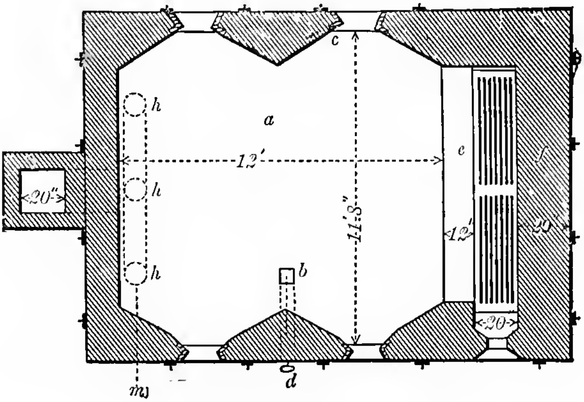


FIG. 96.—SINGLE ROASTING FURNACE. Plan.

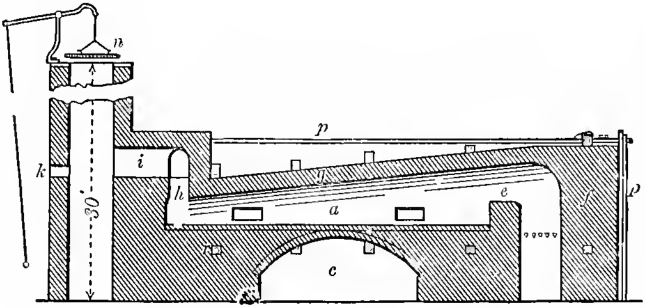


FIG. 97.—SINGLE ROASTING FURNACE. Section.

The bridge, *e*, is from 10 to 12 in. wide, and from 8 to 10 in. high. It separates the hearth from the fire-place. If possible, it should be made of some refractory material, like fire-bricks.

The outside wall, *f*, is often made 12 in. thick, but it is always better to give sufficient substance to the wall on account of the heat, and at the same time to obtain a strong support for the roof; the thickness ought not therefore to be less than 24 in. Economy here would be misplaced.

The roof is generally 20 in. above the bottom, as the greatest distance; if it is less, although the form of the furnace is improved, it is less likely to be durable, unless the work is perfect. The length of the bricks gives the thickness of the arch, that is 8 in. This arch, if all the bricks are placed perpendicularly, will stand a great deal longer than one of 12 in. thickness, when the bricks are laid with the long sides alternately horizontal and perpendicular.

There are three circular openings, *h h*, in the roof, each 10 in. in diameter, communicating with the chimney by the flue, *i*, opposite which is a small door, *k*, for the purpose of cleaning the flue, *i*, from time to time. For the same purpose an opening must be prepared in front. The best way to regulate the draft is by means of a cover, *n*, on the top of the chimney; but when there are two or more furnaces to one chimney, a damper in the flue, *i*, will answer the purpose.

FIG. 98. GRAPPLING IRONS.—Scale $\frac{3}{8}$ inch = 1 foot.

The sulphurets are charged through the roof by means of a hopper. It is of importance to secure the furnace against expansion by the use of grappling irons, of which there are eighteen.

The iron rods crosswise are from $\frac{5}{8}$ to $\frac{6}{8}$ of an inch in diameter, the others, *p*, which are placed over the length of the furnace, are stronger—being 1 in. The grappling irons (represented in Fig. 98) are made of cast iron, and are about 4 ft. long. For the passage of the rods of the lower ends, square holes, *r* (Fig. 97), are provided through the masonry of the furnace. Another kind of grappling irons is shown in Fig. 99. They are of wrought iron, and must be at least 1 in. thick. They are cheaper than those of cast iron, but the latter are preferable.

*Double Furnace.*—This kind is represented by Fig. 100. The heat which escapes from the single furnaces is conveyed through the flue, *e*, over the upper hearth, *b*, of the same size, having the working door on the opposite side. On this second hearth the ore loses a great deal of its sulphur, and is drawn through a discharge hole in the middle of the floor on to the lower hearth, where the roasting is finished. From *b* the heat must pass a third hearth, *a*, before it enters the chimney. It is a drying rather than a roasting place, as the heat is very moderate. Some furnaces of this kind have an auxiliary fireplace for the second hearth, which, for the roasting of auriferous pyrites, is entirely superfluous, but would assist in a chloridizing roasting of silver ores.

FIG. 99. GRAPPLING IRONS.—Scale ⅜ inch = 1 foot.

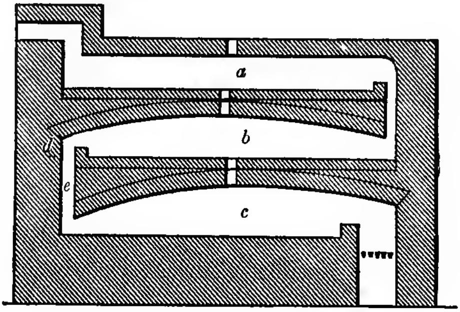


FIG. 100.—DOUBLE ROASTING FURNACE. Scale ⅛ inch = 1 foot.

In Freiberg, Saxony, there were three double furnaces for the roasting of lead ores for smelting purposes. These three furnaces have been combined into one, as represented by Fig. 101. The heat passes the lower three hearths, and then, ascending through the flue, the upper ones. From the flue, *a*, it is conducted over the drying hearth, *b*, built in the same way as *a* in Fig. 100; from *b* the ore is drawn on to the next lower hearth, *c*, through *f*, and is removed after two hours to *d*, where it is again exposed to the heat for two hours.

Every two hours the ore advances a step till it arrives at *e*, where the strongest heat has now already a half-smelting effect upon it. Every two hours one charge of 800 lbs. of roasted ore is obtained from the hearth, *e*. The lower furnaces have two working doors each, the upper one three. There is not much stirring in this operation, and a kind of shovel is more in use than the hoe. The fuel used is coal.

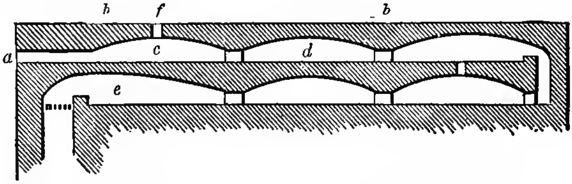


Fig. 101.—Double Roasting Furnace. Scale $\frac{1}{16}$ inch = 1 foot.

**Pontgibaud Furnace.**—A different construction is shown in Figs. 102 and 103. The first is the plan, the second a section, of a furnace built by Mr. Deetkin, in Grass Valley, for the sulphurets from the Eureka mine. The construction is like that of the roasting furnace for lead ores at Pontgibaud. The upper hearth is above the lower, and a continuation of it, interrupted by a step flue, *a*, of 7 ft. 10 in. in height—in the drawing, less. The upper hearth is 6 ft. wide by 39 ft. in length, furnished with working doors upon each side, twelve in number. The chimney, *b*, about 25 ft. high, is built after Deetkin's cheap plan, of 4 in. thickness, 28 in. each side in the clear.

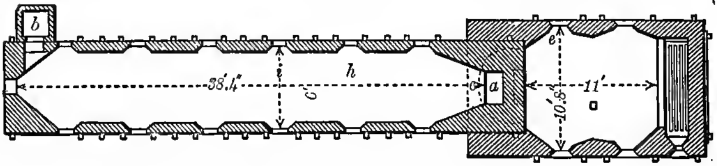


Fig. 102.—Section of Pontgibaud Furnace. Scale $\frac{1}{16}$ inch = 1 foot.

This kind of furnace requires more room than the double furnace (Fig. 100), but the work of stirring is less tiresome, as the roaster is not obliged to constantly step up and down. Another advantage is the extent of the upper hearth, which receives 9 tons of ore without difficulty; whereas the charging of a three or four story high furnace is very troublesome unless favoured by a sloping locality. The rising flue, *a*, Fig. 103, is not necessary; on the contrary, it uselessly consumes a part of the heat. There is also no special advantage in making the second hearth narrower, for although the heat is more contracted, the length must increase for the same quantity of sulphurets.

A roasting furnace ought to consist of at least three hearths, constructed like those shown in Figs. 96, 97, but arranged similar to the lower tract of Fig. 103. By this means it is possible to draw every ten or twelve hours a ton of roasted sulphurets from the hearth near the fire-place. It takes about 24 hours to roast a ton of sulphurets. The only way of shortening the time is by the use of long furnaces, in which a great quantity of stuff is gradually prepared by being moved in successive portions towards the fire-place.

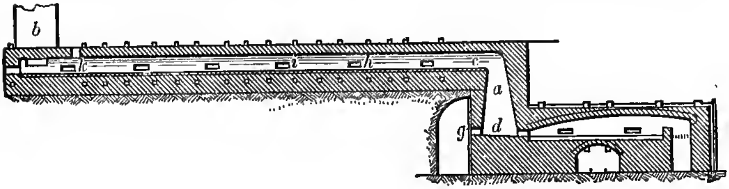


Fig. 103.—Plan of Pontgibaud Furnace Scale $\frac{1}{16}$ inch = 1 foot.

**Küstel's Roasting Furnace** is shown in Fig. 104, which is on a scale of 8 ft. to 1 in. In this furnace there are two very important improvements. The first consists in breaking the straight line of the long furnace. The working doors, *g g*, are placed so that no lateral work is performed; only drawing and pushing on an inclined hearth is required. The ore is introduced through the hopper, *n*, on the upper hearth, *a*, spread equally, and after an hour's time drawn at *g″* and pushed from *g′* upon the second inclined hearth, *b*, and from this upon the third, *c*, in the same way. No stirring is required unless very difficult ore is under treatment. The moving of the ore from one end of the furnace to the other is generally sufficient. The necessary heat is kept up by two or three fire-places, *h h*. The gases pass through the flue, *m*, above the roof into the dust chamber, *d*, and escape through the chimney, *e*. The arrangement of having the working doors at the end of the long sides enables the roaster to do a great deal more work than if tired out by the old way of moving the ore toward the fire-bridge.

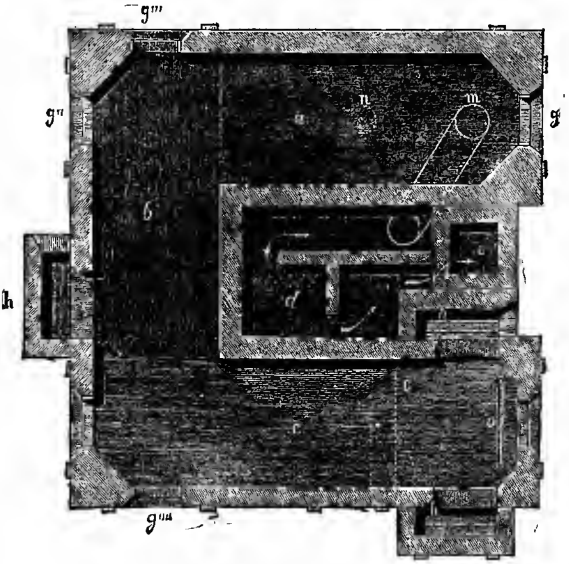


FIG. 104.—KÜSTEL'S ROASTING FURNACE.

The other improvement is the chloridizing chamber, K, shown in Fig. 105, the purpose of which is to shorten the time of roasting. It has been ascertained that the ore at rest in a red-hot condition continues to be chloridized when drawn out of the furnace. The ore falls through the opening, *o*, into the chamber, and remains there red-hot for two or four hours as may be required. Chlorine and volatile chloride metals, which are evolved, pass into the furnace and continue to chloridize the ore all along the furnace. In case there is no sulphur in the ore sulphurous acid gas can be introduced through the pipe, *l*, simply by burning sulphur. The sulphurous gas is transformed into sulphuric acid and liberates the chlorine from the salt.

The inventor claims that a furnace of this kind, by the aid of the chamber, K, can put through from 15 to 20 tons of ore in 24 hours, employing two shifts of three men each, consuming less than one-fifth of a cord of wood per ton of ore.

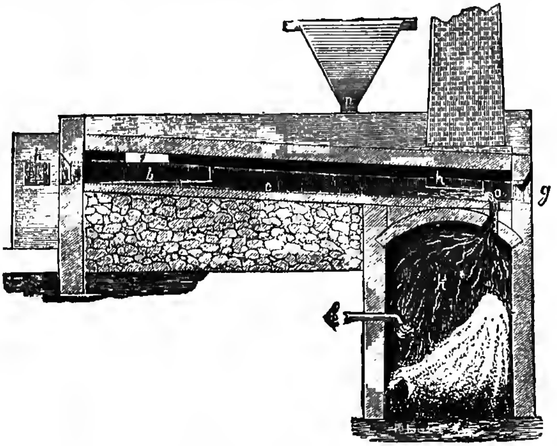


FIG. 105.—KÜSTEL'S ROASTING FURNACE.

**Mechanical Roasting Furnaces.**—It is hardly probable that any of the shelf furnaces already described will find any practical application in the mines, but with certain modifications, so as to allow of a large output, a preliminary roasting might be effected with them (especially if the manufacture of sulphuric acid could be associated with the metallurgical operations). They have the drawback of requiring manual labour, and preference has therefore been given to mechanical roasters, which do their work automatically. Among those which give satisfactory results is the Spence furnace, which is capable of an output of 10 tons daily.

**Bruckner's Cylinder.**—Bruckner's revolving cylinder for roasting ores is used not only in the chloridizing roasting of silver ores, but also in the oxidizing roasting of iron pyrites which contain gold.

Fig. 106 is an elevation of the cylinder in perspective; Fig. 107 is a longitudinal, and Fig. 108 a transverse section. The exterior of the cylinder is a shell of boiler iron, 12 ft. long by 5 ft. 6 in. in diameter. The ends are partially closed with similar material, leaving in the centre a circular opening about 2 ft. in diameter, bounded by a flange projecting several inches. Upon one side is placed an opening closed by a hinged door. Upon the outside of the cylinder are bolted three bands, as shown in Fig. 106, in which the section of the first is square, and that of the third semicircular; the second or middle band is a strong spur gear. Passing through the cylinder are six pipes parallel to one another, in a plane at an angle of 15° to the axis of the cylinder; these pipes also lie in this plane at an angle of from 30° to 35° to the longitudinal axis of the plane, as shown in Fig. 107, where the internal arrangement of the cylinder is seen, a perforated diaphragm being formed through part of the cylinder by means of perforated plates placed between the pipes, the plates being held in place by longitudinal grooves upon these pipes.

The entire cylinder is lined with brick (common building bricks have been found to answer the purpose very well), the bricks being placed in the following manner:—The entire side of the cylinder is covered with one layer, laid flatwise, thus forming a lining about 2½ in. thick; there is an additional layer extending from each end of the cylinder about 15 in. to the centre of where the nearest pipe passes out; then additional concentric layers are added thereon, until the circle is contracted down to the size of the opening in the end, which is also lined, and each layer falls short of the preceding one by about 2 in., thus giving the end linings a conical form, the entire lining being laid in a mortar of one part fire-clay, two

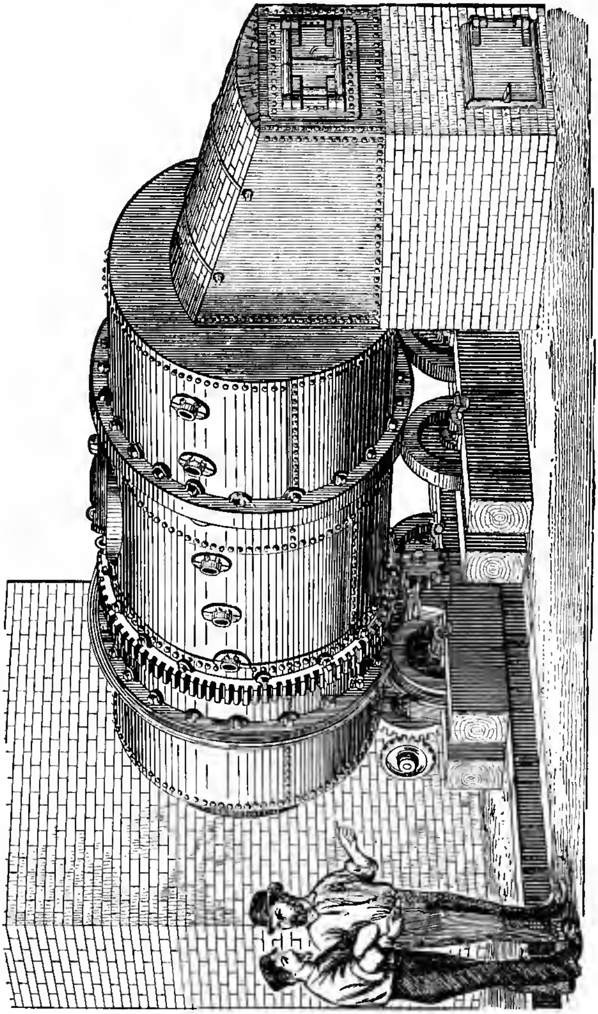


Fig. 106.—Bruckner's Roasting Cylinder.

parts pulverized old fire-brick and water, all thoroughly mixed

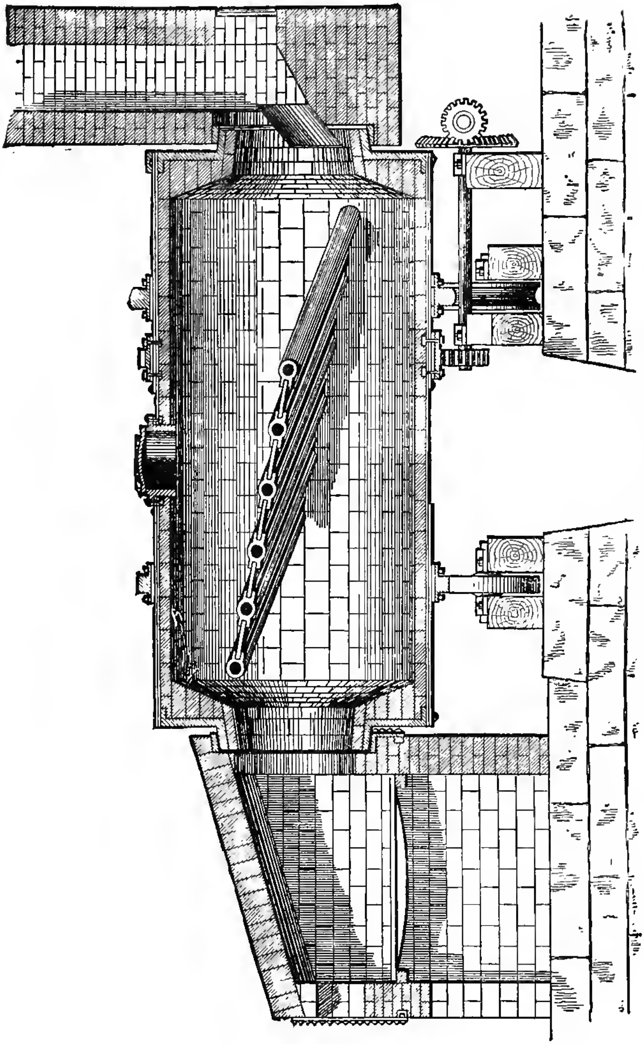


FIG. 107.—BRUCKNER'S CYLINDER. Longitudinal Section.

and beaten. The cylinder is supported upon four large friction rollers, two of which have a groove on their periphery, to loosely receive the semicircular band, and thus holding the cylinder longitudinally in place. The other two friction rollers are made without a groove, and bear upon the square band, thus accommodating themselves to the expansion and contraction of the cylinder, or any irregularities of form, all of which are seen in Fig. 106. Rotary motion is given to the cylinder by means of a pinion placed under the cylinder and gearing into the spur gear-band. Upon the other end of the pinion shaft are placed two bevel wheels, into which gear two match wheels, which latter are loose upon the driving shaft, standing at right angles to the pinion shaft. Either of those wheels can be attached to the driving shaft, thus communicating the speed of revolution of one or the other of the bevel gear as may be desired.

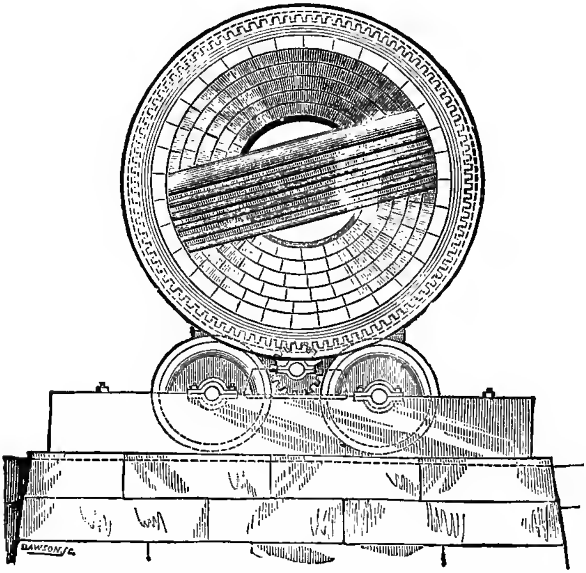


Fig. 108.—Bruckner's Cylinder. Transversal Section.

Inasmuch as by wear or settling the axis of the cylinder may possibly be thrown out of proper line, the following means of adjustment are provided, but not shown in any of the figures, viz :—Each journal box of the friction rollers is held in position by adjusting screws, by which it can be moved horizontally to or from the centre line of the machine, thus giving entire control of the lateral and perpendicular adjustment of the cylinder which they support.

The circular flange of one end of the cylinder loosely projects into a fire-box, best seen in section to the left of Fig. 107. The other end projects into an opening communicating with dust chambers and a chimney. There is placed in the bottom of the flue a shoe projecting into the cylinder, which catches such dust as may fall back, and returns it into the cylinder in lieu of allowing it to escape through the crevice between the cylinder flange and opening into the flue. A door is placed in the flue opposite the opening, through which the interior of the cylinder and its contents can be readily examined at any time.

Bruckner's cylinder has been found to give excellent results in roasting the compound auriferous pyritic ores to be treated by the Plattner process, in which case a small quantity of charcoal is subsequently introduced to the charge, so as to facilitate the decomposition of the resultant sulphate of copper.

**The Ottokar-Hoffman Furnace.**—Mr. Hoffman has also invented an improvement on the ordinary revolving cylindrical furnaces, which can be used to advantage in oxidizing roasting of auriferous ores, as well as for chloridizing roasting of silver ores. In my work on the "Metallurgy of Silver" (p. 173) it will be seen that Mr. Aaron has used a furnace of this construction for roasting the Silver King ores. His improvement consists in combining with a revolving cylinder-furnace two fireplaces, one at each end, and connecting both with the furnace, and also with an escape flue. Suitable dampers are fitted between each fireplace and the escape flue, so that when a fire is built in one fireplace its damper may be closed and the flame directed through the furnace, and thence through the opposite fireplace and its flue to the chimney. After the ore at the end of the cylinder nearest the first fireplace is properly roasted and chloridized a fire is built in the other fireplace and the first one suffered to go down. The damper of the first one is opened, while that of the second one is closed, so that the direction of the heat is changed, and the ore nearest this second fire is subjected to the highest temperature. This enables the operator to roast and chloridize the ore quickly without exposing the ore at one end to an injurious temperature in order to heat that at the opposite end to a sufficient temperature. Fig. 109 is a longitudinal section of the fireplaces and flues.

The revolving roasting-furnace, as ordinarily constructed, consists of an iron cylinder, lined on the inside with bricks, and provided on one end with a fireplace, and on the other with a flue leading to the chimney. The shape of some of these revolving furnaces is prismatic, but in construction they do not differ materially from the cylinder-furnace. There are on the mantle of the cylinder four doors for charging and discharging. After the furnace is charged with ore and the doors are closed, the cylinder is set into a slow revolving motion, while the flames and gases pass in the direction from the fireplace to the flue, and through it into the chimney.

It is apparent that that part of the ore which is nearest to the fire will be exposed to a higher temperature than that on the farther end; and as the ore, by the revolving motion of the furnace, is mixed and turned, but does not change its respective place in the furnace, it is further apparent that no uniform roasting can be obtained, at least not without causing an unnecessary loss of silver by volatilization.

That part of the ore which is nearest to the fire is perfectly roasted and chloridized, while the ore at the other end is from five to ten and more per cent. less chloridized. In order to bring chlorination up, the temperature should be increased and roasting continued for a considerable time longer, which increased and continued heat disposes silver to volatilize. The same ore will lose less silver if the chloridizing roasting is completed in a short time than it will in a long period, and, again, will lose less if the chloridizing roasting is performed at a lower heat; but as these revolving furnaces are constructed (especially if long cylinders are used—say sixteen feet long), the metallurgist encounters great difficulties in treating the charge of ore in the furnace to the best advantage, and is not able to obtain sufficiently satisfactory results if it contains a great deal of antimony, and bakes easily, as the construction of the furnace does not give him the facilities to subject the ore to a uniform heat.

Antimonious ores have to be roasted at a very low heat; but if the fire is kept low enough for the ore nearest to it, the ore at the farther end of the furnace will not have heat enough to roast; and, again, if the fire is kept strong enough for the farther part, the ore nearest to the fire will commence to cake and sustain a heavy loss in silver.

To avoid these difficulties, and to enable the roaster to roast the ore at the required uniform temperature, an improvement in the construction of the revolving roasting-furnaces has been introduced by Mr. Hoffman. Opposite to the fireplace, and on the end of the cylinder where in the old construction the flue is, a second fireplace is erected, so that the furnace, A, is provided with two fireplaces, *b* and *c*, on each end one. Each fireplace is provided with a flue and damper, *d* and *a*, leading to the main flue, *e*, and chimney. It will be seen that by attaching the flue directly to the fireplace, and by shutting the ash-pit and fire-door and raising the damper in the flue during the working of the furnace, the firebox serves alternately as fireplace or flue, according to the way the flame is directed.

To bring the furnace into operation, after charging it with ore the fire is built in the fireplace, *c*, and the damper, *a*, closed. The flame and gases traverse the furnace, A, and pass through the opposite fireplace, *b*, and the flue into the main flue, *e*. After a lapse of an hour or more, according to the character of the ore, a fire is built in the opposite fireplace, *b*, the damper, *a*, opened, and *d* closed. Flame and gases are now forced to pass through the furnace in the opposite direction. This changing of the fire is kept up at regular intervals during the whole time of roasting. While one fire is in operation the other is allowed to go down.

With these double fireplaces and flue arrangements, the ore in the furnace is subjected in both halves to the required roasting temperature, thus obtaining a uniform roasting throughout the whole furnace without the disadvantage of burning one half too much, in order to throw enough heat on the other. The construction of the furnace also permits the roasting of certain classes of ores, which bake easily in an ordinary revolving cylinder, and also prevents the dust from being carried by the draft into the flue during the charging of the furnace.

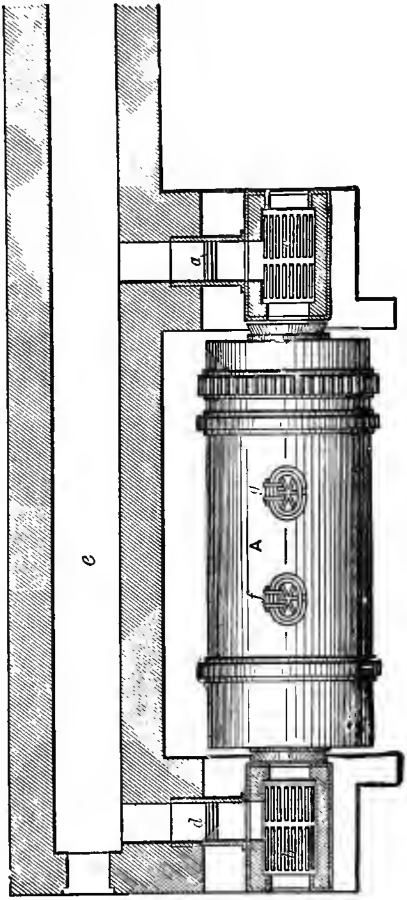


FIG. 109.—THE OTTOKAR-HOFFMAN FURNACE.

In charging ordinary furnaces the doors, *f* and *g*, are opened and placed under the outlets of a hopper. The ore falls in a shower into the furnace, and the draught passing through the furnace carries off considerable valuable dust into the flue. To avoid this, both dampers, *d* and *a*, are left open during the charging of the cylinder, and the draught passes through each fireplace and flue into the main flue independently of the furnace, through which no draught at all is passing.

There is one inconvenience with this furnace,—namely, that when the draught is carried over the fireplace filled with incandescent fuel, some ore dust must necessarily fall on to the fuel and cause clinkers, which have to be removed, and which hardly pay to treat, thereby causing a certain loss of metal.

**The Spence Roasting Furnace.**—In this furnace the roasting of the ores is effected on a series of beds arranged one above the other, and communicating through alternate passages or openings at the end of the beds. A travelling or reciprocating rake is also arranged to propel and agitate the ore on each bed, and to discharge the ore from one bed to the other in succession. Figs. 110—115 illustrate the arrangement.

Fig. 111 shows an exterior side view of the furnace; Fig. 112 a plan view thereof; and Fig. 113 a cross-section of one-half of a double furnace, the other half being shown in outside view; but as the two halves are mere repetitions of each other, it will only be necessary to describe the operations of one of them. Fig. 110 shows a longitudinal section of the furnace-beds, and Figs. 114 and 115 detached views of the stirring and raking instruments.

The construction of the furnace-beds will be seen on reference to Fig. 113. At 1 are the walls of the furnace, in which are fixed projecting fire-clay slabs, 2. Upon these are placed tiles, 3, reaching from one side transversely to the other, and each of convenient length, when put together longitudinally, to make the whole length of the several furnace-beds one above another, so that each of the said beds is made up by one tile only transversely.

Referring now to Fig. 110, the several beds are shown at 3, $3^a$, $3^b$, $3^c$; but each one is not continued unbroken to the end walls of the furnace, there being alternate openings left at 4, 5, 6, 7, and for the present, taking the operation generally, the material, in its ground state, is delivered at H on to the floor, 3, where advancing rakes or ploughs stir it, and subsequently carry forward a portion of the said material and deliver it through the opening, 4, on to the second bed, $3^a$.

The teeth of the rakes are formed of a triangular section, as shown in the enlarged detached view in Fig. 115, the apex of the triangle being in the direction of the motion of such rake longitudinally from end to end of the furnace, the flat sides of the teeth of the rake being in the direction in which it is desired to traverse the ore along the bed of the furnace. When the rake is advancing in the direction of the pointed part of the teeth of the rake the ore will be raked or turned over only; but when the rakes are being traversed in an opposite direction to that previously described a certain quantity of the ore will be carried by the flat side of the teeth of the rake along the floor of the furnace.

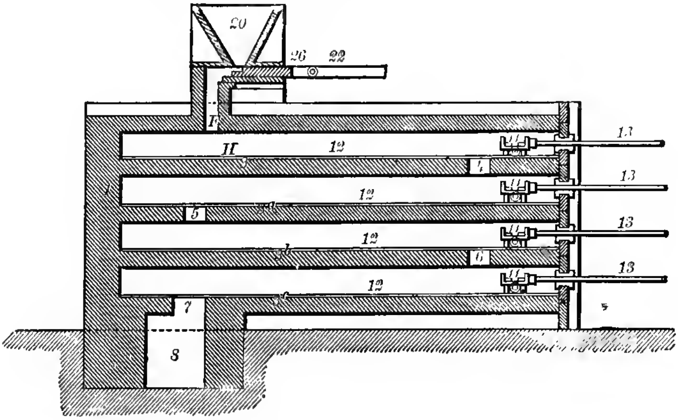


FIG. 110.—SPENCE FURNACE. Longitudinal Section.

The ground material being delivered to the floor, 3, at a point, H, the advancing rakes or ploughs, by means of the angular side of their teeth, stir it, and subsequently, by the flat sides of the said teeth, carry forward a portion of the said material and deliver it through the opening 4 on to the second bed. 3$^a$, where the same operations take place, the material now passing down the opening, 5, on to the bed, 3$^b$, and so through

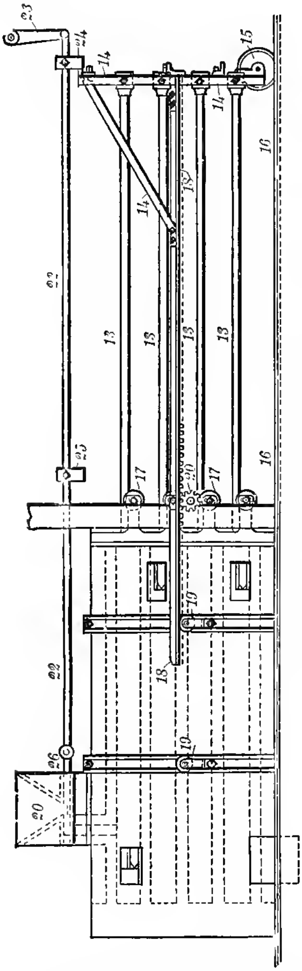


Fig. 111.—Spence Furnace. Side View.

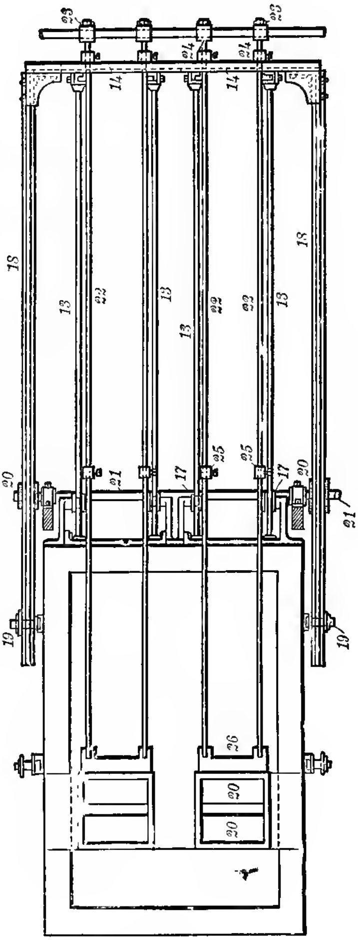


Fig. 112.—Spence Furnace. Plan.

any number of a required series of beds, until it is at last discharged through the opening, 7, into the receptacle, 8. As the alternate openings in the successive beds are on alternate ends of the furnace, the stirring and conveying instruments must be reversed as regards their faces in succeeding beds, whereby the whole series, by travelling in one direction alternately on each bed, stir and deliver the material successively until it reaches the receptacle, 8.

The teeth of the rakes are mounted in angle-iron bars, 11, provided with rollers, 11x, which run upon rails, 12, carried by the projecting supports, 2. To each of these angle-bars are connected rods, 13, attached at their other ends to a frame or carriage, 14, provided with wheels, 15, which run upon rails, 16, on the floor, the said rods being supported and guided by grooved pulleys, 17.

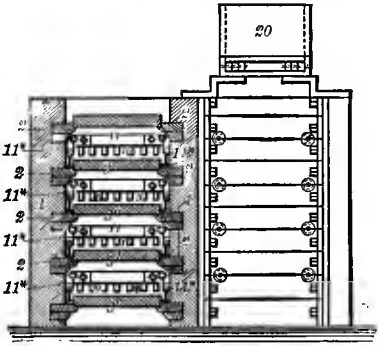
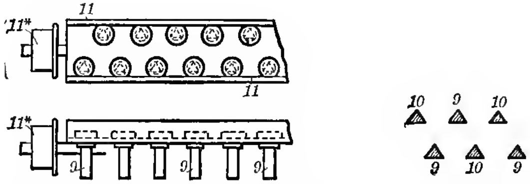


Fig. 113,—Cross Section of Double Spence Furnace.

To the carriage, 14, are fixed toothed racks, 18, situated outside the furnace, and supported at their outward ends by rollers, 19, and in gear with these racks are pinions, 20, on a shaft, 21, driven by the motive power. Motion being communicated to the shaft, 21, the pinions, 20, cause the racks, 18, to traverse the frame, 14, which, as stated, travels on the rails, 16, and thus the rods, 13, are caused to traverse the rakes or conveyers along the several beds of the furnace.

According to the positions shown in the drawings, the carriage, 14, is in its outward or nearly outward position, and the flat ends of the instruments will have delivered a certain amount of material through the opening, 4, on to the bed, $3^a$, the same operation having taken place with regard to the opening, 6, and bed, $3^c$. The carriage now running inward, the sharp points of the ploughs will simply stir the material on the beds 3, $3^b$, while the blunt ends on the floors, $3^a$ will deliver an amount of material through the opening, 5, on to the bed, $3^b$, and at the same time the instruments on the floor $3^c$ will pass an amount of completely calcined material into the receptacle, 8, to be removed at pleasure, and thus the alternate stirrings and deliveries take place at each forward and backward movement of the carriage.



Figs. 114 and 115.—Spence Furnace. Stirring and Raking Instruments.

The feeding apparatus is shown in Fig. 110. At F is a channel leading to the top-floor, 3, and above this channel is a hopper, 20, into which the ground material is from time to time fed. The bottom of this hopper is provided with a sliding plate, 26, formed at its inward end with a ledge, as seen in Fig. 110. This plate is connected to rods, 22, swung upon arms, 23, and each having two stops, 24, 25. According to the position shown, the material rests upon the ledge of the plate, 26, which, when the carriage runs in, is pushed forward by its arrival in contact with the stops, 24, and this action delivers a certain amount of material through the channel, F. On the return motion of the carriage it arrives in contact with the stops, 25, to shift the plate, 26, back, and so on for each traverse. Instead of the plate, 26, there may be a ringed bottom to the hopper or similarly-formed part. The drawings show a double furnace, and the operating parts are the same ; but it may be single.

The shaft, 21, is connected by means of suitable gearing to any source of motive power, so that it may be rotated first in one direction and then in the other, and thereby traverse the rakes alternately from one end of the furnace to the other.

The rakes may move continuously, but it is better for them to remain stationary periodically in the position shown in the drawings, as they are then clear of the material in the furnace, and also out of the direct action of the heat of the furnace, thereby suffering less injury from corrosion.

The furnace, as above described, does not require the application of any external heat. On starting it is brought to the required temperature by the introduction of burning wood or other suitable fuel, after which the temperature is self-sustained by the combustion of the material under operation. By the use of this furnace the whole, or nearly the whole, of the copper, in cupreous pyrites or other sulphides, is converted into soluble sulphate of copper, which may be placed in suitable vessels and subjected to the action of water for obtaining sulphate of copper in solution, which may be precipitated as usual, or manufactured into pure sulphate of copper by removing the sulphate of iron which always accompanies it.

**The McDougall Furnace.**—This apparatus consists of a series of superposed chambers or floors furnished with rakes or agitators and feeders, by means of which the materials under treatment are fed continuously through the series of chambers whilst being submitted to the burning, calcining, or roasting process.

As will be seen in Fig. 116, the rakes or agitators and feeders, *b*, are fixed to a main shaft, B, passing vertically through the chambers, A, which are provided with openings or passages, *a*, from chamber to chamber, alternately at the centre and at the side, so that the materials are passed by the rakes across each floor and from chamber to chamber until finally delivered out of the furnace by the outlet at C.

As it is sometimes desirable to dry the materials before they are fed into the furnace a floor may be provided outside the roof of the top chamber, as shown at D, and this floor may be furnished with a rake, or agitator, or feeder, $b^1$ (or two or more of them), so that the material being fed on to the floor, D, can be dried thereon, and then, by the operation of the rake, fed into the top chamber.

The vertical shaft may be rotated by any convenient gearing, and connections may be made therefrom for operating charging pistons for pushing the material from the hopper or hoppers on to the drying floor or into the chambers. Each chamber may be provided with manholes for access thereto when required.

The shaft and rakes, when made of cast iron and exposed to great heat, are liable to warp and break when strained. To obviate this they are made hollow and provided with wrought-metal pipes fixed in their interiors, forming an inner wrought-metal lining to the cast metal. In order to replace the shaft without removing the floors of the chambers, the shaft at each place where the rakes are to be fixed is provided with shoulders, and the rakes with a forked inner end, which embraces the shaft. A pin or cutter is passed through the forked ends of the rake to secure it to the shaft, and by withdrawing this cutter the rakes may be readily removed from the shaft, which can then be withdrawn from the furnace for renewal or repair.

In order that the shaft may be readily removed for repairs or other purpose after the rakes are removed (without the necessity for breaking the floors) on each alternate floor where the material passes through the passage at the side, the central floor box, through which the shaft passes, is furnished with a flanged movable cover resting on a flange on the floor box and fitting round the shaft. The shaft can thus be readily removed through the floors and out at the top of the furnace.

A loose piece of hard steel may be fixed to the bottom of the shaft working on a loose steel disc, so that these loose pieces when worn can be replaced without necessitating the renewal of the shaft.

To adapt the furnace for use for treating various kinds of materials requiring submission to the heat of the furnace for a longer or shorter period, there are provided, in addition to the feeding hopper at the top, other hoppers communicating with floors at different levels through which the different materials may be fed at the same time, according to the number of the chambers through which they are required to pass.

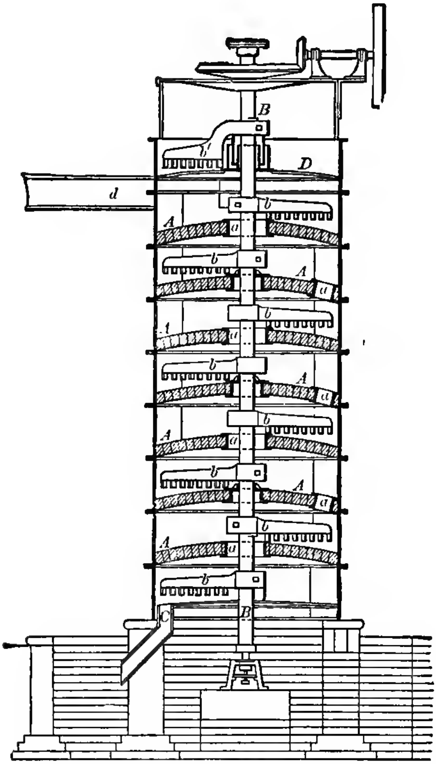


FIG. 116.—McDOUGALL FURNACE.

The same inventor has devised a form of furnace for use when it is desired to burn small pyrites or other suitable material so as to occasion the least possible amount of dust. Instead of the arrangement of floors and rakes above described a chamber is constructed with a long horizontal or approximately horizontal bed, constituting a series of chambers following each other, and provided with perpendicular shafts, rakes and feeders, so arranged as to act upon the material upon the bed and propel it from chamber to chamber. The teeth of the rakes and feeders are set at an angle to feed from their circumference towards the centre of each shaft, and each alternate shaft is made to move at the same speed and in the opposite direction to that of the shaft next to it. Any suitable number of shafts may be fixed in a line as may be required, and the chamber is made sufficiently wide for the revolution of the rakes, and sufficiently long for the number of shafts and rakes required.

The material to be burned is fed in from a suitable hopper at one end of the chamber, and delivered through a hole in the floor near to the last shaft leading to an inclined shoot. By this arrangement the material may be fed into the furnace and carried forward to the last rake in the chamber upon the same plane, and the air to support combustion is fed into the chamber above the level of the material on the floor either at one place or more as is found convenient. The steady and regular moving of the burning material is thus accomplished without its falling from chamber to chamber, and the chief cause of dust is obviated.

The outer casing of the furnace may be made of any suitable material, such, for example, as cast or wrought iron plates, with bearers bolted together, and completely cased inside with fire bricks or fire tiles; and in other respects the feeding of the material and the working of the furnace may be conducted as above described with regard to the vertical furnace.

When it is desired to burn certain kinds of sulphur compounds containing low percentages of sulphur, so that ignition may commence as early as possible on their entering the furnace, and that as much as possible of the heat may be retained within the furnace, one or more additional tiers of the horizontal series of chambers may be used with advantage; this arrangement in effect combining the vertical and horizontal methods hereinbefore described.

It will be understood that more than two series or tiers may be used if desired. The movement of the rakes is the same in each series. When two or more series or tiers are used, the material is fed to and carried forward in the top series to the end of that series, and is then fed through an opening into and is carried along to the opposite end of the next series; and so on with as many series as may be used; the material being finally discharged through a suitable outlet provided in the bottom of the end chamber of the lowest of the series.

Two or more rakes or agitators and feeders can be used in each chamber.

# CHAPTER VIII.

## *THE HYDRO-METALLURGY OF AURIFEROUS ORES: CHLORINATION.*

THE PLATTNER PROCESS OF CHLORINATION, according to Prof. Küstel—Conditions for Successful Chlorination—Assay of Gold Sulphurets—Chlorination Process for Sulphurets and Arseniurets—Roasting of the Sulphurets—Roasting Arsenical Pyrites—Operation of Roasting—Apparatus for Chlorination—Treatment of the Ore with Chlorine Gas—Lixiviation—Precipitation—Cost of Treatment by Plattner's Method—Arrangement of Chlorination Works—The Plattner Process at Plymouth Mine—At the Merrifield Mine—Gold and Silver Lixiviation Works.

**The Process of Chlorination.**—The Plattner process for the extraction of gold by means of chlorination, as improved by Mr. Deetkin, of Grass Valley (an account whereof, to which I am indebted, has been given by the late Professor Küstel *), may be described in general terms as follows:—

(1.) The auriferous concentrates from the stamping mill having been perfectly oxidized, are moistened with water and put lightly, by means of a sieve, into a wooden vat, coated with tar and rosin, and having a perforated false bottom upon which a filter is made, for which there are numerous ways. When filled, a close-fitting cover is placed on top.

(2.) Chlorine gas, produced by decomposing salt and peroxide of manganese with sulphuric acid, is introduced between the false and true bottoms, and made to permeate upwards through the ore mass. After the expiration of from fifteen to forty-eight hours, the gas is found to appear abundantly on the ore mass, and is then shut off, and the vat allowed to remain a few hours under the influence of the gas.

---

* In his "Treatise on Concentration of all kinds of Ores, including the Chlorination Process." San Francisco, 1868.

The cover being removed, pure water is added to fill the vat even with the top surface of the ore; the fine particles of gold, under the action of chlorine, have changed from metal to a soluble terchloride, and in this condition it is drawn off or leached out with water, fresh water being added until a test shows no trace of gold.

(3.) A prepared solution of sulphate of iron—the usual precipitant—is carefully added to this drawn-off solution, and the gold thrown down as a black or brownish precipitate; this is gathered, washed, and melted into ingots of nearly pure gold.

**Conditions for Successful Chlorination.**—The process is thus based upon the property of chlorine gas to transform metallic gold into soluble chloride of gold, and with some kinds of pyrites it is very perfect if well executed, but the following requirements have to be carefully observed:—

(1.) The gold must always be in a metallic state. Quartz, free from other earths and sulphurets, containing very fine gold, can be subjected to chlorination without other preparation than moistening with water, as described farther on. Sulphuretted ore requires a perfect roasting. The presence of lead makes a careful roasting necessary, commencing with a very low temperature. All metals, except gold, must be transformed into oxides. Sulphates are injurious.

(2.) The chlorine gas must be free from muriatic acid. From the generator the gas is forced through clear water, by which the muriatic acid is absorbed. The muriatic acid dissolves the oxides, and causes, when sulphides are present in consequence of defective roasting, the formation of sulphuretted hydrogen, by which soluble chloride of gold is precipitated. The muriatic acid dissolves also oxides of metals precipitated by the addition of sulphate of iron with the gold.

(3.) There must be no other substances in the charge which will unite with the free chlorine, since this would occasion a great waste of gas, and a failure in the desired separation of gold from other metals.

(4.) There must be no reaction in the mass treated with chlorine which will prematurely precipitate the gold before the final solution is obtained and drawn off.

(5.) In a word, it is required that all the gold, and, it possible, nothing else, shall be obtained in the final solution. Precipitation and melting then present no special difficulties.

Generally the concentrated sulphurets from the gold mills are subjected to the chlorination process; but also ores consisting of quartz and free gold, without admixture of other earths or sulphurets, can be treated by this process without any further preliminary treatment than reduction to powder.

**Assay of Gold Sulphurets by Chlorination.**—Of the finely pulverised sulphurets, from five to seven ounces or more are weighed out and roasted on a piece of sheet iron, the edges of which are bent up, and the inside coated over several times with clay water and then well dried.

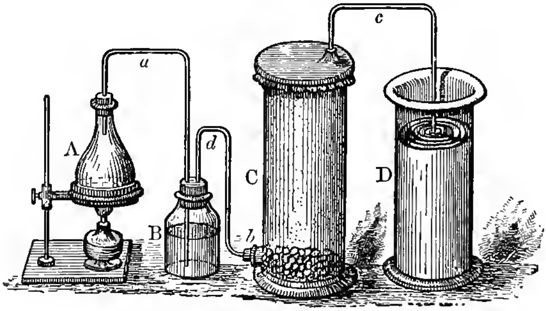


Fig. 117.—Chlorination Test Apparatus.

The roasting may be performed over charcoal or coke, in a small stove, or in a large black-lead crucible, through the bottom of which a hole is cut for the draft. The sulphurets must be stirred with an iron spatula until no sulphurous smell is perceptible; after which a tolerably strong red heat is applied. When cold, the sample must be ground over in an iron mortar and roasted once more at a red heat. When no sulphurous smell is observable the roasting may be considered as finished.

When cold, the roasted ore must be moistened with sufficient water in a dish or cup to make it of a loose or woolly consistence, in which state it is best suited for chlorination. If the roasting be perfect, the metallic gold can be dissolved in chlorinated water and extracted; but in case there be in the roasted stuff a small quantity of sulphurets or arseniurets not decomposed, it is more proper to use chlorine gas. The extraction is performed in the following way:—

Into a glass cylinder, C (Fig. 117), from 8 to 10 in. high, and 2½ in. wide, provided with a neck, *b*, near the bottom and about ¾ in. wide, is introduced a layer of small clean quartz fragments as shown in the drawing, and on this a thin layer of coarse and then of fine quartz sand. This quartz forms the filter; pulverised glass can also be employed. Over the quartz is then placed the roasted and moistened ore, as loosely as possible. The cylinder has a cover of wood or of india-rubber, in which a glass pipe is fixed, as shown at *c*. The longer end of this pipe dips into another cylinder, containing rolled stiff blotting-paper or pieces of blotting-paper, or shavings of wood, moistened with alcohol.

For the generation of the chlorine—say for 20 ozs. of roasted sulphurets—a glass vessel, A, is charged with 1 oz. of pulverised peroxide of manganese, 4 ozs. of muriatic acid, and 1 oz. of sulphuric acid, mixed with 1 oz. of water.

This mixture is shaken up, and the vessel placed on a cup or on a piece of sheet-iron covered with sand, with the neck corked. Through the cork a glass tube, *a*, is fixed so that it shall reach about 2 in. below the surface of the water contained in the bottle, B.

The chlorine gas is forced through the water, and by this means washed from muriatic acid. It is then conveyed through the pipe, *d*, into the ore receiver, C. The apparatus is more conveniently arranged if the three tubes, *a*, *d*, *c*, are each made in two parts, the first and last joining at *a* and *c*, by short india-rubber tubes. The tube, *d*, is connected below, near the cork, *b*, so that its short horizontal part will serve for the discharge of the lixivial when disconnected. As the chlorine is not only disagreeable but also injurious, it is necessary to cover all corks and joints with wheat flour dough. The above quantity of sulphurets (20 ozs.) requires about two hours' roasting, and a receiver, C, 6 in. high and 4 in. diameter.

The operation can be performed in a room without unpleasantness from the chlorine, because, with the alcohol in the cylinder, D, it forms chloral and muriatic acid, which are not disagreeable.

The mixture in the vessel, A, must be moderately heated at the beginning. The gas appears of a greenish colour, and soon fills the cylinder, C, by which time the finest gold is transformed into chloride; but this is not the case with the coarser particles, and with such gold as may be contained in sulphurets not decomposed. For this reason the development of the gas must continue for at least one hour more, before the chlorination is finished.

If auriferous pyrites is operated upon, the apparatus can be taken apart, commencing always with the tube, *a*, otherwise the water from B would be drawn over to A. If compound ores (galena, zinc, &c.), with free gold, are under treatment, the bottle, A, is separated from the apparatus, which remains undisturbed for at least fifteen hours. After this time, when all is taken apart, warm water is carefully introduced over the ore, in order to lixiviate the chloride of gold and other soluble salts. The cylinder must be a little inclined, so that no fluid shall remain on the bottom, and if the neck should be a little too high above the bottom, this can be filled level with the outlet with pitch.

The solution obtained must be mixed first with a few drops of muriatic acid; then a clear solution of sulphate of iron (green vitriol), in sufficient quantity, added, and stirred with a glass rod. The whole is allowed to stand (if possible, warm) till all the gold is precipitated and the solution clear. If a few drops of the sulphate solution should effect a slight precipitate when added to the clear fluid of the precipitated gold, it would prove that too little of the precipitant was used. The solution and precipitated gold are introduced into a filter, and washed with clean water: then the gold is dried with the filter, and washed with clean water; after which the gold is again dried with the filter in a porcelain cup or in an assayer's dry cup, and then burned, with free access of air. When cold, the ash is mixed with one or two hundred grains of test lead and cupelled under the muffle, and the gold button weighed. If gold quartz containing only fine gold and no sulphurets is under treatment, roasting is not required; but it is necessary to extract the metallic iron—which results from pulverising in an iron mortar—by means of a magnet. In place of the cylinder, C, if such cannot be obtained, a bottle, the bottom of which is cut off by means of a string, as shown in Fig. 118, will answer the purpose. Or a wooden box of a square section may be substituted for the cylinder, provided the inside be carefully coated with pitch or asphaltum cement.

FIG. 118. CHLORINATION VESSEL.

**Chlorination Process for Sulphurets and Arseniurets.**—From what has been said before, it will be seen that it is essential that each gold mill should be provided with proper concentrating apparatus, by which the worthless gangue may be separated from the heavier and valuable ore particles. Gold ores generally carry a certain percentage of sulphurets and arseniurets which are auriferous, and these are collected on the concentrators along with some fine gold which has escaped the battery and copper-plate amalgamation. If any silver is in the ore, the mineral particles are also collected on the concentrators; but by this it is not implied that concentration in every case will prove so perfect as to collect all that is valuable in the ore, and a careful watch should be kept over the tailings which leave the mill. Frequent assays of the tailings should be made, so as to guide the operator in his manipulations.

Besides iron pyrites and arsenical pyrites, the ores often contain copper pyrites, galena, zinc blende, tellurides, &c., and whatever is collected on the shaking tables or runners is designated by the generic name of *concentrates*. The nature of the ore particles constituting the concentrates will determine the results of the chlorination process and percentage of precious metal extracted from them.

It is a mistaken idea which induces certain operators to allow the concentrates to lie for any length of time exposed to atmospheric action, whereby a natural decomposition is expected to free some of the precious metals, and where nature is to effect a similar change to what would take place in the roasting furnace. To a certain extent this may be true: the concentrates do oxidize, but they clog together and form hard lumps which have to be re-pulverized, and it is therefore better to dry them at once and then to roast them.

There are not many establishments in existence where the ores are pulverized and treated at once by the Plattner process, without concentration, but since the introduction of the Mears process, which will be described presently, it has been found practicable to carry out the system with certain classes of free gold ores; and no doubt before long, with an economical production of chlorine, proper appliances, and rapidity of working, the amalgamation process (we shall find) will have to give way to chlorination wherever the gangue will permit of such treatment.

**Roasting of the Sulphurets.**—The presence of *galena* and of *copper sulphurets* does not interfere with the chlorination, provided the roasting be well performed. Gold of very low fineness, containing from forty to fifty per cent. of silver, will probably resist the chlorination, unless it is in the finest state of pulverisation, as the silver forms an impenetrable coating of chloride of silver, which can only be removed by a chlorinated solution of salt, which dissolves the chloride of silver and also that of gold. The formation of chloride of silver would prevent the thorough chlorination of the gold particles. The silver combined with the gold is not obtained by the chlorination process, but only the pure gold, for the reason that the amount of silver is generally too insignificant to admit of a remunerative extraction by a hot solution of salt. Coarse gold requires too much time to be converted into a chloride, so that the ore containing such gold is unfit for chlorination. Pan tailings of concentrated sulphurets allow an easy and perfect extraction of gold by this process.

The presence of *lime and talc* (*silicate of magnesia*) makes the chlorination of roasted ore very troublesome, or even impossible, but the addition of salt in roasting removes this difficulty. If the ore contains lead, the lead may be precipitated by sulphuric acid; it is then necessary to allow at least two hours for the precipitation. This is especially required if there is a great deal of lead in the ore and the roasting was performed with salt. After the precipitate is deposited on the bottom, the clear solution containing the gold must be drawn over into another precipitating vat, and the gold precipitated by sulphate of iron; or the gold is first precipitated by sulphuretted hydrogen, by which the lime, baryta, &c., which would be precipitated as a white powder by the sulphate of iron, remain dissolved in the solution.

The concentrated sulphurets must be subjected to roasting while damp, or at least before crusts and lumps are formed by oxidation, in consequence of lying too long. Pan tailings of sulphurets, however, must be dried, either artificially or by exposure to air and sun, and then pulverised by some means so fine as to admit of being sifted through a sieve of from twelve to fourteen meshes to the running inch. If these tailings were subjected directly to roasting they would bake into hard lumps, and the mass would be rendered unfit for chlorination. In all furnaces, at the beginning of roasting, at a low heat, the sulphur of the sulphurets is set free, and combining with the oxygen of the air produces volatile sulphurous acid, which is well known by its sharp, suffocating odour. The metals, by losing a part of their sulphur, are converted into oxides and sulphates, in which the iron sulphates are predominant. Sulphate of iron is the precipitant of the chloride of gold; its presence in the roasted ore is therefore very objectionable, and it is necessary to increase the heat by degrees, in order to decompose the sulphates and to form oxides.

**Roasting Arsenical Pyrites.**—Similar to this is the behaviour of auriferous arsenical pyrites when subjected to roasting. Arsenious acid escapes under the influence of heat and oxygen, while oxides and arseniates remain, the latter being further decomposed to oxides by increased heat. At the same time all metallic iron derived from grinding or stamping must be converted into an oxide.

After the sulphur and arsenic have been expelled, the gold remains in a free, metallic condition, and can be easily detected by pulverising and washing a small portion of the roasted stuff. While the formation of oxides and sulphates or arseniates is going on the gold is set free, and remains so during the whole process. When salt is used in roasting, the gold, according to Plattner, first forms chloride of gold ($Au\ Cl^3$) far below red heat. At 200° C. it loses a part of its chlorine, and at about 240° C. changes into a sub-chloride ($Au\ Cl$), which is not soluble in cold water. At a red heat it is converted into metallic gold.

There is hardly any loss of gold in roasting, and only in cases where the roasting is performed very rapidly are the finest gold particles carried off by the volatile products of roasting. In submitting, however, auriferous silver ores to a chloridising roasting with salt, great care has to be used, as during such roasting large portions of gold may become volatilized, as experiments in Australia and California have shown. Experiments made in the latter country have also shown that sulphurets which have been roasted for more than thirty hours without interruption have yielded 90 per cent. of the gold as compared with the fire assay. The tailings always contained some gold, so that very little or none of it was lost during the roasting.

**The Operation of Roasting.**—Roasting the sulphurets or arseniurets is very simple, the principal aim being a perfect dead roasting—that is, the expulsion of all sulphur. After the furnace has been heated for some hours, the sulphurets are introduced through a hopper above the roof into the furnace, and spread over the hearth. A furnace such as shown in

Fig. 96 takes one ton at a charge. One man is sufficient to attend the furnace. The fire is kept moderate, as the burning of the sulphur creates so much heat that nearly half of the sulphur contained is expelled thereby. On exposing a new surface of the mass (which in a short time becomes dark red hot) to the air, the burning sulphur having a bluish light can be seen distinctly. The hoe is principally used for stirring. It must be as light as possible, and is represented by Fig. 119. The plate, *a*, is made of boiler iron, 10 in. by 4½ and ⅛ of an in. thick, and riveted to the round iron rod, ⅝ of an in. in diameter, and 7 to 8 ft. long. One or two hoes are required on each side of the furnace.

The stirring is performed at intervals of about fifteen minutes. Whatever may be said about the uselessness of continual stirring, the fact must not be overlooked that sulphurets exposed to the air are sooner deprived of their sulphur than those which are nearest the hearth and excluded from undecomposed air. If this were not the case even a partial stirring would be useless. For that reason the intervals must be only of such length as is necessary for the relief of the roaster.

Fig. 119.—The Hoe.

Proper attention should be paid to the regulation of the draught, to keep the quantity of air which enters the furnace in proportion to the heat; and while the sulphur is burning it is better to leave some of the working doors open, so as to supply plenty of air to the burning sulphur, but taking care that the heat is not lowered too much.

There is no advantage in having a very high heat at first, and it is better on single-hearth furnaces to roast at the beginning with a low heat and finish with a high heat; but a proper regulation must be maintained so as to avoid the balling of the ore, or melting into lumps.

The ore should be changed from the front part of the furnace to the back, and *vice versâ*, as there is necessarily more heat near the fire bridge, and this precaution will give the whole ore mass a chance to be exposed to the higher temperature of the furnace.

In proportion as the oxidation of the sulphur and iron approaches completion the temperature decreases, and it is necessary to use more fuel, so as to keep the mass at a good red heat. It takes from twenty to twenty-four hours before the roasting of one charge may be considered finished. If in throwing up the sulphurets in the furnace by means of the hoe or shovel many brilliant sparks should appear, this would indicate that the roasting has been carelessly performed, and it must be continued until this appearance ceases.

In a double furnace (see *ante*), the heat in the lower hearth is always kept bright. One ton is roasted below, and about nine tons are spread on the upper long hearth. There are two roasters busy at the same time, one with the finishing, the other with the preparatory roasting, but they assist each other if needed. When one charge is drawn into the iron car below the hearth, the other load is shortly beforehand pushed down from the upper hearth, *c*, through the flue, *a*, and is now ready at *d* to be drawn into the furnace by means of hoes through the doors, *e e*. There is another door, *g*, behind the flue hearth for the same purpose. This charge is exposed to the preparatory roasting for about twenty-four hours on the upper hearth, and consists of a small part of undecomposed sulphurets and of oxides and sulphates. At a lively heat and with active stirring at intervals, all the base metals ought, after twelve hours' work, to be converted into oxides, and the charge taken out. In this way it is possible to obtain two tons of well-roasted sulphurets in twenty-four hours.

As soon as the charge from *c* is removed into the lower furnace, in which both roasters are engaged, the sulphurets from *h* must be removed to *c*, from *i* to *h*, and so on, until the end of the hearth at *k*, Fig. 103, is ready to receive a new charge of one ton through the charge hole in the roof. Although the temperature is very low, the whole mass on the upper hearth assumes a glowing condition in consequence of the burning sulphur. The roaster performs his stirring regularly from *c* to *k*, or the reverse, from both sides of the furnace. The hoes are made of $\frac{5}{8}$ round iron, and are 6 ft. long for the upper hearth. The final roasting is always performed nearest the fire-place.

**Test for Perfect Roast.**—As the object of roasting consists in expelling by heat all that can be volatilised or burnt in the ore—such as sulphur, arsenic or antimony—and to convert iron, copper, &c. into oxides, and leave the gold in the metallic state, it is necessary to make an assay to see if the mass has been brought to a dead roast.

If silver be present in the ore it will be as a sulphate, and it will not be advisable to push the heat so far as to decompose it, nor should the lead sulphate be decomposed.

The dead roast will have been reached when the roasting has effected the decomposition of the iron and copper sulphates, and this is ascertained by taking a sample and putting it into a glass of water. After the roasted pulp settles and the water clears above it, a few drops of a solution of potassium ferrocyanide may be added, which will produce a green or blue colouring if the roasting is not perfect. Iron sulphate, if left undecomposed in the ore, occasions a large loss of chlorine gas by absorption. If copper sulphate is present, the same test gives a yellowish precipitate, but the presence of iron sulphate makes a green colouration of the liquid, resulting from the blue of the iron.

On filtering some of the roasted ore and adding a few drops of ammonia to the filtrate, if all the copper sulphate is decomposed, the blue colouration peculiar to copper will be absent.

**Roasting with Salt.**—An addition of salt in roasting sulphurets is not injurious to chlorination, but it uselessly increases the expense if mixed with such sulphurets as do not require it. If the ore from which the concentrated sulphurets are obtained contains lime, calc-spar, talc, or heavy spar, it is necessary to introduce a chloridizing roasting, so as to convert these substances into chlorides, thereby effecting a saving of chlorine gas during the subsequent impregnation. In all other cases experience must decide. It is, however, also easily ascertained by an experiment, on a small scale, with two comparative assays, one of which is roasted with five or six per cent. of salt, the other without. The result will show which method is preferable. Lead and antimony do not allow of the use of salt in roasting.

It is sufficient to add five per cent. of salt in a pulverised condition. It is immaterial whether the mixture be made on the upper hearth or whether the salt be added when the charge is exposed to the finishing roasting, as there is no action of the salt, or very little, in the dark-red heat of the upper hearth; but if the roasting is performed in a single furnace, where the heat is less under control, it may be better to introduce the salt by means of scoops into the furnace five or six hours before the end of the operation. The salt must be well mixed with the ore.

The roasted ore, when in the car, is wheeled to a cooling place, which, if possible, must be so much below the floor of the roasting department that the contents of the car may be damped and spread out. After the roasted ore is cooled down it is removed to another compartment for the purpose of moistening. It is then subjected to the process of chlorination.

**Damping the Roasted Ore.**—The roasted ore cannot be treated at once with chlorine gas, for two reasons: first because the sulphurets, if dry, form a more condensed mass than if in a damp condition, and are therefore more obstructive to the ascending gas; and secondly, for the more important reason that the chlorine does not act as vigorously on dry as on damp ore. It is therefore indispensable to moisten the sulphurets after they are sufficiently cool.

For this purpose, the roasted charge—it may be of several tons—is spread in a compartment 8 or 10 ft. square, with sides 2 ft. high, of thin boiler iron, and the water conducted over it by means of a hose, or otherwise. It may require from 4 to 5 per cent. of water. The wetted surface is then turned over several times and mixed with the dry stuff beneath, till it appears nearly uniform. The moistened charge must not create the slightest dust, but, at the same time, the hand should remain dry and clean on handling it. A handful of it, pressed hard together, must form a lump which can be held in the fingers, but which falls into its former loose condition if handled. Should it appear too dry, more water, or in the opposite case, more dry ore, must be added and mixed with it.

Further on it will be seen that the vats in which the chlorination is performed contain a false bottom, on which a filter is prepared. After a charge has been removed, the filter contains a great deal of moisture, which is drawn into the newly moistened charge, rendering the lowest stratum too moist; this causes it to settle somewhat, and hinders the free access of the chlorine when introduced. To avoid this, some dry ore is spread over the filter—that is, the bottom—say 8 or 10 in. deep, and allowed to lie for six or eight hours. Should it be found too dry or too wet, it must be made right by the addition of moistened or dry stuff, as the case may be. A little experience will teach the operator to introduce the proper amount of roasted ore to take up the moisture. This charged part, as well as the moistened, must be subjected to *sifting*, an operation which is requisite for the separation of lumps and crusts formed during the roasting, and of other impurities which might drop in accidentally. Another reason for this sifting is the required loose condition of the ore in the vat, which is best obtained by passing it through a sieve. For this reason the sifting must be performed directly into the vat. The sieve is 12 to 14 in. by 25 in the clear, the sides 5 in. high. The sieve is sufficiently fine, if there are seven to eight meshes to the running inch.

**Apparatus for Chlorination.**—A vat into which the ore is sifted by pushing the sieve to and fro, either on two scantlings laid over the rim of the tub or suspended on four ropes, is represented in Fig. 120.

This figure shows a vertical cross section of a circular vat, 7 ft. in diameter, capable of holding three tons of roasted sulphurets. Above the bottom, *b*, is an empty space over the whole bottom, 1 in. high, formed by the false bottom, *a*, the boards of which are laid together, leaving about ⅛th of an in. space between them. Besides this, there are half-inch holes bored in it, from 10 to 12 in. apart. The boards are supported by short pieces, *c*, leaving sufficient space for the passage of the chlorine. Over the false bottom is spread first a layer of clean quartz, from 1½ to 2 in. in thickness. In default of quartz, another kind of rock will answer the purpose, provided there is no lime or talc in it—which would absorb a considerable amount of chlorine; and if notice is not taken of the character of the rock, the great consumption of chlorine might be supposed to be the consequence of defective roasting. Over the coarse layer smaller pieces are laid, and so on, decreasing in size, till a layer of sand covers the whole, forming thus a filter of from 4 to 5 in. in thickness. This filter remains always in the vat; the shovelling out of the residue, therefore, must be done carefully on approaching the filter bottom. There are two holes communicating with the space below the false bottom. One is

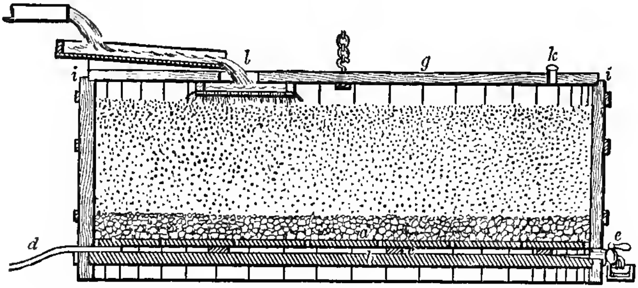


FIG. 120.—CHLORINATION VAT. Scale ⅜ in. = 1 ft.

for the reception of the lead pipe, *d*, by which the chlorine is introduced; the other is provided with a leaden cock, *e*, for the discharge of the lixivium. This discharge side of the vat stands ½ in. lower.

The wooden vat would absorb a great deal of the gold in solution, if the inside were not coated with some material which prevents the soaking in of the fluid. Mr. Deetken, who first introduced this process in California, uses one part pitch melted with one part of tar. This is a cheap and suitable mixture, and is applied hot by means of a brush. It is a matter of course, also, that the boards of the false bottom be coated carefully on all sides, as well as the whole of the inside.

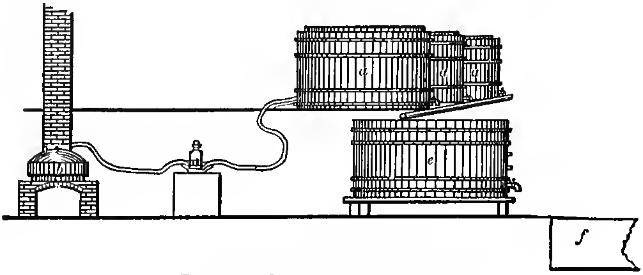


Fig. 121.—Chlorination Apparatus.

There are also vats or tubs 5 ft. in diameter and 3 ft. high, holding 2 tons of sulphurets. Three or more of them are arranged in one row, as shown in Fig. 121, *a a*. They are conveniently managed, and are to be preferred where small charges of custom ore are to be treated. The usual charge of a vat like that shown in Fig. 120 is 6 tons of roasted sulphurets.

Chlorination vats with a greater diameter are preferable to higher tubs of the same capacity, for the reason that a low column of sulphurets assumes a less dense condition; and also because of the greater cubic contents of the free space above the sulphurets, which is filled with chlorine, so that an accidentally greater consumption of the gas can be replaced.

The cover, *g*, Fig. 120, must fit as well as possible in the step of the vat side, but not too tight; the planks, however, have to be fitted together tightly with tongue and groove. For the purpose of lifting, there are generally three or four chains fastened to the cover. Ropes are of no use, as they are destroyed in a short time by the gases. After the ore is sifted into the vat, and spread evenly, it is ready for chlorination.

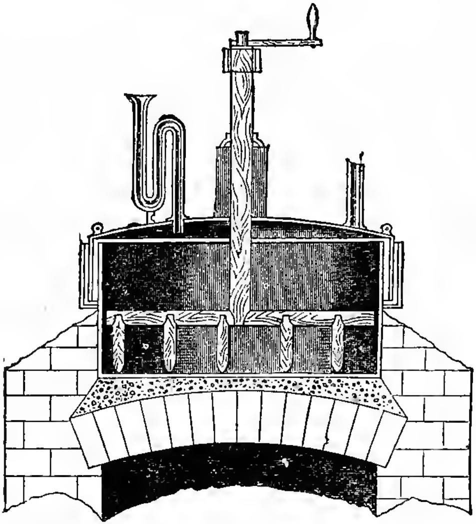
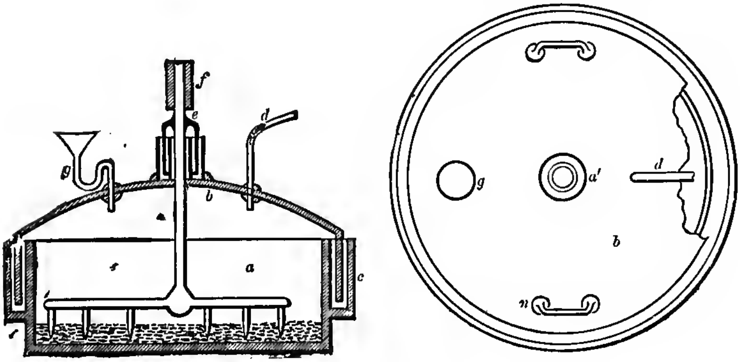


Fig. 122.—Chlorine Generator set in Furnace.

**Treatment of the Ore with Chlorine Gas.**—The chlorine is produced in a leaden vessel, such as is shown in Figs. 122 and 123. Fig. 123, A, is a vertical cross section, and Fig. 123, B, a top view with the cover on. The circular tub, *a*, has an outer ring, *c*, 6 in. deep, for the reception of the ring-shaped side of the cover, *b*. A similar small ring, *a'*, is on the top of the cover, which receives the collar, *e*, fastened to the leaden stirrer, *f*. There is also a short leaden pipe, *g*, bent in the shape of the letter **S**, through which the sulphuric acid is introduced, the outer end forming a funnel for this purpose. Another lead pipe, *d*, conveys the chlorine to the vat. The cover is taken off, and for a charge of three tons of roasted sulphurets the following materials are introduced: 30 lbs. of pulverised peroxide of manganese; 30 to 40 lbs. of common salt, according to quality; 75 lbs. of sulphuric acid, of 66 degrees; and 45 lbs. of water.



A. B.

Fig. 123.—Chlorine Generator. Scale ¾ inch = 1 foot.

The water, salt, and manganese are introduced first, and the generator covered. The bottom is made of 16-lb. and the sides and cover of 8-lb. sheet lead. The two rings, *c* and *a'*, are filled with water, and thereby the contents of the generator shut up air-tight, with the exception of the two lead pipes, *g* and *d*, of the cover. The gas generator stands over a small furnace, as represented in Fig. 122. The sulphuric acid is now introduced through the pipe *g*, Fig. 123, but not all at once Three bottles are generally sufficient to create so much heat that the development of the gas takes place in sufficient quantity. No fire is made as yet under the generator. The chlorine is not conveyed directly to the chlorination vats, but through a purifying apparatus, as represented in Fig. 124.

An ordinary wash basin, or some similar vessel, *a*, receives two ¾-in. lead pipes. One of them, *a*, conveys the chlorine from the generator, and is bent a little upward. The other is bent in the same way, but stands higher. Both ends are covered with a bottle, the bottom of which is cut off. There is sufficient clean water in the dish to stand one half or three-quarters of an inch above the mouth of the pipe, *a*, so that all the gas which enters the space in the bottom is forced through the water, which takes up the muriatic acid. The chlorine then passes through the pipe, *a″*, which is as long as may be required by the distance of the vat, enters the space below the false bottom, and gradually permeates the ore. The water through which the gas passes absorbs, if cold, about two and a half volumes of the chlorine, and is then saturated, but is still good for the purpose of taking up muriatic acid. The warmer the water is the less chlorine is absorbed. It is therefore wrong to introduce a continual stream of cold water into the wash basin, *a*, as is sometimes done, as a good deal of the chlorine is thereby lost. The water in the basin may be renewed once or twice during the operation with warm water.

This apparatus is not only for the absorption of muriatic acid—since, if a portion of the acid should happen to enter the vat, and, forming sulphuretted hydrogen, precipitate metallic gold, this would be converted again into a chloride in presence of abundant chlorine—but the apparatus is an indispensable indicator of the progress in the gas generator. The bottle, *b*, Fig. 124, must show a greenish gas, and the bubbling from the pipe, *a*, must be very lively. If this should not be the case another bottle of sulphuric acid must be introduced, and the addition continued as often as the development of the gas becomes weaker. After the last bottle has been used up a moderate fire must be made below the gas generator. The arch, as shown in Fig. 123, is very flat, and only two inches thick in the middle. Care must be taken to have a half-inch layer of sand over the arch. An open crack would cause the melting of the bottom of the lead vessel. It is also necessary to turn the stirrer, *f*, now and then, carefully, to prevent the caking of the ingredients.

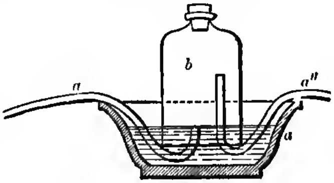


FIG. 124.—WASH BOTTLE.

The vat, after the ore has been sifted in, as before described, is left uncovered. It takes from three to six hours before the gas reaches the top of the charge. The progress can be easily watched by taking samples from underneath the surface. When the smell of the chlorine odour is perceived within a few inches of the surface of the charge, the cover is laid over the vat, and the edge, *i*, Fig. 120, all around the cover, luted with wheat flour dough. If there be any cracks in the cover they must be carefully pasted so that no chlorine can escape anywhere. The only opening not shut is a hole one inch in diameter, *k*; but as soon as the gas commences to escape, that is plugged up and secured with dough.

The chlorine is now permitted to operate on the gold for twelve or eighteen hours. If the vat holds five or six tons, the gas generator can be put in operation before the vat is entirely filled with ore, because the ascension of the chlorine is also slow. The pipe, *d*, is removed and the hole plugged up.

All the apparatus should be examined at intervals to see that there is no loss of gas, or at least whenever such loss may be perceived by the odour. For this purpose ammonia serves. A glass rod dipped into it and carried close around the place where the loss is indicated by the smell, will immediately give off white fumes when in contact with the chlorine, and show the place where it escapes. To prevent the drying of the dough around the cover it may be covered with strips of wet cloth.

For the production of chlorine the following formula is also used: 1 part manganese, 2 parts muriatic acid, 1 part sulphuric acid diluted with 1 part water.

**Lixiviation.**—After twelve, or if the sulphurets contain coarser gold, after fifteen to eighteen hours, the cover is taken off and water introduced. If it should happen in taking off the cover that no gas is found over the ore it will be advisable to shut the vat and to immediately impregnate the ore again with chlorine, as in nine cases out of ten the extraction of gold will fall short. This, however, with proper management, does not often occur.

Before taking off the cover from the vat, the plug can be removed from one of the vent-holes, and by holding a glass rod dipped in ammonia before the vent-hole, the evolution of fumes will indicate the presence of chlorine above the ore.

The water should flow in quickly, and in such a way as not to strike on one point, and thus produce a deep hole in the mass. The cock, *e*, is shut, and the water-flow continues until the surface of the charge is covered and no air bubbles appear. The water is then stopped, and the cock, *e*, opened. A small stream of water running into the vat must replace as much as flows through *e*, and so keep the surface of the ore always covered.

The respiration of the chlorine is injurious, and it is therefore advisable to avoid the inhaling of the gas as much as possible by leaving the room until the gas disappears. The best plan is an arrangement by which the gas can be carried out of the building; for instance, to put a movable wooden pipe 6 in. square about the proposed opening, *l*—or it may be more convenient to fix a lead pipe through the side of the vat, near the top, through which the gas will be forced by the entering water out of the building. In this case the water would have to be conveyed by india-rubber hose through the opening, *l*, of the same size as the hose. In order to prevent the stream of water making a hole into the ore, a perforated wooden distributer fixed to the cover, as shown in Fig. 120, would answer the purpose.

It is advisable to collect as much gold as possible in a small volume of water, and the best results are obtained by letting a shower of water percolate the ore. This, in travelling slowly through the ore mass, takes up nearly all the gold which permits of an easy precipitation. The gradual displacement of the surplus chlorine can be effected through a discharge pipe fixed in the top side of the vat and passing into another leaching vat, and this will allow of the operation being carried out while the cover is on the vat.

A trough below the cock, *e*, receives the solution and conveys it into the precipitating tub, *e*, Fig. 121. The trough must be lined with sheet-lead, avoiding sharp corners, or at least it must be well coated with tar and pitch in default of sheet lead. Great care must be taken to prevent the waste of the solution. Not a drop of it should be seen outside the trough.

**The Precipitating Vat** (*e*, Fig. 121) is a wooden tub like the chlorination vat (Fig. 120), but without a false bottom. The staves must fit together perfectly. A vat 4 ft. in diameter and 3 ft. high is sufficient to receive the solution of three tons of ore. Deetken's ten-foot vat, containing six tons of sulphurets, requires two precipitating vats, the one 5 ft. and the other 6 ft. in diameter, and each 2 ft. high. The vats ought to be lined with sheet lead. A better shape would be a rectangular box, having a half-round somewhat inclined sheet-lead bottom, as this would permit easier and better cleaning. In default of a leaden lining, the vats must be coated with a mixture of pitch and tar, otherwise the wood absorbs some of the gold solution. A better and smoother coating is obtained by the use of the so-called "asphaltum cement," which should be applied twice before it is ready for use, as the cement is too liquid for a single coating. A very smooth surface of the vat is important, else it is difficult to gather all the finely precipitated gold.

From time to time samples are taken in a clean white glass tumbler from the solution at the end of the trough, when it is observed whether an addition of a clear solution of sulphate of iron (green vitriol) causes a dark precipitate. If the solution after the addition of the precipitant should remain perfectly clear, the water supply in the chlorination vat must be stopped, and all the liquid contents of the vat permitted to flow into the precipitating vat.

**Precipitation.**—The precipitant for the gold is a solution of sulphate of iron. It is known also under the name of "copperas," or green vitriol, and is in light green crystals. Dissolved in water it generally makes a muddy solution, and deposits light sediment, which must not be disturbed in drawing off the clear solution from above it. For this purpose a leaden syphon will answer, but it is, however, better economy to prepare the precipitant fresh in the chlorination works. In a barrel or tub of about ten cubic feet contents are put 50 to 60 lbs. of pieces of old wrought iron; then five or six buckets of water are added, with 20 to 30 lbs. of sulphuric acid. This is prepared two or three days before the solution is required for use. One or two buckets of this solution must be poured into the precipitating vats before the gold solution is allowed to flow in, so that the precipitation may begin immediately. After this is added as much of the precipitant as is required, which can be ascertained by taking a sample out of the precipitating vat, passing it through filtering paper, and mixing it with the precipitant. If after a time the mixture should darken, a little more of the iron solution must be introduced into the precipitating vat. The precipitating gold requires some time before it is all deposited on the bottom. The fluid must appear perfectly clear before the water can be drawn off. Generally the mixture stands undisturbed all night. The upper plug of *e*, Fig. 121, is removed, and the clear liquid conveyed into another vessel, *f*, of sufficient capacity, till it is nearly all run out of the precipitating tub through all the plug holes. The discharge must be performed carefully, so that the flow shall be always clear.

It is better if several chlorinations have been performed before the gold is taken out, as with a large quantity of gold there is a less percentage of loss by wastage. The gold is taken out carefully by means of a dipper or scoop and put into a clean porcelain dish or enamelled vessel, and the rest is washed out through the lowest cock. It is well to run a jet of water over the sides and bottom, to wash off all the precipitated gold.

The gold obtained is then introduced into a paper filter, and subsequently dried in an iron porcelain vessel in a warm place or over fire. For melting purposes black-lead crucibles are less suitable than " Hessian " or clay crucibles; from the latter a purer gold is obtained. A little salt, some borax, and saltpetre are added as fluxes.

Zinc, antimony, and lead are not obstructive to the extraction of gold by chlorination, but it has not yet been determined by experience whether or not a great amount of galena would to any extent prevent chlorination. In all cases the presence of galena necessitates a good roasting and a strong finishing heat in order to decompose the sulphates as much as possible. If the roasting is not properly finished, the undecomposed sulphurets and sulphates will absorb a great deal of chlorine, and the chloride of lead and antimony formed will be carried into the precipitating vat. Both are precipitated as a white powder (as chlorides) by dilution with the leaching water. It often occurs that such ore is accompanied by lime or calcareous spar; in this case the lixivium will contain chloride of lime, which is precipitated with the gold as gypsum, unless precipitated beforehand and separated by the addition of sulphuric acid. Heavy spar will probably behave like calcareous spar, and be precipitated by sulphuric acid.

The question in regard to the utilisation of the chlorine which remains in the vat after the chlorination of the gold has been finished must be decided by a practical trial. A vat of 7 ft. diameter and 2 ft. high, when filled with roasted ore to within 6 in. of the top, leaves about 47 cubic feet of space for the chlorine, as the moistened stuff contains more than 50 per cent. of interstitial space. Provided the roasting be well performed in auriferous iron pyrites, the whole free space in the vat will be filled with chlorine after the process of chlorination is finished. This ought to be the case under all circumstances; and if the roasted stuff consumes more gas than usual, it must be replaced by an additional quantity. Having then two chlorination vats, the communication between them for the purpose of conveying the chlorine from one to the other is easily obtained by the lead pipes, each leading from the upper part of one to the empty space (*c*, Fig. 121) of the other, the pipes being provided with cocks. As soon as the water is admitted through the hose into the vat, the chlorine will escape through the lead pipe into the other already prepared vat. It will be necessary then to fix a glass tube, bent in the shape of a horse-shoe, into the upper part of the vat, so that the height of the water above the ore can be seen and regulated accordingly.

The chlorine transferred into the other vat will require a certain additional quantity from the generator.

**Precipitants for Gold.**—As before stated, sulphate of iron is the precipitant usually employed, but hydrogen sulphide answers fully as well. Where animal charcoal is used as the precipitant, there is a difficulty in the separation of the gold from the carbon, the only feasible way being to redissolve the metal by means of *aqua regia*, and again precipitate it by some one of the known reagents for that purpose.

Wood charcoal is a much better precipitant, as it is more easily disposed of by burning, the gold being recovered from the ashes by melting them with borax.

Sulphur dioxide (sulphurous acid) is an excellent precipitant for the gold, which it throws down in the metallic state in the form of a dark powder. It is easily produced by heating strong sulphuric acid and either charcoal or sulphur in an iron retort, the resulting gas being passed into the solution of gold. There are many other substances which throw gold down in a metallic state, like copper sulphide. The copper sulphide* is converted into sulphate, which dissolves, while the gold is deposited in the metallic state. The copper sulphate may be recovered from the liquid by precipitation by means of either hydrogen sulphide or an alkaline sulphide, though in the latter case it would be mixed with some free sulphur, which would then become mingled with the gold, but could be easily burned off.

The copper sulphide may be applied by stirring it into the

---

* Report of the State Mineralogist of California.

gold solution until a test shows that no gold remains dissolved; but a better way would be to let the gold solution flow slowly through a series of small filters containing the sulphide. When the copper sulphide in the first filter is almost entirely replaced by gold, that filter must be emptied, refilled with copper sulphide, and replaced as the last of the series, the former second becoming the first, and so on. The precipitated gold may be freed from remaining copper sulphide by digestion with some warm, strong gold solution. This method offers advantages in the collection of the gold, which is thrown down in a granular condition, and, when washed and heated, assumes the golden colour. Any remaining traces of copper sulphide become oxidized by heating, and may be removed by a little nitric acid, leaving pure gold for melting.

Mr. Aaron holds that gold enough is lost, in most works, by imperfect settling, to pay the cost of this method of precipitating.

In using iron sulphate for the precipitation, he found that after forty-eight hours' settling there remained gold in suspension to the amount of half a dollar for each ton of ore treated. The settling is promoted by the addition of some sulphuric acid to the liquid; also by repetition of the stirring about two hours after precipitation. When the ore contains copper, the liquid from the precipitation vats is conducted to other vats containing scrap iron which precipitates the copper. The cement copper thus obtained always contains gold, and this appears to have produced an impression that iron sulphate does not precipitate all the gold from solution. When sufficient iron sulphate is added, no gold remains dissolved; the gold found in the cement copper must, therefore, be that which had remained in suspense, and it indicates that, in case there is no copper to precipitate, the loss from this cause may be considerable.

Mr. Nelson E. Ferry recommends the addition of molasses to the leach when lime is present; one gallon of molasses to be dissolved in 30 or 40 gallons of water, and kept for use. The quantity to be used must be determined in each case by a laboratory test; if calcium sulphate comes down, either the molasses is in insufficient quantity, or it has not been thoroughly mixed. Examine by transmitted light. Avoid large excess of ferrous sulphate. If the gold comes down at first in a flocculent form it is of no moment: it will soon assume the usual form. The best results are got when the liquid is made slightly acid. The usual practice in such a case is to add sulphuric acid to the leach, and let it stand for a number of hours, then transfer the liquid to another vat, and precipitate the gold with iron sulphate; the gypsum crystallises on the side of the first vat. This method requires a double set of precipitating vats.

**Cost of Treatment by Plattner's Method.**—The expenses of working by this method are rather high, especially where the ordinary reverberating furnaces are still in use.

At a small 8-stamp mill in Amadar county, California, the superintendent informed me that the ore which is run from the battery over copper plates is concentrated, the sulphurets having an average value of £22 per ton. Their treatment cost £4 per ton, leaving £18 profit. The average proportion of sulphurets per ton of ore was 5 per cent., and the yield of gold obtained from the sulphurets about £1 per ton of ore.

**Chlorination Works.**—A site for them must be chosen with reference to the prevailing winds, the fall of the ground, and the ready supply of clean water. As the sulphuric acid and chlorine vapours are destructive to the machinery of the mine and mill, great care is necessary to select a site for the furnace from which they cannot be carried to the mill by the prevailing winds. The direction of the draught is also in that of the winds. For the proper and economical working of the process a fall of at least thirty to forty feet is required. The water supply, of at least thirty-five gallons per hour, is discharged into a wooden tank which will hold a twenty-four hours' run.

The concentrates are delivered at the ore floor of the chlorination works, on a level with the charging hopper of the upper furnace. They contain on an average six per cent. of moisture.

The general arrangement of works for chlorination of auriferous ores is shown in plan and elevation in Figs. 125 and 126. I is a Brückner cylinder, in which the ore undergoes a preliminary roasting to remove such precipitating agents as sulphur, antimony, and arsenic. The leaching vats are at A, the precipitating vats at B, trucks to remove spent ore at C, the gas generator at D, and the waste tub, where the water runs through sawdust, at E.

**Mills and Chlorination Works at Plymouth Mine, California.** — In an interesting account of the operations at these works, which was given by Mr. G. W. Small in a paper read before the American Institute of Mining Engineers, it was shown that if a mine is located near to centres of supplies where fuel, labour and chemicals can be cheaply obtained, the Plattner process can be carried out at a cost of about £2 per ton after the concentrates are delivered at the works. Few mines, however, are so centrally located as those he described, and the cost stated above—£4 per ton—should be taken as a *low average*. Mr. Small's report was as follows :—

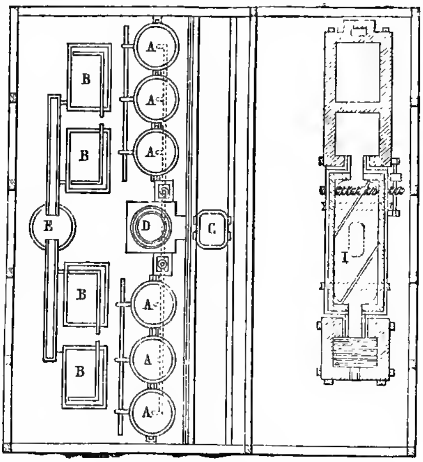


FIG. 125.—PLAN OF CHLORINATION WORKS.

"The ore, as it is raised from the mine, has an average assay-value of $11 per ton, chiefly in the form of free gold. All the ore goes directly to the stamp-mills, of which there are two. The older and larger mill contains sixteen batteries of five stamps each, with one Frue vanner to each battery. The new mill has eight batteries of five stamps and two Frues to each battery. The large mill is driven by Leffel turbine wheels, with a pressure of 80 feet, and a consumption of 600 miners' inches of water. The smaller mill is driven by "hurdy-gurdy" wheels, with a pressure of about 550 feet and a consumption of 150 inches of water.

"At both mills the tailings from the stamps pass over about 20 feet of plates on their way to the Frues. In each set of plates the first or upper one is copper, the rest are so-called silver plates. The bullion from the stamps is about 800 fine in gold and 200 in silver.

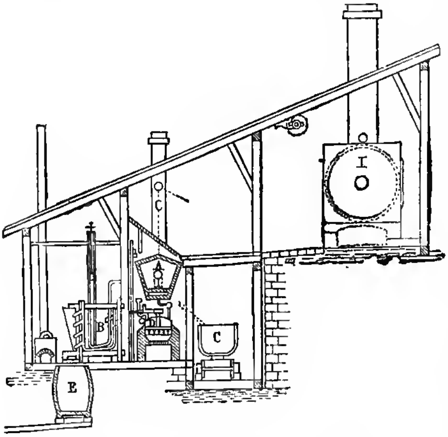


FIG. 126.—SECTION OF CHLORINATION WORKS.

"The concentrates from the Frues average from 1¼ to 1½ per cent. of the ore stamped. They very rarely exceed 2 per cent. I was unable to get the exact assay-value of the concentrates, but it is said to vary between $100 and $200 per ton. The concentrates are treated at the chlorination works at the rate of one hundred tons per month. The capacity of the works is somewhat greater than this, but as the supply of concentrates is limited it is not deemed advisable to work them up any faster. Care is taken to keep the concentrates always damp until they are put into the roasting furnace. If this is not done, a decomposition of the pyrites begins, forming lumps which do not roast, and which consequently cause a loss of gold in the residues from leaching.

"A *Fortschaufelungsofen* * is used for roasting. Its dimensions, including fire-box, are 12′ × 80′. The hearth is one continuous plane, but the charges, of which there are three in

---

* *Anglicè*, a long continuous reverberatory furnace.

the furnace at one time, are kept entirely separate. The furnace-men called the three compartments, the "drying," the "burning," and the "cooking" compartments. In the middle, or "burning" compartment, the ore is spread out very thin, and occupies about double the space of either of the other compartments.

"The furnace is worked by eight-hour shifts, one man on each shift, and one charge is drawn and a new one added in each shift. The charges weigh 2,400 pounds, and carry about 10 per cent. of moisture. The ore averages about 20 per cent. in sulphur, and just before the sulphur ceases flaming (in the second division of the furnace) 18 pounds, or ¾ per cent. of salt is added to the charge.

"The roasted ore from each shift is kept by itself on the cooling-floor until a tankful (about 4 tons) has accumulated from a single man's shift; then that lot is worked by itself. This enables the person in charge the better to control the roasting; for if only one lot out of the three is bad, it is presumable that the fault lies with the workman; but if all three are bad, the probabilities are that there has been a material change in the character of the ore, and the roasting process must be altered accordingly.

"The vats for chloridizing the roasted ore are 9 feet in diameter by 3 feet in height, and are four in number. They are slightly inclined, so that they will drain completely. The bottom of each tank is occupied by a filter about 6 inches thick, composed as follows: Light strips of three-quarter inch wood are first laid in the bottom of the tank at intervals of about one foot. Across these strips are laid six-inch boards, leaving cracks of an inch or more between the boards. On top of this loose floor are placed coarse lumps of quartz, and on top of this again finer quartz material, until a total depth of about five or six inches is obtained. Finally, this "sand-filter" is covered by another loose floor, the boards lying crosswise to the loose floor beneath, and pretty close together. This upper floor is intended merely to furnish a shovelling surface, so as to permit the removal of the leached ore from the tanks without disturbing the filter.

"The ore to be chloridized must be damp (about six per cent. moisture). The working test is, to take a handful of the ore and squeeze it, then open the hand, and if the lump immediately begins to crumble and fall apart (not run) the ore has the requisite amount of moisture. The damp ore is screened into the tanks, so that it will lie as loosely as possible and facilitate the penetration of the chlorine gas. A coarse screen of one-half inch mesh is used for this purpose. The tanks are only filled up to within about three inches of the top. This is to insure that the entire contents of the tank are covered by water in the subsequent leaching, otherwise there will be great difficulty in washing out all the soluble gold.

"As soon as the tanks are filled as stated, they are ready for the introduction of the chloiine gas. This is introduced into the bottom of the tank from two opposite sides, and is continued until ammonia held over the ore gives off dense fumes of ammonium chloride. This usually takes about four hours. When this point is reached, covers are placed on the tanks, and the cracks are luted with a mixture of leached ore, bran, and water. The gas-generators, of which there are two employed at one time in charging a tank, are allowed to work on until they are exhausted; then they are disconnected and the holes in the tank are plugged.

"The tank is usually charged with gas in the morning, and is left standing for two days. On the third day the ore is leached. The tank is first filled with water, and allowed to stand a few minutes so that the water may penetrate all the ore. If no more water is absorbed, the liquor is drawn off at the bottom, care being taken to keep the tank full of water during the entire operation, which takes from four to five hours. In charging the tank, a gunny-sack is laid on top of the ore, where the wash water is afterwards to be introduced, in order to better distribute the water in the tank and prevent its washing and packing the ore.

"The liquor from the leaching-vats is conducted to settling- or storage-tanks, and about 40 pounds of sulphuric acid (66° B.) is added. (Experience has shown this addition of acid to be advantageous in obtaining a clean product in the subsequent precipitation. The chemical reaction is, however, by no means clear.) It is usually allowed to stand for twenty-four hours, but two hours are quite sufficient. It is then run into precipitating-tanks, and the gold is precipitated by a solution of sulphate of iron. The iron solution is added until, after stirring, a further addition produces no purple colour. After the gold is precipitated it is allowed to stand two or, if convenient, three days to settle; then the supernatant liquor is drawn off with syphons into a second settling-tank, where any gold that may have been drawn off by the syphons has a second opportunity to settle. The liquor stands in this tank until it is necessary to run it off to make room for another charge. Very little gold is found in this tank, and it is therefore only cleaned out once during the year. In the meantime, fresh liquor has been run into the precipitating-tanks upon the gold already precipitated there. In this way the gold is allowed to accumulate until the semi-monthly clean-up. Except when it is necessary to have them open, the precipitating tanks are kept covered and locked.

"In making the clean-up, the supernatant liquor is syphoned off, the gold gathered up and placed in a filter of punched iron lined with a sheet of ordinary filter-paper, and washed with water until all the acid and iron salts are removed. It is then dried, melted in crucibles, and cast into bars.

"The works extract from 95 to 96 per cent. of the assay-value of the concentrated sulphides. Two men, on day-shift, attend to all the work of handling the ore after it is washed (the leaching, etc.). The head man receives $3, the other $2.50 per day. Owing to the limited amount of ore allotted to the works, only three tankfuls are leached every four days. The men, however, are employed steadily.

"The sulphate of iron is manufactured on the spot. For this purpose an ordinary wooden tank about 4 feet by 4½ feet, standing outside the building in the open air, is used. The tank is kept full of water and supplied with old scrap-iron *ad libitum*, and for each charge to be precipitated about 40 pounds of acid are added to the tank.

"The precipitating-tanks, which are of wood, are protected from the action of the acids by a coating of 'paraffin paint.'

"I append an itemized statement of the cost of handling the ore. The basis of figuring is 100 tons of ore per month of 30 days. Consumption of chemicals in the leaching department, 24 days in each month :

| | | |
|---|---|---|
| *Roasting* : | | |
| Three men, at $2·50 per day, for 30 days | $225 00 | |
| 1¾ cords wood at $4·25 ,, ,, | 223 13 | |
| 54 lbs. salt at ¾ cent ,, ,, | 12 15 | |
| | | $460 28 |
| *Generator:* The charge is manganese, 30 lbs., salt, 34 lbs., sulphuric acid, 60 lbs.; therefore, for two generators: | | |
| Manganese, 60 lbs. per day, 24 days, at $47 per ton | $33 84 | |
| Salt, 68 lbs. per day, 24 days, at $15 per ton | 12 24 | |
| Acid, 120 lbs. per day, 24 days, at $60 per ton | 86 40 | |
| | | 132 48 |
| Acid for settling-tanks (40 lbs.), and for sulphate of iron manufacture (40 lbs.), 24 days | | 57 60 |
| Wages of leachers, at $5·50, for 30 days | | 165 00 |
| Salary of foreman | | 125 00 |
| Total | | $940 36 |

Or, per ton of concentrates, $\$9\ 40\frac{3}{10}$, or £2 sterling."

**Chlorination Works at the Merrifield Mine.***—These works consist of a reverberatory furnace, 70 feet in length by 10 feet inside, capable of roasting three tons of concentrations in 24 hours, consuming three-quarters of a cord of wood to the ton. There are two chlorine generators, four chlorinating vats, three precipitating tubs for gold, three for silver. The chlorinating vats are 6 feet 8 inches in diameter by 2 feet 6 inches inside height, with a capacity of three tons of ore. The precipitating vats for gold are 6 feet in diameter and 3 feet deep. The leaching tubs for silver are of the same dimensions as the chlorinating vats. The covers are coated with red fire-proof paint, protecting the wood from the action of chlorine. The filters are perforated boards covered with burlaps. After the gold is leached out, the ore, if it contains silver, is transferred to the silver leaching tubs, where it is leached with "hypo" for the silver.

The chlorine gas generators are heated in a water bath, and

---

* From the Report of the State Mineralogist of California.

the exhausted charge is readily removed by means of a stream of water, through the spout in the side, which passes through the wall of the iron pan forming the water bath. The ore is exposed to the action of chlorine during 48 hours.

**Gold and Silver Lixiviation Works.**—A site should be chosen, if possible, on a sloping ground, where the gradient is such as to permit the different departments to be located on different levels. On level ground where these advantages cannot be secured, a building with several storeys stoutly framed must be erected, and the accompanying sketch (Fig. 127) will give a sectional elevation of works arranged as suggested.

The top floor, 1, 1, contains the storage tanks for holding the hypo solutions, and also the precipitant for base metals in case Russell's method is employed for the separation of lead. A A is the storage tank for the hypo solution. B B the storage tank for the sodium carbonate solution. The hypo solution is prepared by dissolving 100 pounds of sodium hyposulphite in 100 cubic feet of water, and this quantity is either diminished or increased according to the character of the ore. The roasted ore is brought in cars on a tramway X, after being hoisted by means of a lift to the floor level of department 2, and filled into the lixiviation tanks, C C, the hypo solution being admitted through the indiarubber tubes, N N, which are connected by means of the iron pipes, M M, with the storage tank, A A.

The lixiviation tanks, C C, have a false bottom, and the silver solution will flow out through P into the base metal precipitating tanks, D D. The lead is precipitated by the sodium carbonate solution flowing from B B through O O, and fed through the indiarubber pipes, Q Q. This third storey is the base metal department.

The precipitating vats, D D, are also wooden tubs, and to facilitate the drawing off of the liquor after precipitation, an indiarubber hose, J, is tightly drawn through the side in the tank and to its upper end is attached a wooden float, which causes the hose to draw the liquid from the surface without disturbing the precipitate at the bottom. When not in use, the float is tied to the side of the tank as shown at $J^1$.

When the base chlorides are first leached with water, the

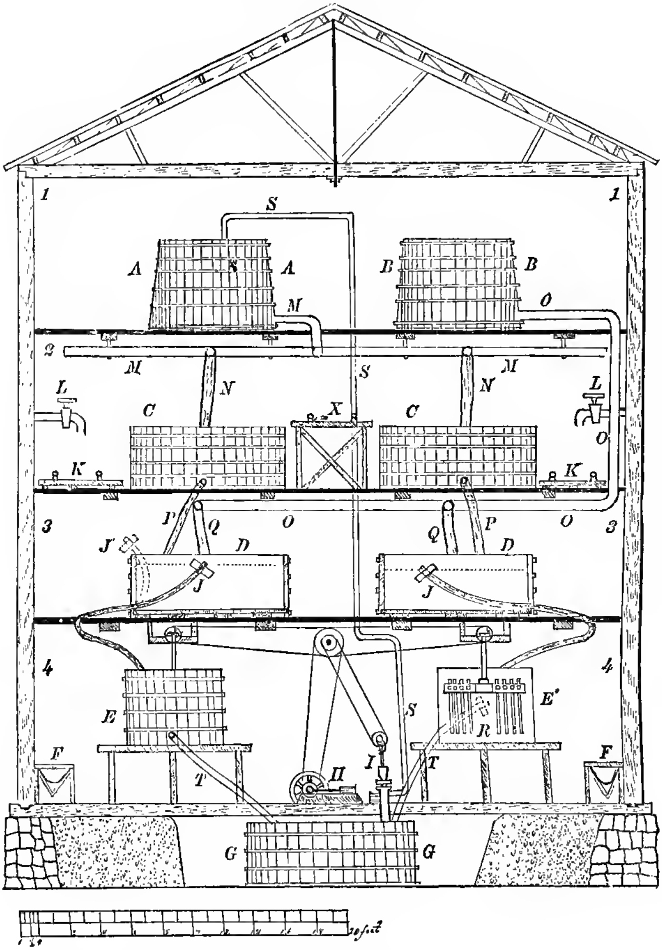


FIG. 127.—GOLD AND SILVER LIXIVIATION WORKS.

water is admitted into the tanks, C C, by means of the water taps, L L, and drawn off into the base metal precipitating tanks through P as before. When leaching for precious metals, the liquor is drawn by the same pipe, but elongated directly into the precious metal precipitating tanks, E E$^1$, on floor 4. For this purpose holes are made on the floor of 3, through which may pass the long indiarubber pipe reaching into the precipitating tubs, E. There the precipitant is added, and the liquor agitated by means of mechanical stirrers, as shown in R, which permits an easier settling of the precipitate. The liquor of hyposulphite is now drawn through the pipe T into the sump tank, G G, where the regenerated solution is pumped back, by means of the pump I, into A, through the pipe S S S. H is the engine furnishing motive power to pump and stirring apparatus.

When the ore is leached, the tailings from C are shovelled into cars running on the tramway, K K, on which they are taken to the hoist and lowered to the dumping ground. The leach vats, C, can also be arranged with a gate covering a large opening in the side whence the tailings can be sluiced out.

The precipitate from E is taken out and placed on the filter, F; and if silver sulphide is roasted in a small reverberatory furnace and then smelted according to directions given in the chapter on Assaying.

In treating gold ores not containing sufficient silver to pay for its extraction, all that will be required are the two storeys, 2 and 4—namely, the leaching tanks and the precipitating tanks—and stirring or pumping machinery will not be required. The operation is generally carried on in one large building, where the lixiviation tanks are set on a higher level, on a strong framework, which permits the operator to get underneath the tank, so as to detect any leakages. No sump tank is needed, as once the gold is precipitated, the lixivium can be allowed to waste. To facilitate leaching, suction pumps or Korting's ejectors are used; but in leaching gold these would only be needed where the pulp is very slimy or mixed with clayey ores.

---

**In the next chapter will be found an account of several improved processes for the treatment of auriferous ores by chlorination.**

# CHAPTER IX.

## *LATER PROCESSES OF CHLORINATION.*

THE MEARS CHLORINATION PROCESS—Treatment of Arsenical Ores at the Deloro Mine—The Adolph Thies Process—Working Pyrites at the Phœnix and Haile Gold Mines—Barrel Chlorination at the Bunker Hill Mine—The Newbury-Vautin Process—The Pollok Hydraulic Chlorine Process—The Swedish (or Munktell) Process—The Rottermund Process—The Ottokar-Hoffman Process—The Von Pateras and Roeszner Process—Kiss's Method: how Distinguished—Hauch's Treatment of Telluride Gold Ores.

**The Mears Chlorination Process.**—B. Howell Mears, M.D., of Philadelphia, when experimenting a few years ago upon roasted pyrites, with a small apparatus in his laboratory at home, met with an accident through the clogging of one of the outlet pipes of the vessel in which he was exposing some ore to the action of chlorine. The stoppage caused a pressure of gas in the vessel, and an explosion followed by which the whole thing was shattered. As the exposure of pulp to gas had been only for a few minutes, he thought to try the thing over again, and for that purpose gathered up the sand, broken glass, and débris; but, as it was inconvenient from lack of apparatus to immediately expose the pulp again, he tested the pulp to ascertain if any solution had taken place, and to his astonishment he found that the action had been about as thorough and complete as he would have expected to find it after hours of simply passing the gas into and through the moistened pulp. This was a discovery, and on it is based the now perfect process called after the distinguished discoverer.

In the Mears process the roasted ore is moistened with a small quantity of water, then charged into an iron lead-lined barrel which revolves, and into which gas is forced

By this means the ore is kept constantly stirred and tumbled about. Several advantages are gained by this mechanical attrition.

(1.) The ore is kept lively, and every particle of it exposed to the action of the chlorine.

(2.) If there be any large, coarse grains of gold present, instead of a coating of chloride of gold forming around them, and thereby preventing further action of the gas upon the gold grains—as is inevitably the case where the pulp lies still in the vat—the chloride of gold, as soon and as constantly as it is formed, is removed, thus keeping up a continuous action of the gas upon the gold, and consequently rendering the dissolving of a grain of gold as certain and, within reasonable limits, as speedy as if all the gold were of the finest powder. This is a very important consideration, for I am convinced that in many cases where sulphurets are treated by the Plattner process, and poor results or high tailing assays obtained, it is because the gold-particles are so coarse as not to be entirely dissolved during the time allowed for chlorination—in other words, that the solution has been hindered by the protecting coat of terchloride of gold enclosing the gold particles and stopping further solution.

(3.) In California a good deal of the gold in sulphurets, or rather many kinds of sulphurets carrying gold and silver, have the gold so highly alloyed with silver that it is not easily dissolved by chlorine. It is almost impossible to dissolve this alloy of gold and silver contained in some sulphurets, even in nitri-muriatic acid, unless the coatings formed on the gold particles be constantly removed by rubbing. Part of the silver becomes converted into a chloride, coats the particles of gold, and retards further action of the chlorine. But this only occurs in an alloy having a certain proportion of silver to gold. It is not always so, but it is common. Roasted sulphurets carrying this kind of gold alloy, when treated with chlorine, present no difficulties in a revolving barrel, because by the mechanical attrition the coating is rubbed off by the sharp grains of sand as fast as it is formed, in a manner similar to the terchloride of gold previously spoken of.

The above advantages are undoubtedly accomplished by attrition, and are easily and speedily effected in the revolving barrel.

The pressure of gas which is applied while the barrel is revolving is gained by pumping in the gas (by means of a peculiarly but simply-constructed pump) directly through one trunnion of the barrel. The effect of this pressure is also to hasten the action. When the charge has been revolved enough the barrel is filled with water, thus liquefying the pulp, and the contents of the barrel are discharged into a filtering vat, through which the solution runs into precipitating tubs, where the gold is thrown down in the same manner as in the Plattner process. Another plan is to run the solution through tanks packed with pulverised charcoal, which of itself precipitates and catches all the gold from the solution, and, after a time, the charcoal is burned and the gold collected. The precipitation by a freshly prepared solution of sulphate of iron is preferred.

Several plants embodying the Mears process are now in successful operation in California, and the process itself has led to further developments, as will appear in the following pages.

**Treatment of Arsenical Ores at the Deloro Mine.**—Mr. Rothwell gives an account of the treatment of these ores, in which the Mears process is adopted for chlorination. The ores are gold-bearing arsenical sulphurets of iron (mispickel), carrying the theoretical proportions of 42 per cent. of arsenic, 20 per cent. of sulphur, and the remainder iron. The gangue is quartz, calc spar, and some talcose, slaty rock, evidently resulting from the decomposition of the wall rock, which on each side of the veins is syenitic granite. The treatment of the ore consists of six processes.

**Crushing.**—The ore as it comes from the mines is dumped over grate bars at the top of the mill building; what is too coarse to go into the rock breaker is broken by hand, and all then goes through the No. 1 rock breaker, which breaks it to a maximum size of 1¾ inches, then over fixed grate bars, which take out all pieces less than half an inch in size. The coarse then goes through two small rock breakers which crush it to a maximum size of about three-quarters of an inch.

**Concentrating.**—The peculiarity of the Deloro is, that the gold is contained for the most part in the mispickel (which carries when closely concentrated nearly $100, or £20, per ton as an average), and this mispickel is much more friable than the associated quartz and calc spar, which contain but small quantities of gold. The consequence is that, when the rock as it comes from the mine is coarsely crushed, we find the fine is composed, for the most part, of mispickel, and the coarse is quartz with a little mispickel. Taking advantage of this peculiarity of the ore, the mill was constructed so that the ore from the second crushers and the fine from the first crushers would go into the No. 1 revolving screen, 20 ft. long by 5 ft. diameter, where it would be sized into *fine* (*i.e.* passing through a mesh of less than an eighth of an inch), and the remainder into two sizes, which would go to coarse jigs, which are intended to separate into a rich and a poor product—the rich after drying joining the fine from the No. 1 screen, and going to the rich rolls to be crushed to the roasting size, and the poor going to the poor rolls and jigs. The object sought to be attained by this arrangement was the partial concentration of the ore without the usual loss in concentration; but as the roasting and chlorinating capacity of the Deloro works is yet far inferior to the mill capacity, it has been deemed wiser for the present to crush in the poor rolls all the ore as it comes from the second rock breakers, and concentrate it in the jigs. The crushing is done in Cornish rolls 36 in. diameter, 15 in. face, steel shells on double cone centres.

The ore, after passing through the rolls, is elevated to the two No. 2 dry screens, which have a length of 8 ft. by 4 ft. diameter. In these the ore is sized. All passing through a 1½ millimeter, which comes out of the end of the No. 2

screens, drops back into the rolls and goes through again. The fine ore is sized in the usual manner, except that the screens are much larger than usual, for the limiting capacity of such works is always found in the screens. Each set of rolls has a capacity of fully five or six tons an hour in grinding hard quartz and mispickel down from say 1 inch to $\frac{1}{16}$th of an inch.

The crushing (and consequently the screening in the No. 1 and No. 2) is dry. The sized ore is concentrated in ordinary Hartz jigs discharging through the bottom, and the slimes in Spitzkasten and Rittinger's tables. Mr. Rothwell says that his own preference would be for other concentrators of greater capacity than these tables, although they are found to do very fair work.

**Roasting.**—The concentrates are taken from the jig room in a tram-car, which, going up an inclined plane, delivers them directly into a hopper over the drying furnace. This is an inclined revolving cylinder * 20 ft. long, 36 in. diameter at small end, and 48 in. at large; and it has a conical addition of 24 in. in length at the small end, making a total length of 22 ft. The fire passes through this cylinder, and the capacity has never been tested to anything like its limit; but no doubt it would dry two tons an hour, if required. It is very economical in fuel.

As the dry ore drops out of the drying furnace, it is continuously raised by an elevator into No. 1 roasting furnace. This is a revolving cylinder 30 ft. long, 60 in. diameter outside, lined with 4½ in. of fire-brick, and with eight shelves through nearly from end to end. These shelves are formed of key-bricks 9 in. long, so that they stand 4½ in. out from the lining. In this furnace the arsenic and the greater part of the sulphur are volatilized, and pass out through a long series of arsenic condensing chambers, and through a centrifugal (Guibal) fan, 8 ft. diameter, 3 in. face, used to make the draft

---

* For a description of the revolving cylinder the reader is referred to my "Metallurgy of Silver."

to the stack. The ore runs from the first cylinder through a pipe directly into the second cylinder, 20 ft. long, by 48 in., with a 4½ in. lining and six shelves, where the roast is completed. The escaping gases pass to a stack, which also serves the drying furnace.

The air which feeds the No. 1 furnace is preheated by the escaping gases of the second cylinder, by passing through an air space between the two arches which form the top of the second roaster dust chamber.

The two roasting cylinders are jacketed, first with an air space, and then with a covering of mineral wool, and paper over that. The whole arrangement of the roasting cylinders, their jacketing, and the plan of utilising the escaping gases to heat the feed air for the first cylinder, are believed to be new, and are found very economical and efficient in practice.

The ability to make a sweet roast (such as is required for chlorinating) in a single operation, in continuous revolving cylinders, has been questioned by some metallurgists. In proof of its feasibility, Mr. Rothwell points out that, in the continuous arrangement above mentioned, ten tons of concentrates have been roasted in twenty-four hours, and to such perfection that in the subsequent chlorination by the Mears process, from 93 to 98 per cent. of the gold was extracted.

**Condensation and Collection of Arsenic.**—It has also been asserted by some metallurgists that the roasting of arsenical pyrites presents many difficulties, but experience with these Deloro ores has shown, on the contrary, that they roast with much greater facility, and in about two-thirds of the time necessary to roast simple sulphurets. They stand almost any amount of heat without fusing, and the arsenic, which forms 40 to 42 per cent. of the mispickel, volatilizing at a comparatively low temperature, seems to leave the mass porous, thus facilitating the oxidation of the sulphur. The arsenic condenses readily in the series of brick chambers between the furnace and the stack.

The use of a centrifugal suction fan through which the furnace gases are drawn, and the draft of the furnace thereby created, is also believed to be a novelty in metallurgy, which at the Deloro has proved itself both practical and economical. It is clear that a blowing fan, which would occasion the escape of arsenical fumes, could not be used in this case.

**Chlorination of the Roasted Concentrates.**—The roasted concentrates are chlorinated by the Mears process, in charges of one ton, in a revolving lead-lined iron cylinder. The chlorine is made from chloride of lime and sulphuric acid, from 40 to 50 lbs. of the former and 50 to 60 of the latter being used to a ton of ore. The pressure in the cylinder rises to about 40 or 50 lbs. per square inch, and falls to 25 or 30 when the roast has not been so perfectly made as is desirable. The operation lasts about two hours, though probably less time than this will be found sufficient to completely chlorinate the gold when it is in fine particles.

**Precipitation of the Gold.**—The lime contained in the ore was found to give rise to quite unexpected difficulties in precipitating the gold from the chloride solution. The usual precipitant, ferrous sulphate, was found to throw down a voluminous precipitate of principally calcium sulphate along with the gold. In order to avoid this, an effort was made to get rid of the lime, first by sulphuric acid. This was too tedious, and, after many annoying delays, the precipitation by charcoal was tried. The chloride liquor is allowed to filter slowly through a mass of charcoal broken to, say, a sixteenth to half an inch. The barrels are kept full of solution by the filtrate being brought from the bottom of one barrel in a rubber tube which terminates a few inches below the top of the next barrel.

The precipitation of the gold is practically complete. The lime does not remain in the charcoal, yet the liquor undergoes such a chemical change by its contact with the charcoal, that the lime is no longer precipitated by either ferrous sulphate or sulphuric acid, but is by oxalate of ammonia.

Several chemists, who have experimented with this Deloro chloride liquor, have considered that hydrogen sulphide, either in a saturated aqueous solution or as gas, or ferrous chloride, would make convenient precipitants, either of these reagents precipitating the whole of the gold without the lime.

The charcoal which collects the gold is burned in an open pan with fire under, and (it is claimed) without loss of gold, at a cost said to be less than six cents per ton.

**Chlorination of Concentrates containing Gold and Silver.**—The roasting of concentrates of this description is carried out at the Deloro with the addition of salt, so as to convert the silver into a chloride. It is difficult to give any indications of how much salt should be used, and this has to be determined by experiment. It is not advisable to charge the salt with the ore into the furnace, but to give a desulphurizing roasting first and wait for the production of sulphates before adding the salt, which is then thoroughly stirred with the hot ore, and after thirty minutes to one hour withdrawn from the furnace on to the cooling floor, where it is piled up in a heap for as long as possible, as experience has shown that a beneficial result is obtained by leaving the ore in its red-hot state for some time, as no doubt the evolution of the chlorine gas, which is perceptible by the smell, continues to permeate the ore and thoroughly chloridizes the ore.

The ore after damping is charged into the lixiviation vats, and the gold extracted, according to the *modus operandi* of the Plattner process as previously described.

**Operation for Extracting Silver.**—After the gold has leached out and the wash water has drained out from the vat, a solution of calcium hyposulphite is allowed to flow into the same vat and by a separate launder is drawn into another precipitating tank.

The hyposulphite " leach," when holding chloride of silver in solution has a peculiar sweetish taste, and when this taste disappears it is an indication that the leach is finished, which is verified by test. Some of the leach flowing from the discharge-pipe is caught in a glass beaker and some solution of calcium sulphide is added. If a precipitate is produced it shows that the leach still carries silver or some other metal, and it is generally best to add some more hyposulphite.

The leaching is rather slow, and with some ores it takes two days, according to the character and silver contents of the ore under treatment.

**Precipitating the Silver.**—This is effected by means of a solution of calcium polysulphide, which throws down the silver and other metals as sulphides, the precipitate looking like dark mud. The addition of the polysulphide to the leach restores the hyposulphite into its original condition, so that the same can be used over again for leaching fresh charges; therefore this process can be considered as a continuous one. Care should be taken to avoid an excess of polysulphide, as its entrance into the leaching vats would cause a loss by reducing some of the dissolved chloride of silver into a silver sulphide, and enrich the tailings, and therefore in carrying out the precipitation it is better to stop the addition of polysulphide before all the silver is precipitated. Whatever silver remains in the solution will not be lost, as it passes through the ore again in the next operation.

A lively agitation of the liquid assists the precipitation, and when this is accomplished the renewed calcium hyposulphite is drawn off and pumped into the storage tank above. The silver sulphide precipitate is generally found to contain some base metal sulphides, and is drawn by means of a discharge faucet on to a filter, where it is washed with hot water, and by means of a press pressed into cakes which are slowly dried. The silver cakes are now roasted in a small reverberatory furnace, to drive off the sulphur, and care must be taken not to raise the heat too high so as to cause the melting of the cakes, which can easily take place if there is any lead in the ore.

The silver cake is now ready for melting, and this is accomplished in the usual way, by the addition of some scrap iron, which, along with some borax, is placed in a black-lead crucible and melted in the ordinary wind furnace. The iron is added for the purpose of combining with the sulphur to form a matte, and sufficient has to be added to combine with the sulphur. The matte collects on top of the molten metal and can be dipped off with a hot iron ladle, and when this is done some more silver cake, previously warmed, can be put into the crucible, and at the final casting of the ingot the matte collects on top of the silver ingot, and on cooling easily separates from it. If the matte is tough and not brittle, this indicates the presence of silver, and the same ought to be remelted, with the addition of more iron, which will generally yield a good button of silver.

**The Adolph Thies Process.**—This in reality is an outcome of the Mears process. The latter, while fairly effective from the metallurgical point of view, is objectionable in some particulars, especially the difficulty of keeping tight joints in the stuffing-boxes, and the consequent escape of chlorine gas through the works. The cost of repairs has also proved excessive.

The system adopted by Mr. Thies is extremely simple, and requires no tight joints liable to leakage. The chlorinator consists of a plain iron barrel lined with lead, and provided with a manhole on one side for charging and discharging the ore. The ore is charged in one-ton lots. The barrel is first partially filled with water, then a sufficient quantity of chloride of lime is dropped in, on top of which the roasted ore is charged. On top of the ore is poured the requisite amount of sulphuric acid, the manhole cover is put in place and thoroughly secured, and the barrel is started to revolve. The reason for charging in this order is to prevent the sulphuric acid and chloride of lime from coming in contact before the barrel has been securely closed. When the charging is completed the barrel is started and is kept revolving until the whole of the gold is dissolved. A very ingenious arrangement has been devised which enables the operator, by means of a clay pop-valve, to test from time to time, without opening the barrel, the presence of an excess of chlorine gas. Mr. Thies operated with some sulphides where each one-ton charge consumed 20 lbs. of chloride of lime and 25 lbs. of acid. The time occupied in chlorinating so far has been six hours, but with more experience this time ought to be reduced to one half.

When this operation is completed, enough water is introduced to nearly fill the barrel, which is again revolved, so as to thoroughly wash the ore and dissolve the chloride of gold. The barrel is then opened and the liquor is decanted off upon large shallow filter beds. Another wash water is then put into the barrel, which is again revolved for a few minutes and then decanted upon the filter. Ultimately the whole charge is turned down in the ordinary way, and a final wash water is given on the filter.

The results so far obtained appear to have been remarkably successful. Subjoined are the results of eight charges, each of which contained a little over one ton of ore of such fineness that 99 per cent. of it would go through a 100-mesh and 60 per cent. through a 150-mesh screen. The assays made of the roasted ore showed \$36·70, or £7 10s. gold per ton.

| | | $ | | s. | d. |
|---|---|---|---|---|---|
| Assay of tailings from first charge gave | ... | 1·56 | or | 6 | 3 |
| ,, ,, second ,, | ... | 1·03 | ,, | 4 | 1⅓ |
| ,, ,, third ,, | ... | 0·78 | ,, | 3 | 2 |
| ,, ,, fourth ,, | ... | 0·52 | ,, | 2 | 1 |
| ,, ,, fifth ,, | ... | trace. | | | |
| ,, ,, sixth ,, | ... | 0·78 | ,, | 3 | 2 |
| ,, ,, seventh ,, | ... | trace. | | | |
| ,, ,, eighth ,, | ... | 0·78 | ,, | 3 | 2 |

In order to further test the advantages of this system, Mr. Thies tried one charge considerably heavier than the others, and endeavoured to filter it direct in the usual way. While he used for this charge nearly double the amount of wash water, and while the time occupied in filtering was much more than the other charges required, the tailings resulting contained \$5·68, or £1 2s. 10d. per ton. There seems to be no difficulty in filtering through a bed of ore from 3 to 4½ inches thick; but when the thickness of the bed is greater, then the filtering of such very fine material becomes almost impossible, hence the advantage of decanting.

The precipitation is done in the usual manner with protosulphate of iron. If ores contain a considerable percentage of magnesia, it gives a voluminous precipitate. The precipitation of lime and magnesia may be entirely prevented if care is taken to keep the tank solution in proper condition. If such is not the case the magnesia shows in the precipitating tank a milky cloud.

The cost of roasting, labour, chemicals, and power, Mr. Thies states did not exceed in his operations \$4, or 16s. 0d. per ton. The actual gold recovered is in all cases within a few cents per ton of the full assay value of the ore, less the amount shown by assay to have remained in the tailings. While the gold obtained in the stamp mill and amalgamating works is seldom purer than 897 fine, the gold obtained by this process is generally over 978 fine.

**Working Pyrites at the Phœnix and Haile Gold Mines.**—Mr. Thies, in a communication to the State Mineralogist of California, gave the following interesting details of his operations at these mines, which, read in conjunction with the information given above, will make his process more readily understood. The Phœnix Mine is in North Carolina, and the Haile Mine in South Carolina.

Mr. Thies worked the Mears process for over two years, but met with difficulties, occasioned by the leakage which resulted from pressure. Meanwhile he tested the effect of *a highly saturated chlorine water* under attrition, *without pressure*, and when, after repeated tests, he found his results as good without pressure as with the highest pressure—and better when he divided the requisite amount of chloride of lime and sulphuric acid, so as to have nascent gas during the time of working—he remodelled the chlorinator, so as to have a sheet-iron cylinder 42 inches diameter by 60 inches long, without any of the complicated adjuncts which the introduction of chlorine under pressure required.

The heads of his chlorinating barrel are cast, and securely bolted to end flanges, and provided with tight and loose pulleys. The bung for the introduction of the roasted ore and chemicals, 6 inches in diameter, is provided with a lead-lined cover, which before rotation must be closed hermetically. The interior of the cylinder is lined with sheet-lead of 10 to 12 pounds per square foot. The capacity of the chlorinator is from 1 to 1¼ tons of roasted ore. Before introducing the ore the chlorinator is charged with from 100 to 125 gallons of water, enough to make an easy flowing pulp. This done, the roasted ore is introduced; half the requisite quantity of sulphuric acid is then poured in, and lastly, half the required chloride of lime, when the bunghole is closed, and the chlorinator set in motion at the rate of about fifteen revolutions per minute.

For Phœnix ores he used 40 lbs. chloride of lime and 50 lbs. commercial sulphuric acid per ton of roasted ore, but he charged 20 lbs. of chloride of lime and 25 lbs. of acid first, rotated 3 or 4 hours, then opened the bung, and charged the other half, having found better results by thus dividing the chemicals. He rotated for 2 or 3 hours longer, and if, by the aid of the lead-valve, free chlorine was found to be present, the cover was removed from the bunghole and the chlorinated ton of ore thrown on a shallow filter 6 by 8 feet, provided with a 5-inch filter-bed, over which the pulp would spread to a thickness of about 4 inches. The filter, before the ore pulp is thrown on it, is first flooded with clear water from below, and when the water stands over the filter the discharge-hole is corked, so that the water acts as a cushion against the ore pulp, prevents the packing of the filter-bed, and admits of a free filtering.

When the chlorinator has been emptied on the filter the cork is removed and the solution allowed to pass into a stock tank below. As soon as the first solution has passed through, so that the ore surface is exposed, from 3 to 4 inches of water are added over the whole surface, and when this is filtered through and the ore surface exposed again, the whole space above the ore, about 11 inches in depth, is filled, which by practice on Phœnix ores was found sufficient to remove all the chloride of gold; but should there be still a reaction with ferrous sulphate, more water must be added.

The filters are lead-lined, 18 inches deep, and have a fall of 1 inch towards the outflow. The bottom is covered with perforated glazed tiles of clay or of mineraline, which is impervious to the action of acids and chlorine. On this rests the gravel filter bed, which is topped off with ordinary clean river sand. To prevent the filter from getting an uneven surface, longitudinal 1¼-inch wooden slats 8 or 10 inches apart keep it in place.

The filtering should be accomplished as quickly as possible; but as this depends generally upon the fineness of the ore treated, no rule can be established. As long as the solutions show the presence of chlorine when the last wash water has passed through the filter, there is no risk of not having clean tailings.

The solutions accumulating in the stock-tank are let off into smaller tanks for precipitation with ferrous sulphate, which should always be regenerated if not active, so as to destroy any ferric sulphate which will dissolve precipitated gold. Ferric sulphate will not dissolve gold in presence of ferrous salts. Care should be taken by examining after 24 hours to ascertain if all the gold has been precipitated, as losses may occur by a partial precipitation.

The tanks for precipitation should not be too deep. A convenient size is 6 to 8 feet in diameter and 3 feet high, holding the solution from about 3 tons of roasted ore. A sufficient number should be on hand to allow the precipitate at least 3 days to settle. After 3 days' settling in shallow vats, the supernatant liquid can be drawn off, and fresh solutions added for precipitation. At the Phœnix the liquor is passed over metallic iron, and the copper recovered as cement.

From the precipitating tanks the precipitate is finally collected, washed as clean as possible to remove the iron salts, dried, and melted. The amount of chloride of lime and acid used at the Phœnix was 40 and 50 lbs. respectively, which is due to the presence of an appreciable amount of chalcopyrite. An excess of acid should always be used, so as to convert all the lime into a sulphate to remain in the filter. The solution should react slightly acid; if neutral, soluble chloride of lime will cause a bulky precipitate with ferrous sulphate.

At the Haile Mine the iron sulphurets are pure, requiring only 10 lbs. of chloride of lime to 15 lbs. of acid, and 4 tons of roasted ore are treated in two chlorinators during 10 hours, and 94 per cent. of the assay value of the ore is extracted.

The success of chlorination, by whatever process, depends on a thorough roast, assuming that clean concentrates are dealt with. It is of the utmost importance that the roasters should have some guide to go by, and to this end they test every charge before drawing by a bright filed iron rod. A small portion of the roasted ore is boiled in water, and stirred with the bright iron; the least trace of sulphates will stain the iron—a sign for the workmen that the roasting is not completed.

At the Phœnix Mine, a revolving pan furnace is used, 12 feet in diameter, with a short reverberatory attached. From two working doors the roaster can rabble the ore. When a charge is finished the ore is discharged through the hollow axis on which the pan revolves, into an outer circle below, and then removed, by scrapers attached to the bottom of the pan, into a car and delivered to the cooling floor, from where it is elevated into the chlorination house. Such a pan furnace roasts 1 ton of raw ore in 12 hours, with a consumption of three-eighths of a cord of wood and 90 cents for labour. The power necessary to drive the pan is a small item, and will not exceed 25 cents per ton of raw ore.

At the Haile Mine, a double reverberatory furnace furnishes 2 tons of roasted ore every 24 hours, with an average consumption of 1 cord of wood at 5s. per cord, and 4 labourers. The cost per ton of roasted ore amounts to 12s.

The cost of chlorination by the barrel process depends chiefly on the number of tons chlorinated per day. Two men can easily chlorinate 4 tons in 10 hours, elevate the ore, and clean out the filters, of which there are 4 to each chlorinator, and having arranged on this basis the work at the Haile Mine, the cost for chlorinating 4 tons daily was as follows:—

| | $ | £ | s. | d. |
|---|---|---|---|---|
| 40 lbs. chlorine of lime, at 3 cents . . | 1·20 | 0 | 4 | 10 |
| 60 lbs. sulphuric acid, at 2 cents . . | 1·20 | 0 | 4 | 10 |
| 2 labourers, at 90 cents . . . . | 1·80 | 0 | 7 | 3 |
| 1 chlorinator. . . . . . | 2·00 | 0 | 8 | 0 |
| Motive power . . . . . | 0·50 | 0 | 2 | 0 |
| Total . . . . . | $6·70 | £1 | 6 | 11 |

Or $1·67½ per ton = 6s. 9d.

Adding to this 12½ cents for sulphuric acid for making sulphate of iron and 20 cents for repairs and wear, we have $2, or 8s. per ton for chlorination, or $4·62½ = 18s. 6d. for roasting and chlorinating 1 ton of roasted ore, representing 1⅓ ton of raw iron pyrite. Within 7 hours from the time the ore is in the chlorinator, the solutions are ready for precipitation, and the tailings clean. The wear on the inner lead lining of the chlorinators is imperceptible. A chlorinator in use at the Phœnix for over five years did not show any wear on the lead.

**Barrel Chlorination at the Bunker Hill Mine.**—It has been found at this Californian mine that the cost of treating the concentrated iron pyrites by the barrel process is equal to the cost by the Plattner method, as appears from the following data given in the report of the State Mineralogist of California:—

ESTIMATED COST ACCORDING TO THE PLATTNER PROCESS.

| | £ | s. | d. |
|---|---|---|---|
| Labour | 0 | 16 | 0 |
| Wood | 0 | 12 | 0 |
| Peroxide of manganese | 0 | 2 | 8½ |
| Salt | 0 | 1 | 6 |
| Sulphuric acid | 0 | 8 | 0 |
| Incidentals—taxes, insurance, ordinary repairs, sulphate of iron, assaying material | 0 | 12 | 0 |
| | £2 | 12 | 2½ |

ESTIMATED COST BY BARREL PROCESS AT BUNKER HILL.

| | £ | s. | d. |
|---|---|---|---|
| Labour | 0 | 16 | 0 |
| Wood | 0 | 12 | 0 |
| Chloride of lime | 0 | 4 | 10 |
| Salt | 0 | 0 | 8½ |
| Sulphuric acid | 0 | 5 | 0 |
| Power | 0 | 1 | 8 |
| Incidentals—same as in Plattner process | 0 | 12 | 0 |
| | £2 | 12 | 2½ |

The percentage of gold extracted was higher by the barrel process, as only 14s. per ton was left in the tailings, whereas, when the Plattner process was used on sulphurets of equal assay value, they were found to contain about £1 8s. per ton. Sulphurets containing about £12 per ton, and worked up to 92 per cent. of the assay value by this method, and tailings from £20 sulphurets, do not contain any more gold than those of a lower grade.

The roasting furnace proper is 9 feet wide and 40 feet long, outside measurement; the thickness of walls is 1½ feet. The top of the bridge wall is 18 inches above the grate bars, and 9 inches below the centre of the furnace arch; the drop to the hearth is 6 inches. That part of the furnace nearest the bridge wall is a finishing oven, with revolving hearth, 12 feet in diameter, inside measurement, and makes one revolution per minute. The furnace is charged in the same manner as ordinary reverberatory furnaces for roasting sulphurets, and at this mine 1 per cent. of salt is introduced with the charge.

During the latter part of the roasting in the finishing oven, different portions of the sulphurets are in succession exposed to the highest action of the heat and oxidizing flame. The capacity of the furnace is 2 tons in 24 hours, and the fuel required five-eighths of a cord per ton of sulphurets.

The barrels are lined with ¼ inch of lead of the best quality, entirely free from zinc. Their outside diameter is 40 inches, outside length 54 inches, inside length 48 inches. There is an opening in the centre of the barrel 10 inches in diameter, for admitting and removing the charge. After the charge has been introduced and the cover of the opening closed, the barrel is made to revolve 13 times per minute for 6 hours, when the mass is discharged into a leaching tank of the usual construction. The barrels hold 1 ton of ore at a charge, which is mixed with 130 gallons of water and 30 lbs. of chloride of lime, costing 2d. per lb., then 36 lbs. of 66° B. sulphuric acid, costing 1¾d., is added. Ordinary labourers are paid 12s. per day.

**The Newbury-Vautin Process.**—In this process also, chloride of lime is decomposed by sulphuric acid in a revolving barrel, and the evolved chlorine made to act on the gold in the barrel itself, rendering the manipulation very simple. The plant required is a hopper, chlorinating barrel, leaching vat or filter, reservoir for the liquid which is drawn from the filter, and the charcoal collector. The hopper is fixed above the chlorination barrel; into this the crushed ore, roasted pyrites, or tailings are poured.

The barrel is made of iron, lined with lead, this again with prepared wood; it is sufficiently strong to bear a pressure of 100 lbs. to the square inch. In form it is cylindrical, and in the centre are two valves directly opposite to each other. One valve is sufficiently large to allow the pouring in of a stream of tailings or sands; the other is smaller, and serves for the introduction of the compressed air. The barrels vary in size according to the quantity of ore required to be treated. The charge falls from the hopper through the larger valve into the chlorinating barrrel.

The chlorine is produced in the barrel by adding to the ore a certain quantity of chloride of lime and sulphuric acid, and enough water is added to produce a liquid mass. The valve cover is then screwed down, and the barrel turned half round, so that the small valve is uppermost. To this is attached an indiarubber pipe, leading from an air pump, and compressed air is forced in until a pressure of about 60 lbs. to the square inch is reached, when the valve is screwed down and the hose disconnected.

The barrel is now set revolving at a speed of about ten revolutions per minute, and the sulphuric acid decomposing the chloride of lime produces chlorine gas, which impregnates the water in the barrel, and the free gold is thereby converted into a chloride of gold, which is dissolved by the water. The time occupied to effect this solution ranges from one to four hours, according to the degree of fineness of the gold particles and the character of the ore.

The chlorinator, on being stopped, is now discharged first of the compressed air and gases, by means of the small valve, to which an indiarubber hose is attached leading outside the building. The large valve is then removed, and the barrel set revolving as before; at every downward turn it discharges a portion of its contents into a shoot leading to the leaching-vat below. When almost all its contents are discharged a few buckets of water are thrown in, which by the revolution of the vessel are swirled around, thus completely washing into the filter all remaining solution.

The filter is an iron vessel lined with lead, with a double bottom. To the lower part of this a pipe is connected leading to a vacuum pump. When the ore from the chlorinator has filled the filter, the air is drawn from between the double-bottoms of the latter, which causes very rapid filtration.

The filtered liquid is drawn off through a pipe into a vat; a continuous stream of water is kept playing upon the tailings in the filter, which by the suction is drawn through them, so that all the chlorides are washed out. To prevent any clogging of the material, the action of the exhaust-pump is made intermittent. The solution as it runs to the vat being continually tested, it is easy to know when the work is completed; then the water is cut off and the pump stopped. The filtration generally occupies about one hour. The stuff in the filter is tipped into a truck running on a line of rails below, by which means it is run to the tailing heap.

The gold is now in the solution in the tank, at the bottom of which is a tap from which the liquid slowly runs through a filter of charcoal. Contact with this material returns the gold chloride to metallic gold, which collects on the charcoal. Copper, lime, magnesia, and zinc are not touched by this reagent, so they pass through in a soluble state. The charcoal after drying is burned, and the gold collected by it is melted into an ingot.

Fig. 128 is a general elevation of the apparatus employed for the extraction of gold by the Newbury-Vautin process. Fig. 129 is a sectional detail of the closed chamber in which the gold is acted on by any suitable reagent such as chlorine. Fig. 130 is a sectional detail through the filtering vessel or leaching vessel when the flow is downwards. Fig. 131 is a modified detail of the filtering vessel with cover when the flow is reversed or taken upwards.

A A are the hoppers from which the pulverised ore is discharged for treatment into the closed vessels, B B, by chlorine, bromine, or other equivalent reagent, in a liquid or gaseous condition, which is applied under pressure and which is caused to react on the ore under a pressure greater than that of the atmosphere by means of an air force pump, C. This vessel, B, shown in detail in Fig. 129, is strongly constructed of iron or similarly suitable material, to carry considerable internal pressure. It is hermetically closed by a manhole door, D, by which the charge of ore is admitted when required, and discharged when sufficiently treated.

Within the iron shell a coating of lead is applied to protect the former against the action of the chemical reagents or salts, and within that another protective coating of wood, earthenware, or suitable material is applied, to prevent the abrasion of the lead lining by the agitation of the ore therein by rotation.

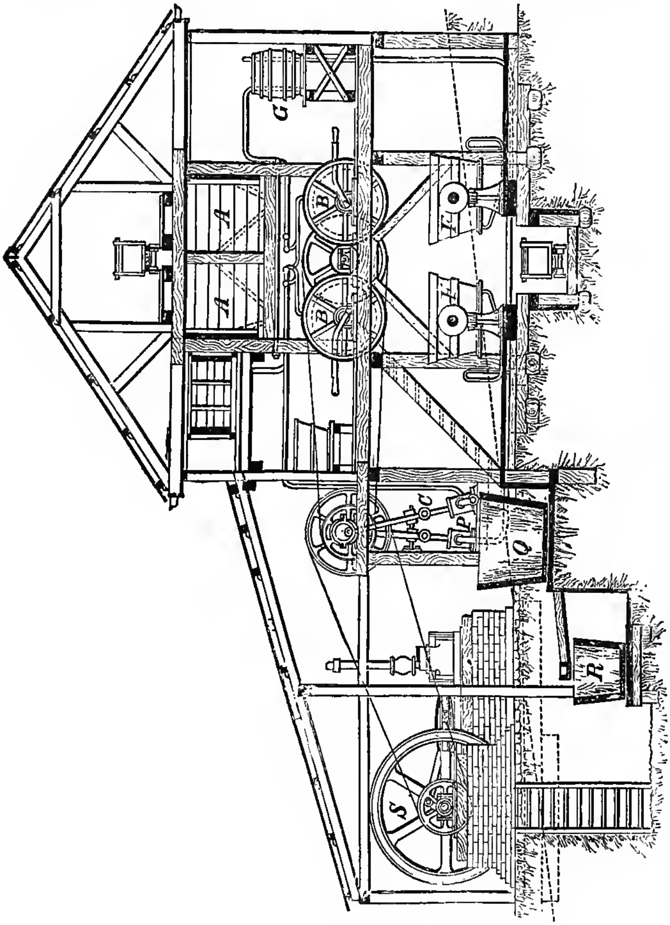


FIG. 128.—NEWBURY-VAUTIN PLANT.

The gas or air communications are effected through a suitable stop valve, E, to which the pipes are connected by a suitable union. By this means, after the admixture of the pulverised ore and the chlorine or other chemical reagent has been made in this chamber, compressed air is admitted from the air compressor, C, until the required working pressure of about five

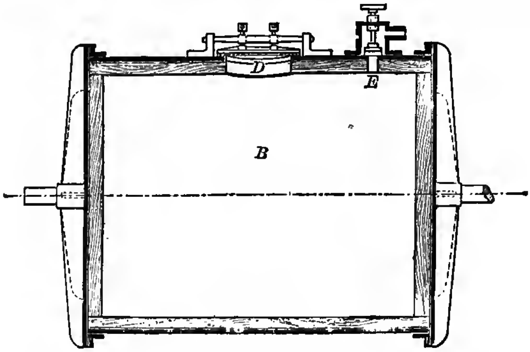


Fig. 129.—Newbury-Vautin Process. Chlorinating Barrel.

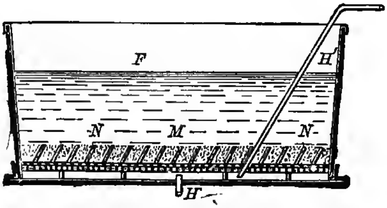


Fig. 130.—Down Filter.

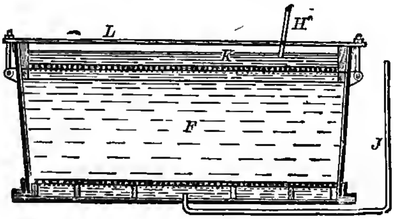


Fig. 131.—Up Filter.

atmospheres per square inch is arrived at. The valve, E, is then screwed down, and the pipes detached, whilst agitation of the contents by the rotation of the vessel proceeds. Before discharge of the contents into the filtering vessel or leaching tub, F F, the compressed air and gas is allowed to pass through this valve, E, by a suitable connection into a solution of lime water in the tub, G, to absorb noxious vapours. The ore so treated is then discharged by inversion of the chamber, B, into the leaching tubs or filters, F F (shown in alternative detail in Figs. 130 and 131).

The leaching or separating of the auriferous solution from the pulverised material mixed therewith, is effected by the direct suction of a pump, P, through the alternative suction pipe, H H (Figs. 130 and 131), a flow of water being maintained as long as required. In the apparatus, as shown in Fig. 130, this operation is carried out by filtration downwards. In Fig. 131, the inverse process is used, the solution being withdrawn upwards through the filtering medium, K, consisting of a perforated wooden diaphragm covered with a suitable filtering medium such as canvas or asbestos, a flow of water being maintained through the supply-pipe, J.

In this case the filtering vessel or leaching tub is conveniently closed by a cover, L, bolted down upon the open mouth of the vessel. The pulverised material and filtering medium are prevented from choking by a reflux action from the suction pump, P, through the suction pipe, H or $H^1$.

After sufficient leaching of the ore or pulverised material has been effected in this leaching vessel, the solid material is conveniently discharged by reversal of the filtering vessel, and taken away by a trolly or other convenient means. To prevent the filter bed, M, as shown in Fig. 130, from being discharged by reversal of the filter, a series of obliquely inclined wooden slats interspersed between the filtering medium are provided to keep it in place even during reversal of the said filter vessel. The auriferous solution so withdrawn from the filtering vessels is delivered into the vat, Q, where it is treated with a jet of steam or air, to drive off any free chlorine or unconbined reagent that may remain in the solution. The solution is then passed through a filter, R, composed of charcoal or other reagents, which is formed of alternate layers of coarse and fine pieces respectively, and is of considerable height and small sectional area. The gold is here precipitated from its solution and is recovered from the material of the said filter by burning or other suitable chemical process.

The power required to work the apparatus may be conveniently furnished by a steam-engine at S.

**The Pollok Hydraulic Chlorine Process.**—After Dr. Mears had shown that chlorine acts more energetically when under pressure, it was found that there was difficulty in obtaining this pressure by pumping in chlorine gas owing to the corrosion of the joints; and it was proposed to obtain the pressure by pumping air into the cylinder and thereby retaining all the chlorine in solution, by which means a solution of a high degree of concentration should be obtained.

Mr. Pollok, who has investigated the subject, disclaims that any such result is obtained, and says it is a mistake to think that by pumping air into the cylinder, the chlorine would be driven into the solution, or that the gas would become liquefied, and in this view he is supported by the high authority of Sir William Thompson.

On theoretical grounds, his reasoning seems to be quite correct, as chlorine gas will only liquefy when the pressure of its own vapours is at 60 pounds to the square inch; which is quite different from the effect of pumping in air at this pressure and mixing it with the gas. On the other hand, as Dr. Mears has shown, when chlorine gas acts under pressure, the effect is beneficial, and it remains for practical experience to demonstrate if pressure when exerted in one form or another on chlorine solutions is equally beneficial in promoting the gold extraction.

In the process with which his name is associated, Mr. Pollok takes the pulverized ore and places it in revolving cylinders in charges of one ton or more, with about one per cent. of bleaching powder, and 1½ per cent. of bisulphate of soda, these being added the one before the other, and after the ore, to prevent the liberation of chlorine before revolving the cylinder.

After the ore and reagents are in the cylinder, the cover is screwed over the charging aperture. When this is done, water under pressure is admitted into the cylinder through a cock, a pipe being coupled to the screwed end of the cock connected to a pump, accumulator, or other source of supply, and the air escapes from the cylinder by a valve placed on the top for that purpose. When the cylinder is full, the air escape is closed, and the cylinder revolved, the accumulator or pump exerting a steady pressure throughout the whole operation. The hydraulic pipe does not interfere with the rotation of the cylinder as it passes through the centre of rotation, to one end of which it is connected by a packing-box that keeps the pipe perfectly tight, but leaves the cylinder free to revolve.

Inside the hydraulic supply pipe, and immediately beyond the packing-box, is placed an automatic rubber valve which is made like a cone slit at the top, and placed pointing into the cylinder; through this the water can enter freely, but it prevents ore and solution from finding its way out of the cylinder into the pipe.

Pollok's claim is, that by the use of hydraulic pressure the whole of the chlorine present is retained in solution, and the high pressure rapidly forces this strong chlorinating liquid into the pores of the ore.

When the chlorination is completed, the hydraulic pressure is shut off. The cock to which the hydraulic pressure was attached, is connected now with a pipe leading to chambers containing slaked lime, and the cock being opened, any excess of chlorine gas will flow out into these chambers and be reconverted into bleaching powder. The ore and solution are discharged on to suitable filters, and the filtration activated either by vacuum pumps or ejectors. The solution is run into precipitating tanks, and the gold thrown down by ferrous sulphate. After settling for twelve hours, the clear liquor is run off through charcoal filters, which serve to arrest any gold that has not completely settled.

The patents of this process have been acquired by the Pollok Patents Gold Extracting Company of Glasgow. According to a communication received from Mr. Pollok, reporting upon working tests made by him on numerous samples of ore, he claims to have extracted from 92 to 99 per cent. of their assay value.

**The Swedish (or Munktell) Chlorination Process.**—In carrying out this process, Mr. Munktell prefers a chloridizing roasting. If the material after roasting and washing is found to contain gold in somewhat large grains, these may be conveniently separated by washing; or by some other mechanical means the heavy metals are removed.

He subjects the material, after being reduced to a sufficiently fine powder and freed from protoxides—whether roasted and separated from the coarse grains of gold or not, as circumstances may require—to the following process. The ore when pulverized should not be too fine, so as to allow liquids to percolate through it.

A small quantity of hypochlorite of lime, or some other salt in which the acid consists of chlorine combined with oxygen, is dissolved in water or in some other suitable liquid, such, for instance, as a solution of chloride of sodium, when the raw material contains silver. Instead of chloride of sodium, chloride of calcium or some other substance capable of dissolving chloride of silver may be employed. The solution should contain about one per cent. or less of hypochlorite of lime. This solution, before it enters the running tanks containing the auriferous material, meets with a highly diluted solution of some acid, such as muriatic acid, sulphuric acid, carbonic acid, or some other acid, the solution being of such a strength that equal volumes of the acid and the hypochlorite of lime will just suffice for the decomposition of the hypochlorite of lime.

The mixed liquids now percolate through the material, whereby the chlorine produced by the mixture, partly in a nascent state, unites itself with the gold, which is thereby dissolved, and is continually allowed to run off at such a rate that only a small excess of chlorine remains in the solution. This solution is led in to the precipitation tanks, where the gold, and any silver present, is precipitated by one of the usual methods.

This method of extracting gold can be carried out in a convenient and cheap manner without the escape of any considerable quantity of chlorine.

The solution may also be stronger and be mixed beforehand, and then, after a longer or shorter time, be poured over the crushed raw material prepared in the manner described. In this case, however, not only does much chlorine escape, which has a very detrimental effect on the carrying out of the process, but also, larger quantities of the solution are required. By allowing the chlorine in a very diluted solution to act on the auriferous substances, and by producing this chlorine at the same moment the solution is used, the process becomes extremely convenient and inexpensive; and the difficulties (Mr. Munktell remarks) which hitherto have always been expe-

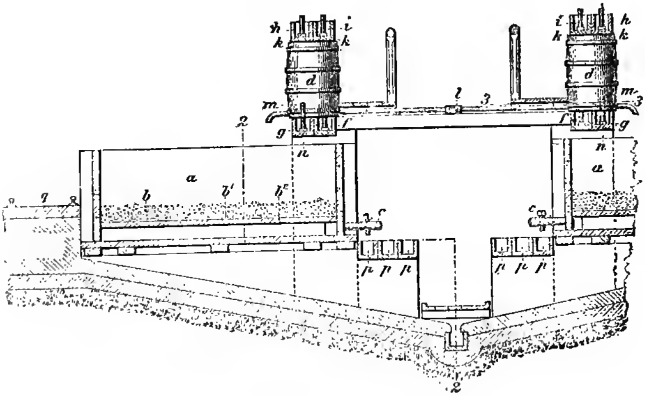


Fig. 132.—Munktell's Plant. Section.

rienced when chlorine is used on a large scale for the extraction of gold are thus avoided.*

The process can be carried out with very simple apparatus. A convenient arrangement consists of tanks having a layer of filtering medium above a false bottom. Over each tank are two vessels by which the quantities of the solutions are measured. They are filled by means of pipes or conduits and discharged into other conduits leading to the tanks. Other pipes or conduits carry water or other liquids to the tanks for dissolving and carrying away the protoxides, if any.

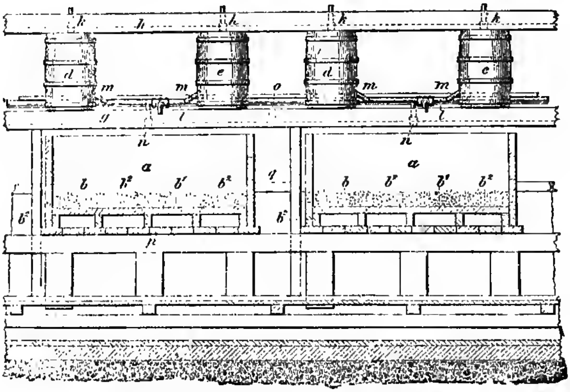


FIG. 133.—MUNKTELL'S PLANT. Longitudinal Section.

Fig. 132 represents a simple arrangement of an extraction apparatus, showing the tanks in a section of the line 1—1 in Fig. 134. Fig. 133 is a longitudinal section thereof along the line 2—2 in Fig. 132; and Fig. 134 represents the same in plan and horizontal section along the line 3—3 in Fig. 132.

*a a* are extraction tanks of wood; *b b* represent filtering bottoms inside the tanks, consisting of a layer of gravel or

---

* Mr. Munktell, however, when making this last statement, was evidently not aware of the inventions of Dr. Mears and others; and even with a properly arranged Plattner plant no inconvenience such as he anticipates need (I think) be apprehended.

quartz spread on perforated boards or plates of earthenware $b^1$, placed on the supports $b^2$. *c c* are outlet cocks. For measuring the requisite proportions of the solutions of hypochlorite of lime and the acid, the tubs *d d* and *e e* are placed on the channels *f f* and *g g* above the tanks. The diluted ready-made solutions of hypochlorite of lime and acid are led separately through the channels *h h* and *i i* into the respective tubs *d* and *e*. As each tub is filled, the plug, *k*, is inserted into the hole in the channel leading to that tub. The channels, *f f*, are used for admitting such acids and salt solutions as are required for dissolving other substances than gold. *l l* are water pipes.

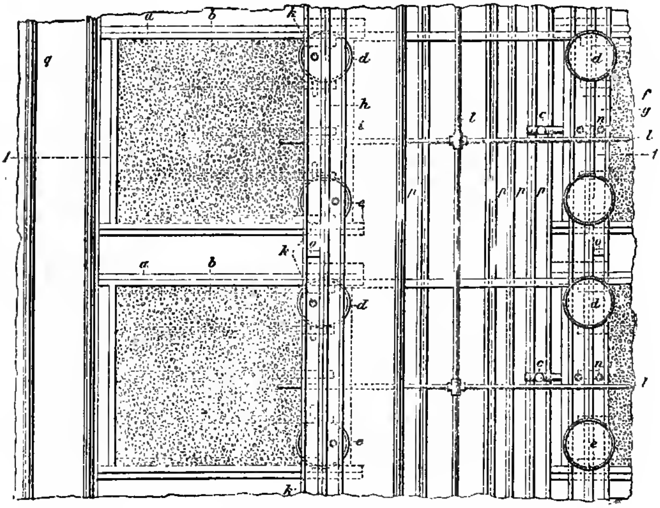


Fig. 134.—Munktell's Plant. Plan.

After the tubs have been filled the process is commenced by removing the plugs, *m*, and the diluted solutions of hypochlorite of lime and acid are allowed to run in definite quantities out of the tubs, *d* and *e*, into the channels, *g*, meeting each other at the openings *n n* and thus running down together into the tanks.

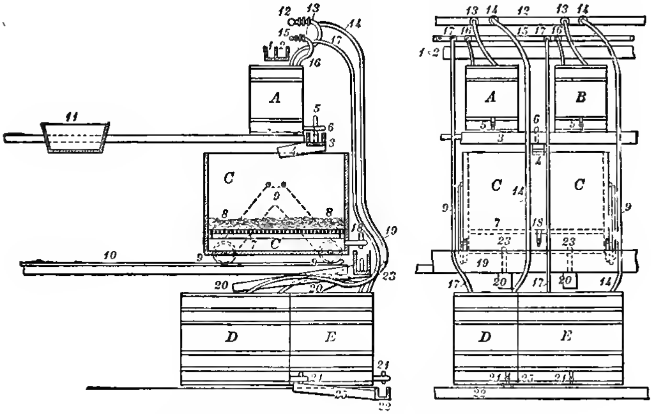
By cross boards, *o*, in the channels the tanks which are not to be operated upon can be shut off. *p*, *p*, *p* are channels to discharge the solutions which are running off or are let out from the tanks, *a*. By means of short shoots placed under the cocks, *c*, the liquid is led into the respective channels, *p*, and from there led to the precipitating tubs to be treated in the usual way; or the gold can be collected on a charcoal filter. Lines of rails, *q*, for transporting the material are also shown on the drawings.

**The Rottermund Process.**—I subjoin a description of this process according to the specification of the patentee. The method, I understand, is now practically applied. As in Munktell's process, the decomposition of the chlorides is effected by dilute acids in open tanks in the mass of the ore, and it is claimed that the chlorine *in statu nascenti* acts with more energy.

The invention (the patentee says) consists of a new or improved process for the extraction by means of chlorine of the precious and other metals from refractory ores, sands, and residues, by an economical continuous operation, by which such extraction is rendered more economical in point of time, material, and labour, with the result of a larger percentage of gold than by any other method. Experience having shown that chlorine is unable to extract the gold from the ores, if the former is combined with metallic sulphurets, unless these are rendered innocuous—that is, removed, prior to the process for extraction of the gold—the ores are prepared in such a manner that the gold becomes accessible to the influence of the chlorine and that it alone attracts the action of the chlorine, the process being so conducted that only sufficient chlorine is employed as may be theoretically requisite to release the gold. The chlorine is used in nascent state—namely, in its fullest strength—so that very weak solutions of chlorine are and can be employed, and the process can therefore be carried on in open vessels, without annoyance to the workmen, and without loss of chlorine. The most finely distributed gold only, or such combined gold as could not be obtained by mechanical means being extracted. The gold which is sufficiently coarse-grained for its specific gravity to come into play, is subsequently extracted by mechanical means.

The process may be divided into the five following principal operations.

1. Roasting.
2. The extraction of the secondary metals.
3. The removal of the protoxide salts.
4. The extraction of the fine gold.
5. The final washing of the residue in order to collect the coarse gold remaining.



FIGS. 135 AND 136.—ROTTERMUND PLANT. Section.

If the ore contains a great deal of zinc, during roasting with common salt, calcined neutral iron-sulphate is added. If a considerable quantity of antimony and arsenic is present, then the ores are sometimes submitted to an oxidising roasting process before they are roasted with common salt.

The roasting may be conducted in furnaces, so arranged that the metallic vapours are condensed in properly adapted chambers, and the resulting sulphurous and hydrochloric acid collected.

Figs. 135 and 136 show two elevations drawn at right angles to each other, and Fig. 137 is a plan of the apparatus employed in the treatment of the ores after the roasting process. The ores, sands, or residues, having been properly roasted, are placed, preferably warm from the furnace, in vats or barrels, C, or in other suitable receptacles, which may be of any size or shape, as may be found most convenient for the ores, sands, or residues under treatment, and preferably arranged so as to be self-dumping. 9, 9, 9 represent the wheels and tipping apparatus; and 10 the rails upon which the vat, C, is run underneath the shoot, 11, for the roasted ores, sands, or residues, and to the dump shoot.

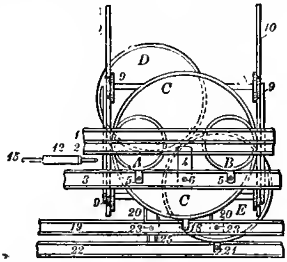


FIG. 137.—ROTTERMUND PLANT. Plan.

These vats or barrels, C, are constructed with false bottoms 7, which are pierced with holes 7 to 8 millimetres in diameter, and both receptacles and false bottoms are constructed entirely of or lined with some material not attacked by chlorine, preferably of wood. Above the false bottoms there is placed a layer, 8, of small stones or other material to act as a filter. The ore in the vats, C, is submitted to the action of dilute sulphuric or muriatic acid of a specific gravity of 1·01 or thereabouts, or other diluted acid coming from the cask, B, and warmed, preferably by steam, to a temperature of about 140° F.

Steam is supplied to the cask, B, by the steam pipe, 15, and steam cock with short hose, 16. This acid solution transforms the protoxides and salts at minimum of oxidation into oxides and salts at maximum of oxidation. The acids used in this part of the process may be those obtained from the roasting of the ores. Muriatic acid is preferable to sulphuric acid, because this avoids the formation of sulphate of lime in a succeeding process. As a substitute for acids, bisulphate of soda may be employed, which is cheaper and easier for transport. The liquid, after the salts are dissolved, generally after the expiration of from two to five hours, is allowed to run into a lower vat containing scrap iron, or other necessary reactives, by which the contained copper, silver, &c., are precipitated. The sulphate of iron so obtained may be used in a subsequent part of the process.

The operation is finished so soon as the effluent liquid ceases to affect permanganate of potash. The ore is then leached with cold water to reduce the temperature to about 60° F., so as to avoid a loss of chlorine.

Two casks or other suitable vessels, A B, composed of wood or some other material not affected by chlorine, and placed above the vats, C, are now filled, the one, B, preferably with dilute hydrochloric acid of specific gravity 1·003, or other specific gravity as may be found most suitable, or bisulphate of soda may be substituted for the acids, and the other, A, with a solution of chloride of lime in the proportion of 0·7 to 1 per cent., or 7 to 10 grammes of chloride of lime per litre of water.

The solution of chloride of lime is supplied to the cask A from the preparing tank (not shown) by the conduit, 1, and the dilute acid is supplied to the cask B from the preparing tank (not shown) by the conduit, 2. The chloride of lime and the hydrochloric acid may be replaced by any other solutions capable of giving out chlorine. The two casks, A and B, should be of about equal capacity, and of such size as is proportionate to the quantity of material to be treated. The vat, C, holds 800 kilogrammes of roasted ores, sands, or residues, the casks, A and B, should hold about 170 litres each.

The two solutions are now allowed to reach the vat C in equal proportions by a common pipe, 4, connected to the conduit, 3, which receives the solutions from the casks A and B through the cocks 5, and supplies them to the pipe 4 by means of the plugs 6. These solutions percolate through the mass of ores, sands, or residues in the vat C into a receptacle, *c*, beneath.

The nascent chlorine formed by the mixture of these two solutions combines with the gold, which has by the previous process been placed in the best conditions for such combination, and a soluble chloride of gold is formed, which is at once run off, thereby avoiding the further contact of the said chloride of gold solution with the ores, which would tend to reduce again to the metallic state the gold contained in the said solution. The finer the gold in the ores, sands, or residues, the quicker is the operation.

The progress of the chlorination can be continuously tested by means of chloride of tin, which gives a precipitate well known as Purple of Cassius. At first the reaction is weak, then greater, attains a maximum, and then diminishes. When the reaction is almost insensible, the operation is stopped.

The liquid collected contains chloride of gold and a little chloride of silver, resulting from the silver which has not been attacked in the preceding process. The silver is precipitated with a small quantity of iodide of potash or with sulphuretted hydrogen, and the silver may be obtained by any known process. The solution of chloride of gold is precipitated by any ordinary method.

If the auriferous ore under treatment is found to contain a great quantity of silver, it is necessary to mix it with common salt in sufficient quantity to obtain the complete conversion of silver into chloride of silver. When the ore is not sulphurous, there is added to it 5 to 10 per 100 of any pyrites before roasting. Afterwards the roasted ore is washed with a concentrated solution, warm or cold, of common salt or hyposulphite of soda. The chloride of silver dissolved is treated by iodide of potash and precipitated as an iodide of silver. The silver can be reduced into a metallic state by pieces of metallic zinc. D is the cask for the copper solutions. E is the cask for the gold solutions. 12 is a pipe for cold water, which supplies water, by the cock and short hose 13, to the upper casks, and by the cock and long hose 14, to the lower casks D and E. 17 is a steam cock with long hose to supply steam to the lower casks. 18 is a cock to convey solutions from ore vat C into the conduit 19. 20 are conduits receiving solutions from vat C and conveying them to the lower casks D, E. 21 are cocks for supplying solution from casks D E to conduit 22, which conveys it to the precipitating tanks, etc. 23 are plugs in conduit 19. 24 are water gates in conduits 1, 2, and 19, and 25 are troughs leading from cocks 21 of casks D to the conduit 22.

**The Ottokar-Hoffman Gold and Silver Chlorination Process.**—If rich auriferous silver ores, in which the percentage of gold is high, almost equal to that in silver, should be subjected to a chloridizing roasting, then impregnated with chlorine gas leached with water, for the purpose of extracting the gold, and finally leached with a hyposulphite of lime for the silver, although a high percentage of silver might be extracted, the yield of gold would only amount to about 50 per cent., more or less.

This result is not easily explained. The gold may be influenced somehow during the roasting by the base metal chlorides preventing the gold from being attacked by the chlorine gas. On the other hand, if the base metal chlorides and the chloride of silver are extracted previously to the impregnation with chlorine, both metals, silver and gold, can be got out very close.

The *rationale* of the process is as follows. The concentrated sulphurets are subjected to a chloridizing roasting in reverberatory furnaces: these furnaces, although old-fashioned stirring furnaces, are considered preferable for this class of ores to any mechanical furnace in use, especially to the furnaces with continuous discharge. The concentrated sulphurets require a very perfect roasting, and Mr. Hoffman claims that the continuous discharge furnaces do not give time enough for very highly sulphuretted ore to become thoroughly desulphurized. Such high grade ores require close attention, and the process must be under perfect control by the roaster. However, it does not take nearly so much time as required by Plattner's gold chlorination.

After the ore has been roasted, it is spread on the cooling floor, and sifted when cool through a sieve of ten to fourteen meshes to the running inch. The sulphurets are heavy enough after roasting to make very little dust during the sifting, so that the inconvenient moistening can be obviated. The lumps are saved till a larger amount accumulates; they are then pulverized in a dry battery and slightly roasted.

The roasted and sifted ore is charged into tanks with filter bottoms, in quantities from 2½ to 3 tons, and leached with water to extract all soluble base chlorides. The water, if saturated with these base chlorides and some salt, of which some quantity is generally left in the ore undecomposed after roasting, acts on the chlorides of silver like a concentrated brine, and dissolves them. To prevent the escape of this dissolved part of silver, Hoffman does not admit the water from above the ore, as is usually done, but from the filter bottom, which, by means of a slight pressure, is forced to ascend through the ore to the top of the vat. In this way the concentrated solution accumulates above the ore, and in diluting it by a stream of water, and permitting the solution to flow out through the filter bottom, the chloride of silver is precipitated on and through the ore, which is then extracted, with the balance of the silver. The operation affects the fineness of the bullion somewhat if there is a considerable amount of lead in the ore, but not materially. The leaching of the silver is done as usual, by leaching the ore with a solution of hyposulphite of lime, and precipitating the silver with polysulphide of calcium.

After the silver has been extracted, the solution of the hyposulphite is allowed to run out till it disappears under the surface of the ore, when clear water is introduced again in order to displace all solution. The delivered ore is then removed from the tank to a dry kiln, where it is left for a time till the surplus water has evaporated. After this it is charged back into the tank, still moist. This second handling and drying cannot be avoided, as the ore after leaching is too wet to permit of a free passage of the chlorine gas; but if the works are arranged properly this partial drying causes neither much delay nor much expense.

The gold in the ore is now in metallic condition, and very bright and clean, permitting a very close extraction. The rim of the tank is provided with a groove, which is open toward the inside, 2½ inches deep and 1½ inches wide. In this groove fits the cover of the tank, leaving, however, a play of one-eighth of an inch around the circumference. The cover being made of inch boards, the staves of the tank will project 1½ inches above the cover. This arrangement facilitates the operation of making the cover air-tight. This is first done with clay, and then a sheet of water one inch deep is poured on the top of the cover, thus making it perfectly air-tight and preventing the escape of gas into the working-room, while the tank is charged with water for the purpose of extracting the gold. The cover is provided with two pieces of 1¼-inch gas-pipe, 6 inches long, and a square opening 6 by 6 inches. During the time of the impregnation of the ore with gas the pipes are closed with balls of clay. As soon as the charge is ready for the extraction of the gold, the balls are removed, and one of the pipes is connected with the hose of the water tank, while the other, by means of the hose, is connected either with another tank already prepared for chloridizing, or with the ash-pit of the roasting furnace. This is done to utilize the surplus of chlorine gas, and to protect the workmen from its very injurious effects. Care must be taken to have sacks placed on the top of the ore right under the water-pipe, and kept in place by weights—two bricks will do—to prevent the stream of water working into the ore. The square opening serves for examining the progress of the gas in the ore, and can be made air-tight by a good-fitting cover and clay.

The chlorine gas is generated in a leaden gas generator, which is not heated as usual, by direct application of fire, but by steam. For this purpose the generator is placed in a tight-fitting box, leaving a space of two inches around the side and the bottom for the circulation of steam. The rim and cover are kept outside the box. The steam enters on one side through a half-inch pipe, while the other side of the box is provided with a one-inch exhaust pipe and an outlet for the water.

Whenever steam can be had in chlorination works, this arrangement will prove very convenient and useful. The operator has the temperature entirely under his control. The least turn on the valve increases or decreases the heat, and, of course, regulates the generation of steam. The discharge pipe of the generator projects a short way out of the box. On this pipe is fastened a piece of hose about 2 feet long, which can be closed with a thumbscrew clamp. The hose lies in a covered trough, which leads outside of the gas-house. In discharging the generator, a small stream of water is permitted to flow through the funnel into the generator, by which the gas is forced through its usual outlet into one of the tanks. When the generator is filled the stirrer is set in motion and the thumbscrew on the hose loosened. In this way the generator can be discharged without molesting or injuring the men.

The cover of the generator is closed tightly with clay, and only removed in case of repair. The gas is conducted through a leaden pipe, intersected with rubber hose. Each tank is connected with the main pipe, and can be disconnected by the use of the thumb-screw clamp. The pipe through which the gas enters the tank is independent of that through which the solution discharges. It is placed higher and as close to the false bottom as possible. After the gas has been a sufficient time in contact with the ores, the gold is extracted by water and precipitated in the usual way with sulphate of iron. The gold obtained is of extreme fineness, varying from 970 to 990 fine.

In treating very rich ores, containing, say, $700, or £140 to £180 gold per ton, the solution carrying out the gold is of a very lustrous yellow colour, and the precipitated gold accumulates on the bottom in spongy lumps of great specific gravity, some of them showing scales of bright gold, and some of these, as the microscope will show, may prove to be crystallized gold. In leaching very little more time is required for rich gold ore than for poor.

If the ore is copperous, considerable copper will be carried out with the gold solution, colouring it green. In order to save the copper, the solution, after the gold has been precipitated and settled. is decanted into the copper tanks; but before doing this it is advisable to draw the solution into a second gold tank, in order to catch the gold which may be accidentally carried off with the stream, and to keep it standing sufficiently long to allow the gold to settle again.

**The Von Pateras and Roeszner Process,** which is based on the solubility of chloride of silver in a concentrated salt solution, is adapted for the extraction of gold, silver, and copper. The chloride of copper, like chloride of gold, is soluble in water, whereas chloride of silver is not, and would, without a salt solution, remain behind in the residue. If, therefore, chlorine is conducted through a salt solution to saturation, the chloridized salt solution will dissolve gold, silver, and copper simultaneously, and is accordingly adapted for ores containing these three metals.

Auriferous silver ores are therefore submitted to a chloridizing roasting, they are then charged into large vats and leached with the cold solution of salt saturated with chlorine. The leach waters are drawn off into the precipitating tanks and diluted with clear water, which produces a white precipitate of chloride of silver. The diluted salt leach has to stand now for twelve to fourteen hours before all the silver chloride has settled, and the fluid above it becomes clear enough to be drawn off into the gold-precipitating tank, where the gold is thrown down by a solution of sulphate of iron. When all the gold has settled, the fluid is drawn into the third copper-precipitating tank, which contains scrap iron, on which the copper precipitates.

This method of precipitation necessitates the subsequent reduction of the chloride of silver, by means of zinc and sulphuric acid, to metallic silver. The large quantity of water required to precipitate the chloride of silver—amounting to nearly fifty per cent. of the volume of the salt leach—dilutes the copper solution too much, and renders the precipitation of this metal difficult. It is better, by means of metallic copper, to precipitate first the silver and gold as metallic gold and silver, and then to draw off the liquid into another tank and precipitate the copper with scrap iron.

Roeszner roasts the ore with salt, extracts a part of the silver by Augustin's method with a hot solution of salt, and then treats the residue with a solution of salt and chlorine, and hot concentrated salt solution, alternately, for the extraction of the gold and the remainder of the silver.

According to Plattner, an auriferous and argentiferous substance, when roasted with common salt, forms chlorides of silver and gold. The chloride of gold is transformed into protochloride of gold, losing two equivalents of chlorine, when heated to somewhat below 200° C., and the protochloride of gold is decomposed into chlorine and metallic gold when heated to about 240° C. This easy decomposition of chloride of gold chiefly accounts for the imperfect yield of gold when treating silver ores by the amalgamation process, or submitting them to a lixiviation with common salt.

Tanikovitz published an account in the *Oesterreichische Zeitschriften* of 1863, in which he advised the submission of the ore first to a perfect oxidizing roasting, while admitting steam into the furnace, then the addition of salt, and chloridizing roasting at a lower temperature. Roeszner advises the leaching of this roasted mass with hyposulphites in preference to a solution of common salt and chlorine water.

Kiss has shown that roasting with salt produces a chloride of gold, which forms with hyposulphite of soda, or lime, a soluble double hyposulphite of gold and soda salt.

To prevent separation of metallic gold from the chloride of gold in the roasting process, the temperature must not be too high during the last roasting period, and some surplus chloride, produced from common salt and easily decomposable metallic sulphates, must be present: if none are in the ore, sulphate of copper should be added. It is advisable to wash the roasting mass with water previous to its treatment with hyposulphites. The chloride of gold, by the action of hot water, is decomposed into protochloride and metallic gold, and though insoluble in cold water is slowly decomposed by it in darkness.*

The distinction between Kiss's method and the Roeszner and Von Pateras method is that the former method is based on the fact that chlorides of silver and gold are soluble in a solution of hyposulphite of lime; and the process has been brought to great perfection in America—as far as the treatment of silver ores are concerned—by the improvement introduced recently by Mr. Russell, who substitutes hyposulphite of soda, with the addition of what he calls the extra solution. His process is described in my work on the "Metallurgy of Silver."

As before explained, in Von Pateras and Roeszner's process the ores or concentrations are submitted to a chloridizing roasting with salt, and the roasted mass is then treated with a solution of common salt saturated with chlorine. This process is modified by first leaching the ore, after a chloridizing roasting according to Augustin's method, with a hot concentrated solution of common salt; the residues are then alternately treated by cold solution of salt saturated with chlorine, and hot concentrated solution of salt, to extract the gold and the remaining silver. Both metals are precipitated from the hot solution by copper; the cement silver is cupelled to drive off all impurities, and the gold separated by parting with acids.

**Hauch's Treatment of Telluride Gold Ores.**—A method of treating these ores which has been devised by Anton Hauch, Assayer to the Hungarian Government, deserves notice here. The gold ores of Nagyag and Offenbanya are highly prized on account of their contents in tellurium, which is rarely met with in mines. Tellurium is important on account of its employment in the manufacture of thermo-electric batteries, and an expeditious and cheap method for its extraction from ores

---

* See Crookes and Röhrig's "Practical Treatise on Metallurgy."

is, therefore, much to be desired. The analysis of the Hungarian ores gave the following results: tellurium, gold, silver, a considerable percentage of quartz, carbonate of lime, and carbonate and sulphide of manganese.

The idea which the inventor carried out was to produce the tellurium by means of a cheap acid solution, from which it could be easily separated. In defining his method, the manganese combinations had to be closely studied. A quantitative determination of the ores gave the following results: 30 to 40 per cent. of quartz; 10 to 20 per cent. of carbonate of lime; 15 to 20 per cent. of carbonate and sulphide of manganese; 5 to 8 per cent. of lead sulphide; 1 to 2½ per cent. of copper sulphide; 5 to 8 per cent. of clay; 1 to 4 per cent. of zinc sulphide; with some cobalt, nickel, antimonium, and arsenic, besides tellurium, gold, and silver. If the ore was submitted to an oxidizing roasting, a portion of the tellurium volatilized, carrying off some gold and silver, which loss could be avoided by condensation.

During the roasting the carbonate of manganese and any manganese sulphide present were converted into manganese oxide, the sulphide of manganese being no doubt influenced by the presence of lime. Under the influence of hydrochloric acid the manganese oxide gave chlorine. It was found also that during roasting a large portion of the gold became metallic, in such a way that fifty per cent. of the gold could be extracted by amalgamation.

In treating the roasted ores with dilute hydrochloric acid in lead-lined wooden tubs provided with a stirring apparatus, a disengagement of chlorine takes place, which converts the metals to be obtained into a soluble condition, with the exception of silver. The surplus chlorine is conducted into tubs filled with water, and this chlorine water can be used as a solvent for the tellurium sponge. After the solution is drawn off from the ore, sulphuric acid is added, which precipitates gypsum and lead sulphate.

A further decantation of the solution from the precipitate is effected, and a solution of iron sulphate is added, which precipitates the gold. The solution is now filtered off, and further treated with metallic zinc, which precipitates the tellurium as black sponge. This tellurium sponge is washed with water acidulated with hydrochloric acid, filtered rapidly, dried and melted, without any further addition, in a platinum crucible, giving raw tellurium, which always contains some lead, copper, nickel, and antimonium.

If the tellurium sponge is first dissolved in chlorine water, then treated for a considerable length of time with sulphuric acid, pure tellurium is obtained and can be melted down as such. After the separation of the largest proportion of the gold and the tellurium, there remains chloride of silver and some gold in a still soluble condition in the solid residues. In treating the damp residues with a solution of iron sulphate, the gold is converted without loss into its metallic state. From the residues, both silver and gold can be obtained by amalgamation, and where practicable can be smelted down with lead.

A practical trial on 7 kilos of telluride ores, containing 21·25 grams gold and 21·06 grams silver, gave the following result:—

The oxidizing roasting in a muffle furnace took an hour and a half and the roasted ore weighed 6·498 kilos; there was, therefore, a loss in roasting amounting to 0·502 kilos, or 7·2 per cent. The contents of the roasted ore were 21·175 grams gold, and 20·25 grams silver; therefore the loss in roasting in gold 0·075 grams, or 0·35 per cent., in silver 0·81 grams, or 3·87 per cent. From these 6·498 kilos of roasted ore only 6 kilos were taken for the operation, containing 19·55 grams gold and 18·70 grams silver. These 6 kilos were introduced slowly and under constant agitation into a mixture of 3 litres water, 2 litres raw hydrochloric acid (20° B.), and 0·3 kilos concentrated sulphuric acid. An active development of carbonic acid and disengagement of chlorine took place, and the mass foamed up considerably. After twenty-four hours, during which time the mixture is repeatedly stirred, 2 litres of water were added, and then left standing quietly for two hours, and the solution, which was not as yet quite clear, was then decanted. The solution was renewed three times and the resulting 10½ litres of brine were treated with 2 litres of iron sulphate solution (25° B.) and well stirred. The metallic gold was completely precipitated after twenty-four hours, and the liquid separated by decantation. The residue after filtering was dried and melted, with an addition of lead; and after cupellation 16·67 grams, or 82·2 per cent. of the pure gold was obtained. This product could have been increased to 90 per cent. if a fourth solution had been used.

The brine decanted from the gold was treated with 2 kilos metallic zinc. The tellurium precipitated after twenty-four hours as a black sponge. After decantation, filtering of the residue, drying and smelting, 30 grams of raw tellurium were obtained, or 0·43 per cent. of the weight of the ore. The richness of the ore in tellurium was found to increase with its contents in gold.

For the precipitation of the tellurium 200 grams zinc were required—that is, 3 per cent. of the weight of ore. The residue, containing some iron sulphate solution, weighed 5·25 kilos; consequently the loss in lixiviation was 0·75 kilos, or 12½ per cent. This residue contained 3·88 grams gold and 17 grams silver. From the solution was obtained 16·07 grams gold, and adding 3·88 grams contained in the residue, makes 19·95 grams gold, which according to the assay contents of 19·55 grams proves a gain of 0·40 grams, or 2 per cent.

The assay showed 18·70 grams silver, and the residue contained 17·03 grams; there was a loss, therefore, of 1·67 grams, or 8·9 per cent. This discrepancy can only be accounted for by the impossibility of obtaining from the ores a perfect assay sample to represent a correct average; and, further, during the assaying process, either in smelting or cupellation, the volatilization of tellurium will carry off some of the precious metal with it.

This method can be carried out on a large working scale, and will prove a cheap mode of extraction for the metals contained in tellurides.

# CHAPTER X.

## *THE ELECTRO-METALLURGY OF PRECIOUS METALS.*

ELECTROLYSIS AS APPLIED TO GOLD AND SILVER—The Electro-Chlorination of Gold Ores: Cassel's Process—Greenwood's Electrolytic Chlorination Process—Electrolytic Precipitation of the Gold—Apparatus for Continuous Lixiviation—Julian Process—Dr. Kiliani on Electrolytic Refining of Copper.

THE success which has attended the separation of copper from its ores by electrolysis has led to researches as to separation of gold by electricity, but so far we are only able to point to one process wherein electricity plays the rôle of the chlorine generator, and decomposes the chloride of gold produced.

It may be said that the first practical application of electricity for metallurgical operations was made in 1865, by Mr. T. B. Elkington, who then introduced his electric copper refining process.

As by this means not alone perfectly pure copper is obtained, but a separation of gold, silver, and other metals is effected, which are often associated with impure copper, the electric refining of copper is now extensively practised, as possessing many advantages over the older methods.

The principle upon which the separation of copper from other metals by electrolysis is effected rests upon the property of copper solutions to deposit their copper with a current of low electro-motive force about three volts, whereas other metals require a higher electro motive force. The copper to be refined is cast into thin slabs, and they form the anodes, being suspended in a tank filled with sulphate of copper solution. The cathodes are thin sheets of pure copper, having the same surface as the anodes. The electric connection being made, and the current turned on, the sulphate of copper solution is decomposed, the copper precipitating on the cathodes, while the sulphuric acid set free dissolves an equivalent proportion of copper from the anodes, whereby the solution would be maintained of a standard strength were it not for the impurities which the copper anodes contain, which also enter into and weaken the solution, so that the bath has to be renewed.

The insoluble material which collects at the bottom of the tank is removed, and the gold and silver which it contains are recovered by other methods.

The foregoing account, of course, is merely an outline of the process of manipulation which has now completely revolutionised the copper refining industry.

**Electro-Chlorination of Gold Ores: Cassel's Process.**—Many attempts have been made to obtain the precious metals by means of electrolysis, and several ingenious methods have been devised, but so far no practical success has crowned these efforts, although the results hitherto obtained on a small scale warrant the expectation that eventually the difficulties which still preclude the working of these processes on a large scale will sooner or later be overcome.

A Glasgow company is developing the Cassel process, which is one of chlorination. The apparatus consists of a large drum, within which are arranged a number of dense carbon rods; these rods form the anodes, or positive electrodes, and are metallically connected with the positive pole of the dynamo, while the negative pole of the dynamo is connected with the hollow iron shaft of the drum, which serves both as axis to the drum and also as negative electrode of the apparatus. This hollow shaft terminates through stuffing boxes in hollow standards or tanks, where finally the gold accumulates.

In carrying out the processes, the drum is charged with 4,000 to 5,000 lbs. of ore, and salt and water are added. This drum is set revolving slowly—eight to ten revolutions per minute. The electric current being turned on, the chloride of sodium decomposes, and chlorine and oxygen are generated at the anode. During the revolution of the drum the ore comes into contact with the carbon elements, which discharge the chlorine from the salt solution. The gold particles thus acted upon are converted into terchloride of gold, which, as soon as formed, dissolves in the solution.

Mr. Cassel, it seems, had to contend with the difficulty that hydrochloric acid was formed during the reaction. This dissolved the iron oxide present in nearly all ores, roasted iron pyrites being the materials generally operated upon. That difficulty he overcame by adding caustic lime, which neutralized the hydrochloric acid as fast as formed, so that no iron could be taken up by the gold solution. As hypochlorite of lime is formed, this again is decomposed by the water present, and gives up its chlorine to be taken up by the gold.

The ultimate products of the reaction are chloride of sodium, which has not been all decomposed, chloride of calcium, terchloride of gold with the gangue at the anode; and chloride of sodium and caustic soda at the cathode. In the iron shaft are bored a number of holes, and the shaft itself is covered with asbestos cloth, which, while preventing the gangue from entering the shaft, allows the dissolved gold to penetrate through the cloth. After the addition of the lime, which precipitates all other dissolved metals present except the gold, the latter metal is rapidly dissolved, and is deposited by the electrical action in the interior of the pipe in a finely divided metallic state; thence it is carried into the hollow standards by means of an Archimedean screw fixed in the pipe. The chloride of gold is decomposed at the cathode into metallic gold and chlorine. The standards are provided with movable doors, from which the gold precipitate is withdrawn, and after drying is melted into ingots.

From this description it will be seen that chlorine can be produced by electrolysis from salt solutions; and although the above method does not seem the most practical way of applying it for the production of the terchloride of gold, it is likely that improvements will follow which will perfect Mr. Cassel's otherwise excellent idea. Gold is more readily attacked by nascent chlorine gas than by chlorine held in solution by water; and it seems that pressure also assists rapid chlorination, as is shown by the discovery made by Mr. Mears, as well as by the Newbury-Vautin process.

**Greenwood's Electrolytic Chlorination Process.**—As I have shown, Cassel tried to produce chlorine by electrolysis, and thereby convert his gold into a soluble chloride; but his method did not succeed, as he mixed his salt with the ore, and tried to do his work by means of a complicated apparatus which did not give the anticipated practical results.

By the Greenwood system the whole work is simplified, as the inventor simply electrolyzes a solution of common salt, or chloride of sodium, when evolution of chlorine takes place at one pole, and hydrogen and caustic soda separate on the other pole. The chlorine so obtained can be led into vats charged with ore according to Plattner's system, or it can be absorbed by water, and the solution then utilised in rotating barrels according to Mears's system.

In generating chlorine by the process devised by Mr. Greenwood, a current of electricity of from five to six volts electromotive force is employed, to decompose a saturated solution of chloride of sodium, or common salt. For this purpose one or more large battery jars are taken, in which are placed a number of carbon rods, and arranged in a circle around the jar, so as to form a suitable electrode, which is connected with the negative pole or terminal of a dynamo or other electrical generator. Inside each battery jar is placed a porous cell of suitable dimensions, in which also is arranged a number of carbon rods to form a suitable electrode, which is connected with the positive pole or terminal of the dynamo or other electrical generator.

The battery, or outer jar, is connected with a reservoir, containing a saturated solution of chloride of sodium, which flows into and enters the jar named at the bottom; and in slowly circulating upwards the solution is decomposed by a current of electricity passing through the said cell. When decomposed, the solution flows cut at the top of the cell, the hydrogen liberated by the process also passing away at the same time.

The inner, or porous cell, is also connected with a reservoir containing water, which flows into and enters the said cell at the bottom, and in slowly circulating upwards, absorbs the chlorine evolved at the positive electrode by the decomposition of the chloride of sodium as above described, and forming a chlorine solution of the strength required.

Where it is desired to produce the chlorine as a gas and not in solution, the gas is led into a gas holder, or led into the charging vats direct.

The flow of water in the inner, or porous cell, is advantageously regulated and maintained in a simple and automatic manner by placing the supply reservoir at a slightly higher elevation than the receiver, into which the chlorine solution is discharged; and according to the quantity of water flowing in, the speed of such flow, and the current of electricity passing through the said cell, the water circulating therein becomes more or less saturated with chlorine, and forms the chlorine solution.

Greenwood's chlorine generator is shown in Fig. 138; and Fig. 139 shows the arrangements and construction of the plant in elevation. Fig. 140 shows a vertical section, and Fig. 141 the horizontal section, of the electrolytic cell for decomposing the solution.

Figs. 140 and 141 show the improved battery jar or electrolytic cell. *a* represents the large battery jar or outer cell; *b, b,* represent the carbon rods arranged round the interior of the jar, and connecting at the top to a ring or frame *c* of suitable conducting material; *c'* is the electric conductor connecting the ring *c* with the negative pole, or terminal of the dynamo or other electric generator.

Inside the battery-jar, *a*, is placed a porous-cell, *d*, of suitable dimensions, and in this inner cell is also arranged a number of carbon-rods, *e, e*, on a ring or frame, $e^2$, to form a suitable electrode, which is connected by means of the electric conductor, $e^3$, with the positive-pole or terminal of the dynamo, or other electric generator.

The battery is shown at A, in Fig. 138, and has its outer jar, *a*, connected with the reservoir, $d^1$, containing a saturated solution of chloride of sodium, which flows into the bottom part of the jar, *a*, through the pipe, $e^1$, and, in slowly circulating upwards, the solution is decomposed by a current of electricity passing through the said cell, *a*. When decomposed, the solution flows out at the top of the cell through the pipe, $a^1$, into the waste-tank, W; the hydrogen liberated by the process also passing away through this pipe at the same time.

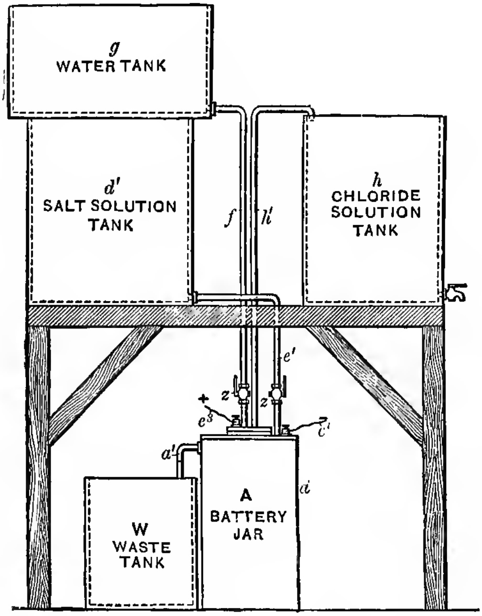


FIG. 138.—ELEVATION OF CHLORINE GENERATOR (Greenwood's Process).

The inner, or porous, cell, *d*, is connected by means of the pipe, *f*, with a reservoir, *g*, containing water, which flows into and enters the said cell, *d*, at the bottom part thereof, and, in slowly circulating upwards, absorbs the chlorine evolved at the positive electrode by the decomposition of the chloride of sodium by the current of electricity, thereby forming a chlorine solution, which is discharged into the receiver, *h*, through the pipe, $h^1$.

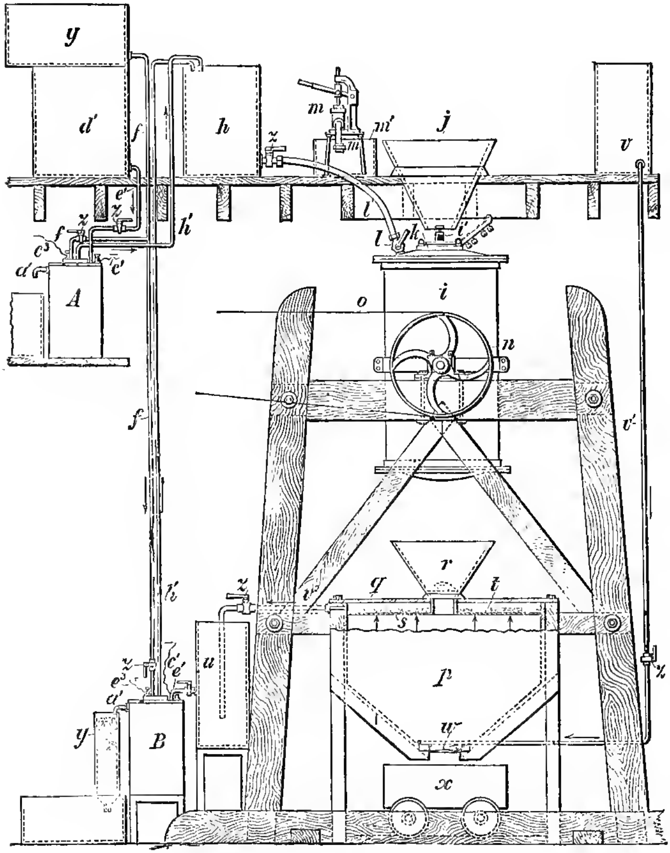


FIG. 139.—GREENWOOD'S PROCESS. Elevation of Plant.

To regulate and maintain the flow of water in the inner or porous cell, *d*, and to effect the discharge of the chlorine solution into the receiver, *h*, in a simple and automatic manner, the water supply reservoir, *g*, is placed at a higher elevation than the receiver, *h*, into which the chlorine solution is discharged, and according to the quantity of water flowing into the cell, *d*, the speed of such flow, and the quantity of electricity passing through the said cell, the water circulating therein becomes more or less saturated with chlorine, and a chlorine solution of any strength can be obtained. Taps or cocks, *z*, *z*, are placed on the pipes, *e* and *f*, respectively, so as to allow of regulating and shutting off the flow of the liquids therein as desired.

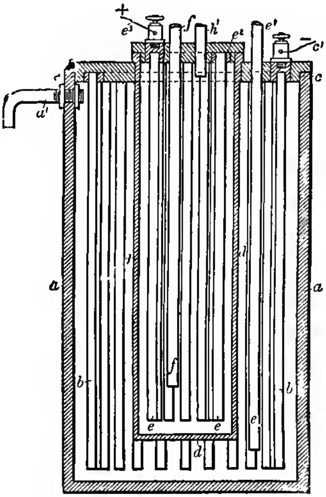


Fig. 140.—Vertical Section of Electrolytic Cell.

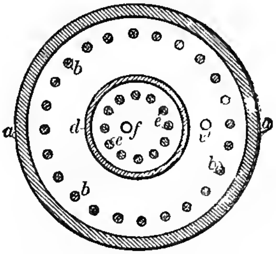


Fig. 141.—Horizontal Section of Electrolytic Cell.

From what has been said, it will be readily perceived that this method of producing chlorine gas would offer a considerable advantage over the methods heretofore employed, where the alkaline or earthy chlorides are decomposed by chemical reagents, involving large expenditure, especially when such chemicals have to be transported to distant countries.

This method, when electricity can be cheaply produced, especially where water-power is obtainable, would combine cheapness and efficiency, reducing the cost of chlorine to a minimum.

What quantity of chlorine gas is actually required to effec tually chloridize an ore would be hard to say in advance, but it is evident that with a pure quartzose ore holding fine free gold the quantity would be reduced to a minimum, whereas certain earthy ores would require a very large quantity, and only experimental researches can determine the volume or weight of chlorine gas to effectually extract the gold contained therein. No doubt, in many cases, a large waste of chlorine takes place, and there is no reason why the same chlorine solution should not be used over and over again till the same has reached its point of saturation, and only then should the gold be precipitated. The chlorinating barrels could be easily arranged in such a manner by means of siphons to decant the gold solution from one barrel into another, or from one vat into the other, and when it is found that the same does not dissolve any more gold, the solution can be led into the precipitating tanks.

As chlorine is soluble to a considerable extent in water, that liquid absorbing at ordinary temperature about twice its volume, and acquiring the colour and odour of the gas, this chlorinated water holds a large quantity of the gas in solution, considering the high specific gravity of chlorine, which is 2·47. In round figures we may state that one cubic foot of chlorine will weigh 3 oz., and therefore a cubic foot of water can hold 6 oz. weight of chlorine in solution when saturated.

If the information is correct that from 20 to 50 lbs. of chloride of lime are employed in the Mears and Newbury-Vautin process to chloridize one ton of ore, and that the bleaching powder under most favourable conditions only contains 30 per cent. of chlorine, this would make from 6 to 15 lbs. of chlorine gas per ton of ore.

I shall now assume a case where 10 lbs. of chlorine gas is required to treat a ton of ore, and that the chlorine is produced by the Greenwood method. From experiments which were conducted in connection with the liberation of chlorine by electricity, the results showed that a current of 421 ampères would be required to liberate 1 lb. of the gas in one hour. This shows a loss of only 18 per cent., that is to say, that according to theory this current should liberate 1·22 lbs. instead of 1 lb.

The electro-motive force required to urge the current through the cell varies with the surface of the electrodes and of the porous diaphragm. I found that at the ordinary temperature the E.M.F. varied as follows with the current passing the cell. I am indebted to Mr. G. C. Fricker, of the Gulcher Electrical Works, for kind assistance in these experimental researches.

| | | | | | | |
|---|---|---|---|---|---|---|
| With | 1·025 | Ampères | there were | required | 3·38 | volts. |
| ,, | 1·5 | ,, | ,, | ,, | 3·81 | ,, |
| ,, | 2·005 | ,, | ,, | ,, | 4·3 | ,, |
| ,, | 2·5 | ,, | ,, | ,, | 4·68 | ,, |
| ,, | 2·98 | ,, | ,, | ,, | 5·1 | ,, |
| ,, | 3·4 | ,, | ,, | ,, | 5·78 | ,, |
| ,, | 4·01 | ,, | ,, | ,, | 6·1 | ,, |

In practice I think it would probably be found most economical to work at such a current density as to bring the E.M.F. per cell down to about 3½ to 3 volts, and with thin porous vessels this could be easily done. The power absorbed would then be about 3 horse-power indicated per lb. of chlorine liberated per hour. This estimate should represent the process at its worst, for an efficiency of 67 per cent. from the cylinders of the engine to the useful electrical work in the cells, and it *may be* found quite practicable to reduce the E.M.F. to 3 volts, and approach nearer theoretical figures in the liberation of the chlorine from the salt solution, so that at its best the power might be estimated at 2½ horse-power per lb. per hour.

Supposing now that 24 tons of ore are to be treated in twenty-four hours requiring 240 lbs. of chlorine gas, or 10 lbs. per hour, this will necessitate a 25 horse-power engine (indicated).

If fuel is procurable at a reasonable price the production of the chlorine will only represent the cost of the fuel and salt, the latter being a cheap commodity; against the costly sulphuric acid of which at least 1,000 lbs. would be required and about 900 lbs. of bleaching powder, if the Mears or Newbury-Vautin process be employed. In distant countries, such as the Transvaal, the cost of these chemicals must represent at least

£24, or about £1 per ton, whereas the cost of producing the 240 lbs. of chlorine by electricity would represent say £5, or about 4s. per ton of ore—a striking illustration of the value of the process for metallurgical operations, if practicable on a large scale.

My experimental researches have shown me that the larger the surface the inner cells present the easier the production of chlorine, and I have accordingly devised the construction of a new cell, which I think could be advantageously employed. To generate chlorine from a salt solution I proceed by preference as follows: I prepare a large cell by taking a stoneware round jar 3 feet in diameter and 3 feet high, shown in the figure at A A, Fig. 142. This is covered with a cover having nine round openings, the central one being one foot in diameter, the other four concentric ones, three-quarters of a foot, and the other four, one-half a foot in diameter. The cover, B, ought to fit snugly round the edge of the pot or outer jar. Besides these nine openings there is also one opening of about 1 to 1¼ inch in diameter in the cover near the circumference.

These nine openings serve for the introduction of the inner or porous jars or cells. Besides these there are 16 to 32 small holes around the periphery of each of the porous jars in the metal cover of 1 inch diameter, which serve for the introduction of the carbon rods, shown at R. The cover having been put on, the carbon rods are inserted, securely fastened, and connected electrically with the negative pole of the dynamo, and the rods are electrically connected among themselves by means of the metal cover, forming the anode. The inner or porous cells, nine in number, are now inserted in their proper places—these cells are shown in section in D, D, D. They are all provided with tightly-fitting metal covers, E, E, and concentric with the periphery each cover carries 16 carbon rods, S S. These carbon rods are connected with copper wire, and when all in place are connected between themselves, as shown at F F, whereby the whole cell is so arranged that the carbon rods, R, plunging into the main outer jar constitute the anode, and all the carbon rods plunging into the inner or porous cells constitute the cathode of the element. If more than one of these cells is employed, they can be arranged in series.

The reason I employ nine or more inner porous cells of small diameter to one outer cell is, that I get a larger surface than when using one large cell, which facilitates the passing of the chlorine from the outer to the inner cell.

After the nine inner cells are put in position and securely luted down, and electrically connected between themselves, connection between the outer and inner cells and the battery is established. A saturated solution of salt is now allowed to flow from the tank, T, into the outer cell through the pipe P, till the same is filled. Water is now turned on from the tank, W, which flows through the tubes, C, C, C, into the inner cells, till they are completely filled, in case a solution of chlorine water is desired. The element is now ready and the electric current is turned on, resulting in the electrolysis of the salt solution, causing a decomposition of the salt, or chloride of sodium, which splits up into chlorine gas, passing into the inner porous cells, and caustic soda. This remains in solution in the outer cell, and with the hydrogen escapes through the pipe, N.

After a short time the water in the inner cells, D D, becomes saturated with chlorine, and the faucet, M, of water, is slightly opened so as to allow a small current of water to flow through the india-rubber tubes, C, and the glass-pipes, H, to discharge near the bottom of the cells, and in this manner force up the chlorinated water through the tubes, O, O, O, leading to one common reservoir. As shown by the arrows, there will be nine tubes leading a steady fine stream of water into the inner cells and nine rubber tubes discharging upward a steady stream of chlorinated water, which is led into a tank to be used for metallurgical or industrial purposes.

As the salt solution in the outer cell, A, becomes gradually exhausted, a small stream is turned on through the pipe, P, which discharges the same through the tube, P′, near the bottom of the outer cell, and the caustic soda solution will discharge, through the overflow pipe, N, into a basin, I I. Single cells of any diameter with only one porous jar inside can also be employed, and these can be connected together, but it will be found in practice more economical to work in the manner here indicated by exposing a large surface of porous cells inside of one outer cell. If more than one large cell is used, they can all be fed from the same salt solution tank through the distributing pipe, U.

The advantage of using a multiplicity of small inner cells instead of one large inner cell will be seen at once if I state that one inner porous cell, 2½ feet diameter and 3 feet high, exposes 22½ square feet of surface, where nine cells of the dimensions above given have a superficial area of 56½ square feet, which is a material advantage.

If it be desired to produce chlorine gas, and not solution, the inner cells are only partly filled with water, so as to cover the orifices, H H, of the glass tubes, and the chlorine gas is led by the pipes, O O, to the pipe, L L, and conveyed to a gasholder or directly to tubs or barrels containing ore.

Any special method of apparatus can be resorted to for treating the ores, which can be done either in stationary tubs by Plattner's method, or in revolving barrels or cylinders. But if chlorine water be used, I propose to use the same water over and over again, until a point of saturation is reached, when the liquor holding the chloride of gold in solution may be allowed to flow through a similar single cell, as described further on, and to decompose the chloride of gold electrolytically.

The gold liquor flowing into the outer cell being decomposed by the current, the metallic gold is precipitated, and the chlorine passes into the inner porous cell to be collected again.

The Greenwood gold extraction plant does not differ much in its mechanical details from the appliances projected by Messrs. Mears, Thies, Newbury-Vautin, and Pollok. In all of them the ore is treated in rotating barrels, except that Mr. Greenwood has his barrel in a vertical position rotating on its short axis—which appears to be a good idea, as the pulped ore is tumbled about better, and the gold thereby better exposed to the solvent action of the chlorine solution, whereas in barrels

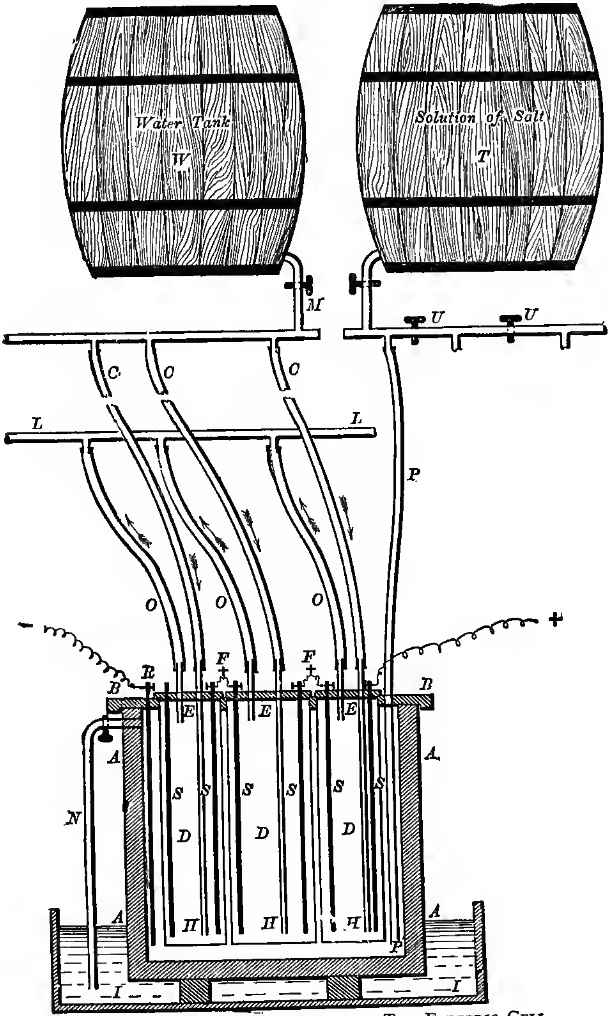


FIG. 142.—GREENWOOD'S PROCESS. THE ELECTRIC CELL.

revolving on the longitudinal axis the ore is apt to slide along the smooth concave surface of the barrel and not so freely disturbed if it "packs."

The ore is charged into the barrel or chlorinator from hopper placed above, which contains the exact weight or quantity required for each charge of ore. This arrangement obviates any weighing or handling of the ore. The chlorinator is in the form of a cylinder, and is constructed of steel or wrought-iron plates, lined inside with pure sheet lead. A further lining of teak or other suitable material is also added, and the outside of the cylinder is well coated with pitch or tar.

The ore is fed into the chlorinator through a manhole, which is hermetically closed after each charge of ore therein, and a chlorine solution is then pumped into the chlorinator through a valve connected therewith, until a pressure of from 80 to 100 lbs. per square inch is obtained. I think that experimental researches will satisfy Mr. Greenwood that this pressure only complicates the apparatus and the operation without any adequate advantages. He also recommends to expel the whole of the air from the chlorinator by means of a small valve which is closed immediately the air is exhausted therefrom, and as it is self-evident that the air would be expelled as the barrel gets filled with solution, the *rationale* of this recommendation is not quite clear to me.

The chlorinator is then slowly revolved for a short time until the gold has been converted into a soluble chloride of gold. The ore and solution are then discharged into a suitable filtering vessel placed beneath the chlorinator. The filtering vessel is a shallow vat substantially constructed of oak or other suitable material, the lower part being cone-shaped, and of the same capacity as the chlorinator already described. The vat is closed by a cover bolted down, in the centre of which is a hopper-shaped inlet for receiving the ore and solution. A perforated wooden diaphragm covered with asbestos cloth or other required filtering medium is fixed from one to two inches below the top of the said vat, through which the auriferous solution filters and flows through a filter of ground asbestos into a suitable receiver. The gold chloride is washed out of the ore by a stream of water which enters the vat at the lowest part thereof and percolates upwards through the ore until the same is perfectly free from any gold in solution. The treated ore is then discharged automatically by means of the large outlet at the bottom of the filtering vessel.

Filtering from the bottom seems to me more advantageous, as on discharging the pulp into the filtering vat and letting the solution drain out a concentrated gold solution is obtained, which is easier to precipitate. To press out any remaining gold solution in the pulp, water should be admitted from the top, and the filtering vessel should be a wide shallow tub, or square tank, so as to present as much surface as possible. It is not easy to filter finely ground ores, and many gold ores have to be ground very finely owing to the extreme fineness of the gold, and if they are clay ores they pack into mud, which is hard to filter. Roasted pyrites which become sandy and gritty do not offer this difficulty, and owing to Mr. Greenwood's ingenious method of precipitating the gold, I strongly recommend the production of as concentrated a gold solution as possible.

**The Electrolytic Precipitation of the Gold.**—From the filtering vessel the gold chloride is conducted into the outer or battery jar of an electrolytic cell, of similar construction to those described in Fig. 140. The electrode of the outer cell is connected with the negative pole of a dynamo, and the electrode of the inner cell is connected with the positive pole or terminal of the dynamo or other electric generator. The gold solution flows into the jar near the bottom, and slowly circulates upwards, and at the same time a current of electricity is passed through the cell. A beautiful reaction now takes place; the gold chloride is decomposed, and falls like a shower of fine spangles to the bottom, while the liberated chlorine passes into the inner porous cell, where it is absorbed by the water circulating therein and forms a chlorine solution. My researches have not as yet sufficiently advanced with solutions containing other chlorides besides gold to determine the effect thereon of the current, and whether they would split up like the chloride of gold. This is recovered in its purest state, 1,000 fine.

The apparatus shown in Fig. 139 represents an elevation (partly in section) of the arrangement adopted for extracting gold by this process.

The receiver, *h*, is charged with the chlorine solution generated by the electrolysis of salt. A is the battery, C$^1$ the conductor from the electrode in the large battery jar to the negative pole of the electrical generator, $e^3$ the conductor from the electrode in the porous cell to the positive pole of the generator; $d^1$ is the reservoir containing a saturated solution of chloride of sodium, which passes to the battery A through the pipe, *e*, and *g* is the reservoir containing water connected by the pipe, *f*, with the porous cell of the battery.

In the drawing *i* represents the chlorinator; *j* represents the hopper above the chlorinator, and the ore is fed into the same through the manhole, *k*, which is then hermetically closed by hand after each charge of ore is introduced therein, and the valves or cocks *l* and *z* being opened, chlorine solution from the receiver, *h*, will flow through the pipe, $l^1$, into the chlorinator, *i*, until it is full; the pipe $l^1$ is then disconnected from the chlorinator and the pipe, $m^1$, of the pump, *m*, connected thereto and a small quantity of water from the cistern, $m^2$, is pumped into the chlorinator until a pressure of from 80 lbs. to 120 lbs. per square inch is obtained therein.

To treat the ore more effectually with the chlorine solution it is advantageous to expel the whole of the air from the chlorinator. For this purpose the chlorinator is provided with a small valve, $i^1$, so that the air contained in the said chlorinator will pass out as the chlorine solution passes in, the said valve being closed immediately the air is expelled.

The chlorinator after being disconnected from the pipe, $m^1$, is then slowly revolved by means of the pulley, *n*, and strap, *o*, from an engine, or in any other suitable manner, until the whole of the gold is dissolved as a gold chloride. The time required for treatment of the ore in the chlorinator varies from about one half-hour to two hours, according to the nature of the ore under treatment. The ore and solution are then discharged into a suitable filtering vessel, *p*, placed beneath the chlorinator. The filtering vessel is a shallow vat, *p*, substantially constructed of oak or other material, the lower part of which is made cone-shaped, as shown, and of the same capacity as the chlorinator.

The vat, *p*, is closed by a cover, *q*, bolted down; in the centre of the cover is a hopper-shaped inlet, *r*, for receiving the ore and solution from the chlorinator. A perforated diaphragm, covered with asbestos cloth, over which is advantageously placed a layer of other suitable filtering medium, such as ground asbestos, is fixed from one to two inches below the top of the said vat, through which asbestos cloth and layer of ground asbestos the auriferous solution filters into a suitable receiver, *u*. The gold chloride is washed out of the ore by a stream of water from a tank, *v*, which water enters the vat through the pipe, $v^1$, at the lowest part thereof as shown, and percolates upwards through the ore until the same is perfectly free from any gold in solution, which can be conveniently ascertained by drawing off a small sample of the solution and testing it in the usual manner. A sliding door, *w*, in the bottom of the vat, *p*, is then opened, and the residue from the treated ore will be discharged automatically by means of the large outlet thus opened into a truck, *x*, placed underneath.

The gold chloride, as water descends through the pipe, $v^1$, passes through the pipe, $v^2$, into the receiver, *u*, and is afterwards conducted from the receiver, *u*, by the pipe, $e^1$, into the outer or battery jar of the electrolytic cell, B, which is similar to the one used for generating chlorine described and illustrated in Figs. 140, 141.

The inner or porous cell of the battery, B, is connected by means of the pipe, *f*, with the reservoir, *g*, containing water which flows into and enters the said cell, *d*, at the bottom part thereof. The gold solution flows into and enters the jar at the bottom and slowly circulates upwards, and at the same time a current of electricity is passed through the cell which reduces and precipitates the gold from the solution, in a perfectly pure state, upon the bottom of the jar, from where it is removed, the interior cell having been taken out previously.

The chlorine, which is liberated at the same time at the electrode in the inner or porous cell, *d*, is absorbed by the water circulating therein, and forms a chlorine solution which passes up through the pipe, $h^1$, to the receiver, *h*.

The water employed for taking up or absorbing the chlorine in the porous cell is advantageously made slightly alkaline.

The electrolyzed solution may still retain a minute quantity of gold undecomposed after passing through the electrolytic cell; the same is passed through a column of charcoal, *y*, in which the gold remaining in solution is deposited.

Taps or cocks, *z*, are placed in all the pipes so as to allow of regulating and shutting off the flow of the liquids therein as desired.

**Apparatus for Continuous Lixiviation.**—Where chlorine is expensive, there will be economy in using the chlorine solution over and over again until it has lost its solvent power, when it may be drawn into the precipitating vats, and the gold thrown down. To accomplish this the solution must be drawn from one barrel to the other, and for this purpose I have projected the apparatus shown in Fig. 143. The barrel, B B, which can be made of wood, well coated inside with asphaltum cement, has the following construction:—A metal pipe, A, which is able to resist the solvent action of the chemicals, is passed through the centre of the bottom of the barrel; its upper portion inside the barrel is bent over in a semi-circle, $A^1$, and this pipe is called the *siphon tap*. Where the pipe passes through the bottom staves, it has a screw, S, 2 to 3 feet in length, working in a female fixed concentric on to the staves, and the whole made air and water tight by means of a stuffing-box. The screw is divided into two equal portions; the centre is marked O, and then the screw is divided, up and down, into inches, these being subdivided into quarter-inches. At a short distance from the portion of the screw which projects outside the barrel, there can be attached to the bottom of the barrel a metal plate, P P, which is also graduated into inches and subdivisions, the numbering being downward. This is called the vernier.

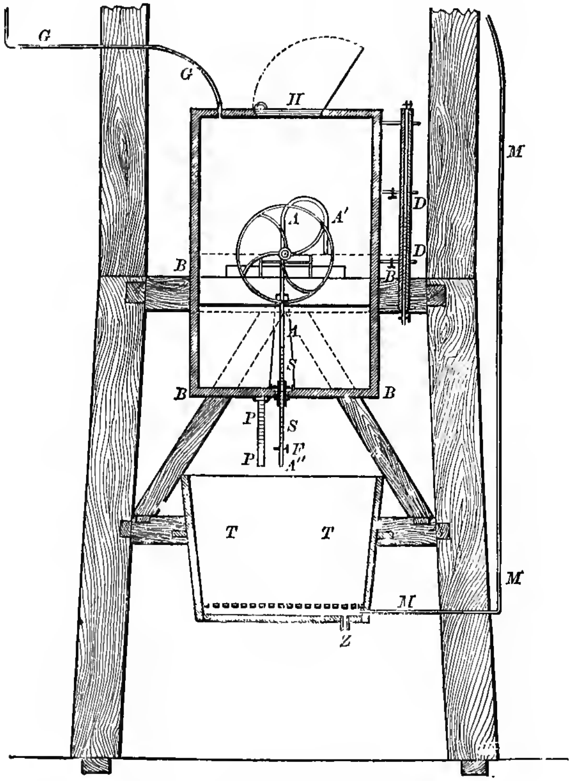


FIG. 143.—APPARATUS FOR CONTINUOUS LIXIVIATION.

Along the barrel is a glass gauge fitted in a graduated metal tube and sockets D D, and the o of the gauge corresponds with the mouth of the short leg of the syphon A$^{1}$, when the o of the screw corresponds with the o of the vernier.

The proper working of the gauge and of the siphon-tap are of the greatest importance in this apparatus, as it is well known that when roasted ores are brought into contact with fluids they swell more or less, and the question of how much they swell is determined by this apparatus, which permits of a perfect decantation of the fluid from the solid.

The gauge should be attached in such a manner that its lower level should reach a few inches below the ore level in the barrel. After the barrel has been stopped and is at rest for 20 or 30 minutes, a perfect separation of the ore and liquid will take place, as roasted ores take a peculiar sandy consistency which allows their quick settling, and the gauge will indicate the line of separation of ore and liquid, and also show any turbidity in the liquid which will indicate that the solution ought to remain at rest a little longer before being decanted. When this is to be done, I take the gauge readings, which, if it shows that the ore and liquid separation line is 8 inches below O or the mouth of the syphon tap A′, to decant the liquid it is necessary to screw the siphon-tap down 8 inches, on a level with number 4 on the vernier, and this will bring the mouth of the siphon on the ore and water separation line—in practice it will be well to keep it a little above, so as not to draw out any ore. If the ore stands say 6 inches above the O gauge line, the screw is turned up 6 inches.

The educt of the siphon-tap A″ is now connected with an india-rubber hose, and the same leads into the next revolving barrel to which the chloride solution is to be transposed. When this is done, compressed air from a small reservoir or pump is turned on through G, the pressure need not exceed 15 lbs. per square inch, and on opening the faucet, F, the liquid is pressed out through the syphon-tap, A′, and forced into the next revolving barrels, where the same operation is repeated.

The ore which remains in the tank still holds gold solution, and to recover this water is added to the tank, and the same discharged into the filter T T and the solution flows through Z into the precipitating tanks. To accelerate the filtering, the air is exhausted by means of a pump from below the filter bottom by means of the pipe, M.

**The Julian Process.**—Amongst the most recent applications of electricity to the amalgamation of the residues from the chlorination process is the system devised by Mr. Julian, of Johannesburg, South Africa. If he had limited his patent specification to the electrical portion alone, he might have the right to claim some originality for his electrolytic cell, but he dips also into the barrel chlorination, according to well-known methods, and not only intends to chloridize his gold so as to bring it into solution, but intends to amalgamate the dissolved gold chloride by adding mercury to the chlorinating barrel, which, certainly, is also a somewhat original claim.

The ore is agitated in a vat or barrel with a small quantity of chlorine, bromine, or iodine, or mixtures thereof, or chemicals which produce chlorine, bromine or iodine, and with some water. The contents of the vats are then subjected to a pressure of air or steam which may be anything above the atmospheric pressure, and the vats put into motion for a short period of time which may vary from a few minutes to several hours according to the nature of the ore.

Mercury or sodium amalgam is next added to the ore and solution either in the same or other vats, and pressure of air or steam again applied to the contents, and the contents agitated as before.

By these means (according to Mr. Julian), the gold which came into solution, in the first instance, is precipitated and amalgamated with the mercury. The vats containing the ore-amalgam are next emptied into a receiver, and the ore-amalgam is washed down an inclined table of amalgamated copper by jets of water, which carry the ore, &c., to the foot of the table, and thence to the electrolytic cells. Part of the amalgam remains behind on the copper plate and the remainder in the electrolytic cells. The gold or silver is obtained from the amalgam in the usual way.

If thought advisable, before adding the mercury, the gold in solution may be leached out, and precipitated by any known means, and the gold or silver remaining in the ore is extracted as above described.

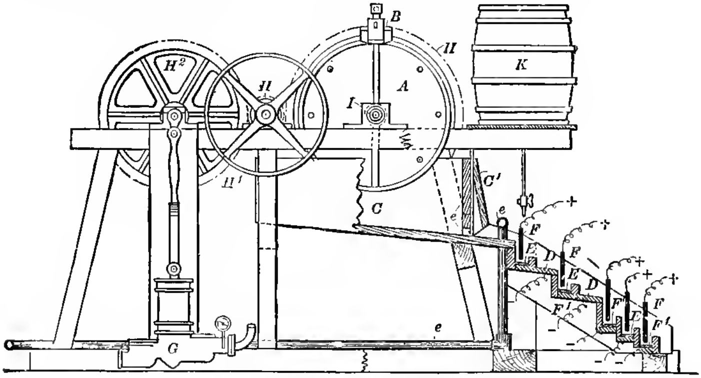


FIG. 144.—JULIAN PROCESS PLANT. Side elevation.

The electrolytic cells have mercury at the bottom of each, connected with the negative pole of an electric generator, and have anodes suspended so as not to touch the mercury. The mercury acts as the cathode, and a constant flow of electricity is kept up between the anode and cathode, and at the same time the ore, &c., passes through the cells.

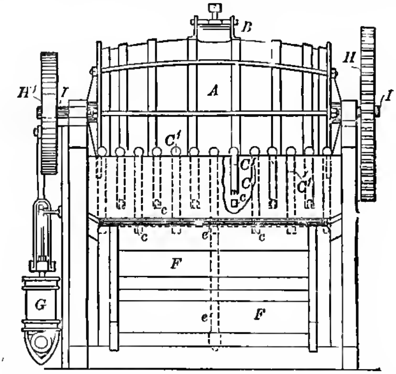


FIG. 145.—JULIAN PROCESS PLANT. Front elevation.

The electrolytic cells are to decompose and amalgamate any gold or silver compounds, and to retain floured mercury and floured amalgam and metallic gold which might otherwise escape with the tailings. When the compounds of gold or silver pass into the electrolytic cells, the compounds are decomposed and the gold or silver goes to the mercury cathode, as in electrolysis, when they readily amalgamate.

Fig. 144 is a side elevation, partly in section, of a plant adapted to effect the purposes of the Julian process. Fig. 145 is a front elevation of the same, and Fig. 146 the plan. The vat is so constructed with an apparatus to agitate its contents, and made of such materials as are not acted upon by the chemicals, employed within it. The rotating drum is shown at A. It is mounted upon a horizontal axis I, and is driven from the belt pulley, H, through the medium of gearing, H H.

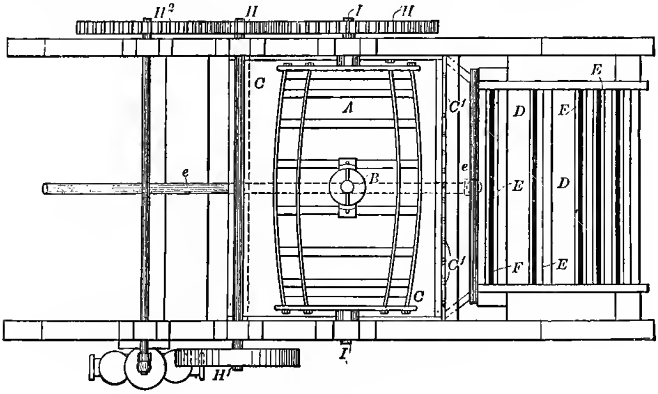


FIG. 146.—JULIAN PROCESS PLANT. Plan.

With the ore in the vat there is added a small quantity of chlorine, bromine, or iodine, or of a mixture of two or all of these substances, or of chemicals which produce chlorine, bromine or iodine. There is also added as much water as will reduce the mass after agitation, to the consistency of a semi-liquid mud.

The manhole of the vat is then closed by a tight cover such as B, and the contents subjected to the pressure of air, steam, or other fluid by means of any suitable device, such as an air compressor, G, which is driven by the gear $H^1$ H $H^2$ and has its outlet connected with the hollow shaft, I. The pressure may be almost anything above that of the atmosphere. About 60 to 80 lbs. to the square inch will suffice in most cases.

The vat is now rotated while under pressure, and its contents agitated for a period sufficient to cause them to assume the consistency of a uniform semi-liquid mud. This period may vary from a few minutes to several hours, according to the nature of the ore under treatment, and the pressure applied to the contents in the vat, one or two hours being a usual time.

In the above operation the gold is wholly or partly dissolved by the action of the chlorine, bromine, or iodine aided by the pressure within the vat, while the silver is converted into an insoluble compound, which may be, for example, a chloride, bromide, or iodide, according to which chemical or chemicals have been employed. The gold which is not dissolved is, however, so prepared, that it will readily amalgamate with mercury, in the following operation.

The manhole of the vat is now opened, and a quantity of mercury or sodium amalgam is added to the semi-liquid mass, either in the same vat or in another one. The manhole is then closed, the pressure of the air, steam, or other fluid again applied to the contents, and the latter are then preferably agitated as before, but only for a few minutes as a general rule.

In this operation the gold is readily precipitated by the mercury and formed into an amalgam, and a portion of the mercury replaces the gold so precipitated. The pressure within the vat also accelerates the operation of precipitation and amalgamation.

If thought advisable, the portion of the gold contained in solution may, before the mercury is added, be leached out and precipitated by any known means, and the gold and silver remaining in the ore may then be extracted therefrom, as above described.

After the second operation the cover, B, is removed and the contents of the vat emptied into a receiver, C, where they are allowed to settle for a short time. The receiver has preferably a sloping bottom, and is provided with outlet openings, *c*, which are controlled by slides, $C^1$.

After settling, the slides, $C^1$, are drawn up, and the ore, amalgam, etc., on issuing from the openings, *c*, are, by means of jets of water, washed down and distributed over one or more inclined plates, D, of amalgamated copper. The water jets may be supplied by a perforated pipe, *l*, placed in front of the openings, *c*. The plates, D, are so inclined that the ore, &c., may be washed off them readily, the inclination being about one inch to the foot. Each plate is connected, preferably, at its foot (as shown), to the negative pole of an electric generator, for a purpose hereinafter set forth.

After the passage of the material over the plates, D, and also, if desired, before its passage over the same, it is caused to pass through a series of electrolytic cells, E, arranged successively one below the other so that a liquid may flow through one and fall into the next. The cells consist of a number of narrow troughs or boxes, preferably made of some non-conductor of electricity, and each provided with a cathode, F′, of mercury (or sodium or potassium amalgam), connected with the negative pole of an electric generator, and with an anode, F, of carbon, platinum, lead peroxide, or other suitable material which is connected with the positive pole of the said generator. The mercury cathode, F′, is placed at the bottom of the cell, and the anode, F, is supported above, and at a convenient distance from it. The anodes, F, thus serve as vertical partitions for the cells, E, and the material washed down from the receiver, C, and plates, D, falling successively into the cells, is obliged to pass through each, between its anode and cathode. If desirable, the anode, F, may be simply attached to the lower edge of a suitable vertical partition of some other material. The number of such cells will vary according to circumstances, but usually four to nine are found to be ample. The cells at the heads of the copper plates D are similar to those below them, but the mercury cathodes F′

are connected to the negative pole of the generator by means of the plates which are extended backwards as shown so as to form the bottoms of the cells in direct contact with the mercury cathodes.

The action of the electrolytic cells and copper plates is as follows:—The ore on leaving the vat may contain some gold or silver or mercury compounds, and also some floured mercury or floured amalgam and metallic gold. To prevent any of these being lost in the tailings, the whole of the ore and solution is passed through the electrolytic cells and over the copper plates, while a strong current of electricity is kept constantly flowing from the anodes to the cathodes. During the passage of the ore and solution through the cells, any gold and silver compounds that may be present are decomposed, and the gold and silver go to the mercury cathode, where they readily amalgamate and are retained. Any floured mercury or floured amalgam and metallic gold that may be present are also retained in a somewhat similar manner.

Part of the amalgam in the solution is retained by the amalgamated copper plates, D. These are used as a guide to the operator in order that he may see what is going on. They should be kept covered with sodium or potassium amalgam. This may be done by allowing a solution of potassium or sodium compounds, such as salt, soda, or potash, to pass through the cells and over the plates while the current of electricity is passing. This solution may be conveniently kept in a vessel, K, placed above the upper cell and provided with a suitably controlled outlet pipe as shown. The sodium or potassium of the solution is deposited and amalgamated at the mercury cathodes, while the element with which it was combined, is liberated at the anodes. The gold or silver may be obtained from the amalgam in the usual way. It is very doubtful that any of the above reactions will take place in practical working, except the amalgamation of the bright gold on the outside copper plates, and perhaps a decomposition of the gold chloride while passing through the cells.

**Dr. Kiliani on the Electrolytic Refining of Copper.**—Dr. Kiliani published, in *Berg Hüttenmännische Zeitung*, a paper which was reproduced in a condensed form in *Engineering* on July 3, 1885. The information on the subject is so scant that some extracts, especially those referring to the separation of gold and silver by electric action from other metals, may be usefully given here.

As to the nature of the process itself, Dr. Kiliani remarks that the basis of the whole matter consists in the simple fact that when an alloy of several metals forms the anode in the bath the electric current does not cause the solution of all the component metals at the same time, but that it makes a selection, and takes one metal after the other in a certain order; and similarly, when several metals are in solution in a bath, the current selects them in a certain order for deposition on the cathode. A fully satisfactory scientific explanation of these facts cannot be attempted, because the whole matter is even yet too little studied, and the materials for a full explanation have not yet been collected.

With regard to the selection of the different metals by the current, Dr. Kiliani says that this "takes place, in general, on the principle that as much energy as possible is created and as little energy as possible is consumed, that is to say, under conditions that metal will be first dissolved from the anode, the solution of which causes the development of the greatest amount of energy (electro-motive force), and that the metal will be first deposited from the solution on to the cathode, the separation of which requires the least consumption of this same energy.

"A comparative measure of the energy required in these cases is obtained by taking the heat of combination of the metals with oxygen to form oxides or salts. The combination heat of the metals with oxygen to form oxides will give the order in which the metals are dissolved:—

"Manganese, zinc, iron, tin, cadmium, cobalt, nickel, lead, arsenic, bismuth, antimony, copper, silver, gold. If gold and silver are present in small quantities in the anode, they will be found in the mud of the bath. Lead in a sulphate of copper electrolyte will form an insoluble sulphate of lead, and also be in the mud at bottom. When the metals are once in solution, their deposition on the cathode takes place in the reverse order, beginning with gold and ending with manganese. If the current exceeds a certain strength all the metals may be dissolved and deposited together. The more neutral the electrolyte is, the more easily will the electro-negative metals be dissolved, and the more easily will the electro-positive metals be deposited.

"If oxides or sulphides are present, they are not acted upon by the current, they simply go into the insoluble mud, or are dissolved by the purely chemical action of the electrolyte.

"The sulphides are mostly good conductors, but not nearly so good as metallic copper. If, therefore, but a small amount of sulphides is contained in the copper anode the current will act only on the copper, and the sulphides will be found in the mud unacted upon, unless by the acid of the bath. If much sulphide is contained in the copper, the current will be more or less divided between the copper and sulphide, and a portion of the latter will be decomposed, with separation of sulphur. In addition to the above secondary reactions of the bath, there are others, some of which are good and some bad, for conducting the process. The current is always striving to decompose the electrolyte into metal (or oxide) and acid, whilst the liberated acid is striving to redissolve the deposited metal or oxide. These two forces are always opposed to one another, and under varying conditions either may gain the upper hand. The resolvent action of the acid, in cases where the components of the electrolyte have a strong chemical affinity, may overpower the action of a weak current.

"Gold, silver, and platinum remain undissolved in the mud when they are not present in considerable quantity, and so long as the electrolyte retains its normal composition as to free acid and dissolved copper. If the liquor becomes neutral the silver dissolves and becomes deposited on the cathode. Bismuth and its oxide go partly to the mud, as insoluble basic salt, and partly into solution, eventually precipitating as basic salt. The presence of metallic bismuth in the anode causes the liquor to become poorer in copper, while the presence of its oxide causes a reduction in the amount of free acid.

"Bismuth does not become deposited upon the cathode, even when large quantities of the basic salt accumulate in the mud, provided the bath be kept in its normal condition as to copper and acid.

"*Tin* dissolves in the bath, and after a while is partly deposited again as basic salt. If the anode contains very much tin, the greater portion remains as basic sulphate, adhering to the anode itself in the form of a deposit of a dirty grey colour while moist, but becoming white when air-dried, increasing rapidly in weight even after long drying at 212° F.; it contains sulphuric acid, and the tin oxide in it is mostly of the variety soluble in hydrochloric acid. The presence of tin, therefore, reduces the amount of copper in the bath without replacing it by any appreciable amount of tin in solution. The tin in solution exercises a surprisingly favourable influence on the copper deposit on the cathode. Copper deposited from a neutral solution of pure copper is rough, irregular, and brittle; but if tin be present the deposits are excellent and tough, even though the deposit give no trace of tin. The resistance of the bath is also much reduced by the presence of tin in the anodes.

"If *Arsenic* be present in the metallic state, it enters the solution as arsenious acid, and only appears in the mud when the solution is saturated with it. Arsenic in the form of arsenic acid combined with oxide of copper, or other oxides, is at once deposited as mud in neutral solutions, since these oxide combinations are non-conductors. Metallic arsenic thus reduces the amount of copper, and increases that of the free acid in the bath, because it goes into the solution without combining with an equivalent of acid, while at the same time a proportionate amount of copper is deposited with liberation of acid. Arsenic does not enter into the copper deposit in the cathode while the bath remains normal as to copper and free acid in a neutral state, or one in which the copper is insufficient, arsenic is deposited with the copper."

# CHAPTER XI.

## *THE CYANIDE PROCESS FOR THE EXTRACTION OF GOLD.*

SYNOPSIS OF THE PROCESS—Conditions which influence Precipitation—Smelting of the Slimes—Treatment of Acid Ores or Tailings—Concentrates—Adverse Conditions which affect the Cyanide Treatment—Practical Results.

**Synopsis of the Process.**—A large amount of public attention has recently been directed to the successful development of the cyanide process in the Witwatersrand gold fields, where it has been mainly adapted to the re-treatment of the tailings.

Before entering upon an account of the process, a few introductory remarks will be advisable.

There are very few chemical means by which gold can be won from its ores, gold being a non-oxidizable metal, which maintains its purity, and is therefore mostly found in its native or metallic state. Its solvents are aqua regia, chlorine, and potassium cyanide. The application of such a corrosive agent as aqua regia for the treatment of ores on a large scale is out of the question, and of the chemical means at our command there remain only the last two. Chlorination processes have been dealt with in previous chapters; and it now remains to describe the methods by which the gold is dissolved in solutions of cyanide of potassium.

That gold when in a fine state of division was soluble in cyanide of potassium was already known in the middle ages, when the gilding of metals was carried out by jewellers and alchemists by the use of gold in cyanide solutions. Coming to later times, several scientific books, dating back to the beginning of this century, mention the solubility of gold in potassium cyanide solutions.

The application of this solvent for the treatment of auriferous ores was first patented in the United States in 1867; but although the process was tried and experimented with by some most eminent metallurgists, no practical or commercial results were obtained. Satisfactory results, however, have been secured in the Witwatersrand gold fields, where the ores carry the gold in a pure and metallic state, in an extremely fine state of division, so that all the conditions exist to make the application of the cyanide process a perfect success.

Its introduction on those fields is due to Messrs. MacArthur and Forrest. For the first patent obtained by them their invention was described as subjecting the auriferous or argentiferous ores to the action of a solution containing a small quantity of cyanide, without any other chemically active agent, such quantity of cyanide being reckoned according to its cyanogen, and the cyanogen being proportioned to the quantity of gold or silver. Subsequently, further patents were granted covering the use of zinc, preferably threadlike, for a precipitating agent, and the use of caustic alkalies for neutralising ores containing acids.

To these methods has now been added the patented process of Messrs. Siemens and Halske, which consists in precipitating the gold by electricity on sheets of lead, an account of which will be found in the next chapter.

Turning to the treatment of the tailings in the Witwatersrand, as the largest proportion—and amongst it the coarser particles of gold—have been extracted by previous plate amalgamation, the precious metal is here found in a very fine state of division, and therefore amenable to cyanide treatment. It must be here remarked that the coarser the gold, the longer it takes to dissolve it; and it is recommendable, therefore, that all ores should, in the first instance, be submitted to plate amalgamation.

When the first cyanide works were erected on these fields, the old accumulated stocks of tailings had to be dealt with. Owing to their long exposure to the atmosphere, changes had taken place in their chemical composition, which caused at the onset some difficulties, but these were soon overcome on the application of the proper remedies. The ores which came from the upper levels, or the oxidized zone, always carry a small proportion of iron pyrite, which, on exposure, becomes oxidized. It is only when free milling ore tailings are taken directly from the battery to the cyanide works that they do not contain any decomposition products and are in a proper condition for cyanide treatment.

The course of the treatment in its successive stages may be thus summarised:—

*First Stage.—Passing an Alkaline Solution* or caustic wash through the ore to the point of saturation, as the tailings always contain a certain amount of organic matter, acid salts, &c. By running on this alkaline solution, a considerable saving is effected in the consumption of cyanide of potassium, and the tailings get the advantage afterwards of the full strength of the strong solution. Strength of this first alkaline solution may be 0·15 per cent. KCy., and may contain 4 ozs. of caustic soda per ton of solution. Caustic soda dissolves out organic matter. Excess of lime destroys cyanide.

*Second Stage.—The Strong Cyanide Solution.*—This solution varies in strength from 0·3 to 0·5 per cent. in KCy. to suit the richness and nature of the tailings under treatment, and the proportion of solution to be run on should not be less than one-third the weight of tailings in the vats. When this solution is run on, though the first solution has drained out, the tailings still contain a considerable quantity of the former solution. This should be displaced by allowing the second solution, or strong solution, to drain down immediately the vats are filled with the strong solution, for say about two hours (according to the capacity of vats and nature of tailings), and when this has been displaced, sufficient strong solution should be run on to make up the required proportional amount.

The vat is now full of strong solution, which, in some cases, may be leached out immediately, though generally it is advisable to leave it in contact with the mass for a short period—say, three hours—to give the solution time to penetrate any lumps.

When this solution is leached out, or only a very small stream is coming away, it should be allowed to continue draining about four hours longer. During these last four hours air is taking the place of the solution, and the gold is in contact with a strong solution of KCy., in presence of oxygen, which produces a more rapid and effectual dissolving of the gold. For proof, take two watch glasses, fill both with cyanide solution of the same strength. In one, place a piece of gold leaf on the surface of the solution, while in the other immerse the gold under the solution. The gold leaf on the surface in presence with air disappears rapidly and is completely destroyed, whilst the other dissolves very slowly.

*Third Stage.—The Weak Solution.*—After the strong solution has been run on, we may safely assume that most of the gold is now in solution, and the object of the third operation is to wash out the dissolved gold. Therefore, after the strong solution has been drained out, sufficient weak solution containing 0·15 per cent. KCy. is run on till the total quantity of solution and washes represent 75 to 80 per cent. of the weight of the ore.

*Fourth Stage.—The Water Wash.*—After the weak solution, water wash is applied, and the quantity so applied is not to be less than 7 per cent. of the weight of the ore, and, indeed, more is necessary.

**The Precipitation.**—The cyanide solutions containing gold, as they flow from the leaching vats, are passed through one or more precipitation boxes. On the zinc shavings the gold is precipitated, and the quantity of solution flowing through each box must be properly regulated. The only satisfactory method of knowing if proper precipitation is taking place is by having the solution regularly assayed. This is an important point, as good results cannot be obtained if the solutions leaving the precipitation boxes are rich in gold.

It is essential for the man on the shift to know the strength of the solution at the different stages of leaching: first, that he may know into which one of the precipitation boxes the solution is to be passed; and then he must ascertain the strength of the solution leaving the precipitation-boxes, in order to let them flow into their respective storage tanks.

The gold in solution increases and decreases in quantity much in the same way and same time as the cyanide in solution increases or decreases. In the precipitation of the gold in the boxes, it is important to keep a sufficient stock of zinc shavings in each compartment, 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 in. in diameter, a hole being punched in the centre for the mandril. It is usual to put bundles of twenty 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 carpenters' 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; or 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 some Lydenburg cupriferous ores, it was found that, by making the solutions up to working strength before passing them through the zinc boxes, the result was that solutions which had before assayed several ounces were reduced to a few dwts. per ton.*

The zinc shavings in the boxes may require replenishing

---

* According to Mr. Feldtmann.

every day to replace the amount consumed, or they may run a week at a time without requiring replenishing.

Having, by passage of the solutions through the zinc shavings, reconverted the gold into the metallic form, the process of collecting the metal and putting it into ingots will be another stage of operations, and this will be described presently under the clean-up.

The exposure of the solution to the zinc after the gold is precipitated can only result in a loss of zinc, and—what is of much greater importance—unnecessary decomposition and loss of cyanide.

The precipitation boxes are set at a slight grade, as shown in the plan. Mr. Philip Argall, M.R.I.A., says that no iron or metal, other than zinc, should be exposed to the solutions in boxes; even the iron wire screens, used to support the filiform zinc, have been shown to cause an unnecessary consumption of zinc and cyanide, through the electro-chemical action induced between the metals.

**Conditions which Influence Precipitation.**—Mr. W. Bettel states that in the precipitation of various metals from cyanide solutions by zinc, the precipitation is influenced by the following conditions:—

(*a*) Different strengths of solvents.

(*b*) Impurities in solutions (which may be derived from ores) affecting precipitation.

(*c*) Influence of rate of flow of solution through extractors,—past a given area of zinc—in gallons per minute.

(*d*) Amount of gold in solution to be precipitated, and maximum precipitation from solutions of various richness.

(*e*) Effect of alkalinity, acidity, or neutrality of liquors on rate and percentage of precipitation of metals from cyanide solutions.

(*f*) Influence on the rate of precipitation of gold of such inert substances as carbonate of lime, clay, oxide of zinc, and coating of zinc, &c.

(*g*) Other physical conditions of zinc, affecting precipitation of metals from cyanide solution.

**Time taken by Treatment on Clean Tailings.**—The method of treatment described below has been adopted at the Rand Central Ore Reduction Company's works, the figures having been kindly supplied to me by the working chemist, Mr. Blomfield :—

| | |
|---|---|
| 1st. *Alkaline Wash.* | |
| Filling with 0·16 per cent. KCy. + 4 oz. Caustic soda per ton solution . . . | time 2 hours. |
| Leaching . . . . . . . . . | ,, 3 ,, |
| 2nd. *Strong Solution.* | |
| Filling with 0·35 KCy. solution and contact . | ,, 5 ,, |
| Leaching . . . . . . . . . | ,, 8 ,, |
| Leaching dry . . . . . . . . | ,, 4 ,, |
| 3rd. *Weak Solution.* | |
| 4 washes at 1 hour each . . . . . | ,, 4 ,, |
| 4 leachings at 4 hours each . . . . | ,, 16 ,, |
| 4th. *Water Wash.* | |
| Filling . . . . . . . . . . | ,, 1 ,, |
| Leaching . . . . . . . | ,, 7 ,, |
| | Total time 50 hours. |

This treatment was applied to 7½ dwts. tailings, residues being from 15 grs. to 1 dwt. per ton; ¾ lb. of cyanide being the consumption per ton of ore.

On these fields it has been found that 12 to 24 hours' contact with the strong solution is sufficient to effect a solution of the gold in the tailings. The subsequent treatment with weak solutions has the object to displace the gold solution and to wash out the already dissolved gold. The displacing of the weak solution by a final water wash tends to reduce the loss of dissolved gold to a minimum, as it displaces the weak gold solution, and the gold which is left in the tailings is generally that still retained in the coarser quartz particles, from which it has not been liberated by stamping, and whatever gold has remained undissolved in the iron pyrites.

The time of treatment of each vat varies, and takes from 50 to 140 hours, according to circumstances, and the size of the tanks employed.

**The Clean-up.**—In making the clean-up, the tray holding the zinc shavings is lifted out from the last compartment, and pulsated up and down in the solution, so that the fine particles of slime and zinc fall through the sieve and settle in the bottom of the box. Before starting this operation, it is advisable to run into the box a sufficient amount of clean water to remove the cyanide solution, as the latter is injurious to the workmen. The Kaffirs, however, put their arms and hands into it without being affected. The zinc shavings are taken out of the tray, which is placed on a rack above the box, so as to allow solution to drain back into the box. They are also rubbed in the water, to remove, as much as possible, all gold adhering to them.

The tray is turned over and brushed down so as to remove any gold adhering to it. The zinc shavings get very hot on account of oxidation, and it will be noticed that steam arises from them. They should be exposed, therefore, as little as possible to the air. The solution of water in the zinc boxes is pumped into settling tanks, where it is allowed to stand for two weeks, so as to give the extremely fine particles of gold which are held in suspension time to settle. In pumping out the precipitation boxes, great care should be taken not to disturb the gold zinc slimes in the bottom. With this aim, the water is pumped out to within 2 inches of the slimes, and the india-rubber suction hose moved into the second compartment, and so on.

The slimes are then pushed back with a scoop to one corner, and the supernated liquor allowed to stand for a while and pumped away again into the settling tanks. The slimes are now scooped into enamelled iron buckets and discharged on to a fine sieve—say 900 mesh—and washed and rubbed into the gold clean-up tank.

After the water is settled in the clean-up tank it is syphoned or pumped off, and the precipitate (called gold slimes) is drawn off through the plug holes on to a calico or linen filter, or into a filter press.

The zinc shavings, some of which will be found to have quite a brown coating of gold, which cannot be removed even by rubbing, are again returned to the precipitation-boxes, and fresh zinc put on top. The gold which sticks to the zinc will be recovered in the next clean-up. After the gold slimes are sufficiently dry to be handled with a scoop, they are dried on an iron plate or on iron pots, and are then ready for roasting and smelting.

The object of the roasting is to oxidize the greater portion of the zinc which, in the form of small chips and shavings, has fallen through the zinc box trays, so as to cause it to combine in the subsequent smelting with the fluxes and leave the bullion fairly fine. Oxidation by the aid of 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. Mr. Feldtmann suggests that 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. Where 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.

Mr. Butters, at the Rand Central works, has a muffle-roasting furnace in which to dry and roast the slimes. The bottom of the furnace consists of a cast-iron pan, and the wet slimes are charged on to this pan, and, when dry, a damper is closed, which turns the flame through an opening in the fire-bridge, under the iron pan, and the slimes are carefully stirred to avoid dusting, and as much of the zinc as possible is driven off during the roasting.

The clean-up is generally made once a month.

**Smelting of the Slimes.**—The dried precipitate is now ready for the smelting process, and graphite crucibles are employed for the operation. The fluxes commonly used are bi-carbonate of soda, borax, and sand.

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 Precipitate. | 2. Very zincy Precipitate. | 3. Very sandy * Precipitate. |
|---|---|---|---|
| Precipitate | 30 lbs. | 30 lbs. | 30 lbs. |
| Bi-carbonate soda | 15 ,, | 15 ,, | 20 ,, |
| Borax | 8 ,, | 12 ,, | 10 ,, |
| Sand | 5 ,, | 5 ,, | — |
| Fluor spar | — | — | 2 ,, |

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 slags, which contain a large percentage of silicate of zinc and soda, corrode the pots, and during smelting there is a heavy evolution of fumes of zinc

---

* According to Mr. Feldtmann's formula.

oxide, causing, most likely, losses of gold. At the Robinson Mine condensing flues connect the furnaces with the chimney.

The several pieces of bullion thus obtained at one clean-up, are, consequently, remelted with borax and run together into one ingot. This remelting 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.

The slags from this operation are difficult to re-smelt, which may be owing to the presence of carbon contained in the zinc. At the Rand Central ore reduction works, Mr. Butters is erecting a small matting furnace, where he intends to smelt the slags with copper ore concentrates and collect the gold in a matte.

The fineness of the bullion resulting from the cyanide process ranges from 600 to 800.

Besides gold, silver, and zinc, there is some lead in the ingots, as the zinc employed contains a certain per cent. of this metal, and also carbon, which also is found in the ingots.

The zinc supplied by the Vieille Montagne Company is the best and purest for the purposes of the process.

**The Treatment of Acid Ores or Tailings.**—Under acid tailings are understood those tailings which contain the decomposition products of the iron pyrites. These products consist chiefly of free sulphuric acid and 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. The reactions which take place, when these salts are in the tailings, will be found explained in a subsequent chapter (Chap. VI.) on the chemistry of the process.

The ores on these fields at a comparatively shallow depth become very pyritic, but outside of iron pyrites the main reef series carries no other sulphur combinations, or, at least, in such small proportions, as to have no practical importance.

The treatment of acid tailings on these fields offers more difficulty than in the case of the "free ore tailings." The oxidation products of the pyrites have to be neutralized by the addition of alkalies or alkaline earths—either caustic soda or lime—with or without a preliminary water washing, to remove such soluble salts or "cyanicides" as may be present.

Caustic lime, in a powdered form, is generally added now on these fields. With very acid tailings—namely, pyritic tailings—which have been exposed for some time to the oxidizing influence of the atmosphere, as much as 2½ lbs. per ton is added. With fresh tailings ½ lb. of lime per ton of tailings is sufficient. In some works the requisite quantity of lime is added to each car load of tailings as the same is hauled up to the leaching tanks. In other works 6 or 10 tons of tailings are dumped into the leaching tank, and the same levelled off and the lime sprinkled over it. The practice of putting all the lime on top of the tailings after the vat is filled, cannot be recommended: the lime forms a pasty mass, and the alkalinity does not penetrate through the total height of the ore, so that in deep tanks it will be found that the top layer, say for one-third of the height of the ore, will be neutralized, whereas two-thirds nearer the bottom will remain acid. The length of treatment varies.

**Concentration and Treatment of Concentrates.**—The question whether it pays to concentrate the tailings before sending them to the cyanide works has not yet been definitely settled on the Rand gold fields. Considering that during the short period of treatment which the tailings undergo, the gold cannot all be extracted from the pyritic particles, it seems rational that a separation of the pyritic matter should be effected, and the concentrates treated separately.

The only mines where concentration previous to cyaniding has been carried out for any length of time are the Crown Reef and the Langlaagte Estate and Langlaagte Block B Mines, where the concentrates are treated separately with cyanide. At these mines I am informed that it pays to follow this method.

Mr. Williams, of the Crown Reef, collects his concentrates by a crude system of classification, which costs him 10d. per ton of concentrates, whereas at the Langlaagte Estate Frue vanners are employed. The manager there informs me that the cost of cyaniding the concentrates amounts to only 17s. per ton.

I believe that when the question has been thoroughly studied it will be found that a system of proper concentration before cyaniding will pay. Up to the present there is not a single mine on the Rand equipped with a proper concentrating plant, as such ores as exist there have to be classified before an attempt can be made to concentrate them. At the Langlaagte Estate, where 22,000 tons of ore are crushed monthly, they get on an average 350 tons of concentrates, or about 1½ per cent.; this quantity could be easily doubled by the application of a proper plant. Most mills have Frue vanners, but this excellent machine cannot do the work alone on these ores.

High extraction by cyanide can be obtained from concentrates, only the process does not seem to act alike everywhere on the concentrates, and possibly the physical nature of the pyrites is not alike in every mine. In some cases it may become necessary to grind the pyrites, so as to liberate the gold, and make it more amenable to cyanide. It is stated that gold contained in pyrites is extracted more easily when the concentrates are mixed with sands. Clean concentrates, therefore, are more difficult to treat, as owing to their high specific gravity they "pack" and resist percolation.

At the Crown Reef mine they collect by means of 3 spitzlutten, about 600 tons of concentrates a month, out of 17,000 tons of ore. A 1½-in. pipe leads the heavy sands and pyrites to settling tanks 30 ft. in diameter and 6 ft. high, which assay 23 dwts. per ton. The main object of this rough concentration is to eliminate the coarse sands so as to submit them to a prolonged treatment with cyanide. The cost of this system of concentration is only £25 per month—namely, the pumping of the water which is needed for the hydraulic classifier—as they are worked by an ascending stream of water. The concentrates, or rather the classified material, by being collected in wooden vats, is constantly under water and does not get oxidized. The material collected is not very coarse, as in the battery they use 900-mesh screens.

From the storage tanks the material is taken to the leaching tanks, and submitted to the action of the solutions for 16 to 18 days. The strength of solutions employed is 0·25, 0·1, and 0·05 per cent. respectively; about 1 lb. of lime is added to each ton of concentrates. Strong solution is run on for 4 days or more, till its strength remains constant.

The residues from these concentrates assay 1½ dwts., giving an extraction of 94 per cent. The consumption of cyanide with these concentrates is about 1½ lbs. per ton. The weaker solutions are constantly circulated through the tanks till a high point of extraction is reached. The Witwatersrand ores carry from 3 to 4 per cent. of pyrites; and I believe the time of treatment could be lessened by employing a somewhat stronger solution.

At the Robinson mine 3 per cent. of the total weight of the ore is caught on Frue vanners, as concentrates assaying 4 to 5 oz. Extensive experiments have been made with cyanide, but the results were not satisfactory, and consequently the concentrates are treated at this mine by chlorination, which costs £3 per ton.

The tailings at the Robinson are elevated by means of a bucket-wheel, and passed through intermediate tanks, fitted with Butters and Mein's distributor; there are six intermediate tanks, and 30 per cent. of the ore goes into the slime pits.

The tailings, which go to the cyanide works, assay 6½ dwts. The slimes assay 5½ dwts.; and Captain Mein tells me that the residues, after cyanide treatment, assay only 1 dwt. This would mean a higher extraction than on any mine on the Rand. He finds that there is a discrepancy of 1 to 2 per cent. between the actual gold produced and that determined by assays.

**Adverse Conditions which affect the Cyanide Treatment.**[*]—1. The action of impurities and base metals in ores prevents solution of the gold and effects the decomposition of cyanide. This will require a complete study by analysis of the component parts of the ore.

2.—It is necessary to examine the condition of gold in residual tailings or concentrates from the cyanide treatment.

(*a*) In this connection it has to be ascertained if a certain percentage of dissolved gold (the auro-potassic cyanide) has or has not been completely washed out.

(*b*) Also if a certain percentage of gold is still present in coarse particles capable of being amalgamated.

(*c*) Also the percentage of gold encased in quartz, or pyrites, to which the solution has not had access.

(*d*) Also the occurrence of gold in lumps of slime after treatment by cyanide; the gold supposed to be precipitated by the action of iron salts producing a ferrous ferricyanide from auro-potassic cyanide and free cyanide, thus explaining some low extractions from weathered ore and concentrates.

3.—The presence of copper combinations is very detrimental to the success of the MacArthur-Forrest process. The ores of the Black Reef series, near Johannesburg, carry a small quantity of copper pyrites, and consequently the extraction of gold by cyanide does not give as good results as on the Main Reef series.

In this connection it is important to mention here a point which the inventors of the cyanide process have strongly emphasised—namely, the selective action of weak solutions of cyanide for gold in preference to other metals. It is a strange fact that, while gold is one of the most difficult of metals to dissolve in acids, it is extremely susceptible to the weakest solution of cyanide, when copper and other metals might remain unaffected in it. The successful use of very weak solutions may have much wider and more important consequences.

---

[*] According to Mr. Bettel.

**The Practical Results.**—Before we begin the treatment of the tailings, we meet with an acknowledged loss of 25 to 30 per cent. of the gold which goes into the slimes. It is true this gold is not lost, for it is still there. It is still an asset which accumulates monthly by the thousands of ounces in the slime-pits; but in the present state of our science and knowledge it is unavailable, and, therefore, for present practical purposes, represents a loss which has to be reckoned with. It may be possible, and quite probable, that in other mining regions this loss may not occur, but I put some emphasis on the point, as investigators in other countries will do well to closely study the nature of the ore they are dealing with before erecting their plant, and thereby probably save unnecessary expense on that head, to say nothing of costly experiments.

To illustrate the question of extraction by an example, I will suppose that we are dealing with a Rand mine whose ores contain an average assay value of 16 dwts. per ton.

I am safe in saying that the average extraction in the battery by plate amalgamation ranges from 60 to 70 per cent., especially from the pyritic ores; and, assuming that the battery recovers 10 dwts. per ton, or about 63 per cent., this would leave 6 dwts. in the tailings. Assuming now that 1,000 tons tailings at 6 dwts. go to the cyanide works, containing 6,000 dwts., 30 per cent. of these, or 300 tons at 6 dwts., containing 1,800 dwts., go to the slime-pit, leaving 700 tons of tailings at 6 dwts., containing 4,200 dwts., which go to the leaching-vats. Of these 4,200 dwts., 70 per cent. is recovered by cyanide, equal to 2,940 dwts., so that the amount recovered from the original 1,000 tons of tailings represents 50 per cent. in round figures, or 3 dwts. out of the 6 dwts. Add to this the 10 dwts. won on the plates, and we have 13 dwts., or a total recovery of 82 per cent. of the gold contained in the ore, which, when the slime treatment shall have been solved will be raised to about 90 per cent., which percentage can be considered highly satisfactory.

Exact figures as to the actual extraction of the gold from the various establishments are not easily obtainable, but from the information I have gathered, I should consider that 85 per cent. would be the average.

**Cost of Treatment.**—This necessarily varies at every plant, and mainly depends (1) on the size of the plant; (2) on the facilities for handling the tailings in charging and discharging; while (3) the principal item is the consumption of cyanide.

The consumption of cyanide, I should judge, varies from ¾ to 1¼ lbs. per ton of tailings treated. The commercial article, as sold in Johannesburg, costs 2s. per lb.

The consumption of zinc is about 0·4 to ½ lb. per ton of ore, and costs 4½d. per lb. in Johannesburg. At the Kleinfontein works the consumption of cyanide is only ¼ lb. per ton. The consumption depends greatly on the washing, and the alkaline wash destroying organic matter. The cost, in large works treating above 10,000 tons and over, would be 4s. per ton; in works treating 5 to 7,000 tons, 5s. to 6s.; in works treating 3 to 4,000 tons, 6s. to 7s.; while in smaller establishments the cost may even be higher.

The cost of zinc precipitation can be estimated at from 1½d. to 3d. per ton.

The Robinson Cyanide works treated, in 1893, 55,200 tons of tailings; in which was contained fine gold, 20180·06 oz., and from which was extracted fine gold, 13872·66 oz.; extraction, 68·7 per cent.; bullion returned, 17921·20 oz.

The cost of cyaniding was as follows:—

| | Total. | | | Cost per ton. | | |
|---|---|---|---|---|---|---|
| | £ | s. | d. | £ | s. | d. |
| Wages (whites and natives, including food) | 3,406 | 11 | 8 | 0 | 1 | 2·81 |
| General stores, assay material, &c. . . . | 1,408 | 9 | 1 | 0 | 0 | 6·12 |
| Fuel . . . . . . . . . . . | 1,204 | 19 | 9 | 0 | 0 | 5·24 |
| Cyanide—64,411 lbs. equal to 1·16 lb. per ton | 5,563 | 0 | 2 | 0 | 2 | 0·19 |
| Zinc—12,521 lbs. equal to 0·23 lb. per ton . | 260 | 4 | 3 | 0 | 0 | 1·13 |
| Contractor (filling and discharging vats) . | 4,325 | 12 | 2 | 0 | 1 | 6·81 |
| Royalty . . . . . . . . . | 3,768 | 1 | 1 | 0 | 1 | 4·38 |
| Total . . . . | £19,936 | 18 | 2 | 0 | 7 | 2·68 |

# CHAPTER XII.

## *THE SIEMENS-HALSKE CYANIDE PROCESS.*

HOW THE PROCESS WAS DISCOVERED—Action of the Electric Current—Why Mercury cannot be used as a Cathode—Conditions which the Metal Cathode must fulfil—The Avode—Electric Current required for Precipitation—Advantages of Electrical Precipitation—Practical Working Results—Scheme of Working.

**The Electric Precipitation of Gold from Cyanide Solutions.***—This process has been in practical operation for several months at the Worcester mine in 1894, when I had the opportunity of observing and testing the operations. At the Meyer and Charlton mine, May Con, and at the Metropolitan, it was expected that the newer process would shortly supersede the zinc precipitation method.

Upon reference to the plans shown in Plates X. and XI., it will be noticed that the works are similar to those in use for the MacArthur-Forrest process, the only alteration being in the extractor-house.

**How the Process was Discovered.**—In 1887 Dr. Siemens, the eminent electrician of Berlin, found that the gold anodes, used in electro-plating at his works, lost weight when standing in the cyanide liquor without any electric current passing through the bath. This fact, of course, corroborated the statements made by chemical authorities that gold is soluble in cyanides, and induced him to try the use of cyanide solutions for the extraction of gold from ores.

He found that the zinc method, as introduced by Messrs.

---

* For much of the information contained in this chapter, I am indebted to a lecture by Mr. A. Von Gernet, delivered before the Chemical and Metallurgical Society of South Africa. Great credit is given to Mr. Von Gernet for the ability he has displayed in developing the process.

MacArthur and Forrest, only gave good results with strong solutions, while the electric precipitation was equally effective with either strong or weak solutions, and its efficiency was not lessened by the presence of caustic soda. He subsequently introduced the process in Europe, Asia, and America, and in 1894 erected a plant capable of dealing with 3,000 tons of tailings per month at the Worcester mine, near Johannesburg.

**The Action of the Electric Current on the Gold Solution.**—The electric current decomposes a solution of a metallic salt, the metal being deposited on the negative pole, while the metalloid is liberated at the positive pole of the electrolytic cell. In a fixed time, a given electric current will deposit a certain quantity of metal, which quantity varies for different metals in direct proportion to their electro-chemical equivalents. This law holds good only for solutions strong in metal, but with very dilute solutions, as in use in the cyanide process, the current does not find sufficient of the metallic compound present at the electrodes, and consequently decomposition of water also takes place. For this reason, to make the efficiency of the precipitation as great as possible, constant diffusion of the solution is requisite.

In order, therefore, to create an artificial diffusion, a mechanical movement of the solution is important, and the most economical and convenient way of effecting this is to allow a slow but steady flow through the precipitation-boxes. But it is still more important to give a very large surface to the electrodes. In fact, a better effect is obtained by doubling the number of plates than by increasing the current tenfold.

**Why Mercury cannot be used as a Cathode.**—Mercury cathodes are not practical, for the reason that such an enormous quantity of this costly metal would have to be employed that the recovery of the gold would become difficult.

To precipitate 100 tons of cyanide solution containing 5 dwts. of gold per ton of solution in 24 hours, about 24,000 square feet of mercury surface is required. If the bottoms of

the precipitating-boxes were covered with mercury, it would be necessary to have it at least a quarter of an inch deep, to make up for differences of level, and thus ensure that the whole surface should be covered. This requires over 200 cubic feet of mercury, weighing 80 tons. At the end of a month there would be 750 oz. of gold in this enormous quantity of mercury; and even straining it most carefully, the gold would be so finely diffused that I doubt whether much of the gold would remain in the straining filter; besides, the practicability of such an operation is questionable. I do not consider the initial outlay of the mercury, and the large loss sure to result when handling such large masses of this metal.

The vertical position in which metallic plates can be placed has the great advantage of keeping the surface of the cathodes clean, as any solid matter entering the boxes in suspension sinks to the bottom of the latter, no obstruction being offered to its downward course. Sheets of solid metal (as copper), coated with mercury, have also been tried, but have been unsuccessful because the mercury, owing to the action of the current, will penetrate the copper and form a dry amalgam which does not adhere to the plates.

**Conditions which the Metal Cathode must Fulfil.**—

1. The precipitated gold must adhere to it.
2. It must be capable of being rolled out into very thin sheets to avoid unnecessary expense.
3. It must be easy to recover the gold from it.
4. It must not be more electro-positive than the anode, in order to prevent return currents being generated when the depositing current is stopped.

The most suitable metal for the purpose is lead, rolled out in very thin sheets, and this is accordingly used in the Siemens-Halske process, meeting all the requirements of the case. The lead sheets are fastened in light wooden frames. There are three sheets of lead, 2 by 3 ft. in each frame, giving each frame a surface of 18 ft., and 87 frames, which are in each precipitating-box, will expose a surface of 1,566 square feet.

Each frame holding three lead sheets at 1 lb. weight each, makes 261 lbs. of lead in each box. See Plate XII.

**The Anode.**—Not less important is the question of anodes. By the action of the current, a metalloid is liberated at the positive electrode, and the latter, when a metal, begins to oxidize. Carbon could be used as an anode, but it will not withstand the action of the current, and soon crumbles into a fine powder, which decomposes cyanide. This finely-divided carbon is in suspension, and cannot be removed from the solution by filtration. Zinc used as an anode forms a white precipitate of ferro-cyanide of zinc by the reaction of zinc oxide upon ferro-cyanide, formed during the leaching. Similarly, iron anodes form Prussian blue by the reaction of oxide of iron and ferro-cyanide. In consequence of this reaction the amount of ferro-cyanide in the cyanide solution does not increase.

From the Prussian blue the cyanide can be recovered by dissolving it in caustic soda, then evaporating the solution, and finally smelting with potassium carbonate.

This last process has been carried out only on a small scale, about 50 lbs. at a time, but a nice clean cyanide of potassium is obtained. In the treatment of tailings this regeneration of cyanide is not of great importance; but with concentrates, which decompose the solution with formation of ferro-cyanide, it will effect a considerable economy.

**Electric Current required for Precipitation.**—In order to precipitate the gold from cyanide solutions only, a very weak current is required, that is to say, a density of about 0·06 ampère per square foot. With cathodes about 1½ in. apart, 7 volt is sufficient to produce this current strength.

The advantages gained by using such a weak current are:—

1. The gold is deposited hard on the plates.

2. The iron anodes are preserved for a long time, as their waste is in proportion to the current strength. In a plant

Missing Page

treating 3,000 tons per month, 1,080 lbs. of iron are destroyed in that period.

3. Little power is required. 746 Watts equal 1 horse-power. A 3,000-ton plant requires 2,400 Watts, equal, theoretically, to 3½ horse-power, and actually requiring about 5 indicated horse-power.

**Advantages of Electrical Precipitation.**—The most important feature of electrical precipitation is, that it operates on the solution quite independently of the amount of cyanide or caustic soda it contains. Precipitation by means of a chemical reaction is invariably more complete with a solution strong in cyanide than with a weak one, but with electricity is absolutely of no importance whatever. Therefore, in the treatment of tailings, very dilute solutions can be used, the only limit being a sufficient amount of cyanide to dissolve the gold satisfactorily. Moreover, however acid the solution may be when entering the boxes, the precipitation takes place equally as well, the same amount of gold being recovered as from a neutral or alkaline solution. There are none of the complications arising from the formation of lime and alumina and hydrate of iron, which, under similar circumstances, sometimes occasion so much trouble in the zinc process.

A solution, containing 0·03 per cent. of cyanide, will dissolve gold just as effectively as a solution containing 3 per cent., provided a longer time is allowed for treatment. In the first case, the decomposition of cyanide in the tailings is much less than in the second, and a corresponding economy is effected. The moisture in the original tailings being usually about the same as that contained in the residues, there is, as a rule, no chance to give a large water wash after the cyanide treatment is finished. The residues discharged contain 10 to 15 per cent. moisture, carrying about 1 to 0·05 per cent. cyanide solution when the zinc process is employed. This is equivalent to ¼ lb. of cyanide per ton of tailings. This last may be reduced to $\frac{1}{40}$ lb. by using as weak wash a solution containing only 0·01 per cent. of cyanide, which strength will be perfectly suitable for electrical precipitation, though difficult to deal with by the chemical method.

Practical experience has taught us that, if we treat an ore containing copper with a strong, say, 2 to 3 per cent. solution of cyanide, the potassic cyanide may be all decomposed, but if we apply a much weaker solution, say ½ per cent., 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 to be both effectual and economical.

The presence of copper in the ores also may affect, to some extent, the second operation in the process, *i.e.*, the precipitation of the gold.

Mr. Feldtmann mentions in this connection the result of some experiments made by him on some cupriferous ores at the Transvaal Company's plant near Lydenburg, 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. extraction on 18 dwts. ore—were obtained with ½ 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.

As I have already intimated, it is my belief that the application of the extremely weak solutions, such as are utilised in this process, will lead to important results in the treatment of the so-called rebellious ores. But further investigation of the subject is required.

The vats hold 135 tons, and it takes 5 hours to fill them. The tailings first get an alkaline wash of 10 tons, after which wash 70 tons of strong solution of 0·05 to 0·08 per cent. is pumped on; and afterward 20 tons of weak solution, of 0·01 per cent. strength, is pumped on. The total quantity of solution used is 100 tons, and it takes 5½ days to leach, filter, and discharge each vat.

**Practical Working Results.**—At the Worcester works, which are in charge of Mr. Oswell, there are in use 5 leaching-vats of 20 ft. diameter with 10 ft. staves, each holding 3,100 cubic ft., 135 tons. One tank is discharged and filled every day. The strong solution used contains from 0·05 to 0·08 per cent. cyanide, and the weak washes 0·01 per cent. The actual extraction of fine gold has averaged 70 per cent., while the consumption of cyanide has been ¼ lb. per ton of tailings treated.

The precipitation plant consists of four boxes, 20 × 8 ft. wide and 4 ft. deep. Copper wires are fixed along the top of the sides of the boxes, and convey the current from the dynamo to the electrodes. The anodes are iron plates, 7 ft. long, 3 ft. wide, ⅛ in. thick. They stand on wooden strips, placed on the bottom of the box, and are kept in vertical position by wooden strips fixed to its sides. In order to effect the circulation of the solution in passing through the box, some of the iron sheets rest right down on the bottom, while others are raised about 1 in. above the level of the solution, thus forming a series of compartments similar to those of a zinc precipitating-box, the difference being that the solution passes alternately up and down through successive compartments.

The iron sheets are covered with canvas to prevent short circuit. The lead sheets are stretched between two iron wires, fixed in a light wooden frame, which is then suspended between the iron plates. The boxes are kept locked, being opened once a month for the purpose of a "clean-up," which is carried out in the following manner :—The frames carrying lead cathodes are taken out one at a time. The lead is removed and replaced by a fresh sheet and the frame returned to the box, the whole operation taking but a few minutes for each frame. By this means the ordinary working is not interrupted at all, and the cleaning out of the boxes, which is necessary in the zinc boxes, is only required at long intervals. The lead, which contains from 2 to 12 per cent. of gold, is then smelted into bars and cupelled. The gold is deposited on the lead sheets as a thin, bright yellow film, which adheres firmly to the lead. The consumption of lead at these works is 750 lbs. per month, equal to 1½d. per ton of tailings, and the working expenses for treating 3,000 tons per month are as follows :—

| | £ | | d. | |
|---|---|---|---|---|
| Filling and discharging leaching-vats | 125 | monthly | 0·10 | per ton. |
| Cyanide | 75 | ,, | 0·6 | ,, |
| Lime | 15 | ,, | 1·2 | ,, |
| Caustic soda | 6 | ,, | 0·5 | ,, |
| Lead | 14 | ,, | 1·1 | ,, |
| Iron | 28 | ,, | 2·2 | ,, |
| White labour | 65 | ,, | 5·2 | ,, |
| Native wages and food | 20 | ,, | 1·9 | ,, |
| Coal | 57 | ,, | 4·6 | ,, |
| Stores and general charges | 41 | ,, | 3·2 | ,, |
| | £450 | or | 3 shillings per ton. | |

At a great many works, where correct sampling and assaying is carried out, the results—that is, the gold won—agree pretty closely with the assay results. At the Worcester the following were the results for the month of August, 1894 :—

| | Ozs. | Dwts. | Grs. | |
|---|---|---|---|---|
| 1,350 tons tailings taken from settling vats containing | 443 | 1 | 0 | gold |
| 1·750 ,, ,, ,, upper tailing dam containing | 498 | 15 | 12 | ,, |
| 3,100 tons tailings which according to assay contain | 941 | 16 | 12 | ,, |
| 3,100 ,, residues ,, ,, ,, ,, | 239 | 7 | 12 | ,, |
| Balance, showing the theoretical output or 74·6 per cent. | 702 | 9 | 0 | ,, |
| Actual gold won, or 74·1 per cent. | 697 | 15 | 15 | ,, |

The tailings which were treated assayed from 6 to 8 dwts. The tailings, or residues, after treatment, assayed from 1 to 2 dwts. The solutions which leave the precipitation-boxes still contain some gold, and the analysis in this case showed that the strong solution contained 4 dwts. 8 grs. per ton of solution. The weak solution contained only 10 grs.

On an average, in these works, strong solutions carry from 4 to 5 dwts., and the weak from 0 to 1 dwt.

In taking samples of the residue, it should be borne in mind that the portion of the residues in which the largest proportion of unextracted gold is contained is near the bottom of the filter-vat, say the last 12 to 15 inches.

The best way to sample is to take a sampling-iron and probe every carload as it goes to the works, and to do the like with every car as it leaves the works with the worked residues.

In the subjoined pages (405 to 411) will be found the details of a working scheme for one week, as carried out at the Worcester works; and to follow the method of treatment, I have chosen vat No. 4, as an example.

On the 20th August, 1894, this vat, which holds 135 tons, has been filled with tailings. It takes about five hours to fill such a tank. One ton of tailings is equal to 27 cubic feet.

| | |
|---|---|
| From 3.15 p.m. till 6.20 p.m. 10 tons of the alkaline wash are pumped into the vat . . . | 10 tons |

| | | | |
|---|---|---|---|
| At | 8.10 p.m. | strong solution is pumped on . . | 5 tons |
| ,, | 10.45 ,, | ,, ,, ,, . . | 5 ,, |
| ,, | 3.30 a.m. | ,, ,, ,, . . | 5 ,, |

### 21st August, 1894.

| | | | |
|---|---|---|---|
| At | 9 a.m. | strong solution is pumped on . . | 5 tons |
| ,, | 1 p.m. | ,, ,, ,, . . | 5 ,, |
| ,, | 5.20 ,, | ,, ,, ,, . . | 5 ,, |
| ,, | 10.15 ,, | ,, ,, ,, . . | 5 ,, |
| ,, | 3.40 a.m. | ,, ,, ,, . . | 5 ,, |

### 22nd August, 1894.

| | | | |
|---|---|---|---|
| At | 9.15 a.m. | strong solution is pumped on . . | 5 tons |
| ,, | 1.30 p.m. | ,, ,, ,, . . | 5 ,, |
| ,, | 6.30 ,, | ,, ,, ,, . . | 5 ,, |
| ,, | 9.35 ,, | ,, ,, ,, . . | 5 ,, |
| ,, | 4 a.m. | ,, ,, ,, . . | 5 ,, |

23RD AUGUST, 1894.

| | |
|---|---|
| At 7.30 a.m. strong solution is pumped on | 5 tons |
| Total strong solution | 70 tons |
| At 1.50 p.m. weak solution is pumped on | 7 tons |
| ,, 7.15 ,, ,, ,, ,, | 7 ,, |
| ,, 1.25 a.m. ,, ,, ,, | 7 ,, |
| Total weak solution | 21 tons |

24TH AUGUST, 1894.

| | |
|---|---|
| At 7.30 a.m. water wash is pumped on | 6 tons |
| ,, 1.50 p.m. ,, ,, ,, | 5 ,, |
| Total water wash | 11 tons |

The tailings are then leached dry and discharged in the morning of 25th August.

It takes, therefore, five and a-half days to leach and filter one tank, and the solutions consist of—

| | |
|---|---|
| 1. Alkaline wash | 10 tons |
| 2. Strong solution | 70 ,, |
| 3. Weak solution | 21 ,, |
| Total quantity of solution | 101 tons |

This is by working the Siemens and Halske process.

## *WORKING SCHEME FOR ONE WEEK AT THE WORCESTER WORKS.*

### **From 20th August to 26th August, 1894.**

### MONDAY, 20TH AUGUST, 1894.

| Tank. | Time. Discharging and Filling. | Time. Discharging and Filling. | Strong Solution. Tons. | Weak Solution. Tons. | Wash Water. Tons. | Remarks. | Volumetric Determination of the Strength of the Solutions. |
|---|---|---|---|---|---|---|---|
| No. 1 | 9.15 a.m. 5 tons.<br>5.40 p.m. 5 ,,<br>12.30 a.m. 5 ,,<br>6 a.m. to | 1 p.m. 5 tons.<br>9.5 p.m. 5 ,,<br>3.30 a.m. 5 ,,<br>7 a.m. Water. | 70 | 10<br>10<br>10 | 6 | | 7 a.m. Tank No. 5. Solution contained 0·03 per cent. K. Cy.<br>11.45 a.m. Tank No. 3. Solution contained 0·010 per cent. K. Cy.<br>1.25 p.m. Tank No. 3. Solution contained 0·024 per cent. K. Cy.<br>2.45 p.m. Tank No. 3. Solution contained 0·030 per cent. K. Cy. |
| No. 2 | 9 a.m. 5 tons.<br>6 p.m. 5 ,,<br>1.30 a.m. 5 ,, | 1.15 p.m. 5 tons.<br>9.15 p.m. 5 ,,<br>5.15 a.m. 5 ,, | 40<br>10<br>10<br>10 | | | | |
| No. 3 | 8.50 a.m. 5 tons.<br>6.10 p.m. 5 ,,<br>1 a.m. 5 ,, | 1.25 p.m. 5 tons.<br>9.25 p.m. 5 , | 15<br>10<br>10<br>5 | | | | |
| No. 4 | Discharging and Filling.<br>3.15 p.m. till 6.20 p.m. from No. 3 solution tank.<br>8.10 p.m. 5 tons.<br>3·30 a.m. 5 ,, | 10.45 p.m. 5 tons. | 10<br>5 | 10 | | | |
| No. 5 | Leaching dry. | | | | | | |

TUESDAY, 21ST AUGUST, 1894.

| Tank. | Time. Discharging and Filling. | Time. Discharging and Filling. | Strong Solution. Tons. | Weak Solution. Tons. | Wash Water. Tons. | Remarks. | Volumetric Determination of the Strength of the Solutions. |
|---|---|---|---|---|---|---|---|
| No. 1 | Leaching dry. | | | | | | 6.15 a.m. Tank No. 1. Solution contained 0·026 per cent. K. Cy.<br>8.30 a.m. Strong solution contained 0·050 per cent. K. Cy.<br>10 a.m. Tank No. 4. Solution contained 0·004 per cent. K. Cy.<br>11.45 a.m. Tank No 4. Solution contained 0·008 per cent. K. Cy.<br>2 p.m. Tank No. 4. Solution contained 0·012 per cent. K. Cy.<br>5 p.m. Tank No. 4. Solution contained 0·016 per cent. K. Cy.<br>7.45 p.m. Tank No. 4. Solution contained 0·022 per cent. K. Cy.<br>9.35 p.m. Tank No. 4. Solution contained 0·028 per cent. K. Cy.<br>10.30 p.m. Tank No. 4. Solution contained 0·03 per cent. K. Cy. |
| No. 2 | 9.30 a.m. 5 tons.<br>5.30 p.m. 5 ,,<br>12.30 a.m. 5 ,,<br>6.10 a.m. to | 1.30 p.m. 5 tons.<br>9 p.m. 5 ,,<br>3.20 a.m. 5 ,,<br>7 a.m. Water. | 70 | 10<br>10<br>10 | 5 | | |
| No. 3 | 9.15 a.m. 5 tons.<br>4.10 p.m. 5 ,,<br>12.40 p.m. 5 ,, | 1.15 p.m. 5 tons.<br>8.50 p.m. 5 ,,<br>3.30 a.m. 5 ,, | 40<br>10<br>10<br>10 | | | | |
| No. 4 | 9 a.m. 5 tons.<br>5.20 p.m. 5 ,,<br>3.40 a.m. 5 ,, | 1 p.m. 5 tons.<br>10.15 p.m. 5 ,, | 15<br>10<br>10<br>5 | | | | |
| No. 5 | Discharging and Filling.<br>3.40 p.m. till 7 p.m. from No. 3 solution tank.<br>8.40 p.m. 5 tons.<br>3.50 a.m. 5 ,, | 10.30 p.m. 5 tons. | 10<br>5 | 10 | | | |

## WEDNESDAY, 22ND AUGUST, 1894.

| Tank. | Time. Discharging and Filling. | Time. Discharging and Filling. | Strong Solution. Tons. | Weak Solution. Tons. | Wash Water. Tons. | Remarks. | Volumetric Determination of the Strength of the Solutions. |
|---|---|---|---|---|---|---|---|
| No. 1 | Discharging and Filling.<br>3.50 p.m. till 6 p.m. from No. 3 tank.<br>8 p.m. 5 tons.<br>2.40 a.m. 5 ,, | 9.45 p.m. 5 tons. | 10<br>5 | 10 | | | 6.15 a.m. Tank No. 2. Solution contained 0·038 per cent. K. Cy.<br>7.15 a.m. Tank No. 2. Solution contained 0·032 per cent. K. Cy.<br>8 a.m. Strong solution contained 0·054 per cent. K. Cy.<br>10.15 a.m. Tank No. 5. Solution contained 0·010 per cent. K. Cy.<br>1.30 p.m. Tank No. 5. Solution contained 0·016 per cent. K. Cy.<br>3.30 p.m. Tank No. 5. Solution contained 0·026 per cent. K. Cy.<br>4.30 p.m. Tank No. 5. Solution contained 0·030 per cent. K. Cy. |
| No. 2 | Leaching dry. | | | | | | |
| No. 3 | 9 a.m. 5 tons.<br>6.20 p.m. 5 ,,<br>12 midnight 5 ,,<br>6.5 a.m. to | 1.40 p.m. 5 tons.<br>9.55 p.m. 5 ,,<br>3.30 a.m. 5 ,,<br>7 a.m. Water. | 70 | 10<br>10<br>10 | 5 | | |
| No. 4 | 9.15 a.m. 5 tons.<br>6.30 p.m. 5 ,,<br>4 a.m. 5 ,, | 1.30 p.m. 5 tons.<br>9.35 p.m. 5 ,, | 40<br>10<br>10<br>5 | | | | |
| No. 5 | 9.30 a.m. 5 tons.<br>5.20 p.m. 5 ,,<br>2.50 a.m. 5 ,, | 1.15 p.m. 5 tons.<br>9.25 p.m. 5 ,, | 15<br>10<br>10<br>5 | | | | |

THURSDAY, 23RD AUGUST, 1894.

| Tank. | Time. Discharging and Filling. | Time. Discharging and Filling. | Strong Solution. Tons. | Weak Solution. Tons. | Wash Water. Tons. | Remarks. | Volumetric Determination of the Strength of the Solutions. |
|---|---|---|---|---|---|---|---|
| No. 1 | 9.30 a.m. 5 tons.<br>5.30 p.m. 5 ,,<br>12 midnight 5 ,, | 1.25 p.m. 5 tons.<br>9.35 p.m. 5 ,, | 15<br>10<br>10<br>5 | | | | 6.30 a.m. Tank No. 3. Solution contained 0·044 per cent. K. Cy.<br>7.45 a.m. Tank No. 3. Solution contained 0·024 per cent. K. Cy. |
| No. 2 | Discharging and Filling. 4 p.m. to 5.45 p.m. 10 tons from No. 3 tank.<br>6.45 p.m. 5 tons.<br>12.10 a.m. 5 ,, | 9.25 p.m. 5 tons. | 10<br>5 | 10 | | | 8.45 a.m. Tank No. 1. Solution contained 0·005 per cent. K. Cy.<br>12 a.m. Tank No. 1. Solution contained 0·008 per cent. K. Cy. |
| No. 3 | Leaching dry. | | | | | | 4.30 p.m. Tank No. 1. Solution contained 0·016 per cent. K. Cy. |
| No. 4 | 7.30 a.m. 5 tons.<br>1.50 p.m. 7 ,,<br>1.25 a.m. 7 ,, | 7.15 p.m. 7 tons. | 65<br>5 | 14<br>7 | | | 6.15 p.m. Tank No. 1. Solution contained 0·024 per cent. K. Cy.<br>7.30 p.m. Tank No. 1. Solution contained 0·03 per cent. K. Cy. |
| No. 5 | 9.15 a.m. 5 tons.<br>2.20 p.m. 5 ,,<br>12.20 a.m. 5 ,, | 1.40 p.m. 5 tons.<br>9.15 p.m. 5 ,,<br>3.30 a.m. 5 ,, | 40<br>10<br>10<br>10 | | | | 8.30 p.m. Strong solution 0.054 per cent. K. Cy. |

## FRIDAY, 24TH AUGUST, 1894.

| Tank. | Time. Discharging and Filling. | Time. Discharging and Filling. | Strong Solution. Tons. | Weak Solution. Tons. | Wash Water. Tons. | Remarks. | Volumetric Determination of the Strength of the Solutions. |
|---|---|---|---|---|---|---|---|
| No. 1 | 9.40 a.m. 5 tons.<br>5.20 p.m. 5 ,,<br>12.30 a.m. 5 ,, | 1.15 p.m. 5 tons.<br>9.20 p.m. 5 ,,<br>3.30 a.m. 5 ,, | 40<br>10<br>10<br>10 | | | | 4.30 a.m. Tank No. 4. Solution contained 0·064 per cent. K. Cy.<br>6.30 a.m. Tank No. 4. Solution contained 0·064 per cent. K. Cy.<br>7.45 a.m. Tank No. 4. Solution contained 0·062 per cent. K. Cy. |
| No. 2 | 9.55 a.m. 5 tons.<br>5.30 p.m. 5 ,,<br>5.45 a.m. 5 ,, | 1.30 p.m. 5 tons.<br>9.30 p.m. 5 ,, | 15<br>10<br>10<br>5 | | | | 11.15 a.m. Tank No. 4. Solution contained 0·046 per cent. K. Cy.<br>12.30 p.m. Tank No. 4. Solution contained 0·044 per cent. K. Cy. |
| No. 3 | Discharging and Filling.<br>3.15 p.m. to 5.50 p.m. from No. 3 tank.<br>7.20 p.m. 5 tons.<br>12.45 a.m. 5 ,, | 9.45 p.m. 5 tons. | 10<br>5 | 10 | | | 3 p.m. Tank No. 4. Solution contained 0·046 per cent. K. Cy.<br>4.30 p.m. Tank No. 4. Solution contained 0·046 per cent. K. Cy.<br>10 p.m. Tank No. 4. Solution contained 0·042 per cent. K. Cy. |
| No. 4 | 7.30 a.m. to 8.30 a.m. 6 tons. Water.<br>1.50 p.m. 5 ,, ,,<br>Leaching dry. | | 70 | 21 | 6<br>5 | | 8.45 p.m. Strong solution contained 0·054 per cent. K. Cy.<br>12.5 a.m. Tank No. 4. 0·036 per cent. K. Cy. |
| No. 5 | 10.10 a.m. 5 tons.<br>5.40 p.m. 5 ,,<br>1 a.m. 5 ,,<br>6 a.m. to | 1.45 p.m. 5 tons.<br>10 p.m. 5 ,,<br>4.25 a.m. 5 ,,<br>7.45 a.m. Water. | 70 | 10<br>10<br>10 | 5 | | 12 p.m. Tank No. 2. Solution contained 0·026 per cent. K. Cy.<br>2.10 p.m. Tank No. 2. Solution contained 0·035 per cent. K. Cy. |

## SATURDAY, 25TH AUGUST, 1894.

| Tank. | Time. Discharging and Filling. | Time. Discharging and Filling. | Strong Solution. Tons. | Weak Solution. Tons. | Wash Water. Tons. | Remarks. | Volumetric Determination of the Strength of the Solutions. |
|---|---|---|---|---|---|---|---|
| No. 1 | 8.30 a.m. 5 tons.<br>6 p.m. 5 ,,<br>11.30 p.m. 5 ,,<br>5 a.m. to 6 a.m. 6 tons. | 1.15 p.m. 5 tons.<br>8.30 p.m. 5 ,,<br>3.10 a.m. 5 ,,<br>7.30 a.m. to 8.10 a.m. 4 tons. Water. | 70 | 10<br>10<br>10 | 10 | | 6.45 a.m. Tank No. 5. Solution contained 0·046 per cent. K. Cy.<br>7.45 a.m. Tank No. 5. Solution contained 0·042 per cent. K. Cy.<br>10 a.m. Tank No. 5. Solution contained 0·034 per cent. K. Cy. |
| No. 2 | 9 a.m. 5 tons.<br>3.35 p.m. 5 ,,<br>11.45 p.m. 5 ,, | 1 p.m. 5 tons.<br>9.5 p.m. 5 ,,<br>4.30 a.m. 5 ,, | 40<br>10<br>10<br>10 | | | | 9.30 a.m. Tank No. 4. Solution contained 0·026 per cent. K. Cy.<br>8 a.m. Strong solution contained 0·060 per cent. K. Cy. |
| No. 3 | 8.15 a.m. 5 tons.<br>5.45 p.m. 5 ,,<br>12.45 a.m. 5 ,, | 12 noon. 5 tons.<br>9.15 p.m. 5 ,, | 15<br>10<br>10<br>5 | | | | 8.15 a.m. Tank No. 3. Solution contained 0·008 per cent. K. Cy.<br>10.45 a.m. Tank No. 3. Solution contained 0·010 per cent. K. Cy. |
| No. 4 | Discharging and Filling.<br>6 p.m. till 8.15. p.m. from No. 3 tank.<br>10.5 p.m. 5 tons.<br>4.30 a.m. 5 ,, | 1 a.m. 5 tons | 10<br>5 | 10 | | | 12.30 a.m. Tank No. 3. Solution contained 0·013 per cent. K. Cy.<br>2.30 p.m. Tank No. 3. Solution contained 0·020 per cent. K. Cy. |
| No. 5 | Leaching dry. | | | | | | 4.30 p.m. Tank No. 3. Solution contained 0·033 per cent. K. Cy. |

## SUNDAY, 26TH AUGUST, 1894.

| Tank. | Time. Discharging and Filling. | Time. Discharging and Filling. | Strong Solution. Tons. | Weak Solution. Tons. | Wash Water. Tons. | Remarks. | Volumetric Determination of the Strength of the Solutions. |
|---|---|---|---|---|---|---|---|
| No. 1 | Leaching dry. | | | | | | 6.40 a.m. Tank No. 1. Solution contained 0·05 per cent. K. Cy.<br>8.15 a.m. Tank No. 1. Solution contained 0·042 per cent. K. Cy.<br>9.50 a.m. Tank No. 1. Solution contained 0·035 per cent. K. Cy.<br>8.15 a.m. Strong solution contained 0·056 per cent. K. Cy.<br>11.15 a.m. Tank No. 4. Solution contained 0·008 per cent. K. Cy.<br>2.30 p.m. Tank No. 4. Solution contained 0·016 per cent. K. Cy.<br>4.15 p.m. Tank No. 4. Solution contained 0·028 per cent. K. Cy.<br>5.15 p.m. Tank No. 4. Solution contained 0·034 per cent. K. Cy. |
| No. 2 | 9.40 a.m. 5 tons.<br>5.30 p.m. 5 ,,<br>12.50 a.m. 5 ,,<br>6 a.m. till 7 a.m. | 1.15 p.m. 5 tons.<br>9.30 p.m. 5 ,,<br>3.45 a.m. 5 ,, | 70 | 10<br>10<br>10 | 6 | | |
| No. 3 | 9.25 a.m. 5 tons.<br>5.5 p.m. 5 ,,<br>12.35 a.m. 5 ,, | 1 p.m. 5 tons.<br>9.20 p.m. 5 ,,<br>3.30 a.m. 5 , | 40<br>10<br>10<br>10 | | | | |
| No. 4 | 9.10 a.m. 5 tons.<br>5.15 p.m. 5 ,,<br>12.20 p.m. 5 ,, | 12.45 p.m. 5 tons.<br>9.10 p.m. 5 | 15<br>10<br>10<br>5 | | | | |
| No. 5 | Discharging and Filling.<br>5.30 p.m. till 8.20 p.m. from No. 3 tank.<br>9 p.m. 3 tons.<br>11 p.m. 4 ,,<br>2.10 p.m. 3 ,, | 10 p.m. 3 tons.<br>3.15 p.m. 3 ,, | | 10<br>6<br>4<br>6 | | | |

# CHAPTER XIII.

## *ERECTION OF A CYANIDE PLANT.*

PLANNING THE WORKS—Slimes—Effect of Stamping on the Ore—Plant—Intermediate Filling—Direct Filling—The Filter or Leeching Tanks—Stock Solution Tanks—Zinc Precipitation Boxes.

**Planning the Works.**—In the erection of a cyanide plant, before planning the same, some essential points have to be considered. Are the works to be erected to treat an old accumulated stock of tailings? Or have they to be laid out to treat the tailings as they come from the battery? In many cases both these points have to be combined.

As tailings reservoirs are generally situated on the lowest point below the battery site, no provision exists below such dams for the erection of works which would permit of further dumping-ground and handling of the stuff by gravitation. At all events, the topographical conditions of Witwatersrand would allow such an arrangement only in rare instances. In localities where the fall of the ground below the reservoirs permits of the erection of the works, I would recommend this to be done, as it permits of the charging of tanks and their discharging by gravitation. In most cases the opposite course had to be resorted to on these fields; the tailings from the old pits or reservoirs had to be hauled up-hill to the cyanide works, steam power being mostly used. The arrangement is simple enough, as the dumping cars are pulled up on an inclined trestle-work above the leaching tanks, and after discharging their contents they run back by gravitation, and are held back by the brake of the hauling drum. In large works five to six trucks, holding 20 cubic feet each, are hauled up at a time. At every mine the mechanical arrangement for the filling of the tanks is different, depending on local conditions. Messrs. Fraser and Chalmers have lately introduced a system of mechanical haulage by means of endless wire ropes which works very well, and which I would recommend in preference to anything I have seen on these fields.

To work old tailings by the cyanide process offers no difficulty to percolation, as they come to the works in the proper condition. They were cleaned of the slimes by the natural system of concentration, which takes place in the collecting reservoirs. It is very interesting to stand at the discharge end of the launder carrying the tailings to the reservoir, and to see how the tailings arrange themselves according to the natural laws of gravitation, and are prepared here for subsequent treatment. At the head of the tailing pit the coarser tailings accumulate, and near the dam the finest, and also slimes. The overflow from the first reservoir is collected in a second reservoir, where the slimy, clayey residue accumulates, which, strange to say, is as rich, and even richer, than the tailings in the first reservoir.

To lay out plans for an accumulated stock of tailings offers no great difficulty, provided there is near by a sloping ground permitting of the discharging from the leaching tanks and their dumping by gravitation. If the country is flat, the re-worked tailings will have to be hauled up an incline again and then dumped. On a flat site it will be necessary to place the leaching vats on masonry sufficiently high to give room for discharge, and gradient for the flow of the leaching solutions to the precipitation boxes.

When works are erected to treat tailings which are discharged from the battery, important appliances have to be resorted to, to prepare them for the cyanide treatment, and before they are collected in the leaching vats, owing to the physical condition of the powdered ore.

The discharge launder which carries the tailings from the battery to the cyanide works should have a grade of at least 3 ft. 6 in. in the 100 ft. to insure a good flow. In a flat country where no grade exists, the tailings should be elevated by means of bucket wheels to the proper height. From the experience gained on these fields, tailing pumps have not given satisfaction; it may be that they were not properly constructed, as I am told that in Australia they are in various places in successful operation. There are on several mines here large tailing wheels in use, and I should consider them the best way of elevating tailings, as they require very little attention and repairs when properly constructed and set.

Supposing that we desire to erect a cyanide plant directly behind a battery, the following grade would be required for doing the whole work by gravitation. Supposing the plant to be located 100 ft. from the battery—

| | Feet. | Inches. |
|---|---|---|
| The grade for the discharge launders will require . | 3 | 6 |
| Masonry for settling tanks . . . . . . . | 6 | 6 |
| Settling tanks . . . . . . . . . . | 10 | 0 |
| Masonry for leaching tanks . . . . . . | 6 | 6 |
| Settling tanks . . . . . . . . . . | 10 | 0 |
| Precipitating boxes and grade for outflow pipes . . | 6 | 0 |
| Total grade . . | 42 | 6 |

To this could be added from 6 to 10 ft. of grade for the storage tanks holding the cyanide solutions, wash and alkaline waters, but these are differently placed, and a lack of further grade would present no difficulties, as will be explained later on.

Plate XIII., showing a section of the cyanide plant of the Princess works, illustrates such a mode of arrangement as is here described.

**Slimes.**—The conglomerates on these fields, after stamping, contain a very large per cent. of slimes. Under slimes are understood the very fine particles of talcose and clayey material mixed with the very fine grains of quartz, iron oxides, and sulphides. If the whole of this fine material be allowed to collect

with the coarser grains, the percolation of fluids through the mass becomes impossible, and, therefore, mechanical means have to be adopted, aiming at a separation of the slimes from the coarser material.

Two methods have been introduced on these fields, aiming at the elimination of the slimes. The one by *direct filling* is the system introduced by Mr. Henning Jennings, the well known mining engineer; and the other *the intermediate filling* adopted by Mr. Charles Butters and Captain Mein, the manager of the Robinson mine.

I will take occasion to remark here that the appliances for the plant were materially changed by Mr. Charles Butters, who has done a great deal for the advancement of the metallurgical treatment of the ores on these fields, as he has introduced a great many practical details, all tending to lessen the cost of the process, and I consider it a very pleasant duty on my part to express my appreciation of his labours and of the good he has done by his work.

The elimination of the slimes has an economic bearing on the gold mining industry of these fields, when it is considered that at least 30 per cent. of the Witwatersrand ores, after crushing, pass away into slime pits: therefore, at the present production of 250,000 tons of ore per month, 75,000 tons go into the slime pits. At the present rate of progress it is almost certain that the tonnage will increase to nearly double this amount, and that within three years the Witwatersrand will be producing 300,000 ounces of gold or one million sterling monthly. If we take the average value of the slimes all around at only 5 dwts. per ton, this represents nearly 20,000 ounces of gold which goes into the pits monthly.

Up to the present no cheap method has been devised to deal with the slimes, so as to win the gold from them at a profit. The question of treating the slimes successfully is simply a mechanical one, as there is no chemical difficulty in the way: on the contrary, the solution of the gold can be easily effected, owing to the fine state of division in which the gold exists in the slimes.

My impression is that the eventual solution of the problem will be a system of agitation in vats fitted with mechanical stirrers, and after agitation and settling the solution will be decanted and a weaker solution added again, again stirred and decanted, till the last wash waters will only show traces of gold. This method will become applicable owing to the introduction of the Siemens and Halske process, which effects the precipitation of the gold from the extreme weak solutions which will have to be used. A successful solution of this problem will add largely to the gold production of the fields.

**The Effect of Stamping on the Ore.**—When ores are stamped in a battery, the resulting product is very uneven, and this is one of the recognised disadvantages of the batteries in use in all gold countries.

The majority of the mines on these fields crush fine, using mostly screens of 900 mesh per square inch, and about 50 per cent. of the ore is converted into slimes. To illustrate the fineness to which stamping reduces the ore I may mention that Mr. Williams, the metallurgist of the Crown Reef mine, informed me that the slimes which flow into their pits do not settle in 24 hours, and that if the overflow is taken and allowed to settle, these second settlings will assay 6 dwts. per ton, being richer, therefore, than the first settlings. He further stated that after the water leaves the second slime pit it still holds in suspension 2 per cent. of the total gold contained in the ore as it comes from the mine. This is the result from careful experiments and trials.

In regard to the difficulty of settling the fine slimes which remain in suspension, it was ascertained at the Paarl Central works, that the overflow from a large tailing dam carried 8 per cent. of the rock crushed in suspension. As the water on these fields does not run to waste, but is used over and over again in the batteries, there is really no loss incurred, as the suspended slimes eventually settle, the water only retaining a certain constant portion.

To give a further illustration of the effects of stamping on the Witwatersrand conglomerates, I will furnish some figures which were communicated to me by Mr. Bettel, a gentleman whose name has also been closely connected with the cyanide process.

40 lbs. of tailings were caught at a battery in a tub, and at least 30 per cent. passed away as slimes, the ore coming through a 900 mesh screen. After drying, it was sifted through a screen of 1,600 mesh per square inch, and there remained on the sieve 1·85 per cent. (1)

It passed afterwards through three sieves as follows:

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| 3,600 | mesh and | there | remained | on the sieve | 27·93 | per cent. | (2) |
| 7,225 | ,, | ,, | ,, | ,, | 20·74 | ,, | (3) |
| 14,400 | ,, | ,, | ,, | ,, | 7·70 | ,, | (4) |

The sands passing the last sieve were panned, and

| | | | |
|---|---|---|---|
| A. Remaining in the dish represented . . | 11·80 | per cent. | (5) |
| B. Finest sand panned away . . . . | 22·34 | ,, | (6) |
| C. Slimes collected from the panning water . | 7·64 | ,, | (7) |
| Total . . | 100·00 | | |

Each of these grades was assayed, with the following results—

| | | | |
|---|---|---|---|
| 1 . . . . . . | 3·03 | dwts. | per ton. |
| 2 . . . . . . | 4·00 | ,, | ,, |
| 3 . . . . . . | 4·40 | ,, | ,, |
| 4 . . . . . . | 4·65 | ,, | ,, |
| 5 . . . . . . | 6·30 | ,, | ,, |
| 6 . . . . . . | 2·85 | ,, | ,, |
| 7 . . . . . . | 2·85 | ,, | ,, |

These figures are very instructive, as they show how very fine the ore is crushed in a battery, and that the material, after passing through a 120 mesh sieve per linear inch, can be concentrated, and will yield a concentrate of over double the value of the sands washed away. No doubt the practical part of these figures will be very soon appreciated on these fields, when concentration will receive closer attention than it does at present.

After this notice of the important part which the slimes play in the metallurgy of the Rand gold fields, I will revert again to the planning of the works.

**The Plant.**—The main features of a cyanide plant are the settling vats, the filter vats, the solution storage tanks, and the precipitation-boxes.

The filter vats are made of timber, or they are brick vats lined with cement. At the Langlaagte Estate and Gold Mining Company, circular excavations were made in the rocky ground, lined with bricks and cemented, forming tanks 40 ft. in diameter and 10 ft. deep, each holding 400 tons of tailings.

Filter vats made of timber will last for years, as contact with cyanide solutions does not seem to destroy it.

I have not been able to obtain the cost of a plant constructed in masonry, but I should consider it more expensive than the timber plant. Where wooden tanks are in use they are placed in such a position that free access can be had to the bottoms in case of leakage, which is an advantage.

The number of filtering vats required for a plant depends on the capacity of the battery, and the time it takes to treat a charge of ore. If we want to treat 100 tons of tailings daily, and it takes, say, 3 days to fill, leach, and discharge a vat, it will require 4 leaching vats of 100 tons capacity each (dimensions 22 ft. in diameter and 5 ft. high), but for safety one extra tank is added. The tendency on these fields is to construct a few large vats for the plant, instead of a large number of small ones. As long as shallow vats were employed, there was no difficulty in shovelling out the tailings over the sides; but with deep vats the bottom discharge was introduced by Mr. Butters.

Before the tailings go to the leaching tanks they have to be freed of their slimes, and I shall now describe the two methods adopted for this purpose.

**Intermediate Filling. Messrs. Butters and Mein's Distributor.**—The first attempts at intermediate filling were made by running the battery tailings to the centre of a circular vat, and allowing the overflow to take place at one point. This did not prove successful, because the sand piled up in a central conical heap, and the slimes settled in the deeper water around the sides of the tank. The next plan was to run the pulp into the vat through a series of stationary launders, delivering at several fixed points. This method improved the distribution, but the result was still unsatisfactory. Then, in order to give an uniform overflow at every point of the periphery of the vat, a circular trough was fixed round the top to collect the overflow and deliver it to a launder.

Each of these alterations was a step in the right direction, but the system of settling could not be considered successful until after the introduction of an automatic revolving distributor. This appliance consists of a central casting, with a vertical spindle A revolving in a footstep bearing B, which casting carried a conical hopper E and a number of radial pipes C with bent ends, as shown in section in Plate XIV. and Fig. 146 *a*.

The distributor is fixed on an iron column in the centre of the vat. The bends at the end of the pipes cause the apparatus to revolve by the reaction of the pulp as it leaves the pipes.

Each pipe has a different length, in order to distribute the pulp over a number of concentric circles. This also has its faults, as it was found that the slimes collected in narrow rings between the outlets of each pipe, giving rings of clean sand alternately with rings of slime. The difficulty was overcome by attaching flattened nozzles to the ends of the pipes, causing the pulp to spread over a wider area, and also by increasing the number of pipes.

As is noticed by this plan, the arrangement is a hemispherical bowl from which radiate 8, 12, to 16 pieces of pipes of various length, which is set in motion by the centrifugal action of the discharging water, something similar to a garden

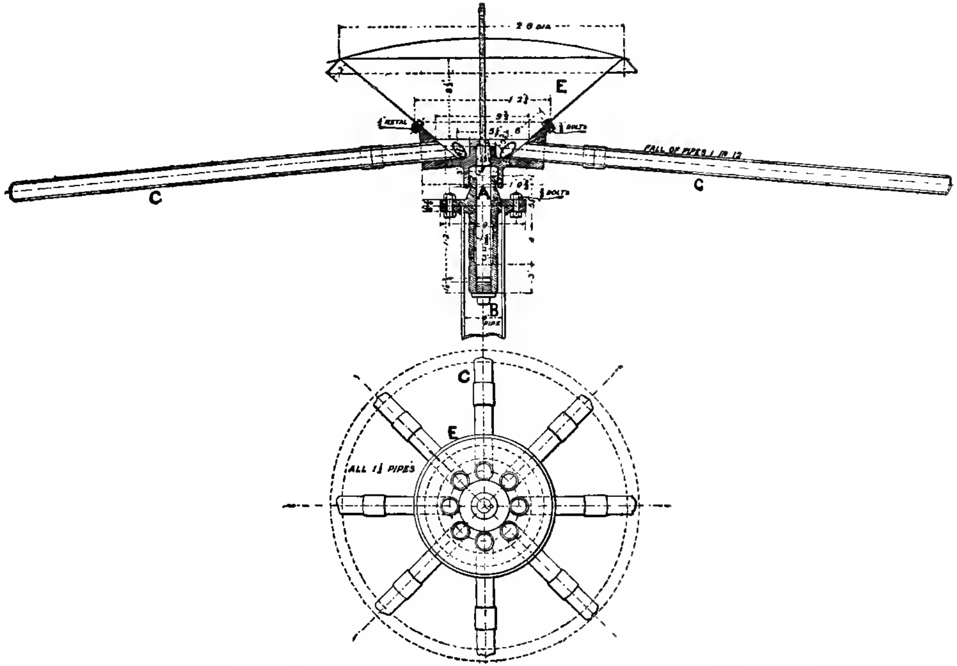


FIG. 146*a*.—MESSRS. BUTTERS AND MEIN'S AUTOMATIC DISTRIBUTOR (SIZE A).

Missing Page

sprinkler, only that it revolves slowly. The bowl is covered with a coarse screen so as to prevent chips or leaves to enter and choke the pipes. The diameter of the discharge pipes is 1½ to 2½ in.

It is necessary to fill the vat with clean water before admitting the pulp. If this is not done, the water is practically stationary, and a constant settlement of slimes takes place until the vat is full and the overflow begins, in which case the tailings in the lower part of the vat will always be more slimy than those in the upper part. For the same reason it is essential that the overflow be continuous until the vat is full of sand; for if any stoppage takes place slime settlement in excess occurs, and a complete layer of slime is formed across the vat which prevents the overlying sand from draining dry. Therefore, when the battery is stopped, an equal quantity of water should be supplied to the vat.

When the pulp is admitted into the tank previously filled with water, the light slime remains in suspension and overflows into the annular ring which surrounds the tank at the top, and from the discharge opening is carried by a launder to the slime-pit.

When the vat is filled with tailings, the outlet pipe below the filter is opened and the water allowed to drain off, the draining taking about fifteen to twenty-four hours. When holes are dug down to the discharge doors, water again commences to flow from the outlet, consequently it is advantageous to dig these holes about six hours before the discharging.

One very important matter is the proper size of vat to be used for a given tonnage crushed in the battery. It is, of course, desirable to catch as large a quantity of slimes with the sands in the tailings as is possible without rendering the product unleachable. When the vats are too small they carry away too much fine sand with the slime; and if they are too large they catch too much slime, which settles in excess. The great difficulty to overcome yet with these intermediate vats is to get the last foot or two near the bottom properly drained, and if discharged and transferred to the leaching tanks in this wet condition, the excess of moisture dilutes the cyanide solution.

To facilitate and hasten the leaching, various devices have been adopted. At the Princess works, where the ground is steep, the drainage pipe has been extended down to the reservoir, thereby causing a natural suction. At the Simmer and Jack works the drainage pipe is connected with a steam exhaust acting like an ejector, so as to cause a vacuum below the filter, and thereby the rate of leaching is increased. At the Worcester works the vats catch from the crushed ore from 75 to 80 per cent. of good leachable tailings, containing 12 per cent. moisture after draining eighteen to twenty-four hours.

The following are the sizes of the intermediate vats erected at some of the works:—

Meyer and Charlton Gold Mining Company, treating 120 tons per day, has 4 vats, each 20 ft. in diameter and 8 foot staves.

Pioneer Gold Mining Company, treating 70 tons daily, has 2 vats, each 20 ft. in diameter, and 14 foot staves.

Worcester Gold Mining Company, treating 70 tons daily, has 2 vats, each 20 ft. in diameter, and 8 foot staves.

Princess Gold Mining Company, treating 85 tons daily, has 2 vats, each 20 ft. in diameter, and 7 foot staves.

The Robinson Gold Mining Company, treating 330 tons per day, has 6 vats, each 24 ft. in diameter, and 11 foot staves.

When all the pulp is running into 1 vat, only about 66 per cent. of the crushed ore is caught, but the whole of this is clean sand and drains sufficiently. If, however, the total pulp from the battery was run into 2 vats, about 80 per cent. of the crushed ore, instead of 66 per cent., would be obtained from the distributing tank. After the water has been leached out, the ore is discharged through bottom discharges into trucks and taken to the leaching tanks. In some localities the distributing tanks are on a higher level than the leaching tanks, and the trucks are then run by gravitation to the leaching tanks. At some works the distributing tanks are at lower level than the leaching tanks, and then the trucks have to be hauled up by steam power.

The framework of the tram lines on which the trucks are hauled up to the leaching tanks rests inside the tanks and on the masonry foundation, and at large works there is generally a double line of rails on top of the tanks. The vats and storage tanks are in the open, and not covered by a building.

The following are the advantages of intermediate filling, as introduced by Mr. Charles Butters :—

1. It is claimed that, by means of Mr. Butters' distributor, from 75 to 80 per cent. of sands, both coarse and fine, with some slimes, are collected in the intermediate tanks, the bulk of the slimes escaping with the effluent water, which is practically free from sands.

2. The water is drained off as near as possible, and when the intermediate vat is discharged through the bottom discharges, the sands during the operation get thoroughly mixed up, thus being in the best condition for treatment by cyanide.

3. Oxidation of pyrites is very slight, so that very little cyanide will be consumed.

To an impartial observer it would appear that the system of intermediate filling would commend itself as the one which is more practical, as the tailings undergo, so to say, a special preparation for the subsequent lixiviation. The expense of transferring the tailings from the intermediate tank to the leaching tank is so slight that it cannot be considered as an important item.

The cost of charging tailings and discharging the residues has been brought down at the Robinson Mine to 10d. per ton of 2,000 lbs., and generally stands in the accounts of other works at about 1s.

Messrs. Butters and Mein's distributors are constructed in three sizes, according to the following particulars :—

Size A distributor has 8 distributing pipes, all of 1½ in. diameter, and is the size of distributor used on batteries up to 30 stamps.

Size B has 12 distributing pipes—6 of 2 in. diameter, 2 of 1½ in. diameter, and 4 of 1¼ in. diameter—and is the size of distributor used on batteries of from 30 to 70 stamps.

Size C has 16 distributing pipes—2 of 2½ in. diameter, and 14 of 2 in. diameter—and is the size of distributor used on batteries of from 60 to 120 stamps.

The above sizes of distributors have been calculated on the average crushings per stamp for the Rand.

**Direct Filling.**—This method, introduced at the works of the Heriot, City and Suburban, Crown Reef, Paarl Central, and Geldenhuis Estate companies, consists in passing the pulp as it leaves the plates into a *hydraulic separator*, a kind of crude spitzlutte. The pulp is here divided into two streams, one overflowing, carrying slimes with very fine sands; the other, consisting of coarse sands, some fine sands and slimes, which are conveyed by means of an india-rubber hose to the leaching tanks, in which one or more Kaffirs are employed to effect the even distribution of the pulp, by moving the hose about to different parts of the vat. The water passes off by adjustable gates fitted inside the vats, carrying with it fine sands, slimes, and some coarse sands. The advantages of the process are:—

1. This method treats pyritic tailings with the minimum of oxidation, as they are not exposed to the action of the air from the time they leave the battery.

2. A second handling of the tailings before treatment is avoided.

3. A preliminary rough concentration, or rather classification of the coarser particles of the tailings is effected.

There is at present a great controversy going on regarding the advantages of direct filling as against intermediate filling, and according to Mr. Bettel the disadvantages of the process are:—

"1. The tailings pack tightly in the vat, and consequently do not drain completely, and a diffusion of the first cyanide solution which is applied takes place at the commencement of leaching, causing loss of cyanide and gold. At the Crown Reef works I noticed that the distribution seemed to be pretty regular, and drainage can be assisted by means of exhaust pumps.

"2. The distribution of the sands and slimes is not so even, and some sands escape treatment, being protected by impervious layers of slime, the cyanide naturally escaping by the paths of least resistance. In leaching tanks where an uneven distribution of slimes and sands takes place, the slimy portion will not drain off; and on discharging such a tank, it is easily noticed that the streaks of slime are saturated with moisture and are still gold bearing, whereas the sandy portion has the solution drained off. The importance of an even distribution and mixture of the pulp can hardly be estimated.

"3. At most of the works where direct filling is introduced, square cement tanks are employed, and the discharging of these is not so practical as the wooden ones fitted with bottom discharges."

In Plate XV., the tailing wheel, vanner room, and cyanide vats at Jumpers mine are shown in section.

**The Filter or Leaching Tanks.**—These are, in most instances, made circular, that form being the strongest. They are from 20 to 42 ft. in diameter, and from 8 to 14 ft. in height, and should be constructed of well-seasoned lumber, with staves 3 to 4 in. thick, having their inner and outer faces cut to correspond to the arc of circle of the tank, and their edges radial to this circle (Fig. 146*b*). The staves are not tongued or grooved, the pressure of the hoops being sufficient, if the tank is well made, to make them perfectly tight. The staves should be at least 1 ft. longer than the inside depth of the tank, and gained from 1½ in. into the bottom timbers, with a chime of several inches.

The bottoms are made of 3 by 9 in. deals, tongued and grooved (Fig. 146*c*), and put together with white lead, or litharge and glycerine. The hoops should be made by wrought iron rods from $\frac{3}{4}$ to $1\frac{1}{2}$ in. in diameter, according to the size of the tank, with threaded ends passing through wrought iron lugs and tightened by hexagonal nuts. When the tanks are of large diameter these hoops are made in sections. The outside of the tanks can be painted in lead paint.

The bottoms of the vats rest on wooden beams 6 by 9 in., placed 18 in. apart. These beams are placed across the stone foundation, and rest in their turn on planks $1\frac{1}{2}$ by 11 in. The planks are put between the stone foundation and the beams to merely ensure a perfectly level surface.

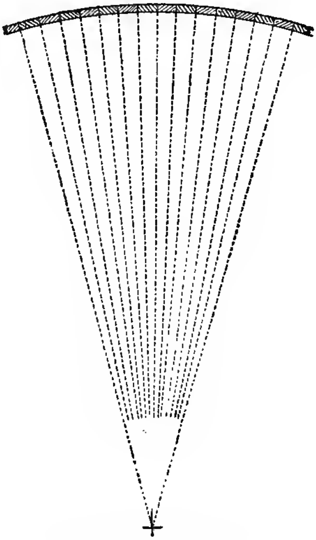


FIG. 146*b*.—STAVES CUT TO CIRCLE.

The construction of these vats should not be entrusted to the hands of any other workmen than experienced coopers.

It is obvious that tanks holding such enormous weights should rest on good foundations, and in every case where

Missing Page

wooden foundations have been used the result has been that the tanks settled, got out of plumb, and leakages occurred.

The filters are constructed of wooden slats, $1\frac{1}{2}$ by 4 in., set 12 in. apart, fastened to the bottom by wooden pins. Grooves $\frac{3}{4}$ in. deep and 3 in. wide are cut in a number of places in the bottom of these slats to allow a free passage of the solution along the bottom. On top of these slats are laid strips of wood 1 by 1 in., only 1 in. apart from each other, making openings 1 in. square. Between the ends of this wooden grating and the inside of the tank an annular space of about $1\frac{1}{2}$ in. wide is left, which is partly filled by a strip of wood 1 in. thick, bent to the circle of the tank. Over this and the slats is placed cocoa-nut matting and burlap, and held by a rope $\frac{1}{2}$ in. in diameter, which is driven into the space remaining between

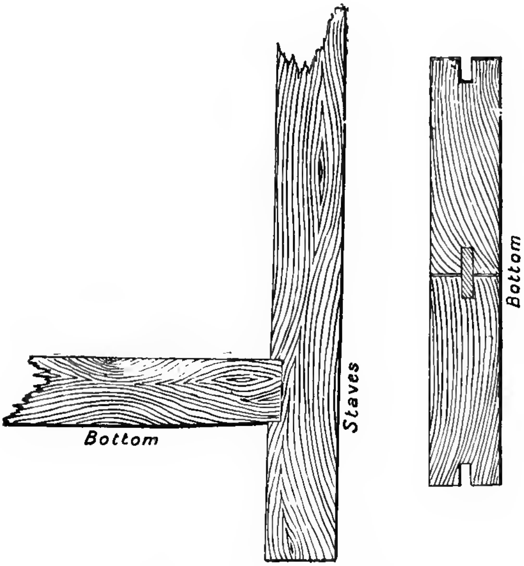


FIG. 146c.—CONSTRUCTION OF FILTER VATS.

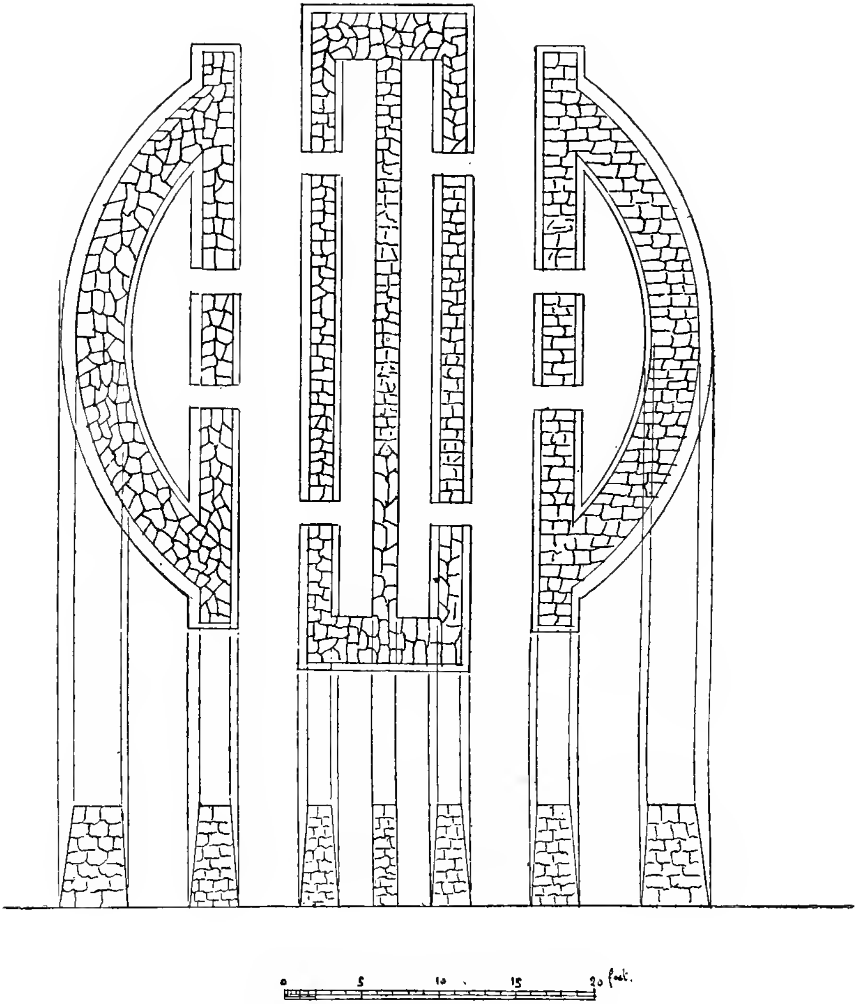


FIG. 146*d*.—STONE FOUNDATION FOR FILTER VATS.

the strips of wood and the staves of the tank. On top of the matting are laid again slats of wood 1 by 3 in., parallel one to the other, about 6 in. apart, their object being to protect the matting from being injured when shovelling the tailings through the man-holes into the trucks below.

The stone foundations are usually 6 ft. 6 in. high above the level of the rails, and are composed of a series of walls closed at their ends, leaving one or two passages underneath for the trucks (Fig. 146*d*).

Each leaching vat has a separate drain pipe, 1 to 2 in. in diameter, and these pipes are so arranged in the extractor house, as to lead the strong solution to the strong extractor box and the weak solution to the weak extractor box. In some works there is one main collecting drain pipe for strong solution and one for weak solution, and the connections are shown in Fig. 146*e*. By shutting the valve, A, leading to strong collecting pipe, and opening valve, B, leading to weak collecting pipe,

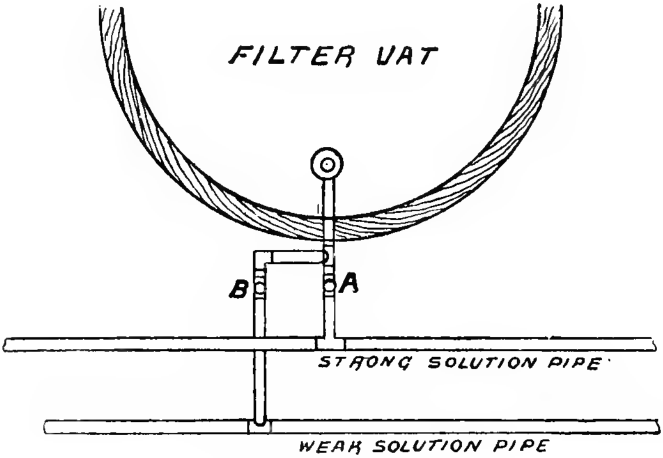


FIG. 146*e*.—SOLUTION PIPES.

the flow is regulated. Filtration is best assisted by causing a vacuum under the filter bed; by connecting the drain pipe with a steam pipe and passing a jet of steam through the same, a vacuum is created under the filter bed.

I should also mention that the best and cheapest method of discharging the tailings from the leaching vats is to sluice them out from a side door, but for this purpose a stream of running water is required, which on these fields is not available.

Mr. Feldtmann * describes a system of discharging tailings from the leaching vats through a bottom discharge door into a launder, whence a copious stream of water carries the residues into the creek below.

The discharge doors can also be made on the side of the vat when the residues are to be sluiced out.

The round wooden filter vats on these fields are discharged by bottom discharge doors, which are closed by means of Butters' discharge lids. According to the size of the vats, there are two, four, six, or eight of these discharge openings to each vat.

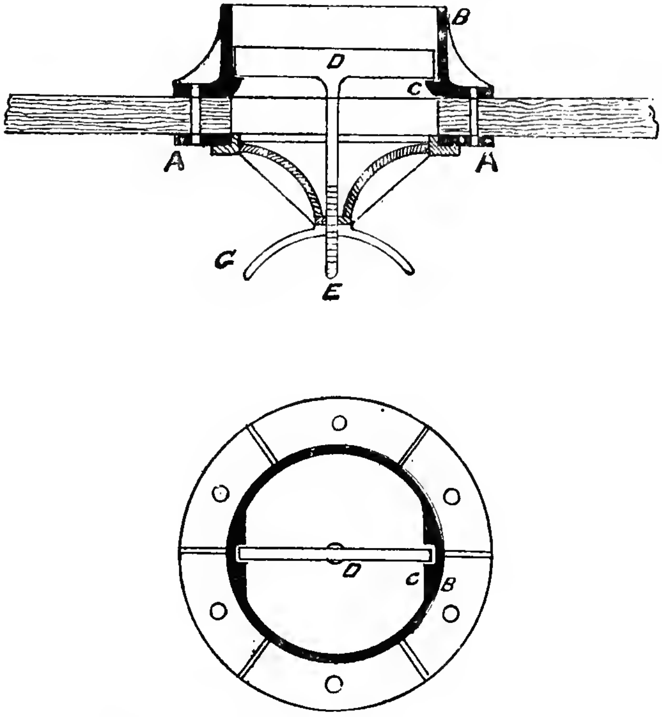
Figs. 146*f*, 146*g* illustrate this. On the bottom side of the tank a cast-iron ring, A, is bolted to the cast-iron cylinder, B, inside the tank. Inside the cylinder is the projecting lug, C, upon which rests the hanger, D, which forms part of the screw, E; the cast-iron cover, when placed in position, is simply fastened by the nut, G, and, screwing the same firmly, the whole arrangement becomes water-tight. The faces of the ring and the cover should be planed, so as to make a good joint. There are other methods of closing the discharge openings. When a tank is to be filled, a clay luting is given inside the iron cylinder, and then the same is rammed full with tailings. When filling the tank with tailings, especially into deep vats, a length of 3 to 4 ft. pipe is put over the discharge holes, and then the tailings are dumped in. It will be easily understood that in discharging a deep tank it facilitates the running of the tailings into an outlet if the same is within a few feet of the surface,

---

* "Notes on Gold Extraction." Argus Printing and Publishing Company, Johannesburg, 1894.

instead of having to push them down 13 or 14 ft. by means of long poles.

The cocks and valves should be of iron.



Figs. 146*f*, 146*g*.—Butters' Discharge Lid (scale 1 in. = 1 ft.).

**Pumps.**—Several varieties of pumps are used to raise the solution from the sumps to the lixiviation tanks, and to provide circulation if needed. Centrifugal pumps are mostly used on the Witwatersrand fields.

**Stock Solution Tanks.**—There are generally three solution tanks at each plant, built very much the same as the leaching tanks, with the exception that they have no filters, man-holes, &c. They are of different capacity, according to the size of the works, and are required to be of sufficient dimensions to store enough solution to keep the works going, without having to run any to waste.

Inside the tanks are gauges indicating the volume of solution. The stock solution tanks are usually 20 ft. in diameter, and from 7 to 14 ft. in height. One is for strong, one for weak solution, and one for alkaline wash. Every foot in height in a 20-ft. tank represents 10 tons of solution of 2,000 lbs. per ton.

To calculate the cubic contents of a circular tank, the following formula is employed. Multiply the square of the radius ($10^2$) with 3·14, and the product by the height of tank (6 ft.).

$$10^2 \times 3{\cdot}14 \times 6 = 1884 \text{ cubic feet.}$$

1 cubic foot of water weighs 62·3 lbs. Therefore,

$$1884 \times 62{\cdot}3 = 117573 : 2000 = 58{\cdot}78 \text{ tons of water.}$$

If we desire to prepare a 0·3 per cent. stock solution, $\frac{117573{\cdot}2 \times 0{\cdot}3}{100} = 352{\cdot}71$ lbs. of cyanide will have to be dissolved in it, making allowance for any impurity in the cyanide. After treatment we find that the solution analyses only 0·16 per cent. Consequently, by multiplying $\frac{117573{\cdot}2 \times 0{\cdot}16}{100} = 188{\cdot}11$ lbs. is left in the solution; and to make the same up to 0·3 per cent., another 164·61 lbs. of cyanide will have to be added.

**Zinc Precipitation Boxes.**—These are made of 1 to 2 in. boards, and are oblong boxes of various dimensions, which have to be in proportion to the quantity of solution which passes through them. In large works, the boxes are 20 ft. or more in length, 3 ft. high, and 3 to 4 ft. wide. There are separate boxes for the strong and for the weak solutions to pass through. At most works there are four of these boxes, placed in what

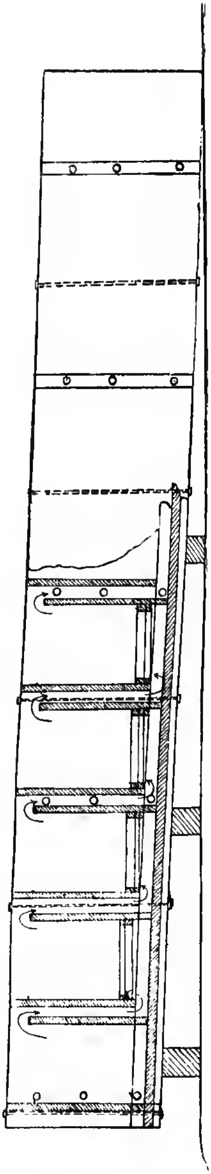


Fig. 146*h*.—Zinc Precipitation Box.

is called the extractor-house, which also contains the machinery, pumps, furnaces, &c.

The precipitation-box is divided into several compartments by partitions and baffle boards, in such a way that the solution is forced to flow upward through the zinc shavings, which are held in trays several inches above the bottom of the troughs. Fig. 146*h* shows the construction of the troughs.

The first division has not got any zinc shavings in the same, as here the solution enters, and any sediment or fine slime which may have passed through the filter settles here. If any intermediary settling tanks are used, as at the Worcester works, this first compartment can be utilised also to hold zinc shavings. From the first compartment the solution flows over the partition, and then down the space, and upward through the tray holding the zinc shavings. The baffle-board is held in place in the position shown in the drawing by being nailed fast to the sides, and reaches a few inches above the level of the solution. From this explanation it becomes clear that the solution has to pass its downward and upward course till the last partition is reached, and from here passes through a pipe to the collecting sump or tank.

The zinc box compartments are fitted with removable trays, made of wooden frames supporting wire screen of $\frac{1}{8}$-in. mesh. The gold in the solution settles on the zinc as a brown coating, and which soon, as it accumulates in a finely powdered state, falls through the screens to the bottom of the trough. In the last partition of each box there is no zinc, but the tray here is utilised to hold cyanide of potassium in lumps to make up its standard strength before pumping the same into the storage tanks.

Over the zinc there is placed a light wooden grating, and the whole trough can be covered by a strong wire netting to secure against theft, as the same can be kept under lock and key.

At some works a wooden launder, covered with a lid and also under lock and key, is attached longitudinally to the box, and from each compartment in the trough when a clean up is made, a plug is withdrawn and the slimes which have accumulated in the bottom are washed through the launder on to a filter and collected. In most works, the clean-up is made differently, as described later on. The zinc trays rest on cleats, several inches above the bottom, and have handles on the sides so that they can be easily lifted out when cleaning up. After passing through the precipitation-boxes the solution is pumped back to the storage tanks, and is used continuously and not run to waste. The dissolved zinc does not accumulate in the stock solution to a great degree, and is probably precipitated in the lixiviation tanks with the charges of fresh ore.

# CHAPTER XIV.

## *THE CYANIDE PROCESS IN OPERATION AT VARIOUS WORKS.*

CROWN REEF WORKS—Simmer and Jack Works—Langlaagte Estate—Rand Central Ore Reduction Company—Dry Crushing.

**Crown Reef Works.**—At these works[*] the tailings undergo the following treatment by the direct process.

The 120 stamp battery crushes on an average 17,000 tons of ore monthly. Of this quantity, 12,000 tons of tailings go to the cyanide works, which assay from 4 to 4½ dwts. The slimes, amounting to 5,000 tons, assaying 3¼ dwts., go to the reservoirs.

The tailings from the battery pass through 3 spitzlutten, one placed in front of the other, and here a classification of the tailings is effected. From the spitzlutten the stream is divided and passed to the separators or spitzkasten, of which there are four, placed in front of the leaching-tanks. At the bottom is a T-piece, with a nozzle screwed on either side, and a rubber hose, the discharge of which can be regulated by a clamp, so that the stream can be diverted into one or the other of the tanks. Mr. Williams claims that he eliminates more slimes by this system of direct filling, and that the tailings which he retains are richer. He claims that 70 tons of his tailings contain more gold than 75 tons of tailings saved by the intermediary tanks, and that is the reason why he discarded them. According to these figures he saves the treatment of 855 tons of tailings every month.

From the spitzkasten the tailings discharge into six large

---

[*] For the information in regard to these works I am indebted to Mr. Williams, the chemist in charge.

cement filter-tanks, each 40 ft. from front to back, 34 ft. wide, and 10 ft. high, measured inside the tank. At the bottom the tanks are narrower, as the inside is built of sloping walls. The side walls are 4 ft. thick at the bottom and 18 in. at the top. Their capacity when full is about 500 tons of tailings.

The sides and walls are of brickwork set in hydraulic lime and plastered with cement plaster. The bottom is of cement concrete at least 12 in. thick. The concrete is made up of three parts of treated tailings sifted, five parts broken stone down to 2 in. cube, and one part of Portland cement. It is mixed, put in place, rammed, and levelled up with a grout mixed two of sand and one of cement. The bottom of the cement-tank slopes to a central gutter, having a 2-in, fall from the front and back to the centre, where a gutter, 4½ in. wide, starts at the division walls of the tank with a fall of 4 in. towards the centre, where is a small pit 15 × 15 in. and 12 in. deep, formed in the cement bottom, connected with a 4-in. pipe bedded in the concrete, and through which the 2¼-in. solution pipe is threaded.

The filter bottom averages 5 to 6 in. in thickness, and is made by filling in with broken stone a 3 by 3 in. wooden combing, laid in cement mortar and bolted to the bottom of the tank all round. Over this is laid the filter-cloth.

Each tank is provided with three iron gates, through which the residues are trammed to the dump. Each gate is suspended by a vertical pulley from a horizontal axle running on wheels, which roll to and fro on rails on the top of the wall.

A cast-iron frame is built into the brickwork round the doorway, and the door made water-tight by wedging against rubber packing. Inside the filter-tanks are laid three 18-in. tramway tracks on longitudinal sleepers, which remain in the filter, and are only removed when the filter-cloth is to be raised. In the doorway is fitted a short piece of rail composed of 2 in. square iron, provided with pins projecting from its lower surface, which pins are dropped into holes in iron plates fitted on the sills inside and outside the doorway. This piece of rail is removed when it is necessary to open the door. To still further reduce the quantity of slimes treated, there are slime gates in the tanks. These are between the doors of the tanks, and are simply a series of wooden slats about 4 in. wide, sliding in a vertical groove 2¾ in. from the wall of the tank. During the filling of the tank with tailings, any slimes which remain in suspension flow over into this sluice-gate, which is raised by additional slats as the tanks fill higher and higher, and pass away by a 4-in. drain-pipe to the reservoir.

Even distribution of the tailings of the tanks is effected by constantly moving the nozzle of the charging hose.

Any leakage of cyanide from the doors of the tanks is provided for by a gutter 6 in. wide being formed in the concrete, outside the wall, and about 12 in. from it. This, starting at one end of the row of tanks, falls about 12 in. in its total length, and terminates in a small brick water-tight tank, 2 ft. square by 2½ ft. deep, from which a pipe leads to the extractor-house. The solution drainage pipes are 2½ in. in diameter, and, until outside the tanks, are threaded through a 4-in. pipe.

The tailings are under treatment for 102 hours. Each tank of 500 tons gets about 400 tons of solutions and washes.

The strong solution contains 0·3 per cent. K Cy.
The weaker solution contains 0·15 per cent. K Cy.
The weakest solution contains 0·05 per cent. K Cy.

The final wash water amounts to 25 tons.

The treatment in the leaching-vats is as follows:—It takes 30 hours to fill one of these vats. No. 3 vat, for instance, holds 550 tons of tailings. Say at 12.30 on the 29th August, 50 tons of 0·05 per cent. of cyanide solution is run on to drive out the water. When the solution is run through 75 tons of strong solution, 0·3 per cent. is put on. On the 30th August, 75 more tons of 0·3 per cent. is put on. On the 31st August, 65 tons of 0·15 per cent. is put on. From the 1st to the 3rd September, 300 tons of 0·05 per cent. solution, inclusive of 25 tons of wash water, are put on. Altogether the treatment takes 6 days, the total quantity of solution employed being 565 tons for this particular tank, in which the tailings assayed 5½ dwts.

and the residues 1·1 dwts. An extraction of 80 per cent. was obtained, which is rather above the average.

The quantity of tailings treated in each tank is tallied by counting the number of trucks which are discharged. From the leaching-vats the solution runs to the extractor-house, where there are three large stock solution-tanks excavated in the ground, brick-lined and cemented, and of the same dimensions as the lixiviation tanks.

Into No. 1 tank are run all solutions up to 0·08 per cent. KCy, which makes up an average of 0·05 per cent. with the weaker solutions.

No. 2 tank receives all solutions from 0·08 to 0·2 per cent. KCy, making up an average of 0·15 per cent. KCy.

No. 3 tank receives everything from 0·2 per cent. upward, and is made up to 0·3 per cent. by adding extra strong solution from the dissolving tank, and the same is agitated by means of a circulating pump.

There is also a vacuum well, consisting of two cylindrical iron receivers connected to an air-pump, in order to create a vacuum in case the filter is stopped up, or when it is necessary to assist percolation, or when it is necessary to dry the ore before discharging.

From the stock solution-tanks centrifugal pumps take the solution either from one or the other, and pump it into the filter-tanks. Before the solution passes into the extractor-boxes, the same flows into three small wooden tanks, where any impurities settle.

At this mine 50 to 60 per cent. of the gold leaving the battery is obtained in the cyanide works. The consumption of cyanide is claimed to be ½ lb. per ton treated and 4¼ oz. zinc per ounce of gold won. The works produced 2,914 oz. gold in August, 1894. The works are nicely planned and splendidly constructed, and the mechanical details have been, no doubt, under the supervision of a skilled man, and may be considered as one of the fine plants on the gold-fields, but they gave me the impression that their cost of erection must be far in excess of those plants which use wooden tanks.

**Simmer and Jack Works.**—When these works were erected the Company had a large accumulation of tailings, the results of five or six years' working, crushing first with 50 and afterwards with 100 stamps.

A contract was entered for the treatment of 100,000 tons with the Rand Central Ore Reduction Company, and under the direction of Mr. Butters, one of the finest and best plants on the gold fields was erected within three months. While in Johannesburg, I made this plant the special object of my study, and I am indebted to Messrs. Torrente and Smart, the chemists in charge, for some of the working details here given, as well as for the illustrations of the works given in Plates XVI., XVII., XVIII.

The plant has a capacity to treat 600 tons tailings daily, and consists of five vats of 600 tons capacity each; three stock solution storage-tanks of 300 tons capacity each; building containing four precipitation-boxes; three 2-in. centrifugal pumps, for returning the solutions from the precipitation-boxes to the storage-tanks; one 4-in. centrifugal pump, for pumping the solutions from the storage-vats to the leaching-vats; two lathes for cutting the zinc shavings; engine to work lathes and pumps and a 60 horse-power boiler to supply steam to engine, and two hauling gears.

There is stabling, also, for 40 mules; besides a large compound to house 200 Kaffirs, a manager's house, and 6 rooms for the white employés.

The tailing vats are 42 ft. in diameter and 14 ft. high, firmly bound with iron hoops 1 in. thick. It takes 10 to 11 hours to fill each vat.

The zinc boxes, four in number, are placed in a building which also encloses the stores, office, engine, lathes, and boiler. They are 24 ft. long, 3 ft. 6 in. broad, 2 ft. 6 in. deep. They are divided in 13 compartments. In every compartment there is a tray with a sieve of 20 to 25 holes to the inch at the bottom, on which the zinc shavings are loosely placed till they reach within 6 in. of the top of the boxes. The trays reach within 4 in. of the bottom of the boxes, this space being left to collect the slimes resulting from the precipitation of the gold on the zinc. The compartments are divided from each other by two boards, the one reaching to the bottom of the box, and the other within 3½ in. This ensures the greatest amount of contact of the solution with zinc, and therefore an almost perfect precipitation.

A 4-in. centrifugal pump is used to run the solutions from the storages into the tailing vats, and three small 2 in. centrifugals, return the solutions from the zinc boxes into their respective storages. A donkey pump is used for cleaning the zinc boxes, and three small tanks, called the clean-up tanks, which are used to settle the zinc slimes.

**Working of the Plant.**—I shall here restrict myself to the actual manipulation of the solutions. The process as carried out at the Simmer and Jack works has to deal (1) with old tailings containing acid; (2) with the tailings coming daily down from the mill, where they keep a certain number of stamps working on free-milling ore (oxidized), and the remainder on pyritic material (blue rock).

The quicklime is ground in a ball mill, constructed by the Gruson works, in Magdeburg, and in every truck of tailings a certain quantity is put, in proportion to the amount of the free acid therein contained.

A good practical way of ascertaining this is to take a given quantity of tailings, say 1 kilo., and place them in an enamelled bucket or basin, adding two or three times their weight in water. Leave them there for two or three hours, occasionally stirring the same, and the blue litmus paper will show if any acid is present. Given the presence of acid: take a weighed quantity of quicklime, finely ground (say 50 grms.), and with a spoon throw a small quantity in the vessel containing the tailings, stirring the same all the time, and keep on adding lime, little by little, until the red litmus paper turns slightly blue, when the acid will be neutralized. By weighing again the remaining lime, the quantity used will be known, and if, say 2 grms. has been used, we shall have that—

Missing Page

Missing Page

If 1 Kg. takes 2 grms. of lime—
1,000 Kgs. (1 ton), will require 1,000 times more, or
2,000 grams, which is equal to 2 Kgs.

Now, if every truck carries, say, ½ ton of tailings, it will require 1 kilo. of lime, or 2·12 English lbs. The advantage of supplying every truck with its proper quantity of lime, is to ensure its even distribution through the mass in the vat.

When the vat is full, it is carefully levelled, and then is ready to be treated.

In the case when lime has been used, the tank is filled with water and allowed to stand full for one hour. By this time the acid has been neutralized, and it can be started leaching. The water from the leaching carries, as a rule, a small excess of alkali, and it is pumped back into one of the storages, to be used again in the next tank. To this water the name of alkaline solution, or caustic wash, is given; and, as a rule, the moisture in the tailings is enough to keep the amount in hand constant, so that very seldom is it necessary to add any more water to it. It is necessary to keep the solution in a storage tank, so as to avoid any loss of gold, as it always carries with it whatever gold may have been left in the filters, &c., from previous treatment. It will also be found that, after treating a few tanks, the solution will contain a small percentage of cyanide.

When the caustic wash is partly drained off, the top of the tailings are turned over with a shovel, so as to break the slimy film which forms; and then follows the solution, whose quantity must be so regulated as to give about ¾ lb. of cyanide (or less) per ton of tailings, and be of a strength of ·25 to ·3 per cent.

This solution is followed by another, which is called weak solution, and whose strength varies between ·08 and ·15 per cent. of cyanide, and is kept on until the tank is ready to leach dry.

Should the solution in the storages get short, a water wash is given to terminate the operation: otherwise the weak solution is used to the last.

The system followed in pumping up the solutions is to keep the tanks always well covered up from the moment the strong solution is first pumped up. As soon as the tailings show, more solution is added until the full amount has been pumped up. This is followed at once with a weak solution, which, in its turn, must keep the tailings always well covered till the tank is ready to water wash or leach dry.

The time employed in these diverse operations is about as follows:—

| | |
|---|---|
| 1st. *Caustic Wash* (*water or otherwise*). | |
| Pumping | 3 hours |
| Contact | 1 hour |
| Leaching | 8 hours |
| 2nd. *Strong Solution.* | |
| Is pumped without stopping the leaching until all is on, takes about | 8 hours |
| (Amount used about 160 tons.) | |
| 3rd. *Weak Solution.* | |
| Is kept up circulating as previously explained about | 40 hours |
| 4th. *Water Wash* (*if necessary*). | |
| Leach dry | 24 hours |
| Time it took to load tank | 12 ,, |
| Total from the time of starting filling till ready to discharge | 96 hours |

(Total amount of strong and weak solution between 500 and 600 tons.)

**Precipitation of the Gold from the Solution.**—From the bottom of the tailing tanks, and under the filter, a pipe 2 in. in diameter carries the solution to the zinc boxes, four in number, as already mentioned.

The object of these four zinc boxes is to return the solutions to their respective storages, for which purpose they are connected:—

2 with the strong solution storage,
1 with the weak solution storage,
1 with the caustic wash storage.

A simple assay will show to the man in charge when to lead the leachings through the one or through the other box. The boxes have a fall of 9 in. in their full length.

**Recovery of the Gold from the Gold Slimes.**—The gold slimes are settled in the slime-tank by means of alum or sodium sulphide. After 12 hours the clear portion is syphoned down, and these slimes are then ready for further treatment. The test of the sodium sulphide is lead paper, which takes a brownish colour when the residues in the tank have been saturated.

To make sodium sulphide:—

6¼ gals. water,
62½ lbs. caustic soda,
41 lbs. sulphur (broken small).

Boil the water with the soda till dissolved, and then add the sulphur slowly, as it is apt to boil over. Dilute to 40 gals. Smaller quantities in proportion.

The slimes are transferred to a special calcining furnace, where they are dried and the zinc slightly oxidized. They are left to roast at a dull red heat for 4 hours, although this time depends upon the quantity and nature of the slimes. After roasting, they are put into iron dishes and mixed with the smelting fluxes.

To 6 shovels of slimes add—

½ shovel fluar spar,
3 shovels borax,
1 shovel carbonate of soda.

If the slimes are hard, and cake, they must be ground fine, so as to mix them well with the fluxes.

If the fire gets too fierce while smelting the slimes, and the pots overflow, keep the cover off, and throw a little borax on the top. This overflowing is objectionable, as the slag cakes at the bottom of the grate and chokes the fires. When all the gold belonging to one lot of slimes has been secured, it is then melted down into one bar.

**Langlaagte Estate.**—At these works, the pulp on leaving the mill passes into three settling-dams in rotation. The lower retaining walls are composed of sand-bags, and two doors are provided in each dam, one for the passage of the full, and the other for the empty trucks.

The discharge of the slimes, when filling the dams, is regulated by strips of wood in the passages, the same system being employed at the Randfontein Estate. Lines of rails are placed inside these dams; and at right-angles to these, immediately inside the lower walls, runs a line of rails sunk in the ground, on which is placed a travelling-carriage, on to which the tipping trucks are run and brought outside the dams, to be picked up by a mechanical haulage composed of an endless wire rope, and carried to the cyanide works.

Here are ten vats sunk in the ground, five on each side, and when the tailings are treated they are emptied by means of a travelling crane, which lowers the body of 25 cubic feet capacity trucks into the vats, to be filled by Kaffirs, and are then lifted out again and placed on their respective carriage, and conveyed by mechanical haulage to the waste dumps. A vat can be emptied in six hours.

The mechanical haulage is operated by a 25 n.h.p. compound engine, with 24-in. stroke and 25 h.p. boiler, consuming about 1¾ tons coal per day.

**Rand Central Ore Reduction Company.**—A prominent position in the treatment of tailings and concentrates is occupied by the above company, which was organised by Mr. Ad. Goerz, mining engineer, representing a powerful German syndicate. The company, who own the Siemens and Halske patents for South Africa, have bought over 1,000,000 tons of tailings, of an average value of 4½ dwts., and 6,173 tons of concentrates, of an average assay and contents of about 4 oz. The technical work is directed by Mr. Charles Butters.

The establishments of the company are six in number, and include (1) their Central works (shown in Plate XIX.)—consisting of a chlorination plant and cyanide works, with workshops and foundry, where everything is made for vats, implements, etc., required for the construction of cyanide plants; (2) works at Maraisburg, which were built to treat tailings from the Main Reef, Aurora, and Aurora West Companies; (3) the Simmer and Jack works, already described; (4) the Worcester works, already described; (5) the No. 1 works, which treat tailings from the Pioneer Company; and (6) the No. 2 works, which treat tailings from the United Langlaagte.

During the year ending June 30th, 1894, the company treated at their six works 311,561 tons of tailings, yielding 52,434·46 oz., of a value of £209,737 16s. 8d.

**Dry Crushing.**—As proper percolation of the solution can only be obtained by having the pulp coarse enough to allow of easy leaching, several establishments in the United States have adopted dry crushing.

Mr. Philip Argall, M.R.I.A. (see *Mining Journal*, London, October 13th, 1894), says: "The best results hitherto obtained on rather friable oxidized siliceous ores have been produced by the following series of machines:—

"(*a*) An ordinary Blake crusher, which reduces the ore to about 1½ in.;

"(*b*) A three-jaw multiple Blake crusher, which brings the ore down to ¾ in.; followed by

"(*c*) A five-jaw multiple, that reduces it to about 6-mesh; and

"(*d*) Rolls that finish the pulp to pass a 30-mesh screen.

"The ore is screened after each crushing, so that the portion reduced to the desired fineness passes direct to the finished ore-bin, instead of, as is too often the case, going to the next machine to be reduced to a further and unnecessary degree of fineness. This plant, crushing to 30-mesh, produces only from 5 to 10 per cent. fine enough to pass 200-mesh.

"With soft, clayey ore, from 5 to 15 per cent. of the finest dust has to be removed before a leachable product is obtained."

The preparation of the ore, in dry as well as in wet crushing, is a very important step in the direct treatment by the cyanide process, and the latter has only been brought to its present state of perfection within the last six months. Owing to the scarcity of water in West Australia, I believe that many mines will have to adopt dry crushing plants, and that the system will reach there a high degree of perfection.

Where wet crushing is resorted to, owing to the large quantity of ore crushed per stamp a large amount of water is necessarily used, and consequently an appreciable quantity of finely-divided gold is carried away in the rush of water.

Mr. Crosse has shown (in a lecture before the Chemical and Metallurgical Society of Johannesburg), that what takes place in wet crushing also takes place in dry crushing. He says:—"I received about three-quarters of a ton of ore from the Roodepart Deep Level Gold Mining Company, which was crushed in a Grusonwerk ball mill; the sieving used had 500 holes to the square inch. It assayed per short ton: Fine gold, 5 ozs. 12 dwts.; fine silver, 16 dwts.; and contained 3·31 per cent. of pyrites, determined as bisulphide of iron.

| | | | | |
|---|---|---|---|---|
| There remained on | A + 1,600 mesh | — 28·35 | per cent. | of the ore. |
| ,, ,, | B + 3,600 ,, | — 22·50 | ,, | ,, |
| ,, ,, | C + 6,400 ,, | — 14·15 | ,, | ,, |
| ,, passed through | D — 6,400 ,, | — 35 | ,, | ,, |

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| A contained | 2·01 | per cent. pyrites, | and assayed | 4 ozs. | 1 dwt. | 16 grs. |
| B ,, | 2·96 | ,, | ,, ,, | 4 ,, | 14 dwts. | 12 ,, |
| C ,, | 4·01 | ,, | ,, ,, | 5 ,, | 19 ,, | 0 ,, |
| D ,, | 4·64 | ,, | ,, ,, | 6 ,, | 16 ,, | 12 ,, |

| | | | | | Proportion of gold to pyrites. |
|---|---|---|---|---|---|
| A contained | 22·03 | p.c. of the gold and | 15·51 | p.c. of the pyrites | 1·143 |
| B ,, | 20·28 | ,, | 18·21 | ,, | 1·182 |
| C ,, | 25·03 | ,, | 17·14 | ,, | 1·196 |
| D ,, | 42·66 | ,, | 49·07 | ,, | 1·196 |

"This table shows that, though the crushed ore was passed through a coarse sieve, the gold bearing portions were more

finely crushed, and that 35 per cent. of the crushed ore passed through a sieve having 6,400 holes per square inch, and this portion contained 42·66 per cent. of the gold and 49·07 per cent. of the pyrites. A milling assay gave 36·6 per cent. of free gold. The sample came from a depth of 700 feet. The ore leached easily.

"The results were as follows:—Total gold in the ore treated 3·509 ozs.; left in residues, 0·446 ozs.; extraction, 88·7 per cent. A cubic foot of the dry crushed ore, weighed 103 lbs. One ton would occupy 19·4 cubic feet."

Mr. Crosse suggests that the best method for dealing with banket would be, to arrange a series of Gates crushers, one Krom roll, and then a short treatment in a ball mill, with a 50 to 60 mesh screen to grind up the pyrites and cement, and so liberate the finely-divided gold.

# CHAPTER XV.

## *THE CHEMISTRY OF THE CYANIDE PROCESS.*[*]

SOLUTION OF THE GOLD—Solubility of other Metals—Louis Janin's Experiment on Silver Ores—Results of Experiment made with Potassium Cyanide on Gold and Silver Ores—Treatment of Pyritic Ore previous to Lixiviation with Cyanide—Treatment of Pyritic Ores by Cyanide of Potassium—Precipitation of the Gold—Decomposition of the Cyanide—Testing the Strength of Solutions—Determination of Gold in Cyanide Solutions—Extraction Tests.

**Solution of the Gold.**—The solubility of gold in a solution of cyanide of potassium has long been known. The presence of oxygen is necessary for the reaction, after the formula:

$$2Au + 4KCy + O + H_2O = 2KAuCy_2 + 2KHO.$$

That is, a double cyanide of gold and potassium is formed. This was proved by the formation, upon evaporating the solution, of octahedral crystals answering to the formula. From the solution the gold is precipitated by filiform zinc. Based on these two reactions, the MacArthur-Forrest Company have obtained the following patent:—

"The invention consists in subjecting the auriferous or argentiferous ores to the action of a solution containing a small quantity of cyanide, as hereinafter set forth, without any other chemically active agent, such quantity of cyanide being reckoned according to its cyanogen, and the cyanogen being proportioned to the quantity of gold or silver, estimated by

---

[*] The chemistry of the process has been described by Messrs. Butters and Clennell in the *Engineering and Mining Journal* (New York) of October 22nd and 29th, 1892, and from their articles (with their permission) the substance of this chapter is extracted.

assay or otherwise to be in the ores under treatment. By treating the ores with the dilute and simple solution of a cyanide, the gold or silver is, or the gold and silver are, obtained in solution, while any base metals in the ores are left undissolved, except to a practically inappreciable extent; whereas when a cyanide is used in combination with an electric current, or in conjunction with another active chemical agent—such as carbonate of ammonium, or chloride of sodium, or phosphoric acid—or when the solution contains too much cyanide, not only is there a greater expenditure of chemicals in the first instance, but the base metals are dissolved to a large extent along with the gold or silver, and for their subsequent separation involve extra expense, which is saved by our process.

"In carrying out our invention practically, we take the ore in a powdered state and mix with it the solution of cyanide in a vessel made of, or lined with, any material not appreciably acted on by the solution. . . . We regulate the quantity of cyanide so that its cyanogen will be in proportion to the quantity of gold or silver in the charge of ore; but in all cases we dissolve it in sufficient water to keep the solution extremely dilute, because it is when the solution is dilute that it has a selective action, such as to dissolve the gold or silver in preference to the baser metals.

"In dealing with ores containing 20 oz. or less of gold or silver, or gold and silver, per ton, we find it most advantageous to use a quantity of cyanide, the cyanogen of which is equal in weight to from 1 to 4 parts for every 1,000 parts of the ore, and we dissolve the cyanide in a quantity of water of about half the weight of the ore. In the case of richer ores, while increasing the quantity of cyanide to suit the greater quantity of gold or silver, we also increase the quantity of water so as to keep the solution dilute. In other words, the cyanide solution should contain from 2 to 8 parts, by weight, of cyanogen to 1,000 parts of water, and the quantity of the solution used should be determined by the richness of the ore. After the solution has been decanted or separated from the undissolved residues, the gold and silver may be obtained from it in any convenient way, such as evaporating the solution to dryness and fusing the resulting saline residue, or by treating the solution with sodium amalgam.

"Having fully described our invention, what we desire to claim and secure by letters patent is: The process of separating precious metals from ore containing base metal, which process consists in subjecting the powdered ore to the action of a cyanide solution containing cyanogen in the proportion not exceeding 8 parts of cyanogen to 1,000 parts of water."

Some time after this patent was issued, patents covering the use of zinc, preferably filiform, or threadlike, for a precipitating agent, and the use of caustic alkalies for neutralizing ores containing acids, or acid salts, were granted to Messrs. MacArthur and Forrest. It will be seen, therefore, that their patents cover substantially three points: namely, the use of dilute solutions of cyanide (not more than 8 parts of cyanogen to 1,000 parts of water); the use of zinc, preferably filiform, as a precipitant; and the employment of caustic alkalies for neutralizing acid ores.

Mr. Louis Janin claims,* that the necessity of oxygen in the reaction which takes place in the solution of the gold in the cyanide has not been proved, and establishes the following formula:—

$$Au + 2KCy + H_2O = KAuCy_2 + KHO + H.$$

More recent investigation has proved that oxygenation of the gold in the process is essential to its success, and the idea has been suggested that an artificial oxygenation of the gold in rebellious ores may lead to the solution of the problem of treating them successfully by the cyanide process.

Mr. R. Feldtmann, in his "Notes on Gold Extraction," gives the following additional formulæ, throwing light on the reactions which take place in the solutions of gold in cyanide:—

When hydrocyanic acid dissolves gold from acid tailings (assuming that the cyanicide has destroyed all the cyanide of potassium),

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

---

* *Mineral Industry*, 1892. Scientific Publishing Co., New York.

an auric hydrocyanide is produced, and this compound is not, or is at any rate only 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 auri cyanic acid may be assumed to form auric potassic cyanide in this way :—

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

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 auri cyanic acid. Probably the hydrocyanic acid, liberated by the decomposition of the potassic cyanide, combines with the auro potassic cyanide to form auri potassic cyanide :—

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

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

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

**Solubility of other Metals and Minerals.**—According to Gmelin, zinc, iron, nickel, and copper are dissolved by potassium cyanide, with evolution of hydrogen; cadmium and silver in the presence of oxygen; and tin, mercury, and platinum not at all. Sulphide of silver is dissolved by strong solutions and a sufficient quantity of weak solution. Silver arsenate $Ag_3 As O_4$, and silver antimoniate $AgSbO_3$ are readily dissolved by potassium cyanide, as are many of the argentiferous arsenical and antimonial minerals found in nature. Chloride of silver dissolves readily, forming chloride of the alkali and a double cyanide of silver and potassium. While metallic silver, when sufficiently fine, dissolves readily in the solution, that found native in ores is not attacked, unless existing in thin laminæ. The oxides and sulphides of copper are attacked by the solution and dissolved, as is metallic copper.

It is claimed that the presence of copper sulphide in a silver or gold ore prevents the precious metals from going into solution. Although experiments have shown that little or no silver or gold is dissolved in certain ores containing sulphide of copper, this question is by no means settled, as artificially prepared sulphide of silver is dissolved in actual contact with the copper compounds. Metallic iron is attacked, but very slowly. Ferric hydrate is not attacked by the solution, but ferrous hydrate, formed in the neutralization of the iron salts by alkali, is attacked by cyanide, according to the reaction :—

$$Fe(HO)_2 + 6KCy = K_4FeCy_6 + 2KOH.$$

Thus ferrocyanide of potassium and caustic potash is formed.

**Louis Janin's Experiments on Silver Ores.***—Mr. Janin made a series of experiments on the applicability of the cyanide process to various types of ores, with the following results :—

*Sample No.* 1.—Grand Central Mine, Arizona. Siliceous ores, containing considerable quantities of lime and manganese. The silver minerals are principally cerargyrite and argentite, and gave an extraction of 92·6 per cent.

*Sample No.* 2.—Christy Mine, Silver Reef, Utah. The silver minerals are chlorides, sulphides, metallic silver, in a gangue of sandstone, somewhat discoloured by carbonate of copper. Extraction, 80 per cent.

*Sample No.* 3.—Horn Silver Ore, Utah. The silver principally as chloride. Extraction, 93 per cent.

*Sample No.* 4.—Tybo, Nevada. The mineral is principally a sulphide and fahlore. Extraction, 71·8 per cent.

*Sample No.* 5.—Gangue (siliceous) with chloride of silver. Extraction, 97 per cent.

*Sample No.* 6.—Ramshorn, Idaho. Containing galena and carbonate of lead, pyrites, and zinc blende. Extraction, 80 per cent.

*Sample No.* 7.—Broken Hill, New South Wales. Chloro bro-

---

* From *Mineral Industry*. The Scientific Publishing Co., New York.

mide of silver in kaolin, quartz, and garnet. Extraction, 99·7 per cent.

*Sample No.* 8.—Broken Hill, N.S.W. A siliceous iron ore containing 38 per cent. Fe O. Extraction, 84·6 per cent.

*Sample No.* 9.—Bullionville, Nevada. Tailings, containing 10 per cent. carbonate of lead, galena, and iron in a siliceous gangue. Extraction, 32 per cent.

*Sample No.* 10.—Bertrand and Geddes, Nevada. Contains antimoniate of lead, with which the silver was combined. Extraction, 11·8 per cent.

*Sample No.* 11.—Argenta, Montana. Contains over 40 per cent. lead. Extraction, 5·7 per cent.

*Sample Nos.* 12 *and* 13.—Belmont, Nevada. Contains arsenical pyrites, pyrite, blende, and galena, with the silver as fahl ore, and arsenical and antimonial ruby forms. Extraction, 35 and 47·5 per cent.

*Sample No.* 14.—Las Yedras, Mexico. Contains large quantities of carbonate of lime, with the silver in the form of ruby silver, and arsenical pyrites. Extraction, 41·5 per cent.

*Sample Nos.* 15 *and* 16.—Ontario and Daly ores, Utah. Contain principally fahl ore, more or less decomposed with some lead and zinc. Extraction, 72·5 and 81·1 per cent.

The conclusions and deductions to be derived from a study of the foregoing are, that silver in oxidized surface ores, or where it occurs as a chloride, is readily attacked by cyanide of potassium, and that where no minerals are present which exert an unfavourable influence, this method may prove economical. It must be confessed, however, that even with these conditions it has but a limited range of usefulness. On the other hand, where lead, oxide of copper, or oxides of iron occur, the results are so poor as to preclude the use of the process.

The results obtained from different samples of silver ore from the same mine vary greatly, for a slight increase of an undesirable element, which would not affect amalgamation in the slightest degree, causes a great decrease in the percentage of extraction by cyanide.

In the case of a number of experiments on Daly ores, the results varied 16·2 per cent., while, with amalgamation, there was a variation of but 2 per cent.

**Results of Experiments made with Potassium Cyanide on Gold and Silver Ores.**—It would seem probable that, in ores containing both gold and silver, only the oxidized surface ores can be treated with success, both the silver and gold minerals from depth proving refractory. With the majority of these ores the consumption of cyanide would be large, as many minerals other than those of silver are contained in them, and would have a decomposing action upon the solution.

Experiments on Gregory concentrates, consisting of pyrites, some arsenopyrite and chalcopyrite, with traces of galena and blende, gave an extraction of 90 per cent. gold and only 3 per cent. silver.

Delamar, Idaho, consisting of a siliceous and limestone gangue, impregnated with silver sulphides and chlorides, and iron pyrites. Results: gold, 90 per cent.; silver, 83 per cent.

Revenue Mine, Montana—Oxidized surface ores, containing a considerable quantity of iron. Gold, 94 per cent.; silver, 5·2 per cent.

Southern Cross, Montana, contains 40 to 50 per cent. of limonite. Gold, 93 per cent.; silver, 50 per cent.

I am not aware, so far, that the cyanide treatment has been applied on a large scale to the extraction of silver ores.

**Treatment of Pyritic Ore Previous to Lixiviation with Cyanide.**—Pyritous ore, if but slightly exposed to atmospheric action, always contains free sulphuric acid and soluble salts of iron. To prevent reactions on the solution, it is necessary to leach these ores with water previous to lixiviation with cyanide of potassium, and before washing with an alkali. If the alkali solution were to be added directly to the ore, the consumption of alkali would be extremely large, and the amount of solution necessary, if lime were to be used, would prove inconvenient to handle.

The alkali solution, assuming that caustic soda is used, reacts on basic iron salts, insoluble in water, according to the following reactions:—

$$Fe_2O_3,SO_3 + 2NaHO + 2H_2O = Fe_2(HO)_6 + Na_2SO_4$$

and

$$Fe_2O_3,2SO_3 + 4NaHO + H_2O = Fe_2(HO)_6 + 2Na_2SO_4.$$

Thus ferric hydrate and sodium sulphate (or calcium sulphate, if lime is used) are formed. Sodium sulphate is soluble and passes off with the wash water, but calcium sulphate remains. The hydrate of the sesquioxide of iron is insoluble in water, and, to all appearances, is unattacked by the cyanide solution; but the hydrate of protoxide is dissolved with formation of ferrocyanide of potassium.

Mr. C. W. Merrill precipitated ferrous hydrate by caustic potash from a cyanide solution. The solution contained but a small percentage of free cyanide, however, as it had already acted on ore and zinc in the precipitation of the dissolved gold, and it is unlikely that this reaction, regenerating the cyanide of potassium, which had been rendered inert by the solution of the iron, would occur in a comparatively strong solution.

**Treatment of Pyritic Ores by Cyanide of Potassium.**—Owing to the rapid decomposition of the pyritic ores when exposed to the atmosphere, precautionary measures have to be taken to treat them effectively. Owing to their successful treatment by the chlorination process, it is at present difficult to say if the cyanide process will eventually prove a serious competitor, as no doubt the process which is the cheapest will maintain itself.

Pyrite ($Fe S_2$) is decomposed by the oxygen of the air and moisture into soluble ferrous sulphate and free monohydrated sulphuric acid, according to the reaction:

$$FeS_2 + H_2O + 7O = FeSO_4 + H_2SO_4.$$

The ferrous sulphate is decomposed by the action of the air to insoluble basic sulphates. In addition, normal ferric sulphate ($Fe_2 3 SO_4$) is produced, which gradually loses acid and becomes a soluble basic sulphate, $Fe_2 O_3, 2SO_3$. There are many basic salts of somewhat complex and doubtful composition formed likewise. Thus in an oxidized ore which has contained pyrite are found sulphuric acid, ferrous sulphate, basic ferrous sulphates, ferric sulphate, and basic ferric sulphates, all of which react upon potassium cyanide.

Sulphuric acid reacts upon potassium cyanide with evolution of hydrocyanic acid according to the reaction:

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

Ferrous sulphate reacts upon cyanide with the formation of ferrous cyanide, a yellowish red floculent precipitate:

$$FeSO_4 + 2KCy = FeCy_2 + K_2SO_4.$$

This ferrous cyanide is attacked by the excess of cyanide in the solution, and ferrocyanide of potassium is formed according to the reaction:

$$Fe\ Cy_2 + 4KCy = K_4FeCy_6.$$

That is to say, one molecule of ferrous sulphate decomposes or renders inert six molecules of cyanide of potassium. Other things being equal, if 1 per cent. or 20 lbs. of ferrous cyanide existed in the ore, some 51 lbs. of cyanide would be rendered inert for the solution of gold, and, in fact, would be lost. This, at the average price of chemically pure cyanide, would cost over £5 to £6 per ton of ore treated.

The ferrocyanide of potassium, formed according to the last reaction, is reacted upon, if sufficient acid be present, by an additional quantity of ferrous sulphate, with production of Prussian blue according to the reaction:

$$3K_4FeCy_6 + 6FeSO_4 + 3O = Fe_2O_3 + 6K_2SO_4 + Fe_7Cy_{18}.$$

This production of Prussian blue gives a blue colour to the surface of the tailings, or to the solution, and indicates at once that the washing and neutralizing operations have not been carried on properly, and that a great loss of cyanide is taking place.

Ferric salts, when present, unmixed with ferrous salts, decompose the cyanide solution with formation of hydrocyanic acid and precipitation of ferric hydrate, according to the reaction:

$$Fe_2(SO_4)_3 + 6KCy = Fe_2Cy_6 + 3K_2SO_4$$

With further decomposition:

$$Fe_2Cy_6 + 6H_2O = Fe_2(OH)_6 + 6HCy.$$

This means that, other things being equal, one molecule of ferric sulphate decomposes six molecules of cyanide. If 1 per cent., or 20 lbs., of ferric sulphate existed in the ore, very nearly the same weight of cyanide, costing £2 to £3, would be destroyed.

If a mixture of ferric and ferrous sulphate, as is probable, exists in partially-oxidized ores, it causes the production, when ferrous sulphate is in excess, of ferrous ferri-cyanide, according to the reaction:

$$12KCy + 3FeSO_4 + Fe_2(SO_4)_3 = Fe_3(FeCy_6)_2 + 6K_2SO_4$$

when ferric sulphate is in excess, the production of ferric ferrocyanide (Prussian blue), according to the reaction:—

$$18KCy + 3FeSO_4 + 2Fe_2(SO_4)_3 = Fe_4(FeCy_6)_3 + 9K_2SO_4.$$

These reactions show clearly that washing by water and neutralization by a caustic alkali must be employed to arrive at satisfactory and economical results. It is more than probable that many of the failures already recorded are due to the lack of these precautions. In addition to these reactions, there are many with unknown compounds, the composition of which cannot be expressed, even where the greatest precautions are used, and the operations supervised with the greatest ability and knowledge.

**Precipitation of the Gold.**—Zinc precipitates the dissolved gold, as the cyanide has more affinity for it than for the gold. Theoretical reaction is:

$$2KAuCy_2 + Zn = 2Au + K_2ZnCy_4.$$

But much more zinc goes into solution than this reaction calls for. According to the chemical reaction, the consumption should be 1 oz. of zinc to 6 oz. of gold. The excessive consumption of zinc must be ascribed to other action than the mere replacement of zinc for gold in the double cyanide of gold and potassium.

There is comparatively little exact knowledge of the reactions taking place in the zinc precipitation-boxes. One fact is known positively, and that is, that hydrogen is evolved. This does not occur, however, when zinc alone is exposed to a cyanide solution, but after gold is deposited on the zinc, or when zinc is placed in contact with iron. In other words, a galvanic couple is formed, the water is decomposed, and hydrate of zinc is formed, which is attacked by the cyanide forming a double cyanide of zinc and caustic potash. The probable reactions may be expressed as follows:

$$Zn + 2H_2O = 2H + Zn(HO)_2.$$
$$Zn(HO)_2 + 4KCy = ZnK_2Cy_4 + 2KHO.$$

The production of caustic alkali explains the increased alkalinity of the solution after passing the zinc precipitation-boxes. It may be considered advantageous to a certain extent, however, as carbonic acid, which decomposes the solution, is absorbed by the caustic potash, with formation of a carbonate of the alkalies. Ammonia is formed also, as is indicated by the strong odour of the gas about the boxes.

The precipitate contains, besides the precious metals, many of the base metals, which may be dissolved by the solution. The principal of these are copper, arsenic, and antimony. When a weak cyanide solution contains copper, 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 the gold is complete.

It is also asserted that, in ordinary cases, the accelerating influence on precipitation of excess of potassic cyanide is probably due to generation of nascent hydrogen.

$$4KCy + Zn + 2H_2O = ZnK_2Cy_4 + 2KHO + H_2.$$

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

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

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 the 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 (it is claimed *), 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 the 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 nitrate of silver solution.

Owing to the large amount of zinc which is dissolved in the precipitating-boxes, one would surmise that the working solutions would become in time very highly charged with zinc compounds. Mr. Feldtmann * remarks on this subject, that, as a matter of experience, it may be stated that they do 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.

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

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.

$$6KCy + FeS = K_4FeCy_6 + K_2S.$$

---

* "Notes on Gold Extraction by means of Cyanide of Potassium," by W. R. Feldtmann.

Mr. T. S. MacArthur has even found that, in very exceptional cases, sufficient alkaline sulphide may be formed to be of hindrance to the action of cyanide on the gold, and has discovered a remedy for this in the addition of metallic (particularly lead) salts, capable of forming insoluble sulphides.

**Decomposition of the Cyanide.**—The compound of cyanogen and potassium is extremely unstable. Not only is it decomposed by mineral acids and acid salts, but by the action at ordinary temperatures of atmospheric carbonic acid, according to the reaction:

$$2KCy + CO_2 + H_2O = K_2CO_3 + 2HCy.$$

Hydrocyanic acid is given off, a portion of which remains in solution and is available for the extraction of gold, but the greater part is dissipated into the air.

The cyanide is easily oxidized to cyanate:

$$KCy + O = KCyO.$$

The cyanate is further oxidized to carbonate according to the reaction:

$$2KCyO + 3O = K_2CO_3 + CO_2 + N_2.$$

The nitrogen given off may cause a still further decomposition, for when a current of nitrogen is passed through a cold dilute solution of cyanide of potassium, hydrocyanic acid is evolved without the nitrogen entering into the reaction. This action, when the presence of a chemical causes a reaction between other chemicals in aqueous solution without entering into the reaction itself, is called hydrolysis, and further reaction must be attributed to this property of caustic alkalies, which are, and must be, always present in a working solution of potassium cyanide. If the solution is boiled with acids or alkalies, hydrolysis of the cyanide occurs rapidly, ammonia and formic acid being formed thus:

$$KCN + 2OH_2 = NH_3 + HCO_2K\ (+ 9{\cdot}5).$$[*]

---

[*] According to T. K. Rose, B.Sc., "The Metallurgy of Gold," London, 1894.

If lime be in solution as cyanide of calcium, carbonate of lime is produced, consequent upon the formation of hydrogen during the precipitation of the gold. Ammonia is also produced in this reaction, which accounts for its presence at the zinc boxes. The carbonic acid evolved attacks the cyanide, as mentioned before, thus causing still greater decomposition of the solvent.

It will be seen, therefore, that the decomposition of the cyanide solution, and loss of the solvent energy of the solution for gold, may be divided under the following heads:—

1. Actual decomposition of the solution:

(*a*) By acids and acid salts present in the ore.
(*b*) By atmospheric carbonic acid.
(*c*) By oxidation.
(*d*) By reaction owing to hydrolysis.

2. The solution of other metals than gold which are not precipitated:—

(*a*) By metals or their compounds present in the ore, as oxides or carbonates of lead, which are first attacked by the caustic alkali; by oxides or carbonates of the alkali; by certain compounds of iron insoluble in water.

(*b*) By the replacement of gold in solution by zinc in the precipitation.

(*c*) By dissolving zinc hydrate, formed by electrolysis, in the precipitation.

It will thus be seen that these losses, under incompetent management, may be frequent, and in many cases may cause unprofitable results.

**Testing the Strength of Solutions.**—The method employed for estimating the amount of cyanide in a solution is based on the capacity of cyanide of potassium to form a double cyanide with silver—which is added to it in the shape of a titrated solution of nitrate of silver—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.

$$2KCy + AgNO_3 = AgKCy_2 + KNO_3.^*$$

As the combining weights of $AgNO_3$ and K.Cy. are 170 and 65·13 respectively, it follows that 170 parts by weight of argentic nitrate may be added to 2 by 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 grms. in a litre, or 1,000 cubic centimetres, to the solution of potassic cyanide to be tested, until a faint precipitate appears, each c.c. of silver solution added will correspond to $\frac{13{\cdot}02}{1000}$ = 0·013 grms. 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 grms. of pure silver nitrate in a litre of water, and 10 c.c. of the cyanide solution be taken for a test, then each c.c. of silver solution added will correspond to 0·1 per cent. of pure K.Cy. 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 c.c., diluting with water to 100 c.c., and then drawing off 10 c.c. for test. Of course, in such a case, 1 c.c. of standard silver solution will indicate 1 per cent. of K.Cy. 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 our estimating the quantity of cyanide present, consequent on the strong alkalinity of the solution.

In estimating very dilute solutions, such as are employed, for instance, in the Siemens-Halske process, I would recom-

---

[*] From Mr. Feldtmann, "Notes on Gold Extraction."

mend the employment of a standard decimal silver solution, as the same would ensure greater accuracy.

By taking 100 c.c. of the ordinary standard solution, diluting the same to 1,000 c.c., 1 c.c. of this decimal solution would be equal to 0·01 per cent. of KCy. The strength of the standard nitrate solution should be occasionally controlled by a standard cyanide solution.

In making the analysis, the solutions are filtered through a little quicklime. On addition of the silver nitrate a white curdy precipitate forms, which redissolves, and it is added till the solution shows a white precipitate.

An easier and more accurate method is by titration with a standard solution of iodine in potassium iodide until a blue coloration is apparent, a starch solution having been added to the cyanide solution. The reaction is as follows:—

$$KCy + I_2 = KI + ICy.$$

The solution may be checked on chemically pure cyanide, or, better yet, on sodium hyposulphite, and the cubic centimeters equivalent to 1 per cent. of potassium cyanide calculated.

If the percentage of zinc in the solution is required, the solution is evaporated to degrees and the residue treated by any of the well-known methods of analysis for zinc.

The cyanide employed is not chemically pure, and the commercial article contains about 80 to 90 per cent. or over. It is customary in some works to dissolve in a small tank several hundredweight of the cyanide, and test the strength of this *storage solution*, and make up from it the stock solutions. The lumps of cyanide are placed on a filter, made of a coarse sieve covered with jute. A pump is used for 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 solution in per cent.;
B being present strength of stock solution in per cent.;
C the strength of dissolving tank solution in per cent.;
D the quantity in tons, lbs., gals., litres, &c., &c., of stock;
$\frac{A - B}{C - A} \times D =$ quantity of dissolving tank solution to be added (in tons, lbs., gals., litres, &c., &c.).

For example, supposing the stock solution to consist of 100,000 galls. of 0·4 per cent. strength, and it be desired to bring this up to 0·6 per cent. by adding some 10 per cent. solution, then—

$\frac{0·6 - 0·4}{10 - 0·6} \times 100,000 = 2127·65$ gals. of the dissolving tank solution.

The subjoined table, which was kindly prepared for me by Mr. Blomfield, chemist of the works, exemplifies the strengths of the different solutions at the various stages while leaching from tailing-vat. This particular lot, which was treated at the Rand Central Ore Reduction works, being somewhat slimey, took a longer time to treat, *i.e.*, about 60 hours :—

---

* See Feldtmann on "Gold Extraction."

| TIME OF LEACHING. | ORIGINAL STRENGTH OF SOLUTION. | STRENGTH OF SOLUTION AFTER CONTACT. | REMARKS. |
|---|---|---|---|
| 1st Weak Solution. | | | |
| | 0·16 per cent. | | There was considerable moisture in the tailings previous to running on the 1st weak solution. |
| 1 hour | | 0·05 per cent. | |
| 3 hours | | 0·05 ,, | |
| 5 ,, | | 0·075 ,, | |
| 7 ,, | | 0·1 ,, | |
| Strong Solution. | | | |
| | 0·35 per cent. | | |
| After 1 hour. | | 0·1 per cent. | |
| ,, 3 hours. | | 0·2 ,, | |
| ,, 5 ,, | | 0·25 ,, | |
| ,, 7 ,, | | 0·24 ,, | |
| ,, 9 ,, | | 0·26 ,, | |
| ,, 11 ,, | | 0·25 ,, | |
| ,, 13 ,, | | 0·3 ,, | Draining dry. |
| | | 0·24 ,, | |

| TIME OF LEACHING. | ORIGINAL STRENGTH OF SOLUTION. | STRENGTH OF SOLUTION AFTER CONTACT. | REMARKS. |
|---|---|---|---|
| | 1st Weak Wash. | | |
| | 0·16 per cent. | | |
| After 1 hour. | | 0 26 per cent. | |
| " 3 hours. | | 0·3 " | |
| " 5 " | | 0·33 " | |
| | 2nd Weak Wash. | | |
| | 0·16 per cent. | | |
| 1 hour. | | 0·33 per cent. | |
| 3 hours. | | 0·3 " | |
| 5 " | | 0·29 " | |
| 7 " | | 0·22 " | |
| | 3rd Weak Wash. | | |
| | 0·16 per cent. | | |
| 1 hour. | | 0·2 per cent. | |
| 3 hours. | | 0·17 " | |
| 5 " | | 0·15 " | |
| | 4th Weak Wash. | | |
| | 0·16 per cent. | | |
| 1 hour. | | 0·15 per cent. | |
| 3 hours. | | 0·14 " | |
| 5 " | | 0·15 " | |
| | Final Water Wash. | | |
| 1 hour. | | 0·15 per cent. | |
| 3 hours. | | 0·1 " | |
| 5 " | | 0·13 " | |
| 7 " | | 0·05 " | |

**Determination of Gold in Cyanide Solutions.**—*Buchanan's method* consists in precipitating a known quantity of solution with excess of argentic nitrate, decomposing the precipitate formed by means of a reducing agent, filtering, drying, and cupelling direct. In detail he found the best method of procedure to be as follows :—195 c.c. of the cyanide solution are transferred to a flask of about 500 c.c. capacity, and mixed with a few drops of potassic chromate. Argentic nitrate solution of any convenient strength, say, 5 per cent., is then added until the characteristic reddish tinge of the chromate shows that the reaction is complete. Then take 10 to 20 grms. of zinc dust or shavings, mix them thoroughly with the precipitate and solution in the flask, and add 2 or 3 c.c. of 10 per cent. sulphuric acid. Allow to stand for 10 minutes, add excess of sulphuric acid to dissolve the remainder of the zinc, filter, wash once, dry, and incinerate on a roasting dish in the muffle and cupel with a little lead. This method does away with the trouble of fluxing, and has the advantage of allowing of a great number of samples being done together. The results on the whole are slightly lower than those obtained by the ordinary precipitation method.

*Crosse's method* consists in taking ½ or 1 litre of cyanide solution, containing gold, and adding excess of nitrate of silver. A precipitate of cyanide of silver is formed, and also argento-auro-cyanide, which is insoluble. This precipitate quickly settles, and is filtered on to a large filter. It is then put in a crucible covered with flux and, say, 500 grs. of litharge. In 10 minutes the fusion is complete, the button of lead obtained is cupelled, and the gold parted from the silver. This method allows of a larger quantity of liquid to be operated upon, is easily carried out, and does not require much watching. In the case of strong cyanide solutions most of the potassic cyanide is decomposed by addition of acid previous to adding the silver nitrate, to avoid the otherwise heavy consumption of the latter.

**Extraction Tests.**—Mr. Feldtmann gives the following instructions as to the best tests for determining if an ore is amenable to cyanide treatment.

It is assumed that the total sample is crushed fine enough to pass a 30-mesh sieve.

1. Assay a portion of the sample.

2. Determine the amount of cyanide it will consume by shaking test. For example: 200 grms. of ore are placed in a glass stoppered bottle with a 100 c.c. of solution of cyanide of 0·5 per cent. strength, and shaken for twenty 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 twenty 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 grms., and, using an alkali solution of 10 grms. commercial caustic soda to the litre of water, each cubic centimetre will correspond to $\frac{1}{10}$th lb. of the same quality caustic soda required to the ton (of 2,000) 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."

It is usual, when reporting the amount of "cyanicide" in an ore, to do 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 colouration of the liquid.

4. Several samples of ore 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 burette clip. A filter is formed over the rubber stopper by means of a piece of sponge, some filter-paper, or some asbestos fibre.

Into such vessels the samples are placed—say, 200 grms. 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 c.c. 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, one, two, and three 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.

# CHAPTER XVI.

## *SMELTING OF PYRITIC ORES.*

THE HUNGARIAN METHOD OF SMELTING WITH IRON PYRITES—Matting Dry Auriferous Silver Ores at Toston, Montana—Treatment of Argentiferous and Auriferous Matte—Spur-ofen for Matte Fusion—The Herresshoff Furnace for Pyritic Smelting—Smelting of Pyritic Ores in Reverberatory Furnaces—Fusion for Matte of Auriferous Pyrites at the Boston and Colorado Smelting Works.

**The Hungarian Method of Smelting with Iron Pyrites.**—I think I may claim for Hungary—my native country—the introduction of a system by which gold is extracted from ores *by direct smelting*, a method which is practised there with iron pyrites containing a small quantity of gold.

When this mineral is heated to a certain temperature without access of air, it loses half its sulphur, and becomes converted into a fusible sulphide of iron. In treating auriferous iron pyrites contained in a quartz matrix, a flux must be added to melt the quartz; a slag will then be formed, and the sulphide of iron will unite with the greater portion of the gold present, forming a heavy regulus, which will subside to the bottom. The stratum of sulphide of iron thus formed beneath the surface of the slag and scoriæ produced by the fusion of the earthy and silicious gangue is technically termed a "matt." If this matt, containing the gold, be now roasted, or oxidized by heating it with free access of air, whereby it is deprived of a further portion of its sulphur, and the product be mixed with a fresh quantity of crude auriferous pyrites, and smelted again, a second matt will be obtained, which will contain the gold from the first and second charges of the auriferous ore. This process may be repeated a third, fourth, or any number of times, until the precious metal has been sufficiently concentrated in the substratum of sulphide of iron to admit of its profitable extraction. The last matt, containing the whole of the gold which has accumulated from the successive charges, and brought, therefore, to any required richness, is roasted and then fused with metallic lead, litharge, or with a mixture of galena and metallic iron. The gold is abandoned to the liberated lead, from the top of which the impoverished undecomposed sulphide and oxides may be skimmed off.

By this process the gold is obtained in a more concentrated form, or mixed with a smaller proportion of lead, than when the latter is used directly as a means of separating and collecting it from the smelted ore. The employment of the iron pyrites in the manner above described is only attended with advantage when it constitutes part of the ore itself, or when it can be readily and cheaply procured in sufficient abundance. When it forms a principal ingredient in the ore, it is manifestly highly advantageous, in point of economy, to employ it as a vehicle for the concentration of the precious metal, while it contributes at the same time to increase the amount of the product.

In smelting for matte, the ordinary crucible furnace cannot be employed, as the matte, as soon as it sinks below the smelting zone, chills and freezes up the tap-hole, and the tapping of a matte furnace becomes a very difficult operation. To avoid this inconvenience, there has been substituted for the crucible furnace the "Spur-ofen," a furnace with a sloping bottom immediately below the tuyeres, which permits the molten mass to flow continually through a narrow channel into crucibles located outside the furnaces. In these crucibles or wells the matte, owing to its greater specific gravity, sinks to the bottom by separating from the slag, and is tapped into moulds, while the slag runs off continuously into the usual slag-pots.

I know only of one locality in the United States, and that is in Montana, where this process has been put into practice and has proved a success.

**Matting Dry Auriferous Silver Ores at Toston, Montana.**—This system was introduced in America by Mr. W. L. Austin, who has given an account of his operations,* which I reproduce here in a somewhat abridged form :—

"During the summer of 1884, while engaged on professional work in Montana, my attention was attracted by the large quantity of dry silver gold-ores awaiting reduction. The available lead-ore supply seemed utterly inadequate for economic silver-lead smelting; besides, competition from outside had brought these ores up to such a figure as to render their treatment unremunerative, nor were many of the ores at all suitable for wet concentration. But to offset these disadvantages, extensive bodies of iron pyrites, within easy access of the dry ore, suggesting the practicability of pyritic smelting. A close examination of the field led to the conviction that such a process was the best means of beneficiating the majority of these ores—namely, those high in silica and low in silver and gold, and to the building of the Toston smelting works.

"The Toston plant was intended solely for experimental purposes, to determine how far the process was adaptable to Western mining enterprises — namely, to quick returns on capital invested, when coupled with other necessary adjuncts, expensive fuel and high-priced labour. At the time of starting these works, no information of the process having been tried in this country, beyond vague rumours of attempts made in Colorado, could be had; therefore there was no American precedent to serve as a guide. To copy the European practice with its large percentage of fluxing-material, small furnace-capacities, and high fuel-consumption, was out of the question. In the spring of 1885 the first furnace was built, a rude affair constructed wholly of sandstone, with water tuyeres, and the approximate dimensions of the ordinary Western lead-furnace. The feasibility of the scheme was conclusively shown by the operations of this furnace. Different styles and forms of furnace were tried, including the Herresshoff patent, and the works were being fitted up for handling a large amount of ore.

---

* "Transactions of the Institute of American Mining Engineers."

"The three years' work at Toston demonstrated that, under conditions existing at that place, the best style of furnace for the matting process is what the Germans call a *Spur-ofen*; that is, a furnace without crucible, which is closed a short distance below the tuyeres, and from which the smelted products are permitted to flow continuously. The many difficulties which presented themselves as long as a crucible-furnace was employed disappeared altogether when that furnace was converted into a *Spur-ofen*, and an outside receiver was attached. A duplicate of the Herresshoff furnace, which does such excellent work at Laurel Hill, New York, was tried and abandoned, on account of the passage communicating between the furnace and receiver becoming constantly choked up. This may have been due partially to the tendency of iron matte to chill suddenly (its fusing point lying above both that of copper and lead matte), and partially also to the use of highly siliceous slags, which do not admit of rapid smelting, and consequently did not furnish a sufficient flow of molten matter from the furnace to the receiver to keep the passage open. On a basic charge and copper matte, the furnace is said to work perfectly. Narrowing the smelting-zone and depressing the tuyeres also assisted operations materially.

"At Toston, the best results were obtained with a very low furnace and large volume of air. Considering the perfection which furnace-building in this country has attained, it should be possible with a furnace of large capacity (about 150 tons in twenty-four hours) to make very satisfactory returns.

"The method of feeding a matting furnace is of the very greatest importance, for, unless precautions are used, the easily fusible sulphide of iron agglutinates the charge above the smelting zone, and chokes up the furnace as fast as it can be barred off.

"The slags made at Toston range in silica from 30 to 48, in protoxide of iron from 27.24 to 49.73, and in lime from 4.7 to 26.98 per cent. Numerous experimental slags were tried, the object being to run the silica up as high, and the lime as low, as consistent with economic work. It was found much more difficult to keep the silver than the gold out of the slag. Slags showing merely a trace of gold (duplicate crucible assays, one assay ton each) were made, when the matte carried as high as 67 dols. per ton in that metal, and 125.5 ounces of silver. Although the silver would run up more often above than below an ounce, it did not, with the proper combination of silica, iron and lime, average as high as 2 ounces.

"The cleanest Toston slags were high in silica and lime, and low in iron. As the losses are usually supposed to be mechanical, a siliceous lime-slag may assist the separation of the metal by reducing the specific gravity of the slag. I am not prepared to say how far the concentration of the precious metals in an iron matte can be safely carried. The richest car load of matte shipped from Toston ran 3.35 ounces gold and 125.5 ounces silver (approximately £40) per ton; but Kerl speaks of concentrated matte made at Kongsberg, carrying from 583.33 to 729.16 ounces per ton (2,000 lbs.), and my experience leads me to believe that a more valuable matte could be produced than that above mentioned as shipped by us. The loss both of gold and silver is almost wholly in the slag, and is usually explained as due to the mechanical adhesion to the latter of small particles of matte. The higher the temperature at which the separation takes place, and the lower the specific gravity of the slag, the better the separation and consequently the cleaner the slags.

"In Europe it has been the practice at most places where the pyritic process is used, to roast, concentrate and desilverize the matte by a complicated series of operations, all having in view the production of bar-silver. In this country, concentration and refining can be omitted from the calculations of the producer, as a ready market exists for the product at almost any of our large copper or lead-smelting establishments. One offer for our Toston matte f.o.b. at works was 20 dols. per ounce for gold contents, and 95 per cent. of N. Y. quotations on silver, deducting a treatment-charge of 15 dols. per ton, and 1.50 for freight. For lead-bullion of the same grade, the best we could do, at the time inquiries were made, was, for silver, 97 per cent. of N. Y. quotations, and for gold the same as in the case of matte; but the freight-charges to market amounted to some 23.40 dols. per ton."

**Treatment of Argentiferous and Auriferous Matte.**—In many mining districts where silver-lead ores (containing gold) are smelted for what is usually called "base bullion," a valuable by-product are the "mattes;" these are of special importance when they contain copper, in which case the material becomes a "cupriferous matte," and has to be treated for its copper, silver, and gold.

The formula of treatment usually consists in converting the matte into black copper by repeated roastings and fusion, and the operation is stopped when a fusion yields but a small proportion of matte. The black copper is refined for pure copper, the details of which process will be found in works treating on the metallurgy of that metal.

The operations may be summarised as follows:

(*a*) Roasting the matte in heaps, stalls, or kilns, according to the methods laid down in previous pages.

(*b*) Fusion of the matte from the first roasting. The furnace employed for this fusion is shown in Figs. 147 and 148, and is constructed on the principle already explained (p. 379), which permits the fused mass to flow out of the furnace as soon as produced.

In the sketch the furnace has only one tuyere, *i*, and the shaft, *a*, is 10 feet high. Its section at the tuyere is 1 foot 6 inches wide by 3 feet 5 inches in length. The shaft widens, and at the feed *c* measures 3 feet. The fused masses are run into the basins, *k*, where a separation of the black copper matte and slag takes place.

The operation with such a furnace—as carried out at the Attenau smelting works, in the Upper Hartz—is as follows: The ore bed is prepared by mixing five tons of roasted matte with five tons of slag resulting from the treatment of copper ores. There are consumed 1½ tons of coke for every charge of 10 tons of ore bed. The smelting result will be—black copper containing the metal which is not contaminated with sulphur; cupriferous matte resulting from the sulphur which remained in the charge; and slag containing the impurities and the iron of the

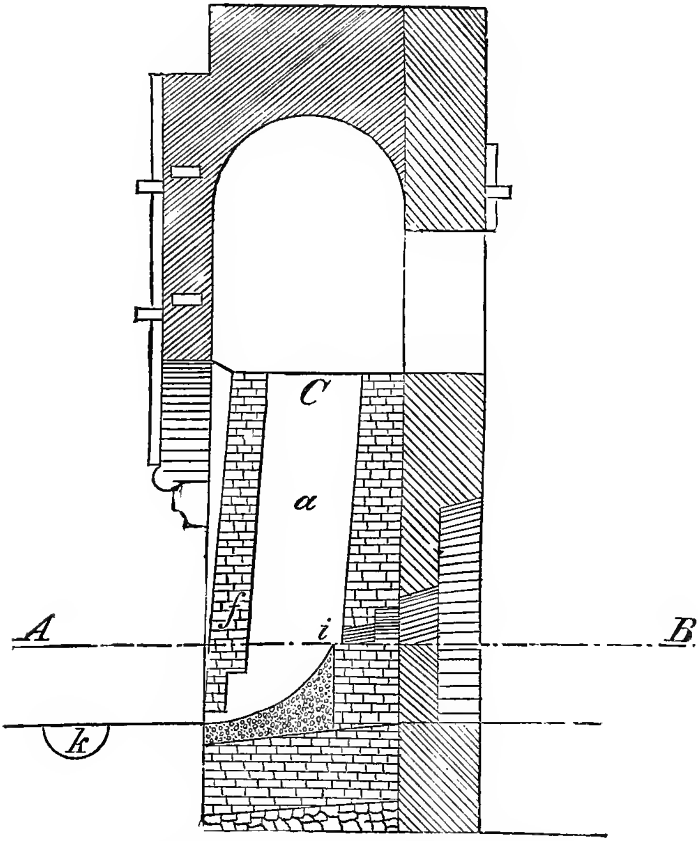


FIG. 147.—SPUR-OFEN FOR MATTE FUSION. Section through *C D*.

matte. The separation in the outside basins is affected according to specific gravity. The black copper is at the bottom, next comes the matte, and on top the slag. From mattes which result from the smelting of lead ores, there will also result some silver lead, which sinks to the bottom below the black copper.

The product of the first smelting operation consists of about 2.500 to 3.000 lbs. of black copper and 1,500 to 2,000 lbs. of matte. The matte produced is roasted and smelted again, and the above operation is repeated four or five times till the matte produced becomes very small, when it is put aside and submitted to a special treatment.

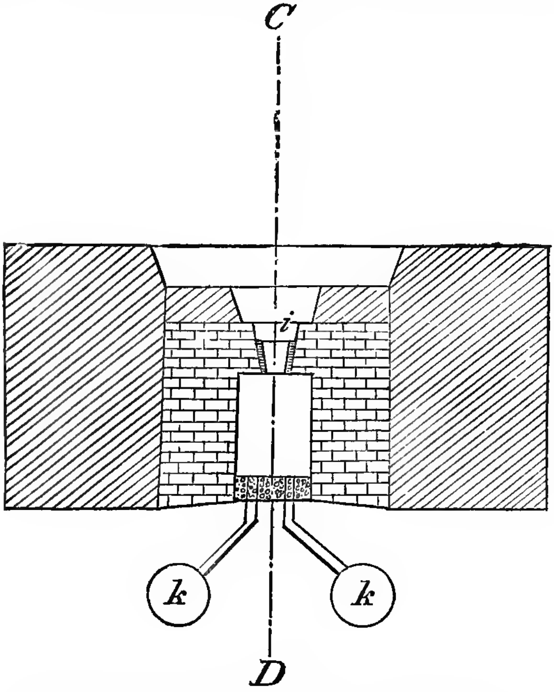


FIG. 148.—SPUR-OFEN FOR MATTE FUSION. Plan through *A B*.

It is obvious that at each subsequent smelting operation an enrichment of copper in the matte takes place, and this concentration rises with mattes containing about 30 per cent. in the first mattes to mattes containing 80 per cent. after the fifth or sixth smelting. In the same proportion as an enrichment of the mattes takes place, the black copper becomes purer at each smelting operation, and poorer in silver and gold. The black copper is submitted to an oxidising smelting so as to separate the impurities, and the resulting metal is afterward granulated; in which state the copper is submitted to a wet process, by which the silver and gold are extracted.

The *rationale* of this process is as follows: The copper granules are placed in a large lead-lined tub with a false bottom; a slow current of dilute sulphuric acid is sprinkled over the granules, which become oxidised on their surface, and the oxide is dissolved by the acid forming a soluble sulphate of copper. The silver and gold contained in the copper separate out as insoluble substances, and form a muddy deposit at the bottom of the sulphate of copper solution tank. This mud, which contains the metals and salts insoluble in the sulphuric acid, is washed on filters so as to free it from the soluble salts, and then mixed while still in the wet state with litharge, and moulded into bricks, which are dried and then smelted in ordinary shaft furnaces with rich litharge, rich plumbiferous matte, roasted iron pyrites, and slag.

The result of the charge is a rich silver-lead and copper matte. The silver lead is sufficiently rich to be cupelled at once, and the copper matte is roasted and treated as above.

It may be mentioned here that the sulphate of copper solution, resulting from the oxidation of the copper granules, is submitted to a regular crystallising process for the production of blue stone.

**The Herresshoff Furnace for Pyritic Smelting.**— The Herresshoff furnace may be regarded as the most modern and complete furnace of the American copper smelter. There are several features about it which are novel, but the most important modification which has been introduced is the movable fore-hearth, which is placed on wheels.

As will be seen from the accompanying drawings, the furnace is a water-jacketed spur-ofen, but instead of a sloping bottom—as in those of German construction—the bottom consists of a circular iron plate, bolted to the lower border of the water jacket, which extends a few inches below the tuyeres; and this bottom is covered with fire-bricks. The furnace allows (on account of the shallow depth between the smelting zone and its bottom) of a rapid outflow of the matte and

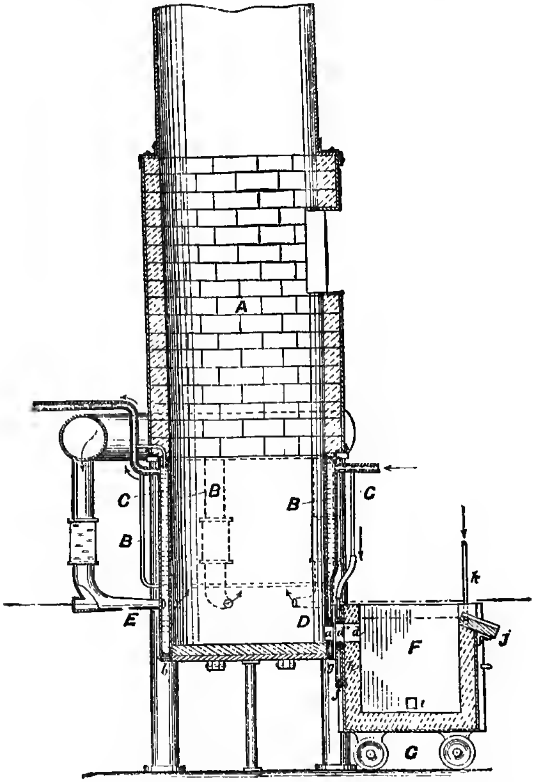
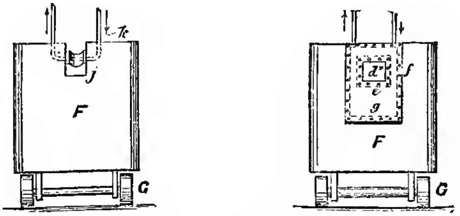


FIG. 149.—HERRESSHOFF FURNACE. Vertical Section.

slag into the well or receiver before any chilling can take place.

The well being on wheels can be readily removed, and rapidly replaced by another when occasion requires it. The front portion of the well which is contiguous to the furnace is also water-jacketed, as it would otherwise be rapidly worn out, this being the portion subjected to high heat.

Fig. 149 is a vertical section, B showing the inner casing of the water jacket and C the outer. The outlet for the molten mass is at *a*, which is an orifice from 4 to 6 inches in diameter, according to the size of the furnace. This aperture, *a*, is formed by placing a thick iron frame, D, into the water jacket, C, which frame is bolted or riveted to the plates C and B, and is clearly shown in Fig. 154. For a furnace which in its main smelting portion is about 6 to 8 feet high, 4 to 5 feet in diameter, this frame, D, should be of such a size that the hole, *a*, will measure 6 by 6 inches, and that the iron of which the frame, D, is composed should measure 1 by 2 inches in thickness. The water jacket of such a furnace is closed at the bottom by a wrought-iron ring, *b*, which is 2 by 2 inches, as shown in Figs. 149 and 153.



FIGS. 150 AND 151.—WELLS OR RECEIVERS OF HERRESSHOFF FURNACE.

The tuyeres, E E, entering through the water jacket, C, are shown in Figs. 149, 152. The discharge opening, *a*, of the furnace communicates with the receiving opening, *d*, of the well or receiver, F. This receiver s a metallic box lined on the inner side with fire-brick and placed on wheels, G. That side of the well which faces the furnace, A, carries a water jacket which is formed by two iron frames, *e* and *f*, and by metallic face plates, *g* and *h*. This water jacket is bolted or otherwise fastened to the well, F, on the outer side thereof, in such a manner that the space, *d*, bounded by the inner frame, *e*, constitutes a prolongation of the opening, *d*, and is likewise in line with the aperture *a*, as shown in Fig. 154. Water is fed into the space between the frames, *e* and *f*, through suitable pipes.

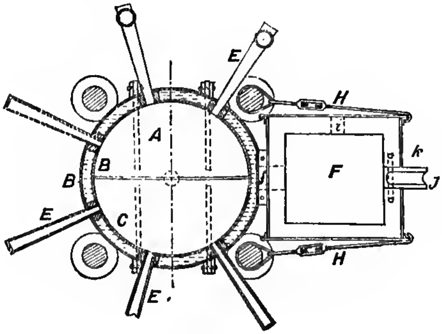
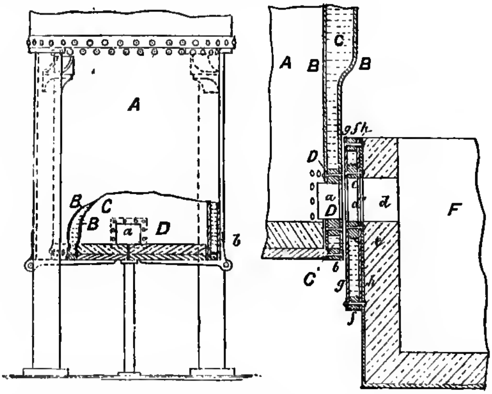


FIG. 152.—PLAN SHOWING THE TUYERES.



FIGS. 153 AND 154.—DETAILS OF FURNACE.

Fig. 152 is the plan of the furnace on the line *c c*.

The molten contents of the furnace will flow through the passage, *a d d*, into the well, F, which is placed close to the furnace, and anchored to the same by preference by means of screw bolts, H, shown in Fig. 152, and as the molten matter passes through the said channel, *a d d*, it will heat the iron frames, D and *e*, but they are protected by the water jackets.

Whatever greater or less space for leakage may exist between the frames, D and *e*, will be luted by the matte that leaks into such space, and thus continuity of the passage to the well will be established, and the well itself protected from rapid destruction by the heated mass.

The well, F, has a lower discharge opening, *i*, and an upper spout, *j*, which spout is cooled by a pipe, *k*, that carries water through it. This spout serves for the discharge of the slag, and this discharge is a few inches higher than the upper edge of the communication channel, *d*, so that when the well is full, the opening, *d*, is covered on both sides with molten material, and the blast in the furnace is completely trapped. In the well a separation of the matte and slag takes place, and when matte appears at the slag spout the tap hole, *i*, is opened, and the matte allowed to flow into moulds.

When the wells are emptied, the inconvenience arises that the blast escapes into them with full force; and to obviate this, covers (not shown in the drawing) are put over the wells, and the slag spout is plugged up with plastic clay, thereby trapping the blast.

**Smelting of Pyritic Ores in Reverberatory Furnaces.**—*Concentrating the Gold in Cupriferous Bottoms.*—This process is in operation at Swansea, and is also employed at some copper smelting works in the United States. When treating auriferous pyritic ores, the process permits the concentration of the gold in the so-called "bottoms." The "ordinary" method of reverberatory smelting is divided into six operations, the "extra" method into seven.

The *ordinary* method consists of—

(1) Roasting the sulphuretted ores, so as to eliminate sulphur, arsenic, and antimony, &c., as much as possible.

(2) Smelting for coarse metal or matte, with the addition of oxidised ores.

(3) Roasting the matte.

(4) Smelting the roasted matte with pure oxidized ores for white metal.

(5) Roasting and smelting the white metal for blister copper.

(6) Refining the blister copper.

In the *extra* process, the fourth of these operations is modified by—

(*a*) Smelting the roasted matte for what is called "blue metal."

(*b*) Roasting and smelting the blue metal for bottoms and regulus.

(1) The *roasting* of the ores is done either in heaps, stalls, kilns, or reverberatories, and the sulphur ought not to be all eliminated, but enough left in the ore for the subsequent operation in matte smelting. The arsenic and antimony are also eliminated during the operation as much as possible.

(2) *Smelting for Matte.*—The ore bed is so prepared that the iron oxide existing in the mass should be carried into the slag, and, therefore, has to be in proper proportion to the silica, alumina, and lime; and when in fusion the slag should be sufficiently light to separate from the matte, but at the same time should be sufficiently viscous to allow of its being skimmed off. Proper attention has to be paid to the quantity of sulphur to be left in the ore, so that the quantity of matte produced should hold nearly all the copper; and when copper ores are too much roasted and contain the copper mostly in the oxidized condition, scorification will take place, and a large amount of copper will go into the slag, unless it is counteracted by the presence of protosulphide of iron, which deoxidizes the oxidized copper.

Experience has proved that very highly siliceous ores can be advantageously smelted in the reverberatory furnace, even those producing a tri-silicate slag, and mixtures which could not run in a cupola furnace can with advantage be treated in this furnace. The same can also be said of the basic silicates which are fused with difficulty in cupolas.

Owing to the great heat which this smelting requires, the fireplace is large in proportion to the hearth area. The ore beds are so prepared as to produce a matte containing from 35 to 50 per cent. of copper. It is advantageous to let the matte accumulate on the hearth, and only tap it once a day, as an accumulation of matte favours the smelting down of the fresh charges.

(3) *Roasting the First Matte.*—The matte can be roasted in lump form in heaps, stalls, or kilns. This roasting has for its object the elimination of some of the sulphur—and antimony and arsenic, if present—and to oxidize the iron, so that in the subsequent smelting operation the matte produced shall be richer in copper.

(4) *Smelting the Roasted Matte for White Metal* is effected in smaller reverberatory furnaces, and the object is to scorify the oxide of iron as completely as possible, and to produce a matte containing over 70 per cent. of copper. The composition of the charge should be such as to permit of the separation of some metallic copper, which means that the quantity of sulphur should be such as not to form a disulphide with all the copper, leaving also a portion of sulphur to combine with iron. When the charge is tapped into the sand moulds, the metallic copper which has separated will accumulate in the bottoms of the moulds nearest the furnace, and hence the name "copper bottoms." These bottoms will contain nearly all the gold in the charge, owing to the weaker affinity of this metal for sulphur, but it will only contain a certain portion of silver, as silver has not so great an affinity for copper as gold, and greater affinity for sulphur than gold.

As silica is required for the scorification of the iron during this operation, it is advantageous to add siliceous gold ores to the charge, but these ores should be free from iron pyrites, although they might advantageously contain oxidized copper ores.

(5) *Roasting and Smelting for Blister Copper.*—This opera-

tion is performed in large reverberatories, and has for its object the expulsion of the sulphur and the precipitation of the metallic copper. When the furnace is charged with white metal, a strong heat is given to gradually melt it in an oxidizing atmosphere. The oxide of copper reacts on the sulphide of copper, producing metallic copper with evolution of sulphurous acid. During the oxidizing period a large proportion of other metals in the matte become oxidized, and during the fusion are absorbed in the slag. When the whole charge is fused, the air ports are closed, and the whole charge thoroughly melted; whatever copper oxide and copper sulphide remain in the molten bath react on one another with the production of metallic copper and expulsion of sulphurous acid. The iron oxides are scorified in the slag, which is skimmed off, and which is rich in copper. When it is noticed that all the matte has been converted into metallic copper, the furnace is tapped and the metal run into sand moulds. The surface of the copper ingots will be found covered with large smooth blisters, from which it derives the name of "blister copper."

(6) *Refining the Blister Copper.*—The refining of this impure copper is now carried out in many establishments by the electrolytic methods, of which an outline is given in Chapter X., p. 348. These permit of the separation of any precious metals which the copper contains, and so far lie within the scope of this work. The refining process for copper will be found described in many of the excellent treatises on copper metallurgy. In substance it consists in the elimination of the foreign substances contained in copper, and is the result of their direct oxidation by means of flame in the refining furnace, and of the reaction of oxide of copper upon the baser metals, having a greater affinity for oxygen than copper itself. The elimination of the sulphur, which has such a strong affinity for copper, is also part of the refining process. It is an oxidizing operation which requires long practice and experience to be successfully carried out.

Mr. Manhes, a French engineer, has introduced a system whereby matte, as produced from the first smelting of the roasted pyrites, is converted by one operation into metallic copper, and for copper ores not containing gold and silver his method is finding already wide application.

**Treatment of Auriferous Pyrites at the Boston and Colorado Smelting Works.***—Among the ores treated at these works were auriferous copper pyrites containing from 2 to 10 per cent. of copper, 2 to 10 ounces of gold, and 2 to 10 ounces of silver. These ores were mixed with tailings, silver ores, and fluxes, and then smelted in reverberatory furnaces for matte. The course of operations was as follows:—(1) Sampling the ore; (2) roasting the ore—large ore being roasted in heaps, small ore in a reverberatory furnace; (3) fusion for matte; (4) Ziervogel's process—including (*a*) crushing and roasting the matte for sulphate of silver, (*b*) leaching the roasted matte and precipitation of the silver, (*c*) washing and fusing the cement

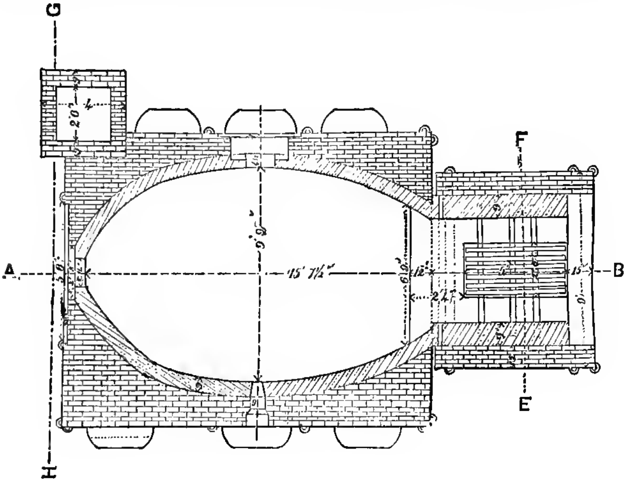


FIG. 155.—REVERBERATORY FURNACE FOR MATTE SMELTING. Plan.

---

* As described by Prof. Egleston in the *Transactions of the American Institute of Mining Engineers*.

silver, (*d*) precipitating the copper, and (*e*) refining cement copper; (5) treatment of the Ziervogel tub residues—including (*a*) fusion for white metal, (*b*) roasting the white metal, and (*c*) treatment of the pimple metal; (6) treatment of the residues of the Ziervogel process by the Augustine process; (7) treatment of the bottoms; (8) treatment of the oxidised copper alloy.

The hydro-metallurgical processes have been abandoned at these works, and have been replaced by more modern methods; but as the pyritic smelting was successfully carried on there, only a description of the various smelting operations is here given referring to the production of matte, white metal, pimple metal, and treatment of bottoms and separation of the precious metals.

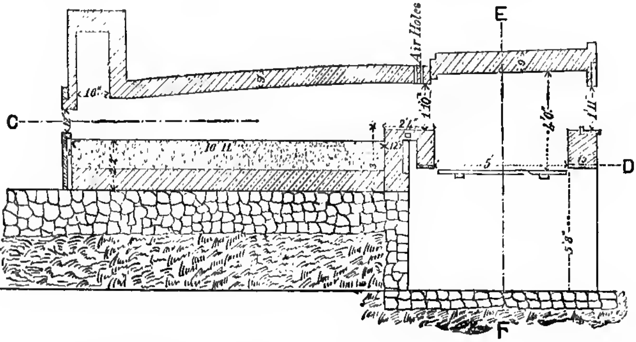


FIG. 156.—REVERBERATORY FURNACE FOR MATTE SMELTING. Section.

*Fusion for Matte.*—The roasted ore is fused in a reverberatory furnace for matte. There are three of these furnaces, which are shown in detail in Figs. 155 and 156. Only two of them are in use at a time. They are constructed to burn wood, so that the fireplace, which is 5 feet at the top of the bridge, is only 2 feet 6 inches at the grate; it is 5 feet long and 4 feet 6 inches deep from the grate to the roof. The opening in the fireplace for charging fuel is at the end of the furnace, and not at the side, as is usual. The fireplace door is of cast iron; it slides in a groove, and is counterpoised with a weight. The bridge is 2 feet 6 inches wide, the fireplace side is 2 feet 3 inches, and the laboratory side 1 foot 10 inches from the roof. Just above the bridge there is a series of openings in the roof, 3 inches by 1 inch, for the admission of air, which follow on the roof the contour of the laboratory in two rows, the outside having eight and the interior eleven holes each. The laboratory is 15 feet 7½ inches long by 9 feet 9 inches wide. The working door is at the end; the two openings at the side are closed for this operation.

In comparing the relative dimensions of the furnace, we find that the surface of the fireplace at the height of the bridge is 25 square feet, and that at the grate 12½ square feet. The laboratory has 143·18 square feet, so that the fireplace, being taken as one, the relation is as 1 to 5·7. In all these fireplaces, with inclined sides, the surface taken as unity is the section at the bridge. As the grate surface is smaller, the relation between the grate and fireplace surfaces should also be given.

Each one of the furnaces has its own chimney, which is 50 feet high. The arrangement of the holes in the roof is a very ingenious one, for as the fireplace is very deep, and is constantly filled with long sticks of wood to a depth of over 3 feet, the wood distils and forms gas, which is burned by the air entering through these holes, so that the fireplace is really a generator for burning wood. Before this method was introduced by Professor Pearce, there was not sufficient air to produce a perfect combustion. Formerly the flue connected with the chimney was constantly burning out, and needed frequent repairs. The immediate effect of the introduction of these holes in the roof was the saving of fuel and more equal distribution of heat. An opening is made in the foot of the chimney for the introduction of cold air. In consequence the repairs to the furnace are very much diminished, not only because the combustion is better regulated, but because the cold air is mixed with the products of combustion on leaving the furnace.

The hearth of the furnace is slightly inclined toward the working door and also to one side. It is made of two layers of brick, upon which fine quartz sand is placed, which is mixed with a small quantity of wood ashes and then agglomerated. When the hearth is made the temperature is lowered, and the charge introduced. This is made up of—

| | | |
|---|---|---|
| Heap-roasted gold ores | 2,000 | pounds. |
| Roasted tailings | 2,000 | ,, |
| Oxidized silver ores | 1,500 | ,, |
| Roasted silver ores | 1,500 | ,, |
| Raw pyrites | 800 | ,, |
| Fluor-spar | 250 | ,, |
| Rich scorias | 500 | ,, |

After the charge is drawn the furnace is repaired, if necessary, with clay, which is beaten in with a ladle-shaped instrument attached to a long handle. Such repairs are usually not made oftener than twice a week. The charge is introduced with a shovel by a side door (in more modern constructions the charge is introduced through the roof). The ore is introduced first and then the rich slags. The charge is so arranged that ten tons of mixed ores will produce one ton of matte. It will not do to make the matte richer, as there are always grains of it in the slag, and the loss would be greater.

The slag is carefully calculated, so that it shall not be too basic, or otherwise it would cut the fire-brick to get silica. The charge is evenly distributed over the surface of the hearth, which is almost at a cherry-red heat. It takes six men, working in groups of three at a time, nearly a quarter of an hour to make the charge. As soon as it is made, the charging-door is built up and luted or closed with sand. The fireplace is then charged, and the furnace is left with the full power of the draught for five or six hours. During this time the workmen clean up the slag, bed and tend to the fire, which requires looking after every twenty minutes. At the end of this time they stir the furnace carefully five or six minutes to bring up everything from the bottom, which should be perfectly smooth to the tool passing over it. This produces the reactions. The furnace is now left in repose for twenty minutes to effect the separation of the scoria and the matte.

If lumps are found, the stirring is done again, and kept up during the firing, or for about an hour. The slag is now drawn with a rabble into moulds prepared for it. The operation for skimming the slag takes about twenty minutes. When the door is opened to skim the slag it is quite hot and fluid, and there is a constant but quiet ebullition of sulphurous and sulphuric acid, the bubbles being about 1 inch in diameter and quite uniformly distributed. Professor Pearce asserts that the larger part of the gas is sulphuric acid. At the close of the skimming, as the slag becomes cooler, the bubbles become larger and less uniform. Just before the skimming, pieces of sheet iron, 3 feet by 2 feet, are placed in front of the slag bed and to one side of it, to protect the workmen from the heat. The casting-bed is made 10 inches deep in front of the furnace to receive the plate slag, which ordinarily contains all the grains of matte. This casting-bed has fourteen divisions, which are connected one with the other. When the slag, which covers the matte to the depth of about 3 inches, is being skimmed, it is very easy to distinguish the matte below, which shows of a dark colour and a more or less brilliant surface. As the rabble goes backward and forward the slag does not close at once over it, and the surface is exposed for a very short time.

When all the slag is drawn off, a new charge of ore is introduced. Four charges are made in twenty-four hours. During each one of the operations, the stirring and rabbling are conducted in exactly the same way. While the slag is tapped the matte is left to accumulate, and is tapped only once in twenty-four hours. When the matte is to be tapped all the doors of the furnace should be opened, so as to chill the last part of the slag a little, so that it will not flow out from the tap-hole. It is then tapped, and made into plates 3 feet long, 14 inches wide, and 4 inches thick in the middle, the bottom being rounded. No slag flows out with it, because it is too much chilled. When all the matte has been tapped the tap hole is closed with damp sand. The charge makes about fourteen plates. The operation of tapping the matte and stirring takes half an hour. Three men per shift of twelve hours are required to work two furnaces. Eight cords of wood are consumed in twenty-four hours. The plate-slag contains on an average 5 per cent. of copper, but is often poor enough to be thrown away with the other slags. It is generally a silicate of protoxide of iron, but is sometimes more basic. The poor slag contains about seven ounces of silver and a trace of gold. It is too poor to treat, and is thrown away. All the slag richer than this is put back into the furnace. The matte contains from 25 to 30 per cent. of copper, twenty ounces to thirty ounces of gold, 600 ounces to 1,000 ounces of silver, and some iron, lead, zinc, and antimony. When the hearth bottom of the matte furnace becomes loose and rises, as it sometimes does, the whole hearth material is taken out, crushed, and treated as ore. The flues of the furnace have to be repaired every two or three months. The roof is made over once a year. The outside walls last a number of years before it is necessary to rebuild the furnace. There are produced from this fusion the copper matte, which passes to the next operation, the plate slag, which is immediately put back into the furnace, and the poor slag, which is thrown away.

*Treatment of the Ziervogel Tub Residues.—A Fusion for White Metal.*—The residues from the tubs consist of oxides of copper and iron, with twenty or thirty ounces of gold and forty ounces of silver to the ton. They amount to about twenty-two tons a week. They are melted in the matte furnace, with rich gold ores of the first class, containing iron with copper pyrites, and variable quantities of gangue, and highly siliceous tellurium ores. All the siliceous pyritiferous ores are selected for this purpose. The ores are all crushed and put through a ¼-inch mesh sieve. The charge is brought to the furnace in alternate barrows of residues and ores, but it is not mixed before charging, as it becomes mixed after it is thrown into the furnace. The charge consists of—

| | | |
|---|---|---|
| Tub residues . . . . . . . | 4,000 | pounds. |
| Raw gold ores of the first class . . | 2,500 | „ |
| Gold ores of the third class . . . | 900 | „ |
| Total . . . . | 7,400 | „ |

When there are no tellurium ores, the charge of gold ores of the first class is made to amount to 3,400 pounds. The treatment is exactly the same as before. A poor slag containing only two ounces of silver and a trace of gold is produced; it is very much poorer than those of the previous fusion. It has otherwise very nearly the same composition as the others, but there is no zinc, either as blende or oxide, in it. The matte contains:—

| | |
|---|---|
| Copper . . . . . . . | 60 per cent. |
| Gold . . . . . . . . | 55 ounces. |
| Silver . . . . . . . . | 130 „ |
| Sulphur . . . . . . . | 30 „ |

It is called white metal. If the matte was made richer in copper the slag would also be richer, and there would be more loss. The tapping is made twice in twenty-four hours. In other respects the labour, fuel, and etc., are the same as in the matte fusion No. 3. This fusion for the treatment of tub residues takes place once a month, and lasts a week. All the plate slag produced during this operation is put directly back into the furnace.

*B. Roasting and Smelting the White Metal.*—At the end of a week all the mattes produced are recharged in large lumps, the charge being about four tons. It is roasted at a dull red heat for about ten hours with admission of air. The reaction which takes place between the sulphide and oxide make a peculiar noise, which can be heard at some distance from the furnace. The operation is termed "roasting" for black copper, but it is stopped half-way. As the sulphur is driven off some metallic copper is liberated.

The slag is very thick, and not more than 200 pounds to 300 pounds are produced. It contains from 8 to 10 per cent. of copper, and is highly basic, often containing crystals of magnetite. At the end of the ninth hour the doors are closed, and the fire-place charged.

The whole furnace is brought to a white heat, so that the whole charge is in intimate fusion. Just before tapping it is rabbled for five minutes, and then tapped into sand moulds.

The tapping is done as before, but moulds are made to receive the matte, as the charge is greater.

In the first three or four pigs there will be found plates or bottoms of metallic copper containing arsenic, antimony, and lead. These bottoms contain nearly the whole of the gold, with a small quantity of silver, from 3 to 5 per cent. of sulphur and 80 per cent. of copper. The matte is pimple metal, and contains about—

| | |
|---|---|
| Copper | 75 per cent. |
| Gold | 2 ounces. |
| Silver | 140 ,, |

For every charge about 600 pounds of bottoms and three tons of matte are produced. This bottom fusion takes three days, making ten days for this treatment of the residues. The labour is the same as in the matte fusion, but more wood is used, four cords being burned in twenty-four hours. Only two operations are made in twenty-four hours.

*C. Treatment of the Pimple Metal.*—The pimple metal is roasted again in the same way, treating it nearly five hours, and making four charges in twenty-four hours. Other bottoms are produced poorer in gold, but containing—

| | |
|---|---|
| Gold | 60 to 100 ounces. |
| Silver | 300 ,, |
| Copper | 75 per cent. |
| Sulphur | 25 ,, |

The pimple metal from this fusion contains—

| | |
|---|---|
| Gold | $\frac{1}{4}$ ounce. |
| Silver | 120 ounces. |
| Copper | 80 per cent. |
| Sulphur | 20 ,, |

the iron being entirely removed. This operation takes one and a half days. The bottoms are treated with the other bottoms. The pimple metal goes to the Ziervogel process B, but is kept entirely separate, because it contains no gold as does that of the process A.

*Treatment of the Bottoms.* — Four tons of white metal from the Ziervogel treatment give 600 pounds of bottoms. These are left to accumulate until they amount to 3,500 pounds, enough for a charge in the small reverberatory furnace.

The fire-place is 6 feet long, 4 feet deep, 42 inches wide at the bridge, and 20 inches at the grate. The bridge is 2 feet wide. The laboratory is 9 feet long, 6 feet 9 inches wide, and connects with the chimney, 2 feet 6 inches square, by a flue. The surface of the fire-place is 21 square feet, that of the laboratory 46.27 square feet; the relation, therefore, as 2.2. The furnace has a working door at the side and a charging door at the end. On the side opposite the working door there is a spout which ends in a wooden tank sunk in the ground, which is 4 feet 5 inches in diameter and 3 feet deep.

The object of the process is to oxidise the lead and other impurities, and to prepare the metal for treatment for gold. The charge is made at 7 A.M. It is first sweated at a low temperature for two or three hours, during which time some of the lead liquefies and runs out of the furnace. It is then left to oxidize for three or four hours. In about seven hours the charge is well melted. The slag, which is skimmed at this time, is composed mostly of oxides of lead and copper, containing from 10 to 15 per cent. of copper, and is sent to operation No. 3. After the slag is withdrawn the bath is beaten with a rabble for about two hours, all the doors being opened to admit an excess of air. It is again skimmed and tapped into water. The "pitch," that is, the condition of the copper, must be such that the whole of the sulphur is eliminated before the oxygen is absorbed. If the pitch is right the globules will be all round and hollow. This point must be seized with the greatest nicety, for if the charge remains too long in the furnace the globules will cast solid, and the charge must then be put back and worked with sulphur. The temperature of the water governs the size of the globules. They are small when it is cold and large when it is hot, but it does not otherwise affect it. It takes about ten minutes to do this casting. The copper flowing from the spout falls on to a pole of green wood held underneath it, so as to scatter the copper.

Care must be taken that the slag does not flow with the copper. To prevent it the doors are opened, so that the slag is cooled until it is pasty. One charge is made at a time, and only one or two per month. The globules contain 1,000 ounces of gold, 600 ounces of silver, and a trace of lead. Twenty tons of white metal give one ton of refined auriferous copper. Three cords of wood are used, one man tends the furnace, one man does the firing.

*Treatment of the Oxidized Copper Alloy.*—The copper globules are oxidized in one of the fine calciners, used for roasting for sulphate of silver. One and a half tons are charged at a time. The oxidation takes thirty-six hours. The globules are put into the furnace in a heap and spread out over the hearth. The charge will be 3 inches deep. The fire-place is charged at once, and the temperature is made as hot as the red bricks will bear, and as oxidizing as possible. It is constantly rabbled. At the end of thirty-six hours a portion is taken out and tested, to see that it will pulverise completely. If it does, the operation is finished; if it does not, the oxidation is continued. The whole of the copper has been transformed by the operation into suboxide, and the charge is increased in weight about 500 pounds by the operation. The grains are black on the outside, but if broken or rubbed the streak is red. The charge is drawn out into an iron barrow, and carried to the store-room. It is placed in bags, packed in petroleum casks, and shipped to Boston. One cask holds 650 pounds. Three cords of wood are used for the process, and two men do the work, one man to each twelve hours' shift. The men are required to bring their own wood.

*Solution of the Oxidized Copper Alloy.*—The oxidized product was treated in Boston with dilute sulphuric acid. This is done in a conical tub lined with lead, having a false bottom. The bottom is hollowed so as to leave as little space as possible. A charge is 1,500 pounds. Over this sulphuric acid at 20° Beaumé is poured. Steam and air are turned on, and the boiling continued for four hours. The whole is not dissolved, but 90 per cent. of the copper will be in solution. It is allowed to settle for an hour, and is siphoned off and a fresh charge put in. Two charges are made in a day. This is repeated until all the oxidized products have been treated. This work is not done at night. The residues are boiled two or three times in the same way to get out all the copper possible. The tub is then cleaned up, and what remains is melted in plumbago crucibles. The bullion is from 600 to 800 fine of mixed metals. It contains from 40 to 50 per cent. gold and 20 to 30 per cent. of silver. This is sent to the mint.

The sulphate of copper is crystallized and sold. The mother liquid is used to dilute the acid used for the solution of the oxides.

The making of these alloys of gold, silver, and copper was first tried in the works, and was given up on account of the high price of sulphuric acid. It was carried on for more than a year in Boston, but has been abandoned, and the separation of gold and silver was subsequently effected by a process invented by Professor Pearce.

**The Economic Advantages of Smelting Pyrites by utilising their Fuel Qualities.**—That pyrites can be set on fire, and that once ignited they will continue to burn, has long been known, and advantage is taken of this property by acid manufacturers, who burn them in kilns, subsequently converting the fumes into sulphuric acid. But it has remained for modern science to show that these same pyrites can also be made to replace, to a large extent, the coke ordinarily used in smelting ores of copper, nickel, gold, &c. This method of smelting sulphide ores, which consists in making available the heat stored up in their iron and sulphur contents, is what is technically known as pyritic smelting.*

Pronounced advantages are claimed for pyritic smelting when applied to suitable ores, as compared with other pro-

---

* The subject has been thoroughly discussed in recent scientific publications, notably in the "Bergund Huettenmaenische Zeitung," vol. liii., p. 42, and in the "Chem. Zeitung," 1893, No. 99. Any one wishing to inquire further as to this method of ore reduction can address Mr. W. L. Austin, Denver, Colorado, U.S.A.

cesses of ore reduction, including a great saving in expenditure of both time and money by avoiding the roasting of ores.

As is well known to everyone engaged in treating pyritic ores, such, for instance, as the great pyrites deposits of Spain and Portugal, or those of Norway and South America, the first step in all such operations is the roasting of the ores. Great heaps of the mineral are piled up and set on fire, and allowed to burn slowly until the pyrites are converted into oxides, which are subsequently smelted or treated in some other way, for the extraction of their valuable contents. This burning, or roasting as it is called, requires weeks, and sometimes months, for its completion, during which time large amounts of capital are tied up in said heaps and, therefore, unproductive. This, in itself, is, in the case of large and important mines, a factor of no small importance, for the year's interest account on the money lying idle in roast heaps amounts to considerable.

The advantage, therefore, of pyritic smelting—which method takes the raw sulphides from the pit's mouth, and with no preparatory treatment whatever dumps them in pieces of any size that a man can lift into a blast furnace, producing therefrom within twelve hours a concentrated product ready for the market, are so manifest as hardly to require comment. All the cost of labour employed in breaking, piling, and burning the raw ore is also saved, and in one operation (or where high concentration is desired, in two operations) the low grade ore is converted into a product (matte) which is only reached in the older methods by breaking, roasting, and repeated smeltings.

A second very material advantage gained by pyritic smelting is the increased capacity of furnaces; for, by utilising the fuel qualities of the ores themselves, very much more material can be run through a furnace of a given size than where tons of coke and charcoal must be consumed together with the ore. It requires time to consume carbon fuels, and where these are omitted from a furnace charge it stands to reason that more paying material (ore) can be added to the charge. Actual practice has clearly demonstrated that when a blast furnace is operated with pyrites instead of coke, from 50 to 100 per cent.

more ore can be passed through it than by the old methods of smelting. This would naturally be expected, because the sulphides of iron, &c., are extremely fusible, and softening at the high temperature produced by the combustion of the iron and sulphur they melt down rapidly before the powerful air blast used, creating an intense heat, which in turn melts the batch of ore next above. Furnaces with an area at the tuyeres of 30 square feet (a size now commonly used for smelting purposes in Western America) will smelt from 150 to 200 tons of raw sulphides in 24 hours. The labour necessary around a blast furnace is comparatively small, so that handling such large quantities of ore in so short a time, and at so small expense, necessarily reduces the cost per ton of ore smelted to the minimum, and permits the handling of very low grade ores.

Under favourable conditions, the cost of producing concentrated matte from raw sulphides—taking the latter from the pit's mouth and handling 200 tons per diem, will not exceed six shillings per ton of ore. It is generally admitted by metallurgists that the most complete extraction of metals from their ores is accomplished where it is possible to employ fusion, and therefore the smelting process is always adopted where admissible. But very often the cost of fuel (coke, &c.) is so great as to preclude the use of this method of reduction : therefore a method of smelting, in the carrying out of which no coke or charcoal is used, is a desideratum. Now admitting that pyrites in itself contains sufficient calorific power to support unaided a smelting fusion (and this has been so clearly proven as not to admit of argument at the present time), it would be manifestly a disadvantage to employ two separate and distinct classes of combustibles at the same time and in the same operation, for there would necessarily be some slight differences in the combustibility of the two materials: one would burn before the other, which would lead to disturbances in the working of the apparatus.

Where sulphides and coke are both burned in a blast furnace, the latter burns first, and the heat thereby produced only serves to melt down the former, which runs through the furnace in the form of monosulphide, and comes out otherwise unaltered. This is a self-evident proposition when one stops to consider the chemical reactions which take place, for the coke being a simple element unites readily with the oxygen of the blast, whereas the pyrites, which are compounds of sulphur and iron, must first have their elements dissociated before they can combine with the oxygen. This is the explanation why sulphide of iron (matte), when melted in a furnace with carbonaceous fuel, comes out in the same form as that in which it was put in, practically of the same quality and in the same amount as charged; and it also explains why it is wasteful and unnecessary to use coke in pyritic smelting, when there is sufficient sulphide of iron in the furnace charge to give the requisite heat for the operation. With a proper arrangement of the ores, fluxes, &c., concentrated matte can be produced without the aid of carbonaceous material in the charge, and, therefore, the saving in cost of treating ores over other methods is apparent, the fuel item alone being often an impassable barrier to the working of low grade mineral deposits.

One of the first things to be considered in all metallurgical operations is the percentage of extraction which can be safely counted upon when comparison is made between the various processes available in a given case. In pyritic smelting we have to deal firstly and mainly with the sulphides or arsenides of iron in one form or another, and secondly with the metals—copper, nickel, gold, silver, &c. Now we know from what has been done in treating mattes, &c., in Bessemer converters, that as long as the sulphide of iron is present it alone is slagged off, being far more easily oxidizable than the latter metals. The same principle holds good in pyritic smelting. The sulphur burns off, or combines with the copper, nickel, &c., while the iron is oxidized to the protoxide, in which form it combines with the silica present to form a slag. As long as sulphur is present, the copper, nickel, &c., on account of their strong chemical affinities to this element, necessarily unite with it to form sulphides, and, therefore, unless the concentration is carried too far, the slag will be found to be practically free from these metals.

Sometimes it is found desirable to divide the operation into two parts, concentrating up to a certain point in the first smelting, and then carrying the treatment to a finish in the second. If the useful, or precious metals, go into the slag in any quantity in the second operation, this slag can be returned to the first smelting, where it is of service in assisting in the even operation of the furnace. Experience has demonstrated that the saving of the useful and precious metals in pyritic smelting closely approximates, if it does not equal, that obtained in the older methods of smelting.

One of the most disagreeable features connected with the treatment of large quantities of sulphide ores by the methods so long in use, is the enormous quantities of noxious gases turned into the atmosphere, devastating large tracts of country, and causing damage to vested interests. Any method of handling sulphide ores by which the sulphurous fumes can be utilised, or at least rendered harmless, would thus be a great improvement. In pyritic smelting almost all of the air is utilised in oxidizing the sulphur and iron, so that the gases escaping from the tunnel head are in a concentrated form, and can be made available for manufacturing sulphuric acid, or readily condensed in suitable towers and thereby prevented from escaping into the air. That these gases are very highly concentrated is evident from the fact that they are often tinged with the yellow colour of the sulphur itself, or burn on the top of the charge; and they are, therefore, in a suitable condition for acid manufacture.

# CHAPTER XVII.

## *THE CUPELLING, PARTING, AND REFINING OF GOLD BULLION.*

CUPELLATION—The English Method—Cupelling Furnaces on the Continent—The Parting Process—At the Refineries of the United States —At the San Francisco Assaying and Refining Company's Works —Gold and Silver Parting at Oker in the Unterharz—Mint Parting with Nitric Acid—Refining of Brittle Gold with Chlorine Gas: Miller's Process—Separating Iridium from Gold in Mint Deposits.

**Cupellation.**—Another important metallurgical operation is the extraction of the gold from the lead, and this has been performed from the most remote antiquity by a process called cupellation. This process, which is very simple in principle, is at the same time one of the most elegant metallurgical operations ever invented. It depends on the circumstance that when silver and gold are exposed in a state of fusion to the action of the air or oxygen, they neither give off perceptible vapours nor are sensibly oxidized, they remain, in short, unaffected; whereas, under similar circumstances, lead and almost all the other metals are oxidized with greater or less rapidity. There is also an important peculiarity connected with lead, which renders it the only metal, except bismuth, applicable to the process of cupellation. This peculiarity consists in the circumstance that the oxide of lead, known as litharge, is fusible at a bright red heat, and in this state absorbs any other metallic oxide with which it may happen to be in contact, but which, without the influence of the oxide of lead, would remain uninfluenced by the heat. For example, if copper be present as an oxide it is taken up by the melted oxide of lead; and if the latter can be separated from the gold and silver, the oxide of copper will be expelled along with it.

This separation may be effected in various ways, but particularly by means of bone-ash, which possesses the peculiar property of absorbing the melted litharge even when mixed with a certain proportion of other oxides, whereas it remains impermeable to the unoxidizable metals. Hence, if a vessel be formed of pounded and compressed bone-ash, and into this vessel a mass be introduced, consisting of a mixture of litharge or oxide of lead, copper or oxide of copper, and native gold (which always contains a certain proportion of silver), and if this mixture be exposed to a considerable heat, the oxide of lead will dissolve the oxide of copper, and both will be absorbed by the bone ash, while the mixture of gold and silver will remain in the vessel, melted, but otherwise unaffected. Hence it follows that gold or silver, or a mixture of the two, may readily be deprived of copper or any other oxidizable metals by cupelling with oxide of lead. For this purpose it is not necessary to employ the oxide itself in the first instance. When metallic lead is heated to a temperature above its melting point, it rapidly combines with the oxygen of the air, the product being in the first place yellow oxide of lead, which appears as a scum on the surface; and when the temperature is raised to about bright redness, this oxide is converted into litharge, or the fusible oxide, which immediately melts. Pure copper requires no less than sixteen or seventeen times its weight of lead to absorb it in the manner above mentioned.

In carrying out the cupellation process on a large scale we do not utilize the absorption power of the cupel alone, but are assisted by another agency, which is compressed air.

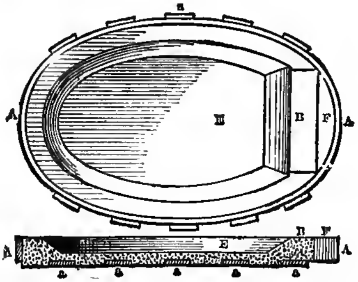
**The English Method of Cupellation** or refining, as carried out in a large refinery located in the heart of the city of London,* is as follows:

In preparing the cupel, a piece of flat wrought iron, about $\frac{3}{8}$ to $\frac{1}{2}$ inch thick and 4 to 5 inches wide, is bent into the form

---

* The refinery of Messrs. Browne & Wingrove.

of an oval hoop and the ends are welded together. The larger diameter of the hoop is in conformity with the size of the cupelling furnace and varies from 4 to 5 feet, and the short diameter from 2½ to 3 feet. Five or six flat bars are fixed to the lower part of the hoop, arranged parallel to each other in the direction of the short diameter. The first bar is placed about 9 inches from one end of the oval, and the others at equal distances between this bar and the other extremity. The hoop is then placed with the cross bars downwards upon a solid floor, and a quantity of pounded and sifted bone-ash is beaten firmly into it with a wooden rammer until it is entirely filled, or the bone-ash is level with the upper edge of the hoop. The bone-ash employed for this purpose is mixed with about one-tenth, by measure, of fern ashes, or one-fortieth by weight of American pearl-ashes, and moistened sufficiently to become coherent by pressure. The fern or pearl ash has the property of giving greater consistency to the bone ash when heated. When the hoop or frame has been well filled with this mixture and solidly beaten down, the surface is carefully scooped out with a trowel all round the centre, so as to form a shallow concavity of about 2¾ inches in depth, leaving a wall of bone-ash all round, about 2 inches in thickness at the top, and 3 inches at the bottom, except at one end, where a thickness of 13 inches is left; whilst the thickness of the sole itself is reduced to one inch above the upper surface of the iron cross



FIGS. 157 AND 158.—CUPEL. Plan and Section.

pieces. At the end of the test where the wall is left 13 inches thick—which is termed the breast—a segment of the bone-ash contiguous to the hoop is removed.

This apparatus is represented in Figs. 157 and 158, of which the former is the plan, and the latter a longitudinal vertical section. In these drawings the letters A A denote the iron hoop; *a a* the transverse bars; E indicates the cavity in the bone-ash; B, the breast; and F, the space where the bone-ash is removed between the breast and the iron. In cupels of more modern construction the bone-ash is not removed from the part F, as the litharge is found to come in contact with the iron ring, on which it exerts a powerfully corroding action, when such an arrangement is adopted. In place of this, however, a round hole is cut in the centre of the breast near F, to which channels are cut.

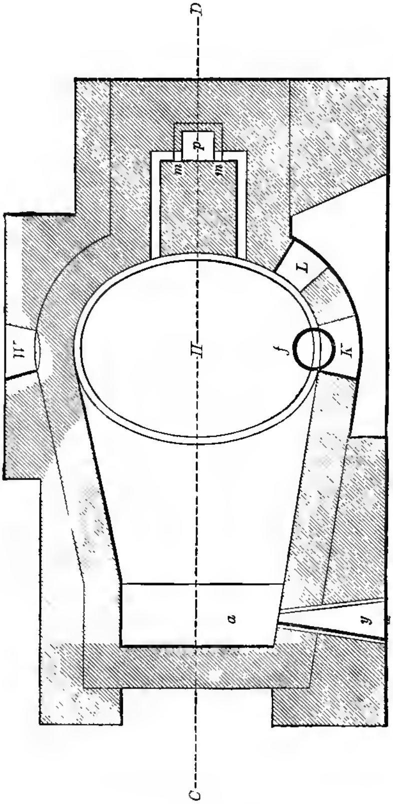


Fig. 159.—Plan of Cupellation Furnace.

When fully prepared, the test is allowed to dry, and is then placed in a furnace, constructed in all respects like a common reverberatory furnace, except that a space is left open in the bed of it to receive the test, and that the width of the arch is much reduced (see Figs. 159 and 160). The test, when placed in position, forms the bed of the furnace, with the long diameter transversely. The test is placed on a small flat bogie car *c* built of iron, whose platform can be raised by means of three jack screws *g*, and the part F of the test projects beyond the platform of the bogie. The whole is run now under the

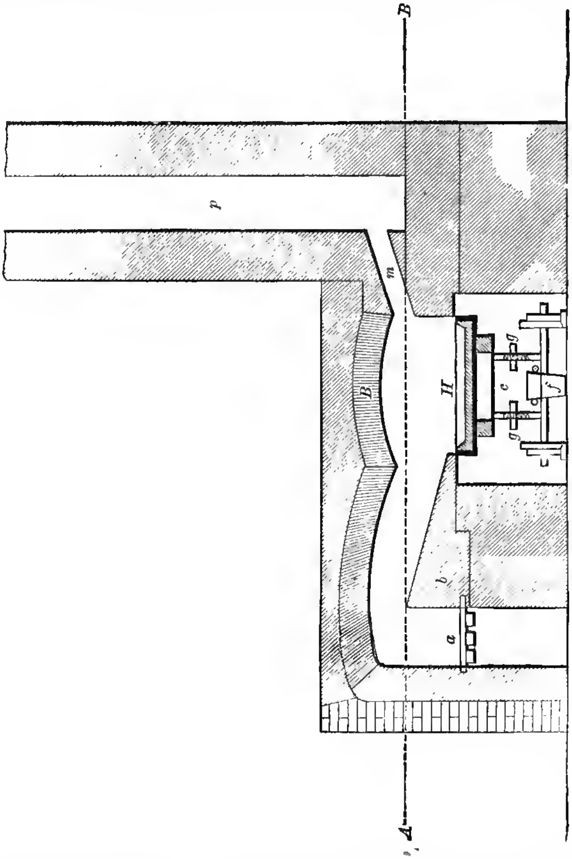


Fig. 160.—Section of Cupellation Furnace.

arch forming the lower part of the furnace, and by means of the jack screws the test is raised into position, thus forming the hearth of the furnace. The fireplace, *a*, of a furnace is generally 18 inches wide by 2 feet 4 inches long, or less, and is separated from the body or bed—and in this case, the cupel—by a fire bridge, *b*, 18 inches wide, which allows the flame to pass directly over the surface of the cupel H, from whence it passes into a chimney *p*. At the side of the furnace, opposite the breast of the cupel, is an opening W through which passes the nozzle of a continuous pair of bellows.

When introducing a fresh cupel or test, the fire must be low and heat must be applied with great caution, or otherwise the bone-ash will split to pieces; and for the same reason the bone-ash must be dried very gently.

When *silver lead* containing gold is to be cupelled, it is first smelted in a cast-iron pot set in a furnace near at hand, and after the cupel is red hot, the metal is ladled in till the cupel is nearly full. The blast is turned on, and oxidation sets in very quickly. At first the lead becomes covered on the surface with a yellow or grayish dross, but on further raising the heat, the surface of the bath "uncovers," and a film of melted litharge begins to appear. Part of this litharge sinks into the bone-ash of the test, which it soon saturates. The blowing apparatus at the back of the test is now set in motion, and forces the litharge as it forms forward to the breast, B, across the surface of which a channel is cut, through which the litharge begins to flow, and runs over at the end through the vacant space, F, into a movable iron pot, *f*, placed on the floor for its reception. The current of air introduced at the back of the cupel not only assists in clearing away the litharge, but supplies the amount of oxygen necessary for its rapid formation. In proportion as the litharge flows away, more of the melted lead is gradually supplied to the test, which is done by ladling it into a channel from the pot outside the furnace through the opening L. After a certain period, the channel first formed in the breast, B, becomes much corroded; it must then be stopped and a second and third channel successively formed for the same purpose. In this manner the operation may be continued until six or eight tons of the rich lead have been refined on the same test.

In dealing with doré silver containing a large percentage of copper, the refining is carried out by first melting the doré silver bullion in a wind furnace in crucibles, and ladling it into the previously heated cupel and adding molten lead in sufficient proportion to carry all the copper away. The proportions of lead necessary to purify the silver are substantially identical with those given in the chapter on assaying of bullion (see page 555).

Towards the conclusion of the operation some striking appearances are often presented. If the gold present is mixed with a considerable amount of silver, the surface of the mass, as it cools, is suddenly thrown into agitation; cones, or little craters, sometimes several inches in height, are thrown up, from which oxygen gas escapes with violence, projecting particles of silver with considerable force. This phenomenon is termed "spitting," or sometimes "vegetation," from the beautiful arborescent forms which are presented, and arises from the circumstance that metallic silver, when melted, has the property of absorbing six times its volume of oxygen, which is ejected with violence at the moment of solidification.

After cupellation, if it is not desired to solidify the metal in the test, the latter can be lowered by means of the jack screws; the bogie is run out from under the furnace arch, and the metal ladled quickly into moulds; or a mould can be placed under the test and a hole punched in the bottom, so as to allow the metal to flow in a regular stream into the mould.

On the Continent, the process of refining is conducted without the use of a cupel, properly so called, no bone-ash being employed. The test, if it may be so termed, is the bed of the furnace itself, which consists of a kind of marl, firmly beaten down into a circular cavity, which slopes from the sides to the centre and is allowed to dry. The roof of the furnace, which consists of a flat dome of bricks, built in a strong circular hoop of bar iron, is movable, by means of a crane. Several tons of lead are introduced at one charging, and after the roof has been replaced, the blast is transmitted through one or more apertures in one side of the furnace. The whole mass of metal is then worked off continuously, without any addition, till the lead is removed, and the alloy of gold and silver is left approximately fine.

**The Parting Process.**—The final operation of separating the silver from the gold is termed " parting," and is generally performed in England, in private refineries, by means of nitric acid, which dissolves the silver without attacking the gold. On the Continent and in the United States refineries the same effect is produced with hot sulphuric acid.

The operation of parting with sulphuric acid is very simple, and if any copper has been left in the alloy, it is separated along with the silver. Its presence is by no means injurious. In this process the proportion of gold must be less than in the parting with nitric acid, and the proportion of the gold to the silver should not exceed one-fifth. Experience has shown that the alloy should not contain more than one-twentieth part of copper. When the silver is not already present in the required proportion—which is never the case with native gold—the requisite excess must be introduced; for when the amount of gold exceeds the proportions above stated, the particles of silver are so enveloped in that metal as to resist for a long time the action of the strongest nitric or sulphuric acid.

The alloy being prepared in the requisite proportions—which is generally done by introducing the excess of silver before cupellation—it is melted in a crucible, and granulated by pouring it into cold water. For one part of the granulated alloy three and a half parts of concentrated sulphuric acid are taken, and the mixture is put into a platinum vessel, which is then introduced into the furnace. The vessel should not be more than two-thirds filled, to guard against the effects of effervescence, which might throw out part of the liquid. It is then covered with a platinum hood, provided with a beak or tube for conveying the gases and vapours into a condensing apparatus. The capacity of the platinum retorts varies. After two or three hours' boiling, varying the time according to the size of the retorts, the silver and any copper that may be present are completely dissolved.

Owing to the great expense of platinum retorts, Mr. Tocchi introduced iron retorts, which can be safely employed, although the former are in general use. He found that the very concentration of the acid presents, of itself, an obstacle to the precipitation of the silver or copper by the iron; and, further, that the interior surface of the retort acquires a coating of silver, so that the iron and the liquid soon cease to be in contact. This point has not been sufficiently studied, but the fact has been placed beyond doubt that the operation succeeds very well in iron vessels, and accordingly, in many refining establishments, no others are employed.

It cannot fail to be observed that, assuming the proportions above given, the quantity of sulphuric acid is much greater than would be strictly necessary to convert the silver and copper into sulphates. Supposing the alloy to be as poor as possible in gold, theory would indicate the following quantities of acid as being sufficient for this purpose:

| | Parts. | | Parts. | |
|---|---|---|---|---|
| Copper ... ... | 50 | would require | 155 | sulphuric acid. |
| Silver . . ... | 949 | ,, ,, | 861 | ,, ,, |
| Gold ... ... | 1 | ,, ,, | 0 | ,, ,, |
| Alloy | 1000 | | 1016 | |

But as the amount of sulphuric acid employed is actually 3,500 to 1,000 of alloy, there is an excess of about 2,500 of acid, intended to hold the sulphates, especially the sulphate of copper, in solution.

When the silver and copper are completely dissolved, the retorts are withdrawn from the fire, and the solution is left to cool, in order that the gold may settle to the bottom. The liquid is then decanted; the gold, which remains in the vessel, is carefully washed, and the water employed in this operation is added to the sulphate solution. Lastly, the purified gold is melted, and after being cast into ingots, is ready for commerce.

The acid liquor containing the sulphates is poured into a leaden boiler containing water and copper shavings. Heat is applied and the sulphate of silver is soon completely decomposed. The precipitated silver is collected and submitted to repeated washings, always adding the water to the contents of the boiler, to be treated in the next operation. Lastly the silver is dried in a small iron pan, and is then melted in a crucible to be cast into ingots. The acid solution, which now contains only sulphate of copper, is evaporated in the lead boilers till it is fit to crystallize. It is then put into the crystallizing pans, and the mother waters are further evaporated to obtain more crystals. This process is continued till the liquid becomes very concentrated, and consists almost entirely of sulphuric acid. In this state it is termed "black acid," on account of its colour, which is due, in great part, to organic matters, dust, and other impurities, which fall in it during the evaporation in the crystallizing pans. The black acids may either be employed to repeat the same operation, or turned to useful account in different manufacturing processes, which do not require that the sulphuric acid be pure if highly concentrated.

At the Mint refineries in the United States the boiling with sulphuric acid is carried out in iron kettles covered with hoods, which are connected by means of lead pipes with a chimney.

With the development of the Comstock mines which produce doré bullion, whose value consisted in about half gold and half silver, a rapid and economical parting process was found desirable; and sulphuric acid being much cheaper than nitric acid, great economy was affected by its substitution.

In the New York Government Assay Office four cast-iron kettles of a capacity of 168 gallons set in brickwork are operated, and care is taken that the alloys treated should not contain more than 8 per cent. of copper. The sulphuric acid is added in successive portions, and when the solution is complete, some waste acid is carefully poured into the kettles to help the settling of the float gold. After cooling, the solution is syphoned carefully into the silver tanks located on a lower level. The subsequent procedure has been thus described :—

To the gold left in the dissolving kettles fresh acid was added. It was then lifted out of the diluted solution of silver in cast-iron scoops (colanders), and transferred to a 60-gallon kettle, in which it was boiled in two successive charges of acid. The resulting weak solutions of silver, &c., were syphoned from the gold, and it was taken out and washed, partially by decantation in a lead-lined wooden tub (30 gallons), and afterward in a filter. It was then treated in the same way again, another kettle being used. A second repetition of the same, in which a third kettle of the same size was used, yielded gold which, when melted, fluxed, and cast into bars, was from 997 to 998½ fine. A coating of bone ash on the surface of the melted metal worked well as an absorbent of the base metal oxides, and the flux.

The weak solution of silver (strongly acid) syphoned from the gold in the three smaller kettles was used on fresh granulations in the large kettles. That which was washed from the gold by decantation and filtering ran into a leaden vat, from which it was syphoned into two tanks on a high platform on the floor below. From these (used alternately) after resting overnight, and thus surely settling any gold that might be in it, the solution was run into the silver-reducing vats already mentioned, on a lower level. The filtrate (weak sulphate of copper) from washing the silver previously treated also ran into these vats. After the strong solution of silver from the large dissolving kettles had been received in them, fresh water was often added to reduce the strength of the whole to about 20° Beaumé. When boiled (by means of steam) for about five hours, the silver was all deposited on the copper plates placed on the bottom and against the sides of the vats, a portion of the copper having gradually replaced the silver in the solution. The next morning this copper solution was syphoned into two large concentrating tanks on platforms on a still lower level, and the silver, after being scraped from the copper plates, was transferred to filters, where it was washed free from sulphate of copper. It was then melted, fluxed, and cast into bars 999 to 999½ fine.

The solution of sulphate of copper in the concentrating tanks was strengthened (by boiling) to about 40° Beaumé, and run into two of the series (12) of crystallising vats. In five or six days nearly all the sulphate of copper was deposited in the form of crystals on the sides and bottom of the vats. The mother liquor was syphoned into a reservoir in the yard for delivery to the purchaser. The crystals, after draining, in order to fit them for packing and sale, were dissolved in water and re-crystallised in another tank and series of vats on the floor below. From these vats the remaining mother liquor, after a strengthening of the first, and a second deposit of crystals, was raised by an ejector to one of the concentrating tanks on the floor above, and mixed with a fresh acid solution of sulphate of copper.

The kettles in the separating room were covered by leaden hoods connected successively with two chambers, each having an upper and a lower apartment, separated by a platform with several feet of coke on it, and five upright pipes 28 feet high, through which the sulphurous and sulphuric acid fumes given off in dissolving the silver had to pass before entering the chimney. These were all supplied with sprinklers. Water which had been made alkaline in sweep washing and mixed with other waste and fresh water in a cistern was forced by a pump through these sprinklers. A damper inserted in the last pipe at its entrance into the chimney gave the means of so regulating the draft that, while it should be sufficient to keep the fumes from escaping from the kettles into the room, it should not take them too rapidly through the apparatus to prevent their absorption by the sprinkled water and conveyance to the sewer. With the whole in good order and the water supply ample, that portion of the fumes unabsorbed and passing out of the chimney could not be felt.

The proceeds of the by-products (blue vitriol and waste acid) sold were sufficient not only to cover the cost of the copper used in reducing the silver, but also that of a large part of the acid.

For about ten years over 1,500,000 pounds of sulphuric acid have been used annually. As about the same weight of nitric acid would have been needed in the old process, and its price was, say, 5 cents per pound higher, the saving effected by the substitution of sulphuric for nitric acid in parting may fairly be estimated at over $100,000, or £20,000, per annum. As a result of these improvements the charges to depositors were reduced from time to time.

**Parting at the San Francisco Assaying and Refining Company's Works.**—Dr. F. Gutzkow, who was in former years the manager of the works in San Francisco, has published an interesting and valuable paper on these operations in the *Berichte der deutschen-chemischen Gesellschaft*, Berlin.

The usual method (he states) employed on a large scale, for a considerable number of years, for separating gold from silver, copper, and other metals, consists in treating the alloy with concentrated sulphuric acid at a high temperature, and precipitating the silver from its dilute sulphuric acid solution by means of metallic copper, the desilverised liquor being used for the preparation of sulphate of copper. This, however, is in many respects deficient, for, in the first place, there is a large bulk of water required to dissolve the rather difficultly soluble sulphate of silver, and this large bulk of liquid requires, of course, large-sized vessels to hold it. Of far more importance is, secondly, the fact that it involves the production of a large quantity of sulphate of copper, for which there is a rather limited market at low prices.

A radical change was therefore made in the method of silver refining, with a view to limiting the manufacture of blue vitriol to the comparatively small quantity of copper contained in the crude, unrefined ingots.

It was found unsuitable to substitute sheet-iron for the sheet-copper in the precipitation, owing to the fact that by the employment of the iron copper is, of course, again precipitated along with the silver, and the separation of the copper from the silver does not admit of any other method readily executed on the large scale than a repetition of the same process again.

Mr. Gutzkow found, however, that the reducing agency of sulphate of iron (green vitriol) can be applied successfully to the solution of sulphate of silver.

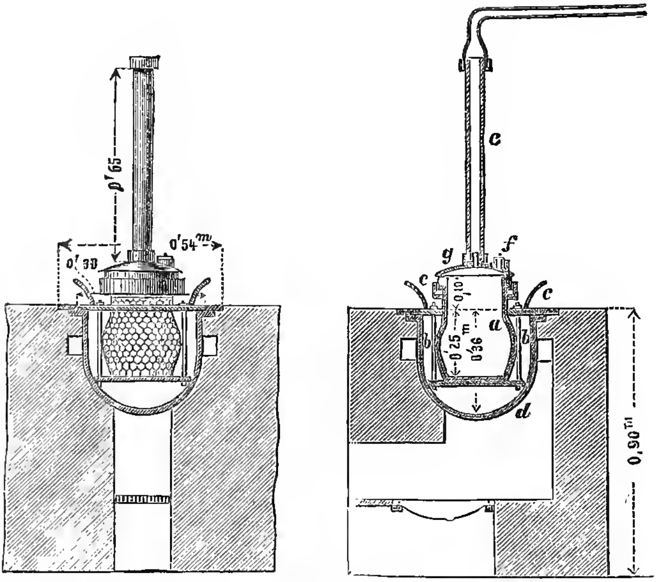
The process about to be described has been employed for a series of years in the works above-named, and by this process several thousand hundredweight of fine silver has been produced. It should be observed at the outset that it is not readily possible, for reasons which are not here further alluded to, to work on the large scale with sulphate of iron upon the solution of sulphate of silver in water. On the contrary, it is required to prepare, first, crystallised sulphate of silver, free from impurities, inclusive of metallic gold in a finely divided state, sulphate of lead, and other substances, insoluble in a solution of green vitriol.

The pure crystallised sulphate of silver is next to be acted upon by a hot and concentrated solution of green vitriol. The very hot, turbid, thickish fluid obtained by the action of boiling concentrated sulphuric acid upon the silver alloy under operation is poured into a large-sized cast-iron cauldron, containing dilute sulphuric acid at 58° Beaumé = 1.617 spec. gr., and previously heated to 110°. A small quantity of water is next added, and after the liquor (having been left at rest for a few minutes) has become clear the solution is syphoned over into another similar cauldron, so placed and arranged as to admit of being thoroughly cooled by means of cold water externally applied. For every hundredweight of silver refined, 10 cubic feet of the dilute acid (sp. gr. 1.617) are taken. The addition of water just alluded to is intended to reduce the very concentrated acid silver solution to the same density, and the quantity of water to be added may be therefore inferred from this explanation.

The addition of water is intended, however, to effect another purpose. Precipitates of sulphate of lead and sulphate of silver are formed, and the latter does not become quite permanent until, first, all the lead which was in solution is precipated; and, moreover, these heavy precipitates greatly aid the throwing down of all substances which render the liquid turbid, and especially the gold. By the means just described, a clear liquid, quite free from any lead and gold, is far more rapidly and completely obtained than by the method usually applied, viz., the pouring of the very concentrated sulphuric acid silver solution into water.

The liquid, having been cooled in the manner described, and reduced to a temperature of from 30° to 40°, is, by means of a pump, transferred again to the upper cauldron, there to be used again as acid at 38° B. At the bottom of the cauldron in which the cooling took place, the sulphate of silver will be found deposited, forming a hard, yellow-coloured crystalline crust about 2 inches in thickness. This crystalline mass is tolerably free from adhering acid, but at the deepest part of the vessel will always be found some strongly acid mother liquor, and this acid is to be again used to dissolve a fresh quantity of silver. The crystalline mass, consisting of sulphate of silver, is removed from the cauldron by means of iron shovels, and placed on the perforated false bottom of a wooden box lined inside with lead, and placed on wheels, so as to be capable of being moved from one place to another. Between the false and real bottom a tap is placed for running off liquid. Along with the crystals, and adhering thereto, is a red powder, chiefly consisting of sulphate of copper.

The next step is to run through, or, more correctly, pour over, the crystalline mass a very hot and very concentrated aqueous solution of protosulphate of iron (green vitriol). The salt of copper is first dissolved, and therefore that liquid is run off separately, afterwards to be used for the preparation of sulphate of copper. As soon as the solution which runs off begins to exhibit the pure brown colour due to sulphate of peroxide of iron, the solution is caused to run into a large and very shallow vessel, in which, on cooling, the largest portion of the silver salt is decomposed, and some metallic silver is deposited in a spongy state, which substance is collected and placed in a large filter. The greater portion of the crystalline mass of sulphate of silver which has been placed in the box is, however, converted slowly on into a dense coherent mass of metallic silver, and the reduction may be considered complete as soon as the vitriol solution which runs off has assumed the green colour it originally possessed. The metallic silver is next washed with pure hot water, then pressed in a hydraulic press, and lastly melted.



Figs. 161 and 162.—Parting Still at Oker.

The iron solution which has collected in the large-sized shallow vessel just spoken of, after having become sufficiently cool, is poured or run into a lead-lined tank, wherein some scraps of old sheet-iron are placed, and is thus again reconverted into sulphate of protoxide of iron, to be used at a subsequent operation. The small quantity of silver and copper separated in the metallic state by this last-mentioned operation is collected from time to time, and put with the crystals of sulphate of silver contained in the vessel wherein they are to be exposed to the action of the sulphate of iron solution. The copper is almost immediately converted into sulphate of copper on coming into contact with sulphate of silver. On the large scale, for every hundredweight of silver reduced from the sulphate, 20 cubic feet of green vitriol solution are required.

**Gold and Silver Parting at Oker in the Unterharz.**—The *Berg und Huttenmannische Zeitung* of 1860 (page 44) gives the *modus operandi* at these works there as follows:—The solution of the alloy is affected in the vessel *a*, Figs. 161 and 162, made of porcelain, which is 34 centimeters high, on top 235, in the middle 340, and at the bottom 283 millimeters wide, and set in an iron frame, provided with handles *c*; and this iron frame sets in an iron kettle *d*.

The cover of the porcelain vessel or retort rests in a rim with water luting; the tube *e* sets in a rim of the cover *g* and connects with a lead tube, by means of which the acid vapours are led outside the building. To prevent the retort from bursting, the vessel is covered with a wire netting and plastered over with clay. At *f* is a small opening.

There are at Oker four of these retorts in operation, each of which is charged with 6.25 kilos of doré silver granules and 12.5 kilos sulphuric acid of 66° B. The firing is done with faggots and conducted with care, as the retorts easily burst. In six hours the metal is dissolved; the solution is allowed to settle for several hours, and is then poured into lead basins where the silver vitrol solidifies. The gold which has separated is boiled again with strong sulphuric acid, washed with water, till the wash waters show no more any silver reactions, is then dried in a porcelain dish, and in quantities of 5 kilos melted in plumbago crucibles; the same is on an average 985 fine.

The silver vitriol is carefully dissolved by heating in lead basins, and then precipitated by means of sheet copper—the cement silver is washed, pressed in conical moulds, well dried, and melted in plumbago crucibles with the addition of some nitrate of soda. For precipitating 100 parts silver 30 parts copper are required.

**Mint Parting with Nitric Acid.**—In the next chapter, on Assaying, I explain that when gold has to be separated from silver, the silver must be present in the alloy in the proportion of at least 2½ to 1, otherwise the gold will so cover the silver as not to permit a perfect separation of the metals even if boiled for some time in nitric acid. The usual proportion in practice is 3 silver to 1 gold. Therefore, if we have a gold alloy weighing 1,000 ounces, 800 fine in gold, this will require—supposing the other 200 parts to be silver—$(3 \times 800) - 200 = 2{,}200$ ounces of fine silver, which has to be melted together with the 1,000 ounces of alloy. After fusion, the molten mass is granulated by being poured into a large vat filled with cold water, the object being to reduce the metal into small particles of various shapes, so as to offer a large surface to the acid, which quickens the operation of dissolving.

The silver granules can be dissolved in glass retorts, with openings leading to a high chimney, so as to draw off the copious nitrous acid fumes. The retorts are placed on a sand bath. The gold remains behind as a brown powder, which after the nitrate of silver solution is drawn off into large stoneware tanks, is washed, dried and melted in Morgan's black-lead crucibles with borax and saltpetre.

To the nitrate of silver solution is added a concentrated solution of common salt—chloride of sodium—which throws down the silver as a white curdy precipitate of chloride of silver. After drawing off the acid this chloride of silver is now reduced, by the addition of metallic zinc and water acidulated with sulphuric acid, to metallic silver, which is very light and spongy. It is collected in filters, washed for some time so as to free it from any soluble salts, and compressed under a powerful hydraulic press into cylindrical slabs, which are dried in ovens. After drying they are heated and assume a metallic lustre, and are now ready for the melting-pot. This operation gives silver nearly 1000 fine.

For making it into coin 900 parts of this silver, after being cast into ingots, are melted with 100 parts of refined copper, which makes the standard American alloy for the silver coin.

**Refining of Brittle Gold with Chlorine Gas.**—Mr. F. B. Miller, assayer in the Sydney branch of the Royal Mint, has invented a method of refining gold which is perfectly suitable for removing such metals as lead, arsenic, antimony, and bismuth. These alloys, if only forming the $\frac{1}{1920}$th part of the whole mass, render the metal brittle, and totally unfit for coinage and the arts.

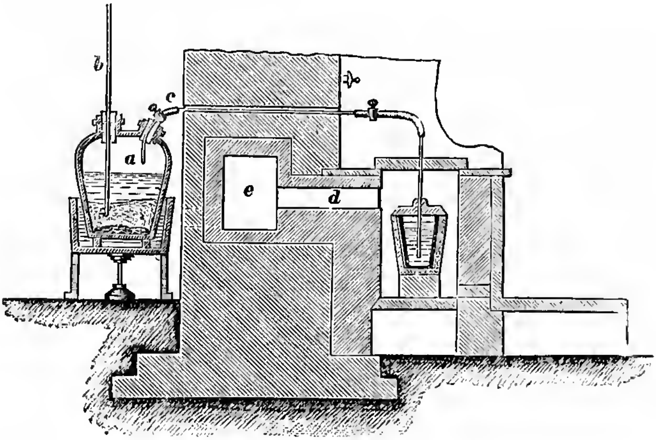


FIG. 163.—CHLORINE REFINING APPARATUS.

The process consists in passing chlorine gas through the metal, while in a molten state, for about four or five minutes. The base metals are thus converted into volatile chlorides which escape; and should there be any silver present it is converted into chloride of silver, which floats on the surface of the metal. To avoid loss of silver, an earthenware crucible should be employed which has been previously dipped in a solution of borax; borax should also be introduced into the crucible with the gold.

The furnace, Fig. 163, contains a special crucible made at the Battersea Works in London, which is capable of holding 600 to 700 ounces of gold, and is covered with a cover having two small holes—through one of these passes the clay pipe which conducts the chlorine gas, and which reaches to near the bottom of the crucible, while the other opening serves as a vent hole for the superfluous chlorine and the volatile chlorides.

In the Sydney Mint there are five furnaces, which are fed from the chlorine gas generator *a*, having a capacity of 120 litres, which is placed in a basin of galvanised sheet iron filled with water; *b* is a glass tube which connects with the acid reservoir furnishing the hydrochloric acid; *c* is the educt pipe; *d* the flue; and *e* the chimney. The educt pipe *c* branches off and connects by means of indiarubber pipes with the other furnaces. The gas generator is of earthenware.

Mr. Miller has described his process as follows:—

"There is no recorded instance of gold having been found in an absolutely pure state. Every natural alloy of native gold contains more or less silver; and in almost all bullion resulting from the melting of alluvial gold the portion that is not gold consists chiefly of silver, with only a very small portion of foreign metals, usually copper and iron, with occasionally a little lead or antimony, and sometimes a trace of tin, iridium, &c. This, however, though true generally, is not always the case with gold obtained from quartz veins by amalgamation, as the mercury occasionally reduces and takes up other metals as well as the gold, which appear in the bullion on melting. Where silver is associated in varying proportions with the gold the question will naturally arise, Is this argentiferous character in any way connected with the geological structure of the district? It is a fact, and certainly a very curious one, whether it arises from accidental causes, or whether it may hereafter be traced to peculiarity in the rocks whence the gold of the different districts is derived, that its quality or fineness deteriorates the further north we go in New South Wales; in other words, it contains more silver and less gold.

"Thus the average fineness of Victorian gold is about 23 carats, that is to say, it contains about 96 per cent. of gold and 3½ of silver, with ½ per cent. of base metals; while, on passing north, we find the average fineness of New South Wales gold to be only 22 carats 1⅞ grains, or to contain 93½ per cent. of gold and 6 per cent. of silver. On going still further north to the colony of Queensland, the average fineness is little more than 21 carats (considerably below standard), or it contains 87¼ per cent. of gold and 12 per cent. of silver, and some contains only 85 per cent. of gold.

"These are averages only. It is not to be supposed that there is a regular and consecutive diminution in fineness with every degree of latitude we go north. There are exceptional localities in the north of this colony where gold is found of a high degree of purity, as at Rocky River, where it is over 23 carats fine, or 96 per cent. Possibly at a future time our geologists may be able to throw some light on these curious facts, and the exceptional cases may then even help in explaining the apparently general rule.

"According to the published returns, 6,820,198 ounces of gold have been received for coinage in the Sydney Mint between its establishment in May, 1855, and December 31, 1868. The average assay of this quantity would be about 943; in other words, it contained 94⅓ per cent. of gold, 5 per cent. of silver, and ⅔ per cent. of base metals. Allowing an average loss of 2 per cent. in melting the gold dust, there would remain after smelting 6,683,795 ounces of gold bullion; and as the silver it contained amounted to 5 per cent. of this quantity, the gross amount of silver in the gold received for coinage was 334,190 ounces, being at the rate of 24,720 ounces per annum.

"The average proportional quantity of silver contained in the gold arriving in Sydney is at present very much greater than that given above, owing to the large amount of silvery gold now being found in various districts. Most of the silver thus naturally present in the gold has hitherto been lost to the colony, owing to the expense in Sydney of the acids, &c., necessary for its extraction by any of the usual methods of refining, which left little, if any, margin of profit on the operation. It therefore seemed desirable to introduce a more economical system, which is here described.

"It is well known that chlorine readily enters into combination with almost every known metal, the action in some cases being so violent as to be attended with vivid combustion. Many metals, such as lead, tin, zinc, and antimony, when introduced into this gas, even at ordinary temperatures, combine with it, forming highly volatile chlorides. The two latter, if in a state of fine division, burst into flame on being placed in an atmosphere of chlorine. Copper also exhibits spontaneous combustion under similar circumstances, but the resulting chloride formed is only slightly volatile. Silver immersed in chlorine gas at ordinary temperatures slowly unites with it, forming chloride of silver; but if the gas be passed over it while red hot, the action is much more energetic, the compound formed being more volatile than the chloride of copper, but much less so than those of lead, tin, zinc, or antimony.

"The method of refining now to be described is based upon these facts. It consists simply in passing a current of chlorine gas through the gold *while in a melted state*, which is easily done by thrusting into the molten metal a small clay tube connected with a stoneware vessel in which chlorine is generated. The chlorine on coming in contact with the silver in the molten alloy at once combines with it, forming chloride of silver, which, being of less specific gravity, rises to the surface of the melted gold, while the latter remains in a purified condition beneath. Chloride of silver has always been considered a somewhat volatile substance, and under circumstances such as those here described it was naturally supposed that it would be either sublimed in the flue or escape entirely up the chimney; but in practice it is found that the volatility of the chloride is not nearly so great as might have been anticipated, and that if it is coated with a layer of fused borax it may be kept melted at a high temperature without any very material loss. The furnace required for the operation is the ordinary 12-inch square gold-melting furnace, the principal points to attend to in its construction being—

"1. That the flue should be as near the top as possible, so as to allow of the crucible standing high up in it without being cooled by the draught.

"2. That the furnace itself should not be too deep, so that when the pot is placed in the fire the bottom of it may not be more than 3 in. above the bars. The covering of the furnace should consist of two fire-tiles, $7\frac{1}{2}$ in. wide and 15 in. long, one of which should have a long slot or hole in its centre for the clay chlorine pipes to pass through. An iron cover will not answer, as it soon becomes much too hot for convenient working.

"The crucibles in which the refinage is performed should be French white fluxing pots* (made by De Ruelle, Paris); ordinary black-lead pots will not answer, owing to the reducing action they exert on the compounds formed. To prevent the infiltration of the very fluid chloride of silver into the pores of the clay pots, they are prepared by filling them with a boiling saturated solution of borax in water, which is allowed to stand in them for ten minutes and is then poured off, the crucibles being afterwards set aside to dry; the borax forms a glaze on the inner surface of the crucibles when they become hot in the furnace. When used for refining these French clay crucibles are placed within black-lead pots as a precaution against loss should the former crack, which, however, seldom happens. The crucibles are covered with loosely-fitting lids with the requisite holes bored through them for the passage of the clay chlorine pipes, &c. Ordinary clay tobacco-pipe stems, from 17 to 22 in. long, have been found to answer well for the purpose of passing the chlorine gas through the melted gold. Of late a pipe made in London to order, $\frac{1}{2}$ in. in diameter, 22 in. long, and $\frac{5}{16}$ in. bore, has been found to answer all requirements. The chlorine generators should consist of the best glazed stoneware acid jars, capable of holding from ten to fifteen gallons, and furnished with two necks. One of these openings should be stopped with a sound cork or vulcanized india-rubber plug if

---

* Special crucibles for the purpose are now manufactured at the Battersea Works, London, for the Sydney Mint.—M. E.

obtainable, through which should pass tightly two glass tubes—the eduction tube and the safety or pressure tube, the length of the latter 8 or 10 ft., and the former being a few inches spliced where necessary by means of vulcanized india-rubber tubing. The other opening, intended for introducing the oxide of manganese, &c., should be closed with a leaden plug, covered with a short piece of india-rubber tube by way of a washer, and well secured.

"Each generator should be charged with a draining layer of small quartz pebbles, down nearly to the bottom of which the pressure tube should extend. On this layer should be placed from 70 to 100 lbs. weight of binoxide of manganese in grains about ¼ in. cube, sifted from powder; this quantity will be sufficient to effect many refining operations, and will obviate the necessity of repeated dismantling of the apparatus. Each generator should be suspended to about half its height in a galvanized iron water-bath.

"The chlorine gas is produced when required by pouring common hydrochloric acid, specific gravity 1·15, down the safety tube, the apparatus being warmed by means of gas burners beneath the water-baths. The gas is conveyed from the generators by means of a leaden pipe fitted with branches to supply the several furnaces, all intermediate connections being formed by means of vulcanized india-rubber tubing, which, if screened from the direct radiation from the fire, stands the heat well, even immediately over the furnaces. All joints between the various pipes and india-rubber tubes are easily secured and rendered perfectly gas-tight with a cement consisting of a thin solution of india-rubber in chloroform.

"Screw compression clamps on the india-rubber tubes give the means of regulating the supply of gas as required, and enable the operator to shut off entirely so soon as the refining is over. The chlorine then having no means of escape accumulates in the generator, and soon forces all the acid up the safety tube into a vessel placed above to receive it, and the acid no longer acting on the oxide of manganese, the supply of gas of course ceases. Two such generators as are here described, and three ordinary gold-melting furnaces, have been found capable of refining daily about 2,000 ounces of gold, containing about 10 per cent. of silver, between 9 A.M and 2 P.M.

"The French crucibles, say size 17 or 18,* duly prepared with borax, having been placed in the cold furnace and slowly and carefully heated to dull redness, the gold, from 600 to 700 ounces to each crucible, is introduced and the fire urged until the metal is melted, the necessary generation of chlorine having meantime commenced by the introduction of a little hydrochloric acid poured down the safety tube into the generators.

"As soon as the gold is melted from 2 to 3 ounces of borax in a state of fusion is poured upon its surface. If the borax is added sooner it acts too much on the lower part of the pot, and if thrown in cold is apt to chill the gold. The clay pipe which is to convey the chlorine to the bottom of the melted gold is now introduced. It is necessary to carefully heat the lower portion of this pipe for some ten minutes before introducing it into the molten gold, or it is apt to split. At the moment of its entering the melted gold the screw compression clamp is slightly loosened so as to allow a small quantity of gas to pass through it, and thus prevent any metal rising and settling in the pipe, which is then gradually lowered to the bottom of the molten gold, where it is kept by means of a few small weights attached to the top. The compression tap is now quite relaxed, and the gas is heard bubbling up through the melted metal, which it does quietly and without projection of globules from the pot.

"Sufficient hydrochloric acid must be added to the generators from time to time to keep up a rapid evolution of chlorine. A rough general rule is to allow one imperial quart of acid of 1·15 specific gravity to every 10 ounces of silver in the alloy operated on. The column of liquid in the safety tube, acting as it does like a barometer, affords a ready means of knowing the pressure in the generator, and of judging of the rate of production of the gas, as well as at once showing by its fall if anything irregular has occurred, such as a leak or a crack of the chlorine

---

* See foot note, p. 523.

pipe or pot. From 16 to 18 in. in the safety tube correspond to and balance 1 in. of gold in the refining crucible. When the chlorine is first introduced into the melted gold, a quantity of fumes are seen to pass up from the holes in the crucible lid; these are not chloride of silver, but the volatile chlorides of some of the baser metals, and they are especially dense when much lead is present in the alloy under treatment, forming a white deposit on any cold substance presented to them. After a time, longer or shorter according to the impurities in the gold, the fumes cease. So long as any decided quantity of silver is present in the molten gold, the whole, or nearly the whole, of the chlorine is absorbed, little, if any, appearing to escape and to be thus wasted, and it is found that the better the supply of chlorine the quicker is the operation.

"It is a curious circumstance that though, in toughening with corrosive sublimate, this substance is only thrown on the surface of the melted gold, yet the whole mass is toughened by its action. It seems essential, in using chlorine, that the gas should pass to the very bottom to effect a complete refinage.

"As soon as the operation is nearly over fumes of a darker colour than those observed at the commencement make their appearance, and the end of the refinage is indicated by a peculiar flame or luminous vapour of a brownish yellow colour, occasioned by the free and now waste chlorine escaping, which can be seen on removing a small plug which fits into a hole in the lid of the pot. This, however, of itself is not a sufficient indication; the process is not complete until this flame imparts to a piece of white tobacco pipe, or similar substance, when held in it for a moment, a peculiar reddish or brownish yellow stain; so long as it gives any other colour the refinage is unfinished.

"When these appearances are observed, usually for gold containing about ten per cent. of silver in about an hour and a half from the introduction of chlorine, the gas is shut off and the pots removed from the fire, the white crucible lifted out of the black one, and together with its contents allowed to stand several minutes, until the gold becomes cool enough to set or solidify. The chloride of silver which remains liquid much longer is poured off into iron moulds. The crucible is then inverted on an iron table, when the still red-hot gold falls out in the shape of a cone; this is slightly scraped and then thrown hissing into a concentrated solution of common salt, to free it from any adherent chloride of silver.

"An alloy containing originally 89 per cent. of gold, 10 per cent. of silver, and 1 per cent. of base metals, will yield on an average a cake of chloride weighing, with a little adherent borax, 15 ounces for every 100 ounces operated on.

"It is necessary very carefully to dry and heat the moulds into which the chloride of silver is poured, as the slightest moisture causes the latter to be violently dispersed while red hot, to the great risk of the bystanders.

"With ordinary care this will never happen, but attention is called to the point as a very deliquescent chloride of iron is apt to form on the moulds.

"The gold is now fine, and simply requires re-melting into ingots.

"As before stated, it is found that all these operations can readily be performed, and about 2,000 ounces refined per day in three common melting furnaces, in five hours; 98 per cent. of the gold originally contained in the alloy operated on is then ready for delivery. The other 2 per cent. remains with the chloride of silver, partially in the metallic state and partly in a state of combination with chlorine, and probably with silver.

"To free the chloride of silver from this combined gold it is melted in a boraxed white-pot, with the addition of from 8 to 10 per cent. of metallic silver, rolled to about one-eighth of an inch thickness. The chloride of gold is by this means reduced at the expense of the metallic silver, chloride of silver being formed, while the liberated gold sinks, and melts into a button at the bottom of the pot. As soon as the whole is thoroughly melted the pot is removed from the furnace and allowed to stand about ten minutes, and the still liquid chloride of silver is then poured into large iron moulds, so as to form slabs of convenient thickness for the next operation, that is, the reduction to the metallic state.

"After the fusion of the chlorides a small quantity of a curious spongiform substance adheres to the sides of the crucible used, probably consisting of sub-chloride of silver, but since it always contains a little gold care has to be taken in pouring off the fluid chlorides to prevent this auriferous sponge from falling out and mixing with them.

"The slabs of chloride of silver are reduced without difficulty by plates of wrought iron or zinc in the usual way. Besides the separation and recovery of the silver as above described, another useful end is gained by this process.

"A very large proportion of the gold of Australia is more or less brittle, an effect generally due to the presence of small quantities of lead or antimony, rendering the bullion quite unfit for coinage or manufacture until it has undergone some process to render it tough.

"The methods usually employed for this purpose are either fusion with nitre and borax, melting with oxide of copper, or the addition of corrosive sublimate (bichloride of mercury) to the melted gold. The two former of these plans are troublesome, from the corrosive action they exert on the crucibles. and the last, namely, the employment of corrosive sublimate, which is that usually employed, is most objectionable, from the dense and highly injurious fumes evolved.

"The passage of chlorine gas through the melted gold is found to effect the complete toughening of the metal by the elimination, as volatile chlorides, of the materials which render it brittle, while the evolution of the deleterious mercurial fumes is avoided.

"In the metallurgic treatment of the precious metals some loss is always sustained, but that incurred in the process here described is found very small.

"The average loss of gold in operating hitherto has been found to amount to 19 parts in every 100,000 of alloy treated, which is considerably less than would be met with in toughening an equal amount of gold with corrosive sublimate in the ordinary manner.

"The loss of silver has amounted to 240 parts in every

100,000 of alloy operated on (containing originally, say, 10 per cent. of silver). There is no doubt that a considerable portion of both these losses would be recovered on further treating the pots and ashes remaining after the operation; and it is found that, as manipulatory skill is acquired, the proportional loss of silver appears to be decreasing. In refining on the large scale, gold containing 10 per cent. of silver, the cost of the operation in Sydney is about five farthings per ounce. The fineness of the gold produced by this process varies from 991 to 997 in 1,000 parts, the average being 993·5 or 23 carats $3\frac{3}{8}$ grains. The remaining $6\frac{1}{2}$ thousandths are silver, and this compares favourably with any of the previously known practical processes, none of which leave less silver than this in the resulting fine gold.

"If the refined gold be subjected to a re-refinage by chlorine, the amount of silver left in it can be reduced to 0·2 per cent., just as in the refinage by the ordinary sulphuric acid process, the same result can be obtained by subjecting the refined gold to a further refinage by bisulphate of potash. For practical working, however, this would probably never be attempted.

"The silver resulting from this method of refining is tough, but its quality varies somewhat according to the gold originally operated on; if the alloy treated contains much copper, the greater part of this remains with the resulting silver, but the other metals are nearly all eliminated.

"The fineness of the silver hitherto obtained has varied from 918·2 to 992 in 1,000 parts, the average being 965·6. Analysis of the silver resulting from the refinage of gold known originally to have contained, amongst the base metals in the alloy, copper, lead, antimony, arsenic, and iron, gave the following results:—

| | |
|---|---|
| Silver | 972·3 |
| Copper | 25·0 |
| Gold | 2·7 |
| Zinc and Iron | Traces. |
| | 1000·0." |

**Iridium in Mint Deposits.**[*]—The small grains of osmiridium, platiniridium, and allied metals of the platinum group, found in many deposits of gold (sometimes, though rarely, in silver), are in mint usage designated by the general term iridium. These grains vary greatly in form, in the proportions of the several metals, and in the amount contained in deposits at different times. Often of definite granular shape, and large enough to be useful in the arts, they appear at other times as scales and coarse powder quite as frequently as very fine powder, the detection of which in gold by inspection is not an easy matter. The ordinary appearance of iridium in a bar of gold is in specks or clots distributed through the metal, but not alloyed with it. It is most plainly marked, usually, on the top of the bar, or that last poured from the crucible.

The quantity of this metal received at the mints in deposits is not absolutely large. As compared with the amount of gold received it is indeed quite small. But in certain specific cases, and taking one period with another, the amount often seems considerable. The history of this metal in mint experience shows that it became specially a matter of consideration by mint officials on the reception of the early shipments of gold from California.

In the first two years of the establishment of the Assay Office (1854—56) the proportion of osmiridium in Californian gold was half an ounce to a million of dollars, but subsequently the average was seven or eight ounces to the million. In the experience of the same office during a period of eighteen years (1865—81) an amount of residuum slightly exceeding 200 ounces of iridium was recovered from various deposits. The total amount, therefore, was not large.

The annoyance, however, caused by the presence of even small quantities of this metal in the gold is considerable. Separable in melting and assaying only by special treatment, and liable to elude casual observation, the risk to an accounting

---

[*] For the matter under this heading the author is indebted to Mr. B. J. Martin, melter and refiner of the United States Assay Office, New York.

officer of excessive debit in weight and fineness is not small. This is apparent when it is remembered that the substance is not an alloy, but is mechanically mixed with the precious metal, and of course, if undetected, is charged as gold.

Yielding no more readily to the action of single acids than to fire, it passes substantially unmodified through the acid refinery, with constant liability to issue in the refined gold, to become there an element of vexatious annoyance to the manufacturer of jewellery, plate, and the like—the practical effect of it in such case being to render impossible the production of a perfectly smooth and polished surface. A costly bit of work, brought to the point of completion, has often had to be cast aside because of the discovery, in the last processes, of grains of iridium, so small as to need a magnifying glass to detect them, yet large enough to vitiate the whole work.

To prevent, as far as possible, these results, great care is first taken at the mints in the inspection of the gold after melting. A double scrutiny by the assayer, first of the bar and subsequently of the cornet or residue from the assay sample, is supplemented by a similar inspection of the bar by the melter and refiner when he receives the metal from the superintendent. This accomplishes two results: the non-distribution of metal containing iridium with that free from it, and its isolation for separate treatment at the time or subsequently. Various methods have been used at the different mint institutions for the treatment of deposits thus affected. The problem is twofold—first, to promptly pass on the matter as it affects the depositor's interest (that he may not wait unduly for payment), while at the same time protecting the interest of the melter and refiner (that he may not be charged with what is not gold); secondly, to separate the obnoxious metal before sending the gold containing it to the refinery, where, once entered, it would contaminate all metal and all apparatus, and issue, in greater or less amount, in the fine bars of gold and silver.

The separation of the iridium is effected during melting, by allowing the deposit to stand for some time in a molten condition undisturbed. The grains of iridium slowly settle to the bottom of the crucible, and by careful pouring the mass of the gold is obtained free, while the king at the bottom, containing the iridium and some gold and silver, can be treated separately. This method is most successful when the grains are large and well defined. In other cases the assay cornets, or small portions of the melt containing iridium, will be dissolved, and from the proportions found the amount in the whole deposit may be calculated. One objection to this method is the lack of uniformity in the distribution of the iridium through the mass of the metal.

Yet another method, practised for many years at the Assay Office in New York, is to melt with the bar containing the iridium two or three times its weight of silver, and, after stirring, to allow the melt to stand for some time. By the addition of silver to the gold the relative gravity of the iridium is largely increased; and in the operation, if properly conducted, very fine powdery particles of it will settle with the large grains to the bottom of the pot. Then, with care in pouring, the bulk of the melt will be free from that substance.

The remains in the crucible--or the king—is subjected to another remelt, with the addition of more silver, until the iridium is collected in a small button. The kings from a number of settlings are allowed to accumulate until a convenient opportunity, when they are operated on in mass. The silver being dissolved out by nitric or sulphuric acid, the residual grains are treated with aqua-regia, and the gold and platinum (if present) are precipitated. The final grains are washed and dried.

It is an item of interest, however, that from a little more than 200 ounces of these grains, after the removal of the silver, there was obtained about 52 ounces of sponge platinum.

# CHAPTER XVIII.

## *THE MELTING AND ASSAYING OF GOLD.*

MELTING AND ASSAYING—Fusion with Borax—Melting Furnaces—Assaying of the Gold Ingot—Several Processes of an Assay—Weighing the Assay Piece—Valuation of Gold Alloys—Preparation of the Assay Piece—Cupellation—Parting of the Assay—Professor Roberts-Austen on the Parting Process—Parting Assays—Gold containing Oxidizable Metals—Assaying Gold Quartz—Assaying by Scorification—Forms of Cupels—Weighing the Button of Precious Metal.

**Fusion with Borax.**—When the gold is received from the gold washer, in the shape of gold dust, or when it is separated from its ores by amalgamation or otherwise, it has to be subjected to a series of metallurgical operations to obtain it in a state of perfect purity. The first of these is that of fusion with borax, and casting the metal into ingots, by which it is obtained in a convenient form, and freed at the same time from earthy and other impurities. For this purpose the gold dust from the washings, or the spongy gold obtained by amalgamation, is mixed with a little dried borax, and is introduced into a black-lead crucible, Fig. 164, previously heated in a common melting furnace.

FIG. 164.—PLUMBAGO CRUCIBLE.

The heating of a *new* crucible, made of plumbago or black-lead, is called the *annealing*, and is accomplished by placing the vessel bottom-side up on the grate of the melting furnace, and building up a slow fire around it until the crucible is dark red. On cooling it will be found to have lost its black colour, and to be of a greyish tint. Several crucibles are usually annealed at the same time. This simple operation is necessary, as otherwise, if a new crucible is suddenly heated, it will fly to pieces and cause loss of gold as well as trouble in collecting the precious metal.

The fuel generally used in mining regions for melting of gold is charcoal, but where coke is available it is preferable.

**Melting Furnaces.**—A convenient furnace is shown in Fig. 165. It consists of two cylinders of sheet iron, placed one within the other. A represents the outer and B the inner cylinder—the latter lined with a coating of fire-clay about one inch thick. Both cylinders are provided with a bottom, and are fixed together at the top, air-tight, by a horizontal hoop or flat ring of metal, E E, in such a way as to leave an equal space, C C, between their sides and bottom. The interior cylinder, B B, is pierced at about the middle of its depth with eight holes, D D, which pass through the lining of fire-clay, and all point to the centre of the furnace, where the crucible, H, is placed on a piece of fire-brick, kept in its position by a little fire-clay, and surrounded with fuel. For a small furnace of this kind charcoal must be employed as fuel, and should be broken into pieces about as large as a walnut. The air is blown into the opening, F, by means of a double-action bellows, or a simple rotating fan may be used. By this means the air is driven in a steady equable current through the holes, D D, into the cavity of the furnace. The heat which may thus be produced with a furnace having an internal diameter of only a few inches is so intense as to be capable of melting manganese, or several ounces of cast-iron, with the greatest ease.

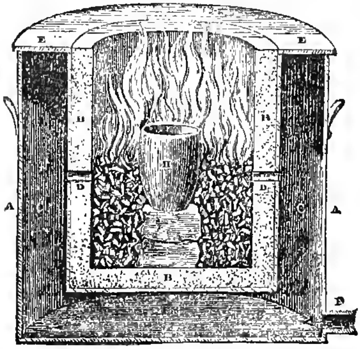


Fig. 165.—Melting Furnace.

A very convenient furnace for the fusion of small quantities of gold bullion up to 100 or 150 ounces, and for the melting of ore assays, is shown in Fig. 166. Being made in sections, it is easily packed, and is of simple and durable construction, being built up of fire-clay and strengthened with hoop-iron bands. In the one which I use in my laboratory I can place with ease four Battersea No. 10 clay crucibles for ore assays. Owing to its portability this furnace is specially adapted for use in distant mining regions. It is made in three sizes, namely: B, 10 inches diameter, 20 inches height, 67 lbs. weight; C, 11½ inches diameter, 22 inches height, 90 lbs. weight; and D, 13½ inches diameter, 26 inches height, 144 lbs. weight. In the illustration, *a* is the socket to fix on iron chimney; *b* the door for putting in crucible, and for cooking; *c* the crucible; *d* the door for regulating the draft; *e* the iron grate.

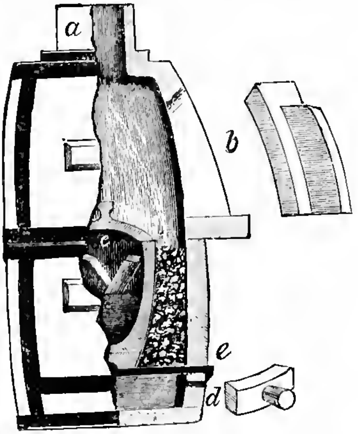


FIG. 166.—PORTABLE WIND FURNACE

FIG. 167.—INGOT MOULD.

A melting furnace can also be built up of ordinary bricks like an ordinary wind furnace, but it is advisable to line the inside with good firebricks or firestone.

I have used to advantage circular sheet-iron furnaces lined with firebricks, having, a few inches above the grate bar, an opening closed with a door, which opening was filled with removable firebricks. This opening served for the insertion of a muffle; and it will be readily understood that such a furnace is economical to use, as when the ingot is cast, the muffle, is inserted in the red-hot furnace, and in a few minutes the process of cupellation can be carried out in the same furnace. After the cupellation is finished, the fire is allowed to go down somewhat; the cover is taken off; and a sand bath placed on top of the furnace in its place. The parting of the cornets can be effected in this manner, and the melting-furnace can be made to answer various purposes. It is a desirable apparatus, therefore, for a distant mining camp, where laboratory facilities are generally lacking. It is also advisable to build over the flue leading to the chimney a permanent sand bath of bricks.

After the fusion of the metal, a slag, consisting of the grosser impurities, accumulates on the surface; this, when too fluid, must be thickened by the addition of a little bone-ash, and then skimmed off. The skimming of the slag is effected by means of a ¾-inch iron rod having a flat coil at its end. This flat coil is heated, and the surface of the red-hot fluid slag is slightly touched with it, when the slag will stick to it, and on withdrawing the skimmer from the pot, the flat coil with the adhering slag is pressed on a stone slab or a piece of iron. When the slag has hardened, the skimming is continued until the molten metal presents a clean surface. During the skimming care must be taken not to let the slag on the skimmer get too cold, as then there is danger of some gold adhering to the slag, if the latter comes in contact with the metal. A little practice will enable the assayer to carry out this operation with nicety.

The metal is now cast into ingots by being poured into iron ingot moulds (shown in Fig. 167), previously warmed, and oiled by wiping the internal surface with a piece of tow dipped in sweet oil.

The removal of the melting pot from the furnace with the crucible tongs (shown in Fig. 168), as well as the pouring the

FIG. 168.—CRUCIBLE TONGS.

molten gold into the mould, requires a certain amount of practice and dexterity, as the operation must not allow time enough to chill the gold in the pot when once it is out of the fire. When the quantity of gold is large, the molten mass ought to be stirred in the pot when fused, and just before pouring. An iron rod is heated at one end to bright redness and then introduced into the molten mass and stirred gently, which insures a perfect uniformity of the ingot when cast.

The black-lead crucibles which have been used in a series of melting operations retain some globules of gold, and these are preserved with a view to the subsequent extraction of the precious metal from them.

When gold contains certain impurities, if they consist of easily oxidizable metals, these are generally removed by the addition of saltpetre in the last stages of the melting operation, which generally exerts a powerful oxidizing action on some of the so-called base metals.

A strange accident once happened to me with a quantity of gold coming from a quartz mill. The bullion contained a rather heavy proportion of iron pyrites, and I added some saltpetre as a flux. Wishing to ascertain if the metal had started, I tried to remove the cover from the crucible, when some charcoal dust fell into the pot. The result was a rather sharp explosion; as inadvertently I had produced a gunpowder mixture by means of the sulphur in the pyrites and the nitrate of potash, which (including the charcoal) were all mingled in a state of incandescence. Needless to say, the melting pot was blown into fragments, and the gold scattered about.

The gold bullion which comes from quartz mines carrying iron pyrites, if not carefully cleaned, will contain some of these, especially that portion which comes from the clean-up pan. In such cases the melt can be proceeded with in the ordinary manner without skimming the slag. After casting the ingot, a layer of matte is found below the slag, which can be hammered off from the top of the bar, but as some of it will stick to the ingot it should be remelted with some saltpetre.

The matter is pulverised, roasted, and fused down with litharge and fluxes, and the resulting button cupelled. It will generally be found that this matte is rich in gold.

It very seldom happens that black-lead crucibles of well-known makers are not sound, but I have come across melting-pots with "pin-holes" in them, which are difficult to detect before using, and when the crucibles are in use the metal will be found running out after it is too late to prevent the waste. As a precautionary measure it is always advisable to have in the bottom of the ash-pit a black-lead dish, such as can be easily cut out from the lower part of a large crucible.

The molten gold in the pot has a greenish lustre, but when the gold is impure, spots or rings in constant motion are visible on surface. These are due to the oxides of the base metals. In such case some more borax or saltpetre should be added, and if the impurity is lead, some bone-ash, which will absorb the lead as an oxide.

**Assaying of the Gold Ingot.**—The ingots obtained in ordinary metallurgical operations are hardly ever of perfectly pure gold—namely, 1,000 fine (or 24 carat). The nearest approximation to this standard is the gold extracted from the Mount Morgan Mine, in Queensland, and the ingots resulting from sulphurets treated by chlorination. Usually the ingots are alloys of gold and silver, contaminated with copper or lead. In placer mining operations very often the ground is strewn with old tin cans and other odds and ends of metal. These materials being washed through the sluice-boxes, it not infrequently happens that the quicksilver gets contaminated with some tin, or antimony, or a little iron, which entering the ingot will produce a base bar. Occasionally, also, the gold contains platinum, iridium, or rodium, which vitiate the assay results.

**Several Processes of an Assay.**—The assaying of gold means its quantitative determination in an alloy. The following metallurgical method is adopted in the Royal Mint and other establishments where assaying is regularly practised, it is not only more expeditious than the wet analysis, but admitting also of a greater number of assays being simultaneously conducted. The method consists of the following processes :—

(1.) The accurate weighing and preparation of the assay piece or sample. In case of gold ingots two of the corners are chipped off with a chisel, one from the top and one from the bottom.

(2.) Cupellation, or removing the copper and other base metals by oxidation or absorption.

(3.) Parting by inquartation, or separation of the silver from the gold by dissolving the former metal in nitric acid.

(4.) Final weighing of the resulting "cornet," and the applying of necessary corrections.

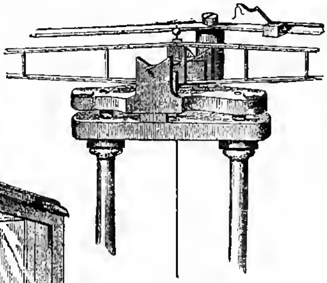
**Weighing the Assay Piece.**—It is evident that, as the amount of metal to be operated upon is extremely small, it is of the utmost importance that the weight of the assay piece should be very accurately determined, both before and after the process. For this purpose very delicate balances are used. The form most generally approved[*] is shown in Fig. 169, which is capable of indicating $\frac{1}{20000}$ part of an assay pound of half a gramme (7·716 grains). The balance employed must be extremely sensitive, sure, and quick—qualities which require a very light and carefully constructed beam. The beam in the balance shown is 10 in. long, and usually weighs about 125 grains. The scale pans rest in stirrups of palladium, and can be removed by means of forceps. The supports at the centre and ends are shown on a large scale in Fig. 170. Each stirrup hangs from two steel points resting in agate cups, and the agate knife edge at the centre does not come in contact with the agate palates until the supports have been removed from below the scale pans by means of the handle moving in a vertical slot in front of the balances.

Two systems are employed for expressing the composition of gold and silver ingots, and the weights used in "weighing

---

[*] As made by Becker, of New York, and by Oertling, of London, and other firms.

in" and "weighing out" the assays are usually so prepared that the results obtained directly represent the composition, rendering calculations unnecessary. Thus when reports of gold assays are required on the trade system, the weight of metal employed (varying from 5 to 16 grains)—called the "assay pound"—is divided into twenty-four parts, called carats, and each of these is subdivided in carat grains and eighths of carat grains.



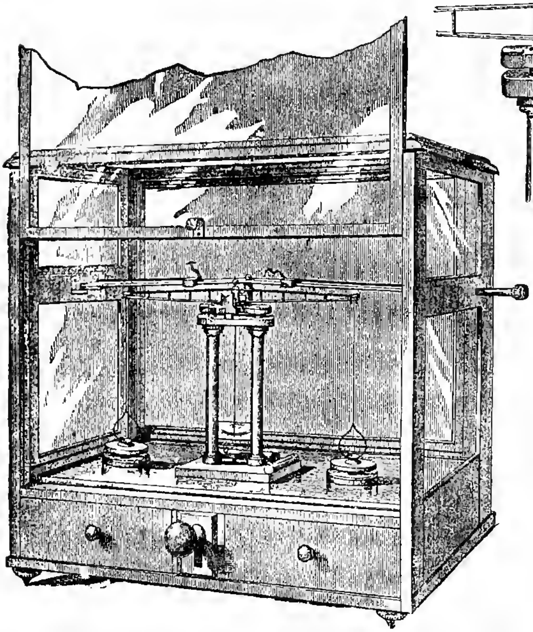


FIG. 169.—ASSAY SCALES.

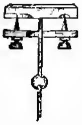


FIG. 170.—STIRRUPS OF SCALES.

If, however, assays are required on the millesimal system, the weight taken is called 1,000, and in weighing the cornet after the completion of the assay subdivisions of this 1,000 are employed.

The 1,000 weight, taken as the assay pound, is generally ½ gramme, and is the centre piece of an ivory box, containing all the subdivisions on this weight on the millesimal system.

**Valuation of Gold Alloys.**—In England the standard of the alloys of gold is calculated in fractions of unity expressed in carats. Perfectly pure gold is taken as unity, and this is assumed to consist of 24 fractional parts or carats, each of which is divided into 4 imaginary grains, and these are again subdivided into eighths and "excess grains," so called in contradistinction to the primary divisions of a carat or "carat grains." One carat grain contains 60 excess grains; thus, the divisions of the original weight or "assay pound" taken will be as represented in the following table:—

| Assay Pound. | Carat. | Carat Grain. | Eighth. | Excess Grains. | Decimal Equivalent. |
|---|---|---|---|---|---|
| | | | | | Per Cent. |
| | | | | 1 | ·017 |
| | | | 1 | 7·5 | ·130 |
| | | 1 | 8 | 60 | 1·042 |
| | 1 | 4 | 32 | 240 | 4·167 |
| 1 | 24 | 96 | 768 | 5760 | 100·000 |

From the above the meaning of the term "excess grain" will at once be evident. It is the $\frac{1}{5760}$th part of the weight of metal taken for assay, this ratio being the same as that borne by a troy grain to a troy pound. For accurately reporting the amount contained in a given alloy, it is necessary to use subdivisions less than eighths. "Excess grains" are therefore employed, and their number is the same as the number of actual grains of gold contained in a troy pound in excess of the amount represented by the report in carats, grains, and eighths.

The standard alloy of the English gold coinage contains 11 parts of gold to 1 of copper, or in other words, 22 parts of gold to 2 of copper; it is therefore said to be 22 carats fine. When assays are reported on this system, which is called the "trade system," the actual amount of gold is not given, but the metal is described as so much better or worse than standard, to indicate which the letter B or W is prefixed. Thus, suppose an assay to be given as B 1 ct. 3 grs. 5 eighths + 3·5 excess grains. This means that it actually contains 23 cts., 3 grs., 5 eighths, and 3·5 excess grains, in the total amount of 24 carats. The percentage composition of such an alloy would be 99·66 per cent. of pure gold and 0·33 per cent. of alloying metal (generally copper or silver).

This system is still retained in the Mint, but has been replaced by a decimal system. Perfectly pure gold is termed "1000 fine," and the report upon any sample of alloy indicates the number of parts of pure gold or silver in 1000 which the sample contains. The English coin standard of 22 would be equal to 916·667. In France and in the United States, the standard coin is expressed by 900 of gold and 100 copper.

The following table, which gives the values of the divisions used in the old method expressed in decimal fractions, will be found convenient for effecting any required conversion in the case of gold. As an example of its use, suppose it be required to convert the above trade report into its equivalent on the decimal system. Then—

| | Thousands. |
|---|---|
| 23 carats will be equivalent to | 958·22 |
| 3 carat grains ,, ,, | 31·25 |
| 5 eighths ,, ,, | 6·51 |
| 3·5 excess grains ,, ,, | 0·61 |
| | 996·59 |

| Carat. | Decimal Equivalent. | Carat Grains. | Decimal Equivalent. |
|---|---|---|---|
| 1 | 41·667 | 1 | 10·417 |
| 2 | 83·333 | 2 | 20·833 |
| 3 | 125·000 | 3 | 31·250 |
| 4 | 166·667 | 4 | 41·667 |
| 5 | 208·333 | | |
| 6 | 250·000 | Eighths. | Decimal Equivalents |
| 7 | 291·667 | | |
| 8 | 333·333 | 1 | 1·302 |
| 9 | 375·000 | 2 | 2·604 |
| 10 | 416·667 | 3 | 3·906 |
| 11 | 458·222 | 4 | 5·208 |
| 12 | 500·000 | 5 | 6·510 |
| 13 | 541·667 | 6 | 7·812 |
| 14 | 583·333 | 7 | 9·115 |
| 15 | 625·000 | 8 | 10·417 |
| 16 | 666·667 | | |
| 17 | 708·333 | Excess Grains. | Decimal Equivalents. |
| 18 | 750·000 | | |
| 19 | 791·667 | 1 | 0·174 |
| 20 | 833·333 | 2 | 0·347 |
| 21 | 875·000 | 3 | 0·521 |
| 22 | 916·667 | 4 | 0·694 |
| 23 | 958·222 | 5 | 0·868 |
| 24 | 1,000·000 | 6 | 1·072 |
| | | 7 | 1·215 |
| | | 7·5 | 1·302 |

**Preparation of the Assay Piece.**—It is usual to make first a preliminary assay to determine the quantity of silver necessary. For this purpose half a gram of the alloy is weighed out, and if the colour of the gold indicates that copper is present, thirty-two times its weight of finely-rolled lead is taken, the alloy wrapped up in it, and cupelled. By the colour of the resulting button an experienced assayer can judge very closely of the fineness of the gold, and add the necessary quantity of silver, as will be presently explained. If a bar is pure, its colour will enable the operator to do away with this preliminary assay.

That preliminary question being disposed of, the corners of the ingots, if of malleable gold, are hammered out on a small anvil with a clean flat hammer, and rolled out into two flat strips between a pair of rollers (Fig. 171). This facilitates the assayer in cutting up his assay pound and weighing it out to a nicety. The shears used are shown in Fig. 172. If the gold does not contain about 2½ times its weight of silver, it is necessary to add silver up to this amount. For instance, if a bar is supposed to contain 700 parts of gold and 300 parts of silver, it is necessary to weigh out 1,450 parts of chemically pure silver, and add this to the assay pound. If no copper is present in the alloy, a very small piece of copper weighing about 30 milligrammes is added, to prevent brittleness in the alloy in the subsequent lamination; but if copper is in the alloy no addition is required. The two metals are then wrapped up in a piece of finely rolled sheet lead, and placed on a tray divided into numbered compartments, which correspond to the positions of cupels in the muffle; the number of the compartment in which each assay is placed being noted with a view to its subsequent identification.

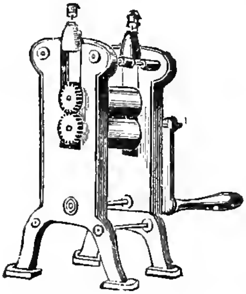


Fig. 171.—Rollers.

**Cupellation.**—The assays having been thus prepared are now ready for *cupellation*. A certain number of cupels are placed in a muffle and heated there to redness. The assays having been placed in the cupels, the opening of the muffle should be loosely closed by a piece of hot fire-brick, or by pieces of charcoal. After a short interval all the assay pieces will be melted, and in each cupel will be seen a round mass of molten metal, on the surface of which appear bright patches of litharge; these are produced by the oxide of lead which is constantly forming, and passing off from the centre to the edges of the assay, where it meets with and sinks into the porous matter of the cupel, leaving always a new surface exposed to the current of air which is carried through the muffle.

Fig. 172.—Shears.

At the same time there rises and passes through the openings of the muffle a white vapour which is caused by the volatilization of a portion of lead. When this vapour is very thin, and rises with great rapidity, it is an indication that the heat is too great, and part of the draught must be stopped; on the other hand, when it is thick and heavy, and hangs sluggishly over and around the cupel, the temperature is too low, and the draught must be increased.

As soon as the greater part of the lead has been thus converted into litharge and absorbed, the remaining bead of rich alloy suddenly becomes opaque and glowing, and is traversed by iridescent bands produced by the extremely thin films of fluid litharge resulting from the last traces of lead. This is a sign that the process is on the point of completion, and means must now be taken to increase the heat of the furnace for a short time, so as to insure the expulsion of the last remaining portions of lead. But if silver be present in the assay, and if it be desired to estimate also the amount of that metal, the increase of temperature towards the end of the process must be of short duration, otherwise a considerable portion of the silver would soon volatilize. Immediately after the disappearance of these bands the globule becomes steady, after which the peculiar action known as "brightening" takes place, when the metal is observed to emit a brilliant flash of light, and it immediately solidifies. If the cupellation has been well performed, the gold—or the mixture of the two precious metals—is now in a state of almost chemical purity.

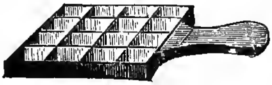


FIG. 173.—CUPEL TRAY.

When the cupellation is terminated, the mouth of the muffle ought to be closed, and the whole allowed to cool until the buttons become solid; or when only a few assays are performed at a time, the cupels containing the buttons of pure metal may be drawn to the front of the muffle, and allowed to cool slowly. They are then withdrawn, and placed in numerical order on the cupel tray (Fig. 173), which avoids errors being made. Some assayers mark and count the cupels from the right, or the reverse, and after some practice mistakes never happen. This refrigeration must not be too rapid, otherwise there may be sensible loss by "spitting, sprouting, or vegetation," an effect, however, only produced when a large proportion of silver is present. To insure gradual cooling, it is occasionally convenient to invert over the cupel containing the bead another heated cupel. Some assayers remove the cupels from the furnace while the metal is in a molten state, but the practice cannot be recommended, for besides the danger of upsetting a cupel and the "spitting" of the assay the buttons are found more malleable when cooled gradually.

I am indebted to Mr. A. Leichter, assayer, of Elko, Nevada, for the following communication:—"Five years ago," he says, "I discovered the instantaneous cooling of assay buttons (silver) to prevent vegetating or sprouting. I have never read anything of the kind before, and I am anxious to know whether I am the first discoverer of the process. It consists as follows: When at an assay on the cupel the 'blick' appears, at that moment, no sooner nor later, the cupel is instantly withdrawn out of the muffle, no matter how the temperature is in the room, or even into a draught of air, so long until button brightens (rebrightens); then it is held back into the muffle's best heat for two to three seconds, and the sample is done and will not vegetate. This will save considerable time to an assayer who has a great deal of work to perform."

The button is detached from the cupel after cooling by a pair of steel forceps, cleaned with a brush formed of stiff bristles, and flatted as presently described. If the process has been successful the button should be round and bright upon its upper surface, but somewhat rough or crystalline on the part by which it was attached to the cupel, from which it ought to be removed without difficulty.

The presence of iron, tin, nickel, or zinc, in the alloy under assay materially interferes with the success of the operation, as these metals will not pass into the cupel by the aid a even of large quantity of lead, but from their rapid oxidation will either volatilize, or their oxides will accumulate on the surface as a sort of slag, in which particles of precious metals may be entangled. In this case the assay becomes what is technically termed *foul*, the cupel not being able to absorb the metallic oxides as they are formed. To obviate this evil, the preliminary process of scorification should be resorted to, when the presence in the alloy of any of the above metals is known or suspected. A convenient weight of native gold or rich alloy to operate upon is 10 to 15 grains.

After cupelling the lead button from bullion or an ore assay, the cavity of the cupel shows different colours according to the presence of base metals in the ore. If the ingot was pure, or the mineral a nice clean quartz, the cupel will be covered with *yellow* coating, the button detach easily, and not adhere to the cupel, or only slightly.

*Copper* will show itself by a dark dirty green colour. *Antimony* will not only stain the cupel a yellowish tint, but cover it with slag, and when present in large proportion will crack the cupel. Hard lead is very difficult to cupel, and will freeze the button. *Arsenic* exhibits the same characteristics. *Iron* will produce deep red brown fine grains, which are readily distinguished in the cupel. *Manganese* will give a characteristic blue tint. *Nickel* will give a greenish tint, and also slag; and when slagging takes place the cupellation does not proceed regular, and the silver will spread out in a flat irregular layer on the cupel, and can be only detached after cooling by breaking the cupel, when fresh lead has to be added on a new cupel and the silver refined. *Tin* is hard to melt, and acts very much like antimony. *Zinc* gives a yellow stain, but different from lead, and eats into the cupel, which it will spoil. *Palladium* gives a crystalline button and a greenish tint.

The lead employed must be in all cases free from silver, being such as has been derived from pure litharge, otherwise it would be quite impossible to estimate the true composition of the alloy.

The assay of gold furnishes results which are more accurate than those obtained in the cupellation of silver, the loss of gold by volatilization being very much smaller, and scarcely any of the metal is carried into the cupel by the lead.

**Parting of the Assay.**—After the preceding operations, the button of alloy which remains is composed of gold and silver—not indeed absolutely pure, for whatever precautions may be used it will still retain a small quantity of lead, and frequently also traces of copper. For practical purposes, however, it may be regarded as a pure alloy of the two precious metals, and in this view it now only remains to separate the gold from the silver.

It has been stated that when silver and gold are intimately mixed together, as in an alloy, it is necessary that the silver should be present in the proportion of at least 2½ parts to 1 part of gold, otherwise its particles will be so enveloped and protected by those of gold that the nitric acid will be prevented from exercising its solvent action upon them. Care should always be taken before the introduction of the assay piece into the furnace that the requisite proportion of silver is present, and if this be the case it will be sufficient at once to subject it to the action of the acid.

But if the silver be present in a less or much greater proportion, then, as a preliminary step to the operation of parting, the alloy must be brought to the proper standard, either by the addition of the necessary quantity of pure silver, or by adding more gold, as the case may be; for if the proportion of silver exceed 3 parts, the gold will be precipitated by the acid as a dark powder, instead of retaining its compact form, and thus the accuracy of the estimation of the amount of gold present will be impaired.

It is hardly necessary to add that the amount of silver requisite can only be determined when the approximate assay is known. This may be determined by a wet analysis, a rough assay, or by the touchstone; or experienced assayers are able to determine with sufficient accuracy by observing the colour and the facility with which the metal can be cut with shears.

The button is now flattened by hammering on an anvil, three strokes being so given that the first shall extend it to about the size of a sixpence, and the other two reduce the thickness still further at two opposite edges, in order to facilitate its introduction into the laminating mill. These flattened buttons are generally annealed before rolling, and by the rolling operation the thickness is reduced to about one-eighth of an inch, and when all the laminæ have been placed in slits on a wrought-iron tray, they are introduced into the muffle and annealed. The operation renders them sufficiently flexible to allow of their being twisted into a small spiral cornet, by rolling between the finger and thumb, or by means of a pair of round-nosed pliers.

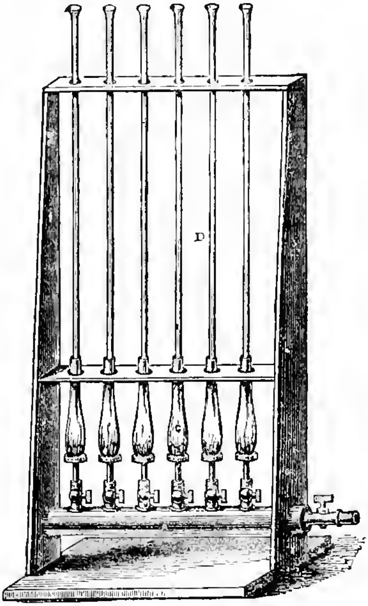


Fig. 174.—Parting Flasks.

To effect the parting of the assay, each assay piece, or cornet, is boiled separately in strong nitric acid, which must be free from chlorine, sulphuric acid, or sulphurous acid. The nitric acid is contained in a small glass flask, called a parting flask. Fig. 174 represents a convenient apparatus for boiling six cornets separately. The large tube, A, is connected with the gas supply, and from it rise a number of small rose Bunsen burners. On these rest the parting flasks, C, into each of which is introduced from 2 to 3 oz. of nitric acid of 1·26 specific gravity. The long tubes, D, dipping into the mouths of the flasks, are intended to condense as far as possible the acid vapours, the condensed liquid running back into the flasks. In a case where a large number are in action at once, they should be connected with a flue to carry off these vapours. The acid should continue to boil from three to five minutes after the glasses have cleared of fumes; the acid is then poured off, the cornets washed with hot distilled water, and again boiled from fifteen to twenty minutes in nitric acid of about 1·30 specific gravity. As the ebullition is at times very violent, and the cornet, owing to its spongy nature, extremely fragile, it is generally recommended that a piece of charcoal be introduced into each flask. This, however, is objectionable, as it may cause the formation of nitrous acid, which by absorption in the acid will even dissolve portions of the metal. This source of error is completely eliminated by using instead of the charcoal small balls of porous earthenware, as recommended by Field.

The last acid having been poured off, the flask is completely filled with distilled water, and a small crucible of smoothly-finished porous clay is placed over the mouth of it. The two vessels thus adjusted are then inverted, so as to allow the cornet to fall gently through the water into the crucible, and by a dexterous movement of the hand the flask is withdrawn, in such a manner as to prevent the overflow of any liquid from the little crucible. The water is now carefully decanted, and as much moisture removed as possible; the crucible containing the cornet is then heated to redness in the muffle. Under this final heating it is not fused, but shrinks in bulk, loses its brown appearance, and assumes the peculiar colour and lustre of gold; while at the same time it is rendered more compact, less brittle and fragile, and when cold can be removed by a pair of forceps to the scales, in which it is weighed with the same precision and care as the original alloy.*

Professor Roberts-Austen, F.R.S., in a lecture delivered before the Society of Arts on March 31st, 1884, gave the following interesting data respecting the parting process :—

"This action of nitric acid on an alloy of gold and silver was certainly known to Geber and the early alchemists, but the

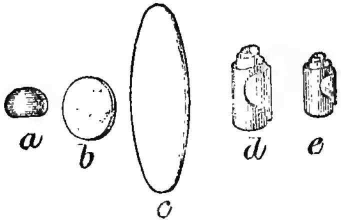
---

* Materials required for assaying may be obtained of Messrs. J. J. Griffin and Sons, Limited, of 22, Garrick Street, London, W.C., or of Messrs. John Orme and Co., 65, Barbican, London, E.C.

first official mention of the use of the parting assay appears to be in a decree of Philippe de Valois, in the year 1343, confirming its use in the French mint. There is, then, this difference between the assay of gold and silver. In the case of the cupellation assay of silver, the button of metal has only to be removed from the cupel, and when the adhering bone-ash has been removed by the brush it passes direct to the balance.

"After the first stage of the assay of gold on the cupel, the object to be attained is mainly to secure a button, *a*, of gold and silver in a convenient form for submitting to the subsequent operations. The alloying stage would, however, be just as effective if it were conducted in a small non-porous receptacle, such as a small crucible of glazed porcelain.

"The subsequent operations are, flattening the button, *b*, annealing it, rolling it into a strip, *c*, and annealing it a second time. It is then coiled into a spiral, or cornet, *d*, and treated by two successive portions of nitric acid, in order to remove the silver; after this the spiral of spongy gold, *e*, which retains the original form given to the silver alloy, is heated to redness, when it becomes bright, and is sometimes so coherent that it may be unrolled without fracture."



Scale about ½.

FIG. 175.—BUTTON AND CORNETS.

The weight of the cornet will indicate very nearly the amount of gold in the alloy if the assay has been properly conducted, but not with absolute exactness. Even when every precaution has been used, the gold contained in the cornet is never chemically pure; it still retains a very small quantity of lead and silver, and frequently also traces of copper, in consequence of which the weight of the cornet may exceed by $\frac{1}{2000}$th part the true result; and when it is considered that in assaying alloys the quantity taken is often not more than 7 or 8 grains, it will be seen that even this small error might lead to serious disappointment in estimating the proportion of gold in large masses of bullion. To ascertain the amount of the error, and thus arrive at a perfectly correct estimate, it is usual to assay, simultaneously with the alloys, a number (generally three) of what are termed *proofs*, consisting of weighed portions of perfectly pure gold, approximately equal in amount to that present in the assay pieces. The cornets obtained from these proof assays are weighed, and the excess or deficiency in weight over that of the gold which was introduced indicates the amount of correction which it becomes necessary to apply to the other assays. This correction is liable to vary, according to the temperature of the furnace and other causes; but it usually ranges from 0·2 to 0·5 in 1,000. It may be observed that it is not essential to employ *absolutely* pure gold, provided the difference from purity be known. Thus the corrections to be applied to a gold assay will usually be one-half of a thousand.

The actual amount of metal retained—called surcharge—is greatly influenced by the amount of copper present in the alloy under examination, and varies considerably with the heat of the furnace and other circumstances. In an alloy rich in gold, which requires therefore only a small quantity of lead, the surcharge is slight; when, however, a large quantity of copper is present the surcharge becomes negative, in consequence of the absorption of gold by the cupel; and for alloys containing 68 or 70 per cent. of gold the surcharge is zero. Rossler has shown that the gold lost in the cupellation process increases with the amount of lead used, and decreases as the silver present is increased; and these conclusions are confirmed by the results of experiments recently made in the Royal Mint. He also considers that the gold lost is generally greater in amount than the silver retained by the cornet.

Professor Roberts-Austen gives the following formula for the corrections to be applied to a gold assay when checked by means of a *proof* or *check*:—Let 1,000 be the weight of alloy originally taken; $p$, the weight of the piece of gold finally obtained; $x$, the actual amount of gold in the alloy expressed in thousandths; $a$, the weight of gold supposed to be absolutely pure taken as a check, which approximately equals $x$;

$b$, loss or gain in weight experienced by $a$, during the process of assay; $k$ variation of "check gold" from absolute purity, expressed in thousandths. Then the actual amount of fine gold in the check-piece $= \text{A}\left(1 - \frac{\text{K}}{1{,}000}\right)$, and the corrected weight will be $\text{X} = \text{P} - \frac{\text{A K}}{1{,}000} \pm 6$, the 6 being added or subtracted according as it is a loss or gain.

If (A) be assumed to be equal to (X) this equation becomes

$$= \frac{\text{P} \pm \text{B}}{1 + \frac{\text{K}}{1{,}000}}$$

**Parting Assays.**—It is frequently necessary to determine not only the amount of gold or silver contained in a given alloy, but also the amount of each of these precious metals present in the same sample. The assays made with a view to obtain this information are termed "parting assays."

*Gold and Silver.*—If the quantity of silver in the alloy submitted to the operation of parting be more than three times that of gold, the separation may still be effected by nitric acid, but the gold will be left in a state of powder. It may, however, be washed, ignited, and weighed. Should the amount of silver be small compared with the gold, and the assayer wishes to avoid the trouble of inquartation, that is, the adding of the requisite proportion of silver, the alloy may at once be subjected to the action of aqua regia, when the gold will be dissolved, and the silver thrown down as an insoluble chloride. The clear solution containing the gold is then decanted, and when the excess of acid has been expelled by evaporation, the gold can be precipitated by sulphate of iron.

The determination of the amount of silver contained in gold by the dry method of assay is a somewhat complex operation. One or two gold assays in the ordinary way are made, and at the same time two more assay pieces are passed through the furnace, but without the addition of silver. By this means all the oxidizable metals are in both cases removed, and the difference in weight between one of the cornets resulting from the ordinary assay, and the button of mixed gold and silver as drawn direct from the furnace, must be due to silver originally present in the alloy.

The silver contained in the solution of its nitrate resulting from the common process of parting, may readily be recovered by adding an excess of chloride of sodium, which will throw down the whole of the silver as white insoluble chloride. This precipitate, which speedily becomes slate coloured on exposure to light, is collected and put into a vessel containing water acidulated with hydrochloric acid, some zinc or iron is introduced, and the silver is rapidly restored to its metallic state, in the form of a dark grey powder.

The supernatant liquid having been decanted, this fine metal is washed with dilute hydrochloric or sulphuric acid, in order to remove traces of iron or zinc; but as the removal of the former of these metals may be effected more perfectly, it is now always used in preference to the latter; after washing with water, and drying, the silver is fused in a crucible with the addition of a little carbonate of soda or nitre, and cast into bars.

*Gold and Copper.*—The alloys of gold and copper are cupelled like the alloys of gold and silver; but as copper has a very great affinity for gold it is necessary to use a larger proportion of lead to insure its oxidation when combined with gold than when united with silver. This proportion varies according to the standard and the temperature. It is admitted that for the same standard there must, under similar circumstances, be twice as much lead used in the cupellation of gold as in that of silver. Thus 14 parts, at least, ought to be employed in common furnaces for an assay of gold coin which contains 0·100 of copper. There is no inconvenience in employing a little more, as it does not increase the loss of gold. However great the proportion of lead may be that is added to the cupreous gold for the purpose of cupellation, the button always retains a very small quantity of copper, of which a fresh cupellation does not free the gold, and which occasions what is termed the surcharge. This surcharge being very slight it can be neglected in assays of minerals, but it is necessary to take notice of it in the assays of alloys. But it is known that the presence of silver much facilitates the separation of copper from gold, and it is rare that an alloy of cupreous gold does not contain a little silver, which must be separated; and when this is not the case a small quantity of that metal can be introduced into the alloy, so as to be in about the proportion of 2½ parts to 1 of gold. When an assay is to be made of an alloy of gold and copper, a sufficient quantity of silver should be added to fulfil the condition according to the presumed standard, which is approximately determined by a preliminary assay and then cupelled with lead.

TABLE FOR PROPORTION OF LEAD TO BE EMPLOYED IN THE CUPELLATION OF GOLD AND COPPER.

| Gold in Alloy. | Lead required. |
|---|---|
| 1000 thousandths | 1 part. |
| 900 ,, | 10 parts. |
| 800 ,, | 16 ,, |
| 700 ,, | 22 ,, |
| 600 ,, | 24 ,, |
| 500 ,, | 26 ,, |
| 400, 300, 200, 100, 50 thousandths | 34 ,, |

**Gold containing Oxidizable Metals.**—Where gold contains platinum with copper, or other oxidizable metals, the latter will be removed with difficulty by cupellation, and silver must be added. Double the weight of the assay pound in silver must be taken, and the cupellation must be conducted at a high heat on the addition of the proper quantity of lead.

Mr. Chaudet, who has made experimental researches on the alloys of gold, silver, copper, and platinum, states that the copper can be more completely separated, and less silver lost, by cupelling at a high temperature, with the least possible quantity of lead, than by employing more lead and working at a lower temperature. He states that it is almost impossible to separate all the copper, and no advantage can be obtained by increasing the quantity of lead. When almost the last traces of the copper are separated the button must be cupelled afresh with a small quantity of lead; but a small quantity of silver is nearly always lost. In all cases, in order that no lead shall remain, it is necessary to leave the assay button some few minutes in the muffle, after the cupellation is finished.

The alloys of gold and silver containing platinum exhibit certain characteristics during cupellation, or parting, which indicate the presence of this metal. If there is only two per cent. of platinum in the alloy, the button remains flat on the cupel, and if this button is dissolved in nitric acid, the solution is coloured straw yellow. The peculiar iridescence of the button at the termination of the assay is not as bright and lively, and is much longer, than when no platinum is present; and on solidifying the button remains dull and tarnished, and does not exhibit a bright lustre with a partly crystalline structure. When the alloy contains more than a hundred parts of platinum to nine hundred of gold, the cornet after annealing is of a pale yellow colour, tarnish silver grey.

The following method serves for detecting platinum in admixture with gold, and other heavy matters obtained by washing auriferous gravels, or in prospecting: If gold is present, collect it with mercury; take the residue and boil it in aqua regia, evaporate to dryness, add some dilute hydrochloric acid, boil and filter; to the filtered solution add a strong solution of sal-ammoniac. If a bright yellow, or reddish yellow, granular precipitate falls, platinum is present in the sand.

*Auriferous bismuth and lead* are either directly cupelled, or, with a very small percentage of gold, first scorified and several buttons concentrated to one.

*Auriferous tin* is oxidized in the muffle, and scorified with the addition of granulated lead and borax, and then cupelled.

*Auriferous iron, steel, or pig iron* is oxidized with nitric acid, the resulting fluid evaporated to dryness, and the dry mass well scorified with eight to twelve parts of lead, two to three parts of borax, and one part of glass.

*Auriferous mercury* is scorified sometimes after previous distillation with a very gradually increasing temperature, and with eight parts of lead.

*Gold containing rhodium* is fused with three to four parts of silver parted as usual with nitric acid, and the well-washed and dried gold fused at a red heat in a platinum capsule with bisulphate of potassa or soda, whereby the rhodium, together with the remnant of silver, is dissolved, with brisk evolution of sulphurous acid, and the formation of a brownish red to black salt. The fluid mass of salts is poured off from the gold, and the fusion with bisulphate of soda again repeated, whereupon the fluid salt appears but slightly coloured. After this is again poured off, the gold is boiled a few times with distilled water, dried, and ignited. In order to be certain of the complete removal of the rhodium, the gold is again mixed with three parts of silver, and the above process repeated. A trace of rhodium imparts to the bisulphate of soda a yellowish colour, and when this happens the globule of gold suffers a loss of weight compared with the previous weighing.

*Gold containing iridium* is dissolved in aqua regia, whereby the iridium remains behind as a black powder. This is well washed, and the gold precipitated from the diluted solution with protosulphate of iron.

The iridium obtained in this way, when reduced by hydrogen at a low temperature, oxidizes slowly at a red heat, and then becomes soluble in aqua regia. It is fused with great difficulty before the oxy-hydrogen blow-pipe. It is very seldom found alone, but usually in association with osmium.

*Osmium* is separated from iridium as follows. The alloy is mixed with a small quantity of chloride of sodium, placed in an open glass tube, and heated over a charcoal or gas flame. Into one end of the tube is led a stream of chlorine gas, which attacks the osmium and iridium and converts them into chlorides, and as the osmium chloride is volatile, its vapours are led through the other end of the tube into a flask containing ammonia solution, to be decomposed into osmic and hydrochloric acids, which combine with the alkali. The iridium chloride remains in the tube in combination with the sodium chloride, and when cold these are dissolved out by water, some sodium carbonate is added, and the whole evaporated to dryness and ignited. After boiling with water and evaporating the same, a combination of iridium oxide and soda is left, which is reduced at a high temperature by hydrogen. The soda is dissolved out with water, leaving iridium behind, which is annealed at a white heat.

*Gold containing palladium* is alloyed with three parts of silver, and treated like an ordinary gold assay, when the palladium as well as the silver goes into solution. After the latter has been removed from the diluted solution by chloride of sodium, the former may be precipitated by metallic zinc.


FIG. 176.—MORTAR.

Its presence in acid solutions can be detected by the addition of a few drops of mercuric cyanide, which gives a pale yellowish white precipitate of palladious cyanide Pd $Cy_2$. This on heating becomes metallic palladium. Hydriodic acid and potassium iodide throw down a black precipitate of palladium iodide, and the reaction is very marked in the most delicate solutions.

**The Assaying of Gold Quartz.**—First of all the quartz is finely pulverised by triturating it in an iron mortar, shown in Fig. 176. This operation is much facilitated by heating the quartz to redness, and then plunging it in cold water. Having pulverised a few thousand grains, it is usual to make at least two assays of the sample to test the correctness of the result. For each of these weigh 500 or 1,000 grains on the pulp scales, Fig. 177, according to the richness of the ore, and well mix on a clean piece of paper with four times the weight of litharge or red lead, and double the weight of carbonate of soda,

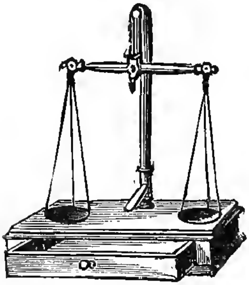


FIG. 177.—PULP SCALES.

and quarter the weight of powdered charcoal. The precise quantities are not very important, but it is better that the carbonate of soda should be in excess than that it should fall short of the proportion given. The following is a very good mixture :—480 grains of the gold quartz, 1,500 grains of litharge, 1,000 grains of carbonate of soda, and 100 grains of powdered charcoal. The mixture is then introduced into an assay clay pot (Fig. 178) and covered with fused and finely pulverised borax, and fixed on a piece of fire-brick in the melting furnace.

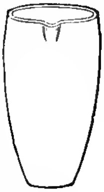


FIG. 178. CLAY CRUCIBLE.

The mixture above mentioned, when introduced into the crucible, should not fill more than two-thirds of its capacity. This precaution is necessary on account of the effervescence which takes place, in consequence of the displacement of the carbonic acid from the carbonate of soda by the silica, and the combination of the carbon with the oxygen of the litharge. The heat is continued till perfect fusion is effected, and towards the last the temperature is raised to bright redness, until no further effervescence occurs, and a clear liquid homogeneous slag is obtained. The crucible must then be immediately removed from the fire (otherwise the unreduced litharge would be liable to cut through the pot), and as dexterously and rapidly as possible the contents of the pot should be poured into the mould shown in Fig. 179. The mould has the shape of a cone, and on cooling it is turned over, when the slag will be found solidified in the upper portions and the lead button in the bottom. By hammering the mass on a clean anvil the slag will readily detach itself from the lead button, which is now ready for cupellation.

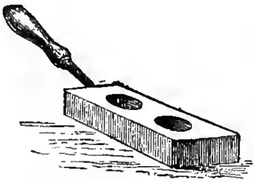


FIG. 179.—BUTTON MOULD.

The proportion of oxide of lead to be used in the fusion will vary with the amount of oxidizable substances present; but this compound should in all cases be added in excess, since, if the slag retains any traces of an alkaline sulphide, it will retain a part of the gold. Phillips states that, for the assay of iron pyrites, about 30 parts of oxide of lead are necessary, whilst for mispickel, zinc blende, copper pyrites, grey cobalt, and sulphide of antimony, from 15 to 25 times their weight only may be employed.

FIG. 180. ROASTING DISHES.

When auriferous pyrites is the subject of examination, it must first be reduced to fine powder, and then roasted in a shallow dish of refractory clay, heated to low redness in a large muffle, until the odour of burning sulphur ceases to be evolved.

As the sulphur burns away, the temperature is gradually raised to bright redness. The pyrites is thus converted into oxide of iron. Of this product 1,000 grains or more may be taken and mixed with 500 of dry carbonate of soda, 300 to 500 of litharge or red lead, and 5 per cent. of charcoal, or a corresponding quantity of granulated metallic lead without charcoal, and lastly about 500 of fused borax. This mixture is melted as before, and the fuel is either charcoal, anthracite, or coke.

The principal objection to this method of assay is the large amount of lead which is left for cupellation, since pure iron pyrites requires, when thus treated, 8½ parts of lead, and sulphate of antimony and grey copper ore yield 6 to 7 parts. This inconvenience, as well as the trouble of roasting, may be avoided by the cautious and gradual addition of nitrate of potassa, which effects the partial oxidation of the mineral, and enables the skilful assayer to procure a metallic button of almost any required weight. The nitre, however, if employed in excess, would determine the oxidation of all the metallic and combustible substances contained in the mineral, except the gold, which is never present in sufficient quantity to form a button. The exact amount of nitre to be added must depend on the nature and richness of the ore. As a general rule, 2½ parts of nitre are sufficient to completely oxidize 1 part of iron pyrites; and 1½ and ⅔rds their weight respectively, in the case of sulphide of antimony and galena, are sufficient to produce the same effect on these ores.

Alloys of gold containing tin and zinc must also be fused before cupellation with nitrate of potassa, so as to oxidize these metals; and by adding about 16 parts of lead when the mixture is quite melted, they are separated in the state of slag or scoria, leaving, as before, a button of lead containing the whole of the gold, which is now ready for cupellation.

**Assaying by Scorification.**—Scorification—or, in other words, the conversion of the silica and other impurities into a scoria—is simply another method of fusion, in which, however, metallic lead, instead of its oxides, is employed; and the oxidation of the various substances to be removed is produced by the aid of atmospheric air, whilst the litharge necessary for the fusion of the earthy and siliceous matters is formed by the oxidation of a portion of the metallic lead. For this operation, instead of a crucible, a shallow cup-like vessel, termed a scorifier, is employed.

**The Scorifier.**—This vessel is made of refractory clay, and made as compact in structure as possible, in order to resist the corrosive action of melted litharge. The scorifier with its contents is heated in the muffle of an ordinary assay furnace, and as many assays may be introduced at one time as there is room for in the muffle.

Fig. 181 represents, in front elevation and vertical section, the form of furnace at present employed in the Royal Mint, which was arranged by Field, the late assay master, and which is convenient both for this purpose and for the subsequent operation; the muffle used in it is shown at M. This last is an arched oven of fire-clay, the mouth of which is slightly contracted, and it is closed at the back; the closed extremity being pierced with a number of cylindrical holes, in order to insure a free circulation of air through the interior. It is advisable that these holes slope outwards, as shown in the figure, in order to prevent any fuel from falling into the muffle. The top is roughed in order that it may be strengthened by a coating of fire lute. It rests on a plate of fire-clay, beneath which is an iron girder plate, C, placed upon the fire-bars, B. The furnace is made of wrought iron about one-eighth of an inch thick, the whole being lined with fire-bricks. It will be seen that the furnace is provided with five distinct openings: the lower one, A, in order to clean the ash-pit; the two side ones, D D, to remove two fire-bars to admit of the fuel being dropped into the ash-pit; and the two, E and F, to charge the muffle and furnace respectively with cupels and fuel. By means of these perfect control is obtained over the fire, and any required course can be given with facility to the in-going current.

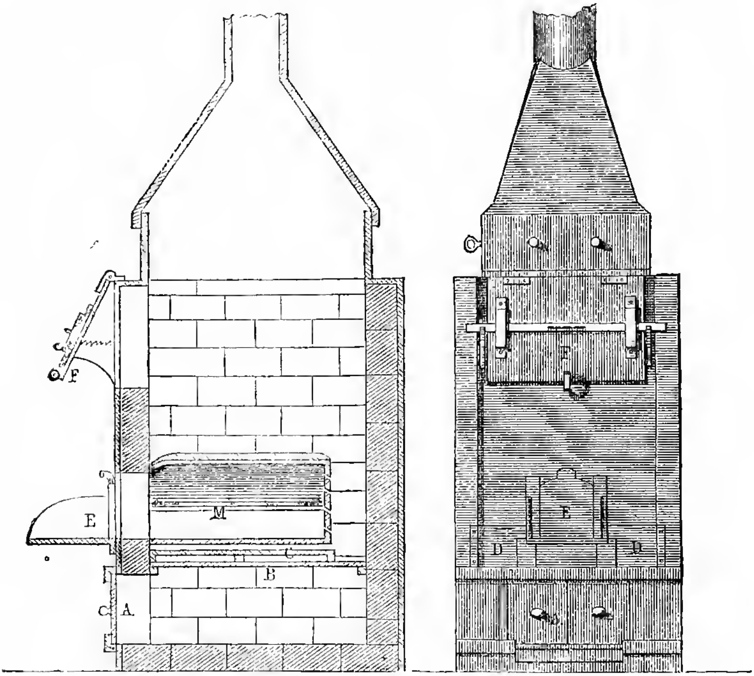


Fig. 181.—Muffle Furnace.

The Battersea muffle furnace (illustrated in Fig. 182) is a portable furnace very suitable for laboratory work. It is built up in sections which are easily packed for transportation. In the illustration *a* is the socket on which to fix iron chimney; *b*, the door for fuel; *c*, the muffle; *d*, the door for muffle; *e*, the muffle arch; *f*, the door for stirring fire; *g*, the door for regulating draught; *h*, the support for muffle, and *i* the iron grate.

**Process of Scorification.**—Before introducing the ore into the scorifier it is reduced to powder, and a determined weight, about 50 grains, is intimately mixed with from three to five times its weight of granulated lead (the larger quantity being required for poor ores), and one tenth its weight of borax; this, having been placed in the scorifier (Fig. 183), is covered with from 150 to 200 grains of granulated lead, care being taken that the lead employed contains no silver. The scorifiers charged with this mixture are then placed in the muffle, and the door at the mouth being closed, they are strongly heated for a quarter of an hour. Fig. 184 shows the scorifying tongs, with which the scorifiers are easily handled. By this time the lead will be completely melted, and the

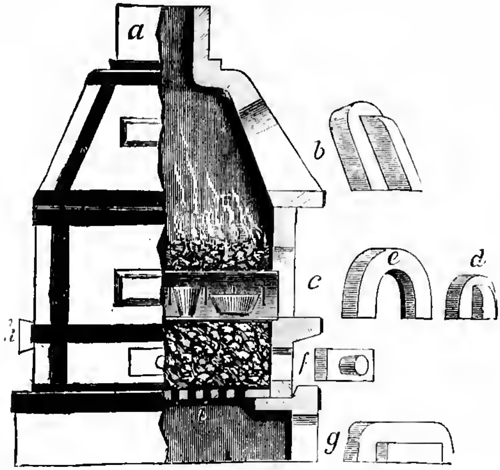


FIG. 182.—BATTERSEA MUFFLE FURNACE.

mouth of the muffle is again opened. Scorification is now effected by the action of the current of heated air passing through the muffle; in other words, the lead is oxidized, and the foreign metals present—whether in combination with sulphur or oxygen—are attacked and dissolved in proportion as this oxide is formed. At first the slag is frequently solid, but it gradually becomes soft, and more and more liquid in proportion as the quantity of the oxide of lead increases.

It is frequently desirable, when the surface has become covered with litharge, to introduce 15 grains of anthracite or charcoal, as by this means a portion of the lead is reduced to the metallic state, and by sinking through the mass of slag takes up any particle of precious metal which may be present. Towards the conclusion of the operation the muffle is for a short time strongly heated, so as to render the slag or scoriæ completely liquid. When a small iron rod, heated to redness and placed in the mixture, is found on being withdrawn to be covered with a slight film of scoria, which runs off clean, the scorification is known to be sufficiently advanced. The scorifiers are successively withdrawn from the muffle by means of tongs, and the contents are rapidly poured into circular ingot moulds of the form represented in Fig. 179. When cold the adherent litharge is detached from the button of lead by a few blows with a hammer. This button contains all the gold and silver which may have been present in the ore, and these are separated from the lead by cupellation.

FIG. 183.
SCORIFIER.

FIG. 184.—SCORIFYING TONGS.

When the material to be scorified is highly carbonaceous, such as the "sweep" or refuse constantly accumulating in the Mint, it is advisable to calcine it previous to the addition of the lead.

If the ore be poor in quality, the whole of the precious metals contained in a large quantity may be concentrated in one small button of lead by repeating the process on successive portions of ore mixed with the requisite amount of lead, and introducing into each assay the button previously obtained. A final button containing all the precious metals from the several quantities of ore will thus be procured. The process may be repeated a third, fourth, or any number of times, till the button of lead thus obtained becomes as rich in the precious metals as may be required. It is necessary, however, that a certain amount of lead be present, with a view to the next process, that of cupellation. In many cases as much as ten times the weight of the ore is employed, but the button may be reduced in size by a single operation to one-sixth or even one-eighth of the original lead. Indeed, it is the chief recommendation of this process, as compared with the method by fusion with litharge, that however small may be the proportion of lead, the slag produced never contains any oxysulphides at the close of the operation, and therefore seldom retains the slightest trace of either gold or silver. The process of scorification has therefore been justly characterised as one of the most exact methods that can be employed, being simple and effective, and applicable, without exception, to the assay of all kinds of auriferous and argentiferous ores.

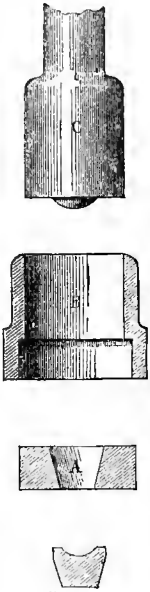


Fig. 185.

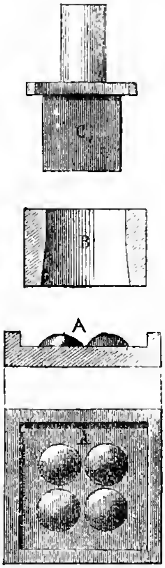


Fig. 186.

Cupel Moulds.

**Cupels and Cupellation.**—Cupels for assaying are made by pressing moistened bone-earth into a mould of cast iron or gun metal which varies in construction. Two forms at present in use in the Royal Mint are shown in Figs. 185 and 186, one-half their natural size. In the first of these, after the space in the piece of metal, A, has been rather more than filled with bone-ash, the guide, B, is placed over it. The plunger, C, having been introduced into this guide, the whole is placed under a common lever in the press, by which means the bone-ash is compressed into the form shown in the figure. Formerly the plunger was struck several times with a hammer in order to give the required compression. It is desirable to give a slight twisting motion to the plunger on removing it.

The second form of mould, shown in Fig. 186, has been introduced in order to lessen the labour of the furnace manipulation. It will be seen that each cupel is square, and is provided with four depressions. Each of these, however, is rather smaller than the one already described, the amount of metal operated upon in the two cases being about 7·5 to 16 grains respectively. The square cylindrical piece, B, being placed on the plate, A, is filled with bone-ash, and with the plunger, C, is introduced as before into the press, the cupel when formed being five-eighths of an inch deep, square, and slightly tapered at the bottom. The cupels, when formed, should be kept in a dry place for several months, as it is found, when any method of hastily drying them is adopted, that they almost invariably crack when introduced into the muffle and cause the assay to spurt.

Cupellation is performed in a muffle furnace, as described above. At the time of lighting the furnace the requisite cupels are successively introduced into the muffle by light tongs, the floor of the muffle being covered with a thin layer of bone-ash before the introduction of the cupels. The temperature of the muffle having been raised to an orange red heat, the assay pieces—either in the form of buttons mixed with lead, as obtained by scorification, or as samples of alloy wrapped in lead foil, as already described — are successively introduced into the cupels, care being taken that they are so arranged as to

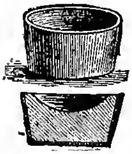


FIG. 187.
CUPELS.

avoid possibility of confusion on their subsequent removal. It is advisable that while the muffle is being thus charged the door should be partially closed by means of a hot fire-brick, in order to check as much as possible the chilling of the cupels which are near to it. The requisite amount of lead may all be introduced as a wrapper of the assay piece. Of course when the alloy for assay consists of a button obtained by a previous process of fusion or scorification, this addition of lead is unnecessary, as a sufficient amount is already contained in the button.

By the previous process of fusion with litharge—or that of scorification, if deemed preferable—the earthy impurities will have been removed in the form of slag or scoria, and nothing now remains but the gold, or a mixture of gold and silver, in combination with the lead and any other metals that may be present. The process of cupellation, as already stated, consists in the oxidation of the lead and other oxidizable metals, and their absorption by the cupel. It is, in fact, a condition essential to the success of the assaying process that the whole of the oxides should be absorbed by the cupel, and thus removed. But this power of absorption varies according to its texture and the care with which the material has been prepared. On an average it may be calculated that it will absorb about its own weight of fused litharge. This, though not strictly correct, will help the experimenter as regards the amount of lead which ought to be mixed with the assay. It is evident, on the one hand, that this must not exceed the absorptive power of the cupel; on the other hand, the actual amount within this limit must depend on the nature of the alloy upon which the cupellation is to be performed.

**Weighing the Button of Precious Metal.**—The silver button obtained by cupellation of the lead button in an ore assay, if 250 grains of pulp are taken as the assay weight, is freed from the adhering bone-ash by hammering edgeways, and is then weighed. The weight, multiplied by 1·16, gives the amount of ounces per ton of ore of 2,000 lbs., which is illustrated by the following example. For instance, a button is found to weigh—

300 parts
50 ,,
6 ,,
———
356 × 1·16 = 412·9 ounces per ton of ore.

After weighing, the silver button is introduced into a glass tube, adding about half an ounce of pure nitric acid, and heated by the alcohol flame. It soon begins to boil, emits reddish-brown vapours, and leaves the gold, if any in the assay, in undissolved particles of a black colour in the tube. The nitric acid containing the dissolved silver is poured off slowly, and the tube filled with distilled water. When all the particles of gold have settled, this water must be poured off carefully and the tube filled once more with water to the brim.

The tube is then covered with the dry cup and quickly turned over. The gold falls to the bottom of the cup, but being very light and sometimes in minute particles, the tube must be kept for awhile in this position until no suspended particles are visible. It requires some practice to lift the tube without spilling any water, which would invariably carry out some gold. The easiest and surest way is to gradually lift up the tube till the water, the brim of the tube, and the dry cup are level. A slide of the tube, with a dexterous upward movement, leaves the gold in the cup undisturbed. A slight tapping of the cup will bring the gold particles together; the water is then poured off, and the cup dried over the alcohol flame till the gold assumes a yellow colour. The gold is now carefully weighed and calculated upon as follows:—

The gold is found to weigh, for instance, $\frac{35}{1000}$, and the silver button, before dissolving, $\frac{356}{1000}$. If the gold is subtracted from the silver which contained this gold, we thus find the pure silver—

356 − 35 = 321 silver × 1·16 = 372·3 ounces per ton.
and 35 gold × 1·16 = 40·6 ,, ,,

To find the value, the ounces of gold must be multiplied by 20·67,[*] and those of silver by the market value per ounce. Therefore

Silver = 372·3 ounces × $1 = 372·30
Gold = 40·6 „ × $20·67 = 839·20

Total value $1211·50 per ton.

In assaying gold ores the button will not dissolve in nitric acid. In this case it must be melted after weighing on a cupel with the addition of three times its weight of pure silver, with a little lead foil, and then dissolved and treated as above described.

In case the ore for the assay has been weighed out by half an ounce—equal to 240 grains—the calculation is made in the same way as before, with the exception that the number 1.215 must be substituted for 1·16. The procedure of the preceding example would be as follows:—

The weight of the button was 321. This multiplied by 1·215 will give the amount of ounces per ton of ore of 2,000 lbs.

32 (321) × 1·215 = 390.

The quantity of fluxes used for 250 grains of ore will also serve for half-ounce assays, which can be as follows:—

ORES OR TAILINGS CONTAINING BUT LITTLE SULPHURETS.

| | |
|---|---|
| Ore . . . . . . . | 250 grains. |
| Glass . . . . . . . | 125 „ |
| Flour . . . . . . . | 8 „ |
| Litharge . . . . . | 1½ ounces. |
| Soda . . . . . . . | 1 ounce. |

ORES CONTAINING ABOUT 50 PER CENT. OF SULPHURETS.

| | |
|---|---|
| Ore . . . . . . . | 250 grains. |
| Glass . . . . . . . | 125 „ |
| Iron . . . . . . . | 50 „ |
| Litharge . . . . . | 1½ ounces |
| Soda . . . . . . . | 1 ounce. |

---

[*] One ounce fine gold 24 carats, or 1000 fine, is worth £4 4s. 11·4545d.

### CONCENTRATED PYRITES, NOT ROASTED.

| | |
|---|---|
| Ore | 250 grains. |
| Litharge | 2½ ounces. |
| Soda | 350 grains. |
| Borax | 240 ,, |
| Powdered nitrate of potash | 400 ,, |
| Salt | 250 ,, |

The salt is placed on top as a covering, and the assay is gradually heated. If iron matte is formed, the assay must be repeated, and the proportion of nitre increased.

# CHAPTER XIX.

## *CHEMICAL EXAMINATION OF GOLD ORES.*

TESTS FOR PRESENCE OF GOLD—Reagents employed—Preparation for Testing—Action of Sulphate of Iron; of Protochloride of Tin; of Oxalic Acid—Tests for Metals associated with Gold: Copper; Silver; Platinum—Application of Reagents—Quantitative Determination of Gold by the Wet Process—Determination of Metals, other than Gold, in Ores—Quantitative Estimation by Specific Gravity.

**Tests for Presence of Gold.**—A rude method of ascertaining the presence of gold in crushed quartz, or earthy ore, is by washing it in a hand-basin or horn spoon; but gold is often present in the matrix in grains or particles so minute that it cannot be detected by the eye, and sometimes it is quite disguised by admixture with other metals or minerals.

In these cases it is necessary to resort to chemical examination. In commencing mining operations the first point to determine is, of course, whether gold exists in any appreciable quantity; and if this be decided affirmatively, the next point is to determine the proportion or quantity of gold contained. The first process is termed testing for gold, or the qualitative examination of a suspected mineral; the second is the quantitative examination, or assaying process, which has just been described.

Sulphate of iron, protochloride of tin, and oxalic acid are the tests or reagents pre-eminently employed in seeking to determine the presence of gold.

**Preparations for Testing.**—Before applying these tests, the substance supposed to contain gold must be brought into a state of solution; and this can only be done by means of nitric or hydrochloric acid, as already mentioned. If the substance consists of earthy and quartzose matter, this must be reduced to powder by trituration in a mortar before it is subjected to the action of the solvent; but if the matter under examination be simply a metallic alloy, it can be dissolved without any previous preparation, and an excess of acid should be avoided. When the solution is effected the liquid should be evaporated to about one-eighth of its original bulk, and then diluted with 3 or 4 ozs. of water. The action of the reagents is as follows:—

(1.) **Sulphate of Iron or Green Vitriol.**—If a few crystals of this salt be dissolved in distilled water, and dropped into the suspected solution, the result is the precipitation of the gold, if any be present, in the form of a dark brown powder, which is metallic gold in a very fine state of division, as already described in connection with assaying of gold. The deposit formed when the sulphate of iron is added may be corroboratively proved to be gold by its being insoluble in nitric acid, but readily soluble in aqua regia.

(2.) **Protochloride of Tin.**—If to another portion of the nitric hydrochloric solution be added a small quantity of a solution of protochloride of tin (commonly known as salts of tin), there will be immediately produced, if any gold is present, a dark brownish-purple precipitate, known as purple of cassius. This substance is used in enamel and porcelain painting, and for giving to glass a tinge of a fine red colour. Its colour, though not a brilliant purple, but rather a reddish-brown, is characteristic, and after being once seen is not likely to be mistaken. Its appearance, when the chloride of tin is added to the liquid, affords an infallible proof of the presence of gold, for a very minute portion of that metal gives a manifest reaction when this test is employed.

When the first test (sulphate of iron) has been applied, and its evidence attested by the solution of the precipitated gold powder in aqua regia, the protochloride of tin may be employed to produce the purple of cassius in this solution also.

(3.) **Oxalic Acid.**—This substance, either in crystals or dissolved in water, causes, when added to the solution of gold, the precipitation of any of that metal in a form which varies with the temperature and strength of the solution.

The action takes places very slowly in the cold, but on applying heat there is immediately a copious evolution of carbonic acid gas, while hydrochloric acid is also formed during the process.

A crystal of oxalic acid, wetted with a solution of gold, becomes soon covered with a thin film of the metal. There are also other reagents which act on gold.

**Tests for Metals usually associated with Gold.**—In examining an ore or alloy supposed to contain gold, it is often an object of great importance to determine the nature of the metals with which the gold is associated. These may be of high value on their own account, and, according to the quantities in which they are present, may greatly affect the value of the ore or mineral in question.

(1.) **Copper.**—It has been shown that copper is almost always associated with gold, even in quartz, and that copper pyrites is one of the substances frequently mistaken for gold. When dissolved in acids, however, it gives characteristic reactions which render its presence easily distinguishable. One of the readiest tests is to introduce into the solution a piece of clean iron, when, if copper be present, it will be deposited on the iron in the metallic state, an effect which is due to a simple exchange of place between the metals, the iron passing into solution while the copper exhibits itself in the solid form.

Again, when ammonia is added in excess to a solution in which copper exists, it communicates to the liquid a rich deep blue colour. Ferrocyanide of potassium produces with copper a brownish red precipitate, even when the metal is present in very small quantity. Carbonate of soda precipitates copper from its hot solutions in the form of an apple-green compound, which is a carbonate of copper known, when artificially formed, as *verditer*, and when it occurs native, as *malachite*. Copper ore in the latter form exists abundantly in Australia—not, indeed, mixed with the gold, but constituting valuable mines, from which the ore is sent over to this country to be smelted in South Wales.

If a platinum wire be dipped in a copper salt solution containing free hydrochloric acid, and then held in the flame of a Bunsen burner, the flame will show an emerald green tint. This test is applicable to the detection of exceedingly minute quantities.

(2.) **Silver.**—It has been shown that gold is invariably alloyed with this metal, sometimes to a very large amount. In its separate state it is readily distinguished, not only by its white colour, but also by its specific gravity, which is only 10·4, or about one-half that of gold. It may be easily recognised by its chemical behaviour with reagents, in which respect it differs from gold by its solubility in nitric acid at all temperatures, and in hot sulphuric acid. On the contrary, with hydrochloric acid it forms a white curdy precipitate, which is the chloride of silver. If the nitric acid employed to dissolve it contains the least hydrochloric acid, the solution will become turbid by the formation of the chloride. Hence, when a mineral containing gold and silver is submitted to the action of aqua regia, the appearance of this white precipitate will immediately indicate the presence of the latter metal. The chloride of silver is soluble in ammonia, and may thus be distinguished from many other white precipitates; or it may be further tested by putting the precipitate into a crucible with carbonate of soda, and exposing the mixture to a strong red heat, when a button of pure silver will be obtained. By careful manipulation the amount of silver present may, in this manner, be accurately determined. If the mineral containing the silver be dissolved in oil of vitriol, the metal is readily detected by inserting a few fragments of copper, which will cause the precipitation of the silver in a pulverulent state.

(3.) **Platinum.**—This is another metal frequently associated with gold, and as it is one of the noble metals, and ranks in price between silver and gold, an ore which contains it in any quantity is of considerable value. The specific gravity of platinum is about 21·50—higher, therefore, than that of gold; indeed platinum, osmium, and iridium are the heaviest of all known metals, all three possessing about the same specific gravity; possibly osmium is slightly heavier than the other two. Platinum is of a light, steel-grey colour, less ductile than gold or silver, but more tenacious; indeed it will support greater weights on equal thicknesses of wire than any metal excepting iron and copper. It is distinguished from gold not only by its colour, but also by its extreme difficulty of fusion; it does not melt by itself in the highest heat of a furnace, but softens sufficiently to admit of forging and welding, and in the arc of flame of the voltaic current, or before the oxyhydrogen blowpipe, it admits of being fused even in considerable masses. On the other hand, it resembles gold not only in its high specific gravity, as already stated, but also in the fact that it resists the action of the simple acids, and is soluble only in aqua regia. This circumstance, together with its great infusibility, renders it of important use in many of the arts, and indispensable for various purposes in the laboratory. It is the metal universally employed for apparatus which require to be exposed to high temperature and powerful chemical agencies, without undergoing any change. It is, therefore, very important that miners in the gold fields should look carefully for this metal, which, being destitute of any glittering appearance, is likely to escape the notice of an ordinary observer.

There are, however, certain chemical reactions by which platinum may be readily distinguished and separated from gold in a solution. Sulphate of iron and oxalic acid, which precipitate gold, do not precipitate platinum. When the latter is dissolved in aqua regia, and the acid neutralised by carbonate of soda, the metal is deposited as a black powder, if the mixture be boiled with tartaric acid and soda. Further, the addition of chloride of ammonium and alcohol to a strong solution of platinum causes the deposition of a yellow crystalline precipitate, which is characteristic of this metal.

**Application of the Reagents.**—Having thus described the properties and characteristic reactions of the different metals which it is desirable to look for as being frequently associated with gold, let it now be assumed that the substance to be examined is a piece of auriferous quartz. This must be first reduced to powder, and then boiled for some time in an earthenware or glass dish with aqua regia. The solution is then diluted with water, passed through a filter, and allowed to cool. If any silver be present it will remain in the filter as a white precipitate mixed with the quartz.

To the liquid which has passed through the filter a solution of carbonate of soda is now added until no more effervescence takes place. This will precipitate all other metals which may be present except gold and platinum, which will remain in solution. The liquid is again filtered, and a solution of oxalic acid added until it ceases to produce effervescence and has a sour taste; then boil, and if gold be present it will be precipitated as a dark powder. The platinum, if any be present, will still remain in solution.

Decant or filter the liquid from the gold precipitate, and add to the former protochloride of tin, when a reddish brown colouring will appear if platinum be present. Or by boiling with tartaric acid and soda, the platinum will be thrown down as a black precipitate.

It has been stated that if silver be present it will be found on the first filtering mixed with the quartz. Wash this with ammonia, which, if copper be present, will produce a deep blue tinge. To the solution which comes through add hydrochloric acid until the smell of ammonia disappears, and the silver will be thrown down as a white curdy precipitate.

It is evident that other methods and reagents may be adopted. For example, the original solution in aqua regia may be concentrated by evaporation, until it is very much reduced in quantity; then add about three-fourths of its bulk of spirits of wine, and lastly a saturated solution of chloride of ammonium. By these reagents the platinum will be thrown down as a yellow crystalline precipitate, while the solution filtered from this, and treated with sulphate of iron or boiled with oxalic acid, deposits gold.

By carefully weighing the gold obtained, the amount present in a given quantity of the ore or alloy may be exactly determined.

**Quantitative Determination of Gold by the Wet Process.**—I will here give the method as indicated by Fresenius in his " Quantitative Analysis."

The quantity to be operated upon should be small, 12½ grains being a very convenient quantity, as it is only necessary to multiply the result by eight to obtain the percentage composition. If the gold be mixed with quartz or earthy matter, so much of this should be taken as may be judged from preliminary experiments to contain the amount of native gold above mentioned; and this must be triturated in a mortar with great care before subjecting it to the action of the aqua regia, as the quartz has to be reduced to a state of minute division, so as to liberate all the gold if possible, and not to leave it enveloped with silica, which would prevent the action of the acid.

Fresenius recommends that the substance under examination, having been weighed, be warmed with hydrochloric acid, nitric acid being gradually added until solution is effected, or it may be repeatedly digested with strong chlorine water. This latter method is especially applicable when the gold is present in very minute quantities, or mixed with other oxides which it is wished to leave undissolved.

The gold is always weighed as a metal, and its compounds may be brought into this state (*a*) by ignition, (*b*) by precipitation as metallic gold, (*c*) by ignition preceded by precipitation as tersulphide of gold.

(*a*) This method is applicable to all compounds of gold which do not contain a fixed acid. The compound should be heated in a covered porcelain or platinum crucible very gently at first, but finally to redness; the residue is pure gold. These results are the most accurate.

(*b*) In the method of precipitation as metallic gold, two cases may occur which may conveniently be considered separately, namely: (1) That in which the solution is free from nitric acid; (2) that in which it contains nitric acid.

(1.) If the solution does not already contain free hydrochloric acid, add a little; then add a clear solution of sulphate of protoxide of iron in excess; heat gently for a few hours until the precipitated fine gold has completely subsided; filter, wash with hydrochloric acid first, and afterwards with water; dry, and ignite. It will be found preferable to form the deposition in a porcelain evaporating dish rather than in a beaker.

(2.) When nitric acid exists in the solution, it should be evaporated on a water-bath to the consistence of syrup, adding from time to time hydrochloric acid. Dissolve this residue in water containing hydrochloric acid, and proceed as directed in (1). It should be noticed that cases may occur in which the residue does not dissolve to a clear fluid, but this is of no consequence.

(*c*) In the case in which ignition is preceded by deposition as a tersulphide, a stream of sulphuretted hydrogen should be transmitted in excess through a dilute solution of the compound under examination. The precipitate thus formed is speedily filtered off without heating, washed, dried, and ignited, as directed in *a*.

In the method *b*, the liquid should be tested before pouring off, to make sure of the last traces of gold having been deposited. This may be done in the following manner: when the whole of the precipitate appears to have settled to the bottom, a few drops of the supernatant liquor are taken out on the end of a rod, placed upon a surface of clean white porcelain, and tested with a drop of protochloride of tin. If no purple precipitate be formed, it is a proof that the whole of the gold has been thrown down. If a dark brown colouring, but still no subsidence, be produced, this will indicate the presence of platinum. If any precipitate be observed, more of the sulphate of iron or oxalic acid must be added to effect a complete precipitation of the gold contained in the solution.

The pure gold after ignition is placed in a watch-glass or small capsule, and accurately weighed in a pair of delicate scales, which should be capable of turning with a difference of at least a hundredth part of a grain. Supposing the gold to weigh 10·75 grains, and that of the amount of alloy submitted to experiment was 12·5 grains, or the eighth part of one hundred, it is evident that 10·75 × 8 = 86 is the percentage of pure gold present.

**Determination of Metals, other than Gold, in Ores.**—This is effected by powdering the sample very finely, and dissolving 10 grams in nitric acid, which need not be concentrated.

*Lead.*—To a portion of the solution which is somewhat diluted, add sulphuric acid. A white precipitate indicates lead; as insoluble lead sulphate. If instead of sulphuric acid, hydrochloric acid is added and a precipitate is formed, when this precipitate has collected and been digested with ammonia, on remaining unchanged it indicates that it is chloride of lead.

The soluble salts of lead behave with reagents as follows:—Caustic potash or soda precipitate a white hydrate freely soluble in excess. The carbonates of potassium, sodium and ammonium precipitate lead carbonate, insoluble in excess. Hydrogen sulphide and ammonium sulphide throw down black-lead sulphide.

*Zinc*, when dissolved in nitric acid will give a white precipitate of sulphide of zinc with ammonium sulphide, which is insoluble in caustic alkalies. Hydrogen sulphide causes no precipitate in zinc solutions of free mineral acids; but in perfectly neutral solutions, or with zinc salts of organic acids, such as acetate, a white precipitate is formed. Owing to the insolubility of sulphide of zinc in caustic alkalies, this test will readily distinguish zinc from all other metals. Caustic potash, soda, and ammonia, give a white precipitate of hydrate, freely soluble in excess of the alkali. Potassium and sodium carbonate give white precipitate, insoluble in excess. Ammonium carbonate gives also a white precipitate, which is redissolved by an excess. Potassium ferro-cyanide gives a white precipitate.

*Bismuth.*—A white precipitate of a hydrate is obtained when the solution in nitric acid is decomposed by an alkali. This precipitate is soluble in strong nitric acid. On diluting the solutions with water, a basic salt is precipitated, and an acid salt remains in solution.

The presence of bismuth in nitric acid solutions can be easily detected by adding water, or by the black precipitate of bismuth sulphide, insoluble in ammonium sulphide, which its solutions yield when exposed to the action of hydrogen sulphide.

Therefore if to a portion of the solution ammonium is added, the formation of a pulverulent, powdery, and cloudy precipitate indicates either lead, bismuth, iron.

*Iron* forms two classes of compounds—namely, the ferrous compounds and the ferric compounds.

The ferrous salts are thus distinguished : caustic alkalies and ammonia give nearly white precipitates, insoluble in excess of the reagent, rapidly becoming green, and ultimately brown, by exposure to the air. The carbonates of potassium, sodium and ammonium, throw down whitish ferrous carbonate, also very subject to the change. Hydrogen sulphide gives no precipitate, but ammonium sulphide throws down black ferrous sulphide, soluble in dilute acids. Potassium ferro-cyanide gives a nearly white precipitate, becoming deep blue on exposure to air. The ferro-cyanide gives at once a deep blue precipitate.

The ferric salts give the following reactions :—caustic, fixed alkalies, and ammonia give a reddish precipitate of ferric hydrate, insoluble in excess. The alkaline carbonates give a similar reaction, the carbonic acid escaping. Hydrogen sulphide gives a white precipitate of sulphur, reducing the sesquioxide to monoxide. Ammonium sulphide gives a black precipitate, slightly soluble in excess. Potassium ferro-cyanide gives Prussian blue. Tincture of gall-nuts strikes intense bluish-black with the most dilute solutions of ferric salts.

*Manganese.*—The characteristic test for this metal, when in nitric acid solution, is the addition of ammonium sulphide, which throws down an insoluble flesh-coloured sulphide of manganese. Caustic alkalies and ammonia give a white precipitate, insoluble in excess, becoming brown when shaken in the air. The carbonates of the fixed alkalies and carbonate of ammonia give a white precipitate, which does not change colour, and is insoluble in excess of carbonate of ammonia.

*Nickel* solution gives a deep blue precipitate with ammonia insoluble in excess. Caustic potash in excess gives a light green precipitate, and the potassium and sodium carbonate give a similar precipitate. The ammonium carbonate gives also a green precipitate, which dissolves with a blue colour in an excess of the reagent. Potassium ferro-cyanide gives a greenish white precipitate. Potassium cyanide gives a green precipitate, soluble in excess, when it takes a yellow colour, from which hydrochloric acid will throw it down again. Hydrogen sulphide will not throw it down from a concentrated solution. Ammonium sulphide throws down black nickel sulphide, soluble in excess, when the solution will assume a brown colour. The nickel sulphide thus precipitated is insoluble in dilute hydrochloric acid, but is soluble in hot nitric acid and aqua regia.

*Cobalt* will also give a blue precipitate with ammonium, but soluble in excess with a brown red colour. Potash gives a blue precipitate, which on heating turns violet and red. Ammonium carbonate gives a pink precipitate, soluble in excess. Potassium ferro-cyanide gives a greyish green precipitate. Potassium cyanide gives a yellow-brown precipitate, soluble in an excess; the addition of hydrochloric acid will not throw it down. Hydrogen sulphide will not throw it down from a concentrated solution. Ammonium sulphide throws down black sulphide of cobalt, insoluble in dilute hydrochloric acid.

*Mercury* gives a yellow precipitate with caustic potash or soda. Copper also precipitates mercury from its solutions. Hydrogen sulphide and ammonium sulphide give a black precipitate, insoluble in ammonium sulphide. Ammonium carbonate gives a white precipitate, insoluble in excess.

In treating an ore we also have a residue which remains insoluble, and which is not soluble in nitric acid. After decanting the nitric acid solution which has been tested for the metals as above indicated, the residue is heated so as to drive off the free nitric acid, and water and hydrochloric acid is added, which is now heated. The solution is filtered and hydrogen sulphide is added. If a precipitate is obtained which is soluble in ammonium sulphide—this precipitate is Arsenic, Antimony, or Tin.

If an ore is dissolved in hydrochloric acid and filtered, and hydrogen sulphide and ammonium or ammonium sulphide poured in, a yellowish precipitate indicates Zinc or Iron.

*Arsenic* is present when hydrogen sulphide is passed into a solution of arsenious acid, as above, and it throws down a bright yellow precipitate, soluble in ammonia. Cupric sulphate, on the addition of some alkali, gives a brilliant yellow green precipitate, soluble in excess of ammonia.

*Antimony* can be precipitated from its solutions as black powder by metallic zinc or iron. Copper also precipitates it as a shining metallic film, soluble in potassium permanganate, from which hydrogen sulphide precipitates it with a red color. Hydrogen sulphide precipitates antimony from its solutions with an orange or brick-red colour, soluble in ammonium sulphide.

*Tin* in solution as stannous chloride is determined by terchloride of gold, which throws down a brownish-purple precipitate, called purple of Cassius. Hydrogen sulphide and ammonium sulphide give a black-brown precipitate, soluble in ammonium sulphide containing excess of sulphur.

*Sulphur* by the action of the nitric acid is oxidized into sulphuric acid, which can be determined by the addition of barium chloride.

*Lime* is determined after the separation of all the other metals by the addition of ammonium oxalate, Having given the reactions by which most metals, as well as sulphur and lime, may be recognised, I add some general directions for determining qualitatively the same ingredients, when it is desirable to establish the character of an ore.

The solvents usually employed are hydrochloric acid, nitric acid, and aqua regia; and the solutions after filtering are kept separately. In examining an alloy, it is best to employ nitric acid. If we dissolve a piece of metal in nitric acid, and a white residue is observed at the bottom of the solution flask, the same is produced by either antimony, tin, or arsenic, as these three metals are converted by the action of nitric acid into insoluble oxides. If a small portion of the solution is poured into a test tube, and a small quantity of water added, a white precipitate will indicate bismuth.

There are certain ingredients in ores which do not dissolve in acids, such as silicates and sulphates of alkaline earths, and to render them soluble they are boiled with a concentrated solution of carbonate of soda. If any residue remains after such a solution it will be found that it is easily soluble in hydrochloric or nitric acid.

If an ore is to be examined which has a sulphate of baryta gangue, or other earthy sulphates, it will be best to fuse the mineral first with six to eight times its weight of carbonate of soda, whereby the insoluble barium sulphate is converted into a barium carbonate and sodium sulphate, which last is easily extracted with water, and the barium carbonate becomes easily soluble in dilute hydrochloric acid.

The metals heretofore enumerated are divided into various groups according to their behaviour with various precipitants; for their separation the following reagents are used:—

*Group I.*—When hydrochloric acid is used for their precipitation from nitric acid solutions the following chlorides are thrown down:—

Silver as white precipitate, completely precipitated.
Mercurous chloride as white precipitate, completely precipitated.
Lead as white precipitate, incompletely precipitated.

*Group II.*—Hydrogen sulphide throws down the following sulphides:—

| | |
|---|---|
| Lead as black precipitate. | |
| Mercurous oxide as black precipitate. | |
| Bismuth as black-brown precipitate. | |
| Copper as black precipitate. | |
| Antimony as orange precipitate. | Soluble in sulphide of ammonium. |
| Arsenic as yellow precipitate. | |
| Stannous oxide as brown precipitate. | |
| Stannic oxide as yellow precipitate. | |
| Gold as black-brown precipitate. | |
| Platinum as black-brown precipitate. | |
| Tellurium as black precipitate. | |

*Group III.*—If we have an ammoniacal solution, containing chloride of ammonium, the following metals will be precipitated as sulphides by means of sulphide of ammonium:—

Zinc as white precipitate.
Iron as black precipitate.
Manganese as flesh red precipitate.
Cobalt as black precipitate.
Nickel as black precipitate.

In examining a solution of ore, after some reagent has been added and a precipitate is obtained, enough is added to cause complete precipitation; the precipitate is collected on a filter and washed with water, and the filtrate is examined for the various metals it may contain, the indications being given in the order of the three groups here indicated.

*Group I.*—When a solution of nitric acid has been made, a portion, after filtering, is digested slightly with hydrochloric acid. The precipitate, as above indicated, is chloride of silver, mercurous chloride, or chloride of lead. The acid is carefully decanted, water is added, and the beaker allowed to stand till the precipitate is settled, it is then decanted, fresh water is added and the precipitate boiled with it and the solution filtered. Chloride of lead has gone in solution, and is precipitated with chromate of potash as yellow chromate. The chloride of silver and mercurous chloride have remained behind on the filter, and by pouring ammoniac solution on the filter the chloride of silver is dissolved out, and can be reprecipitated from the filtrate by the addition of nitric acid. The mercurous chloride remains on the filter and turns black on contact with the ammonium solution.

*Group II.*—The solution which remained after the metals of the first group had been separated is digested with hydrogen sulphide, and if a precipitate is obtained this is filtered and the precipitate examined. The precipitate is digested with sulphide of ammonium, and the metals it contains fall into two groups, namely (1), those soluble, and (2), those insoluble, in sulphide of ammonium.

(1.) As to those soluble in sulphide of ammonium. *Arsenic* is present when a yellow precipitate is obtained if a small quantity in a test tube is treated with hydrochloric acid, which precipitate becomes soluble in ammonium. *Antimony* is present if the precipitate obtained with hydrochloric acid is mixed with a little soda and fused on charcoal with the blowpipe, when a small brittle metallic button is obtained, which during fusion gives off white fumes. *Tin* is present if the precipitate is treated as above on charcoal, and a malleable button is obtained. *Gold* is recognised by adding sulphate of iron to the acid solution, when a brown powder of metallic gold is precipitated. *Platinum* is recognised when chloride of ammonium is added to the acid solution, evaporating to dryness and digesting slightly with alcohol, when a yellow powder of ammonium chloride platinum remains, which on heating turns to black metallic platinum.

(2.) As to those insoluble in sulphide of ammonium. The precipitate is heated in nitric acid, and if an insoluble residue remains it is collected on a filter. This may contain one or other of the following metals: *Lead.*—If a small portion of the solution, taken in a test tube, is neutralised with ammonium and sulphuric acid added, a white precipitate will indicate the presence of this metal. *Bismuth.*—After adding to a small portion of the solution an excess of ammonium, boil it, and if a white precipitate is obtained which is insoluble in an excess of ammonium, filter and dissolve the precipitate in hydrochloric acid and evaporate to dryness; add water. A cloud or milkiness will indicate bismuth. *Copper.*—If the above solution, after adding ammonium in excess, gives a blue colour it indicates copper, and a red-brown precipitate will be obtained if the solution be acidified with hydrochloric acid and cyanide of potassium is added. *Cadmium* is detected in presence of copper, by adding cyanide of potassium until the solution is colourless, and adding sulphuretted hydrogen. A yellow precipitate indicates cadmium.

The insoluble residue may consist of *Sulphur*, which is yellow or grey yellow; *Lead Sulphate* has a white heavy powder produced by the action of nitric acid on lead sulphide; or *Mercurous Sulphide*, whose presence is detected if dissolved in aqua regia, excess of acid evaporated, and on testing the residue with a clean piece of copper foil for mercury, which will cause a white film.

*Group III.*—The precipitate obtained by means of sulphide of ammonium is collected on a filter, washed and transferred to a beaker, and treated with cold dilute hydrochloric acid; and if a residue remains, this is collected on a filter. The solution, after heating, is digested with soda solution, and if a precipitate is obtained, this also is collected on a filter. *Zinc* is indicated when hydrogen sulphide gives a precipitate in the solution. The precipitate obtained with caustic soda being dissolved in hydrochloric acid, chloride of ammonium added and ammonium, there will remain in solution *Manganese*, which can be reprecipitated by sulphide of ammonium. The precipitate obtained with chloride of ammonium and ammonium is dissolved again in hydrochloric acid, and the solution is examined for *Iron*, which will give a blue precipitate with ferro-cyanide of potassium. The residue obtained in the first instance, when the precipitate obtained by means of sulphide of ammonium was treated with hydrochloric acid, is examined for nickel and cobalt; for *Cobalt* the residue is dissolved in aqua regia, a portion of the solution evaporated, the residue acidulated with acetic acid and dissolved in water. On adding potassium nitrate, after ten or twelve hours a yellow powder of a cobalt-potassium oxide separates, and nickel remains in solution. *Nickel* is thrown down as a green precipitate by soda solution.

**Quantitative Estimation by Specific Gravity.**—The high specific gravity of gold permits of its being estimated approximately in an ore or mineral of known specific gravity. The additional weight must be due to the presence of gold, and hence the amount of the latter may be estimated.

Supposing we have a rich specimen of gold quartz, and wish to know how much gold the same contains, we proceed as follows: we take the weight of the piece in the air, then suspend it by a string from the scales and let it plunge in a tumbler of water and note its weight. The principle, therefore, is based on the well-known theorem of Archimedes: "When a solid is immersed in a fluid it loses a portion of its weight, and this portion is equal to the weight of the fluid which it displaces; that is, to the weight of its own bulk of that fluid." The weight of the piece of quartz when weighed in water will be less; the lessened weight being determined, nothing more is required than to find by division how many times the latter is contained in the former, and the quotient will give the density or specific gravity of the sample under examination. If we find that this is no higher than 2·6, it will be unnecessary to proceed further; but if higher, the amount of gold present may be calculated from the difference.

Let A represent the weight of sample in air in ounces; let B represent the weight of sample in water in ounces; let C represent the specific gravity of gold; let D represent the specific gravity of the quartz; let X represent the weight of gold in sample.

Then A less X represents the weight of the quartz;

$\frac{X}{C}$ represents the weight of water displaced by the gold in the sample when weighed in water;

$\frac{A-X}{D}$ represents the weight of water displaced by the quartz in the sample when weighed in water;

$\frac{X}{C} + \frac{A-X}{D} = A - B =$ the total water displaced by the sample; or, to put the equation in another form—

$$X = \frac{DCA - DCB - AC}{D - C}$$

Knowing the numerical values of A, B, C, and D, substitute them in the equation; then multiply, subtract, and divide as indicated to find the value of X. By applying this method to a sample of gold quartz containing no impurities, such as pyrites, &c., the approximate amount of gold in the specimen can be determined without crushing. The specific gravity of gold can be taken as 19·2 and of quartz as 2·6.

Let us suppose—

$A = 26$ ounces, $B = 22$ ounces;

$$X = \frac{(2{\cdot}6 \times 19{\cdot}2 \times 26) - (2{\cdot}6 \times 19{\cdot}2 \times 22) - (26 \times 19{\cdot}2)}{2{\cdot}6 - 19{\cdot}2};$$

$C = 19{\cdot}2$ specific gravity of gold; $X = 18{\cdot}5$ ounces of gold;

$D = 2{\cdot}6$ specific gravity of quartz;

$$Q = \frac{DC(A - B) - D \times 2{\cdot}6}{19{\cdot}2 - 2{\cdot}6};$$

$X =$ weight of gold;

$Q =$ weight of quartz;

$$Q = \frac{2{\cdot}6 \times 19{\cdot}2\,(26 - 22) - 24 \times 2.6}{19{\cdot}2 - 2{\cdot}6} = Q = 7{\cdot}95 \text{ ounces}$$

of quartz.

# CHAPTER XX.

## *PHYSICAL AND GEOLOGICAL FEATURES OF GOLD-PRODUCING COUNTRIES.*

THE SIERRA NEVADAS—The "Mother Lode" of California—Hot Springs of California and Nevada—Other Features of those Countries—Death Valley—Hot Mud Springs—Telluride Ore Veins of Colorado—Origin and Position of Gold Deposits—Characteristics of Gold Deposits—Placer Mines of California—Formation of Gold Deposits—Gold in Australia—Chemical Geology of Gold.

**The Sierra Nevadas.**—As this mountain range of California—a region 300 miles long by 50 miles broad—owes its celebrity to the enormous amount of precious metals which have been produced from its numerous ravines, gullies, rivers, and quartz lodes, some observations here upon the geology of its mountain chains, and the distribution and probable origin of the precious metals, may prove of interest to the reader.

There is probably no country on the face of the earth which in respect of natural features offers to the student so many points of attraction as California, while for scenic effects it has no equal.

Among the numerous quartz lodes, the most prominent are those found near the line of separation of the granites and the schistose rocks. One great quartz lode—the "Mother Lode" of California—having a thickness of from 2 to 20 yards, can be traced for about sixty miles, cropping out in some places like an immense wall. Professor Whitney, State Geologist of California, says of this vein :—

"It is by no means a continuous bed or vein of quartz, but rather a series of nearly parallel belts or lenticular masses, with barren intervals between them, but yet arranged nearly in the same course, so that a straight line drawn in a direction of north 27° west from Jackson, in Amadar county, would either cut or pass through very nearly the whole of them.

"Yet this lode can be traced, by a succession of massive croppings and other easily recognised tokens of its presence, along a nearly straight line for a distance of about sixty miles. Throughout its entire length—from the Princeton and Josephine mines, on the Mariposa estate, to the Keystone mine, near Drytown, Amadar county—there occur at intervals vast wall-like outcrops of quartz, forming an impressive and picturesque feature of the landscape. The vein, in its upper portion, pitches toward the east at an angle of 45° to 50°, gradually approaching a vertical position as depth is attained.

"In thickness or width it varies from 1 foot to 50 feet of solid quartz, but sometimes expands to a width of 100 feet or more, in which case the vein matter is partially filled with fragmentary slate and broken quartz. It is everywhere regularly walled, and uniformly carries a well-defined 'gouge' or selvage of argillaceous clay, greatly facilitating the extraction of its ores. The western or foot wall is slate or greenstone, and the eastern or hanging wall, metamorphic slate, sometimes termed 'bastard granite' by the miners.

"The vein matter of the mother lode consists of grey or white quartz, impregnated with about 1 per cent. sulphurets, and rarely producing 'specimen rock' or free gold-bearing quartz. Generally the gold is disseminated throughout the vein stone in such minute particles as to be invisible to the naked eye.

"The ore does not occur in a continuous channel, but in a series of chimneys or pay shoots, separated by stretches of low grade ore, barren quartz, or argillaceous slate. The longitudinal inclination of these chimneys is sometimes northward, and at others southward. In vertical continuity none of these pay shoots have, up to the present time, been completely cut off; and the experience of over thirty-five years' exploration on this lode has demonstrated the fact that the pay shoots may continue to a depth where mining, unless there should be a great improvement in the grade of the ores, will cease to be profitable.

"The Sierra Nevada itself consists mainly of a central axis of granite, disturbed at intervals by volcanic action, in which is found only a small quantity of quartz. Along the western slope of this mass are thick bands of slates and shales, alternating with masses of serpentinous and trap rocks."

The slates of this locality belong (according to the same authority) to the Jurassic period, although in all probability some of those in Plumas county and elsewhere are of the Triassic. The great metallic belt lies on this western slope, commencing at about the Tegon pass, and extending to the northern limit of the state.

Judging from the immense quantities of quartz débris which are accumulated in the auriferous gravels of the country, it is certain that quartz accumulations must have formerly been in greater abundance and covered a larger area than would be inferred from the appearance of the existing veins of to-day.

In regard to mineral wealth, California occupies a prominent position, not only with reference to the quantities of the precious metals, but also to those metallic elements which occur but sparingly in other countries. Besides gold and silver, platinum, mercury, copper, lead, antimony, arsenic, tungsten, tellurium, molybdenum, bismuth, chromium, manganese, iron, nickel, cobalt, and zinc, are found in various ores.

If we look at the fissure veins on one hand, and observe on the other the immense masses of erupted rocks in these mountains, one cannot fail to suspect a connection and ascribe the production of fissures to volcanic forces, or to earthquakes that accompanied the eruption of molten rocks. But how, may we ask, were these cracks filled with vein matter? Neither the lateral infiltration hypothesis nor the injection hypothesis is sufficient to account for all peculiarities met here; but if we consider with Elie de Beaumont and H. Daubree that veins are the product of hot waters which, charged with mineral salts, entered these immense fissures from beneath and filled them to the brim, we may easily explain the presence of the quartz, of carbonate of lime, and of carbonate of lead and copper, as there is not a single metallic carbonate that would not be a little soluble in water charged with free carbonic acid.

The occasional banded structure of the veins is also thereby explained, the deposits being made at first upon the walls of the fissures, and gradually filling up to the centre. If the character of the mineral water changed after a certain period, a change of the nature of the deposit would of course be the result, and thus the banded structure of various ores—which also is seen occasionally in veins—would be accounted for. The deposition of vein matter from the hot waters is partially due to a loss of temperature of the water, partly to a loss of carbonic acid in contact with the air at the fissure surface. The formation of the metallic sulphurets might be explained by the subsequent entering of waters charged with sulphuretted hydrogen, converting the metallic carbonates into the sulphurets or sulphides.

No other country abounds so largely with hot springs as do California and Nevada, and it is especially in the vicinity of mining districts that they are encountered. Who would doubt that the system of thermal springs was formerly much more extensive in California than at present, when one sees the glaring and decisive marks they have left, the cones they have built, the coatings produced? A fine specimen of a snow-white coating over palæozoic limestone is seen on the eastern slope of the Buena Vista mountains, three miles east of Cerro Gardo, Inyo county.

On subjecting the surface conditions of California to a critical examination, one cannot fail to be struck by the great number of thermal springs. Indeed, there are few countries in the world with so large a number within an equal area. It is true, Montana and Idaho, with the famous geyser regions, contain also a considerable number; and Nevada still more. But these territories, more or less, resemble California in regard to volcanic formations. There is no doubt that a close connection exists between the latter and the thermal springs, both being due to the fact that the earth crust in this region covering the molten interior is thinner than elsewhere. Waters, after penetrating this crust, become heated, charged with salts, and are driven through other fissures to the surface by the power of the generated steam; and it is to this relatively thin crust that we may look for an explanation of the numerous earthquakes occurring in California.

Alexander von Humboldt, in describing his travels through the equatorial regions of South America, says that "the hypothesis of the relation between the volcanic formations and the existence of thermal springs seem *not* to be well founded," simply because he encountered springs of nearly boiling temperature issuing from Azoic rocks at Mariara on the Orinoco. He certainly would have formed a different opinion had he travelled through New Mexico, Nevada, and California, where gigantic peaks and wide-spread flows of volcanic material form most prominent features. It may safely be assumed that it is the exception for thermal springs to issue within very large distances from volcanic formations.

A fact of no little interest is the occurrence of a number of lakes in Eastern California containing as their main constituent carbonate of soda. These lakes, and a number of alkaline and saline flats, are situated east of the Sierra Nevada, and west of the great parallel ranges known as the White Mountains, Inyo and Argus ranges, which are filled with splendid fissure veins. The alkaline lakes are Owens Lake, Mono Lake, Black Lake, and the slough of Bishop's Creek.

In Lower California, as in Eastern and Central Nevada, and in Utah, we find vast depositions of salt, saline efflorescences, and alkaline deposits as well as salt-water lakes. A striking feature is Saline Valley, in Inyo county. If one stands upon the summit of the Beveridge range of the eastern slope, and looks towards the east—but down, down until the senses swim with the awful depth and distance—Saline Valley is seen stretching along the base of these mountains, a great basin, a desert valley, rimmed by volcanic hills, its wastes of billowy sand, its sloping dry washes of rock and boulder, its shining patches of soda and salt blazing under the burning sky. It is a desolate, dreary region, covering nearly 200,000 acres of sterile land. I mention it in this connection because it is related to the mineral deposits of the Inyo mountain ranges, in that its levels, where not buried under sand and rocks washed and shaken into it by generations of earthquakes and waterspouts, are vast deposits of almost pure soda and salt many feet in depth, the product of the lime formations of the Inyos.

Thousands of acres of soda have been located, and in one place a mine of the purest salt, of unknown depth, covering a thousand acres. As seen from the adjacent mountains it shines in the sun with a fierce glare, and is clearly visible for over fifty miles. Enough salt is here already prepared for the markets of the world, and each year adds to the already wonderful accumulation. The only question of its utility is one of cheap transportation. The soda fields here will also be of great value, as they can furnish fertilising material to redeem a State from sterility.

There are also enormous deposits of borax in a great many localities of California and Nevada, which are being worked on an extensive scale. Some borax deposits were found in Death Valley, Inyo County, which is one of the most remarkable geographical localities on the face of the earth. Having several years ago worked mines in that section of country, I deem the locality to be one of peculiar interest, and deserving of some special description here.

**Death Valley.**—This valley takes its name from the circumstance of a company of emigrants entering it on their way from Salt Lake to California in the year 1850. Very little was then known of the mountain passes, and the party made the fatal mistake of attempting a more direct route than the well-known emigrant road. They little knew the dreadful experience which was before them, or the sufferings they were to endure. The valley was to them a *cul de sac*, a region wholly unexplored. While seeking an outlet, they met with dangers and difficulties wholly unexpected and almost insurmountable. Finding it impossible to take their waggons over the mountains, they abandoned them, and while some of the party climbed the rugged and roadless passes, others, seeking water, miserably perished.

Those who escaped, in relating the horrors of their journey, told romantic stories of mines of gold and silver, all generally exaggerated, but which have induced others to visit the locality in search of the mythical treasures. Bennett, one of the emigrants, drank at a running stream of clear water, on the pebbly bottom of which (he said) he saw lumps of glittering gold. An unlikely story, for gold is seldom, if ever, seen under such circumstances. Another said he found a piece of white metal, which he took with him, not knowing its nature or value until months after, when, being at Los Angelos, and requiring a new gun-sight, he delivered the metal to the gunsmith, and was informed that it was pure silver.

This story, more absurd, if possible, than the first, has caused many parties of explorers to visit Death Valley in search of the "Gunsight Lode," which has never been found. While these expeditions have generally ended in disappointment, they have led to knowledge of the country, and the discovery of mines of antimony, silver, and gold, and some very important borax deposits. Telescope, Calico, and other mining districts were the result of such expeditions, and all these regions, I may say, have been closely examined and studied by myself.

Death Valley proper lies within the area bounded by the meridians 116° 30′ and 117° west longitude, and parallels 35° 45′ and 36° 30′ north latitude. Its direction is nearly north and south; length from Furnace Creek south 40 miles, average width 8 miles. The formation is a stratified, sedimentary rock—sandstone and limestone. I have only had occasion to explore its western border along the Panamint Range, whose highest point culminates in Telescope Peak, 11,047 feet above Death Valley, and whose topmost peak I ascended on the 18th of May, 1875, to make barometric observations.

The climate is most distressing to mankind. While in winter it is quite pleasant, in summer it is almost unbearable

The dryness of the air is so excessive that moisture is withdrawn from the body faster than it can be supplied through the system. From this cause frequent cases of death have occurred even when water was abundant, as it could not be drunk fast enough to supply the drain caused by the dessicative power of the dry hot air. I have had a similar experience in a neighbouring region; and I have known also instances of prospectors setting out to explore the Coso Ranges, who in crossing Panamint Valley have been seized with a fever, followed by delirium. Becoming bewildered, they have wandered about aimlessly, beholding in the far distance lovely images of green vegetation, brooks of clear water, and with blazing eyes and panting breath they have hurried on to reach the spot, even divesting themselves in their despair of every particle of clothing, till under the burning heat of the desert they have fallen in death.

Little vegetation can be found a short distance from water, excepting sage brush. Near the creeks only grass, willow, and mesquit brushes grow. The heat is intense, averaging from 95° to 105° Fahrenheit in the shade, and I have noticed it at the Modoc mine 126° in the month of July. Water can be found in abundance by sinking wells in almost any part of the valley.

The Indians mark the site of springs by placing large white stones at the mouth of each cañon leading from the valley in which there is water; and on looking up the cañon, other stones are seen similarly placed, which lead to water if followed. Elsewhere it is useless to look for natural springs.

**Hot Mud Springs.**—Near Coso District are hundreds of these springs, of great interest, some in constant motion, boiling and bubbling up mud. There is one oblong basin, 150 feet long by 75 feet wide, filled with clear alum water, which ebbs and flows every few minutes. It gradually rises from 4 to 5 feet, and as slowly sinks again. A white rock thrown into the spring can be seen to sink for a minute or more. The ground round about is hot. Half a mile west lie extensive banks of sulphur. From crevices steam issues, and on the rocks sulphur in beautiful crystals sublimes. Similar mud springs are found in the Colorado Desert. Four miles distant are low hills of obsidian, several extinct volcanic craters, and walls of lava and pumice-stone, all showing that volcanic agencies were once very active at this locality.

**Telluride Ore Veins in Boulder County, Colorado.**—This county is remarkable for its valuable belt of telluride ores, carrying gold and silver, which is found in the southern part of the county, the formation being three miles in width by about thirteen in length. Only twice before in the history of mining had telluride ores been encountered in any quantity—in the mountains of Transylvania, and in Calaveras County, California—but in neither instance did they appear in such novelty and abundance as in Boulder County.

The telluride belt, as at present known, extends in a northerly direction through Gold Hill, which is five miles from its southerly extremity. The country rock, which at the northern part of the belt is micaceous or gneissic schist, is in the remaining portion principally gneissic granite. All the lodes carry mineral of a similar character; but they may be divided into three classes, each distinguished by the kind of rock inclosing it. First, lodes in gneissic granite, embracing the richest and largest number of mines; second, lodes between granite and porphyry contacts; third, lodes in micaceous or gneissic schist, including the remaining veins, among which is the Smuggler, probably the largest telluride vein yet opened in the country.

Mr. John K. Hallowell, in a published monograph of Boulder County, gives the following description of the geological structure of the rocks of the district:—

"At a time when the system of elevations known as the Rocky Mountains had no existence, the space now occupied by them was an ocean of vast extent, but dotted here and there by islands of primitive granite, the earliest or oldest rock known. Around these islands sedimentary rocks were being formed, which were of various thicknesses and extent, and which afterwards, by pressure and heat, became metamorphosed. Such was the condition of the granite mountains Gold Hill and Big Horn, just east of it, in the olden age. Boulder City to-day is but ten or twelve miles east of them. This granite island shoaled off from twenty or twenty-five miles south. Both on the east and west sides of this primitive formation heavy beds of sedimentary rocks (afterwards metamorphic) were deposited, gradually filling the ocean bed to a level, when on the southern part of this island they overlaid it many feet in thickness. But compared to the thickness of the strata built upon the sides of the granite, the overlying part was a mere shell. The metamorphic rock on the east side only came as far north as Boulder Creek. North of that, and east of the mountains named, there was none. As this ocean gradually shallowed, on the metamorphic rocks was deposited the sediment that afterwards made the cretaceous rocks of the plains and foothills. At this place or island on the east side, a few hundred feet only of this age of sedimentary rocks were laid directly upon primitive granite. The other formations are entirely wanting. At this time we have a shallow, marshy sea, with granite islands showing a small distance above the water-level. Abundance of vegetation in surrounding marshes, which afterwards made the lignite coal beds of the cretaceous measures. That this condition of things continued for a long time, and with some slight oscillations of level, is evident by the different beds and quantity of the coals. Finally came the contraction and wrinkling of the earth's crust, the result of which was the Rocky Mountain system, which, as it arose from the ocean depths, brought up as dry land the whole system of the plains from just west of the Missouri River, as well as the adjoining lands through to the Pacific Ocean.

"The force that elevated the mountain range carried the granite islands up also. In the movement large fissures opened in the granite, miles in length, and parallel with the longest axis of the island. The metamorphic rocks broke at their thinnest place, viz. just over the granite, and stand to-day uptilted upon each side of it. Those strata to the east dip towards the plains, those west towards the range proper.

"The different dips of these metamorphic rocks puzzled me for over a year, as when working in Clear Creek County in 1880 I noticed it, and could get at no legitimate cause, although in South Clear Creek, a short distance above the forks, I could see where the strata broke; and at this place, on the north side of the creek, I found primitive granite overtopping with knife-blade streaks of tellurium ores. At that time I could not understand all of its significance, until a study of the foothills at Boulder City led me to connect the two points, and so solve the problem of the structure. At Clear Creek this split of the metamorphic rocks turned west a couple of miles, then south-west several miles. The bed of South Clear Creek occupies the chasm to the mouth of Chicago Creek, and this creek follows the rent in the metamorphic rocks almost to its source. Now this split and fissured granite held water like a sponge, the motion of the rocks gave great heat, the heated waters sought the surface through these fissures, at the surface the thermal waters rapidly evaporated, leaving their solids in the fissures. Thus were formed the veins of this tellurium belt.

"Here occurs a beautiful illustration on a stupendous scale of the theory of mineralogists, that the contents of veins are derived from atoms carried in aqueous solutions from the surrounding country rocks—that the contents of the vein are thereby governed by the character of the country rock. Here in the primitive granite exclusively do we get the veins carrying the tellurium ores and the sulphides contained in the other veins. In the parallel metamorphic rocks between the tellurium belt and the range we get the veins with the ordinary sulphides, but no tellurium.

"In one place only are tellurides known where the outcrop is in recent formations—that is, in the Bassick Mine of Custer County, Colorado. To my mind this indicates that the source of the ore supply extends down through all the overlying rock, to a fissure in the underlying primitive rock.

Consequently the deposit of ore in the Bassick Mine will be found to be inexhaustible by any means known at this date that man can work with.

"There are more of these tellurium belts in Colorado, some of them, perhaps, as extensive and rich as the described veins of Boulder County, but nowhere so accessible and well proven."

**Origin and Position of the Gold Deposits.**—Auriferous rocks, in whatever part of the world they be found, almost always exhibit a striking resemblance to each other; and although at times gold is discovered with granite and other rocks as a matrix, it is generally associated with quartz. These rocks are of the Silurian period, and it has been observed that the richest deposits are found where rocks of this age have been disturbed by volcanic action. "The most usual original position of gold" (says Sir R. Murchison) "is in the quartz ore veinstones that traverse altered palæozoic slates, frequently near their junction with eruptive rocks. Sometimes, however, it is also shown to be diffused through the body of such rocks, whether of igneous or of aqueous origin. The stratified rocks of the highest antiquity, such as the oldest gneiss or quartz rocks, have very seldom borne gold; but the sedimentary accumulations which followed, or the Silurian, Devonian, and Carboniferous (particularly the first of the three), having been the deposits which, in the tracts where they have undergone a metamorphosis or change of structure by the influence of igneous agency or other causes, have been the chief sources whence gold has been derived."

It is noteworthy that the great gold-fields of Australia extend without interruption along the slopes of the range separating the eastern seaboard from the Continent, while the gold-bearing mountains of California and British Columbia extend in an unbroken line along the western shore of America. Indeed, the great American Cordilleras, which is auriferous and metal-bearing throughout its whole length, extends in an unbroken line from Behring Straits in the north to Cape Horn in the south, bordering the coast of the Pacific Ocean along the North and South American Continents. The Australian range extends southwards across Bass's Straits, and northwards it crops out at intervals in the various islands, finally terminating in Japan. Thus the great basin of the Pacific has on each side an elevation of volcanic origin, from which at intervals gold is obtained.

**Characteristics of the Gold Deposits.**—The chief sources from which gold is obtained, are, firstly, alluvial deposits; and, secondly, quartz rock—commonly a vein of greater or less thickness, intersecting beds of granite or layers of slate. The former, consisting of sand and gravel, have been produced by the disintegration of the siliceous, granite, and other igneous and metamorphic rocks, and have been transported by the agency of water from the mountainous districts. In these beds it is invariably found in the metallic state, but is never quite pure. It usually contains a certain proportion of silver, and not unfrequently iron, copper, and small quantities of several other metals.

In the Californian gold region the gangue of the auriferous veins is invariably quartz, which is generally crystalline in structure, though sometimes partially vitreous. The veins are irregularly distributed among the metamorphic slates, but occur chiefly in the neighbourhood of crystalline and eruptive rocks.

The quartz of a veinstone is generally ribboned, so as to form a succession of layers parallel to the walls of the lode, and frequently some one or more of these laminæ are more productive than the rest. The metallic minerals, ordinary iron pyrites, blende and galena, arsenical pyrites, copper pyrites, and some silver ores, invariably accompany the gold.

"How gold came to be present in quartz veins" (says Professor Whitney) "is very hypothetical. All that can be said is, that it was evidently deposited from the solutions which gave rise to the formation of the inclosing quartz; and this appears the more evident from the presence of that metal in pyrites inclosed in siliceous incrustations, as well as from the fact of large quantities of gold having been formed in the interior of the stems of trees, which in deep diggings are often converted into pyrites.

"The constant presence of iron pyrites in auriferous veins, and when so occurring its invariably containing a certain amount of gold, suggests the possibility of this sulphide being in some way necessarily connected with the solvent by which the precious metal was held in solution. It has been shown that finely divided gold is soluble in the sesquichloride of iron, and more sparingly in the sesquisulphate of that metal. It is also well known that iron pyrites sometimes results from the action of reducing agents of the sulphate of that metal. If, therefore, sulphate of iron, in a solution containing gold, should become transformed by the action of a reducing agent into pyrites, the gold, at the same time being reduced to the metallic state, would probably be found inclosed in the resulting crystals of that mineral."

**The Placer Mines of California.**—In the early days of Californian history all the gold was obtained from the alluvial deposits, and after these became exhausted and the deep gravel mines were attacked, attention was drawn to the quartz mines. The "shallow placers" were formed at a more recent period than the "deep placers," which were deposited by an entirely different river system from that now in existence. These ancient rivers evidently flowed at a much higher level, and frequently at right angles to the valleys of the present day. The beds of these rivers—which bear the name of the ancient Californian Pliocene rivers—belong to the Tertiary age, and are most likely of the later Pliocene epoch. Their apparently unlimited deposits of metalliferous gravels are now worked on a gigantic scale by what is known as the "hydraulic process," of which an account has been given in the first chapter of this work.

**Formation of the Gold Deposits.**—In further elucidation of this question, it will be well to give the views of

Professor B. Silliman.* It is susceptible of proof (he says) from numerous well-established facts that at the close of the geological epoch, just prior to the appearance of man upon the earth, the whole of the western slopes of the Sierra Nevada Mountains were, below a certain horizon, covered by a vast spread of alluvium, owing its origin, probably, to the action of extensive glaciers, which have left the evidence of their former presence everywhere in the higher Sierra. The glaciers furnished the transporting power that brought from above the fragments which, by long-continued action of running water, were worn into the smoothly rounded boulders, gravel, and sands forming the gold-bearing alluviums.

The melting of the glaciers as their lower skirts reached the warmer zones furnished the water for these ancient rivers, whose beds are now found far above the level of the present river system, and whose courses are generally crossed by the valleys of the modern streams. This condition of things continued long enough to permit the accumulation of beds of gravel, the gold-bearing alluvium, to a depth and extent unknown anywhere else in North America; and, if we speak of auriferous deposits, unequalled elsewhere in the world. Of the thickness of this accumulated material we have evidence in numerous places, where it has been protected from the action of subsequent denudation by a capping of volcanic materials. In many such places it reaches a thickness of 500 feet. Usually, however, it has been denuded to one-half of this thickness, often less, and in many regions has been swept completely away.

Subsequent to the glacial and alluvial epoch to which the gold-bearing gravels are referred, there was a period of intense volcanic activity, the evidence of which is seen most conspicuously in the Table Mountains of California, so-called, which are cappings of basalt forming highly characteristic ranges, which portions of the ancient gold-bearing gravels are extensively explored in Tuolunine County by tunnels driven beneath the basalt cappings into the ancient river beds.

---

* As given in Mr. R. W. Raymond's Official Report on Mines and Mining.

Following the outpourings of the volcanic matter there has evidently been an epoch of very active denudation by running water, which has broken up and removed the volcanic cappings, leaving them entire only here and there, as landmarks showing the ancient levels, and sweeping away also vast areas of the old alluvium and redistributing it as secondary or shallow placers at lower levels. This denudation was probably consequent on the sudden disappearance of the vast system of glaciers which up to that time crowned the entire range of the Sierras with ice. It was greatly more energetic in the southern portion of the Sierras than in the northern, where the mass of ancient alluvium remaining is much greater than it is in the former region. The extent of the ancient alluvium, as well as the energy of the power which produced it originally, and subsequently denuded it, becomes apparent on a study of the phenomena, carrying to the mind an overwhelming conviction.

It was these extensive deposits of gold which attracted the attention of the first adventurers, and were by them called "hill diggings;" but their real nature and significance were not at first fully understood, and, being generally much above any sources of water supply then available for washing, they received but little attention. Especially were they overlooked while the rich spoils derived from their secondary removal by denudation were available with no other means than the miner's pick, pan and shovel, upon those productive "bars" of adjacent rivers and in rich " gulches," where the gold (derived in large masses from the denudation of the ancient alluvium) lay open to the first comer in a concentrated form. So complete was the removal of the gravel in some of the southern counties that the gold, left behind by its weight, lay upon the naked rock, covered only by a few inches of vegetable mould—as at Mokolumne Hill, where, in the limits of a single claim 15 feet square, the precious metal to the amount of £10,000 fell, in one instance, to the lot of a single adventurer.

**Gold in Australia.**—From the researches of Mr. Selwyn, the Government geologist, it would appear that the gold-bearing strata of Victoria belong to a geological period much more ancient than that of the auriferous rocks of California. They belong, in fact, to the lower Palæozoic or Silurian epoch, and are (as Sir R. Murchison observed) almost identical in character with the Ural mountains. These rocks of Victoria are remarkable for the almost total absence of limestone bands, the number and extent of the quartz veins, and the constantly recurring protrusions, at short intervals, of granitic and occasionally of plutonic trappean rocks.

The quartz veins vary in thickness, from that of a sheet of paper to many feet; the thick veins being found in the lower portions of the series, but they are as a rule less productive than the thinner ones in the upper beds. The mines are frequently worked to immense depths.

Although the metamorphic rocks of Victoria appear to belong to a more ancient geological epoch than those of California, it should be observed that the drifts or deep diggings of the two countries seem to belong to the same geological epoch, and are both covered by a volcanic mass. Mr. Selwyn has concluded that there must exist two distinct series of quartz veins; the older ones having been formed previous to the Miocene period, and being comparatively barren; while the newer ones formed subsequently, but previous to the Pliocene period, are productive.

South Australia, Western Australia, Queensland, Tasmania, and New Zealand—especially the latter—are also gold-producing countries.

**Chemical Geology of Gold.**—Professor Phillips, in the Proceedings of the Royal Society for 1868, has given the following as the result of his examination of the geology of California, regarded from a chemical point of view:—

"(*a*) Quartz veins have generally been produced by the slow deposition from aqueous solutions of silica on the surfaces of the inclosing fissures.

"(*b*) From the general parallelism with its walls of the planes of any fragments of the inclosing rock which may have become imbedded in a vein, it is to be inferred that they were mechanically removed by the growth of the several layers to which they adhered, and that a subsequent deposition of quartz took place between them and the rock from which they had become detached. In this way were introduced the masses of rock known as 'horses.'

"(*c*) The formation of quartz veins is due to hydro-thermal agencies, of which evidences are still to be found in the hot springs and recent metalliferous veins met with in various parts of the Pacific coast.

"(*d*) From the variable temperatures at which the vacuities in their fluid cavities become filled, it may be inferred that they are the result of an intermittent action, and that the fissures were sometimes traversed by currents of hot water, whilst at others they gave off aqueous vapour or gaseous exhalations. This is precisely what is now taking place at Steamboat Springs in Nevada, where the formation of a vein is in progress, and from which currents of boiling water are often poured forth, whilst at other times the fissures give off currents of steam and heated gases only.

"(*e*) That gold may be deposited from the same solutions which give rise to the formation of the inclosing quartz, appears evident from the presence of that metal in pyrites inclosed in siliceous incrustations, as well as from the fact of large quantities of gold having been formed in the interior of the stems of trees, which in deep diggings are often converted into pyrites.

"(*f*) The constant presence of iron pyrites in auriferous veins, and when so occurring its invariably containing a certain amount of gold, suggest the possibility of this sulphide being in some way necessarily connected with the solvent by which the precious metal was held in solution. It has been shown that finely-divided gold is soluble in sesquichloride of iron, and more sparingly in the sesquisulphate of that metal. It is also well known that iron pyrites sometimes result from the action of reducing agents on the sulphate of that metal. If, therefore, sulphate of iron, in a solution containing gold, should become transformed by the action of a reducing agent into pyrites, the gold, at the same time being reduced to the metallic state, would probably be found enclosed in the resulting crystals of that mineral.

"(*g*) The silica and other substances forming the cementing material of the ancient auriferous river-beds, have probably been slowly deposited at a low temperature.

"It should be mentioned that Professor H. Wurtz differs from Sir R. Murchison as to the origin of gold, and has suggested a very plausible theory, ascribing it to a totally different cause. Having observed that aqueous solutions of sulphate of sesquioxide of iron were capable of retaining in solution a small but appreciable amount of metallic gold, and coupling this with the general belief that gold and gold-bearing sulphurets have been of aqueous deposition, he suggests that all the gold was held in solution in a 'Prozoic thoroughly oxidated ocean,' the sulphurets of iron now so abundant in crystalline rocks being dissolved in it as persulphates, and this persulphate in its turn being quite capable of retaining in solution all the gold in the earth's crust. When, however, in this lifeless ocean organic existence commenced, together with the accompanying deoxidising processes, the ferric became ferrous salts, and these being incapable of retaining gold in solution, it was deposited. Wurtz refers to one of the two richest gold-fields yet discovered—that of California—as disproving Murchison's general conclusion, that all gold-fields are of the Lower Silurian Age, the Californian gold-bearing formations being as recent as the Jurassic."

As a conclusion of the whole matter, one may say that the origin of gold is so far completely enveloped in mystery. Scientific research has not as yet thrown sufficient light on the question *how* the gold originated in the rocks, and the chemical hypothesis above enunciated is an *hypothesis* only, without having (to many minds) any substantial evidence upon which it may be accepted as solid, demonstrable truth.

**The Conglomerate Beds of the Witwatersrand.**—The Witwatersrand (White Water Ridge) gold fields are situated on the central plateau of the Transvaal, on a well-defined ridge having an elevation of 4,500 to 6,500 feet, and running nearly east and west, which forms the northern fringe of a basin whose southern edge is the Heidelberg range, about 30 miles distant. Up to the present the length of the east and west axis of the basin is not yet determined. To the north of the Witwatersrand are large masses of diorite, and in the basin are numerous dykes of this material, filling cross and lateral fissures, and "faulting" the reefs in numerous places. The gold-bearing formation consists of beds of sandstone, quartzite, conglomerates (locally called banket), and frequently shales.

The northern part of the basin is strongly tilted, and has been uplifted by the intrusive rocks, but at a shallow depth the strata flatten.

The gold is carried in payable quantities in several of the most northern conglomerate beds, called the main reef series, which are interstratified in the sandstone and quartzite, and these consist of a mass of waterworn quartz pebbles, which are cemented together by quartz sand, argillaceous and talcose matter, and oxide of iron, which in depth changes to iron pyrites. The gold is contained in the cementing material, and not in the pebbles.

Nearly all the mines on these fields are now in the pyritic zone, as the upper levels, where the free milling or oxidized ores occur, are now worked out in most of the mines, and the average contents seem to remain the same, with the difference that more of the gold seems to be locked up in the iron pyrites, and it is not yet determined if with greater depth the per centage of pyrites will increase and the free gold diminish.

The deep levels like May Deep, Village Main Reef, Champ d'Or Deep, Durban Roodeport, and Henry Nourse Deep, at a depth of over 900 ft., and Geldenhuis Deep, do not show at 500, 600, to 700 ft. depth any remarkable changes, and the cores from the deep bore-holes, which probed the reefs to a depth of 2,200 ft,, are said to show a large percentage of free gold.

I regret that space does not permit me to enter into a detailed description of this vast and interesting gold-bearing area, but suffice it to say that explorations appear to have sufficiently demonstrated that the gold mining industry will be able to maintain itself for the next twenty to twenty-five years, and that profitable mining will be carried on in the majority of the now paying mines till a depth of over 2,000 ft. be reached, and, in some instances, where the ores are the richest, even to a vertical depth of 3,000 ft.

The dykes do not seem to thicken at greater depth, and do not seem to affect the beds, as was supposed. The dip of the beds at greater depth seems to be 30 degrees on an average, and, therefore, over the greatest portion of the Witwatersrand every additional 1,000 ft. of vertical depth means a distance on the dip of the reef of say 2,500 ft. From this the conclusion can be drawn, that, if the banket beds will pay to be mined to a vertical depth of 3,000 ft., the Main Reef series will be worked for 7,000 to 8,000 ft. on the incline, and some eminent authorities maintain to 10,000 ft.

## CHAPTER XXI.

### *MILLING OPERATIONS IN THE TRANSVAAL.*

Gold Mills in the Transvaal—Mode of Treatment—The Deep Level Crown Deep Mine—Results of Concentration—Treatment of Slimes—Crushing the Ore—Grizzlies—Crown Reef Mine—Ore Sorting Floors—Langlaagte 160-Stamp Battery—City and Suburban Battery—Paarl Central Battery—Champ d'Or Deep Level Battery—Water Power—Inside Amalgamation—Duty of Stamps.

**Gold Mills in the Transvaal.**—There were in 1894 about 2,500 stamps at work in the Witwatersrand district alone, and the numerous other localities where gold is found also give employment to a number of stamps, with the prospect that the number of mills in operation in the state will continue to increase. There is no question that the region may be considered as the richest gold country yet discovered.

The milling operations differ very little from those carried on in the United States, and as some of the largest batteries have been erected by American and English makers—principally Messrs. Fraser and Chalmers, and the Sandycroft Foundry—the experience gained in California, the Black Hills, and other localities has been applied to the Witwatersrand ores. In every respect these gold fields may be proud of the achievements which have been accomplished in every engineering department, the operations of which are being conducted by some of the best-qualified and most experienced men in the world. Everything which pertains to perfect milling operations exists there, and the only deficiency is to be found in the concentration department, where no doubt much more will be done.

So far no machinery has been found to supplant the battery, and I regret that I have to point out in connection with the treatment of the Witwatersrand conglomerates the one and only great fault of the stamp-mill—namely, the unevenness of the product which is obtained in crushing in the battery. I have already dealt with this point in the chapter on the erection of a cyanide plant, and, owing to the irregularity of the product, it has been impossible to obtain satisfactory results by concentration up till now, as no attempt has been made to introduce a proper system of sizing or classification.

A great many of the batteries are equipped with the Frue vanner; some have 2, some 3, and some 4 vanners for every 5 stamps, and, as far as it goes, this excellent machine is doing all it can; but still the tailings which leave the vanners assay 3 to 5 and over 7 dwts. at some of the richer mines. At the Alaska Treadwell mine the tailings, after leaving the Frue vanner, only assay ½ dwt., so it clearly proves that it is not the fault of the machine, but that the character of the Witwatersrand ores requires a preliminary classification. There is no question that it will pay to concentrate instead of allowing the tailings to go direct to the cyanide works.

At the Crown Reef mine, they found the cost of extracting the concentrates by Frue vanners too high, so they introduced a system of classification by means of spitzluten, whereby the heaviest particles of pyrites and sands are separated at a cost of 10d. per ton. It is clear that during the short period of treatment which the tailings undergo in the cyanide works, the extraction from the pyritic portion is unsatisfactory. It must pay, therefore, to concentrate, as the concentrates undergo special treatment and a high percentage of extraction is obtained.

Reckoning on a basis of 600 tons, which are obtained at this mine as concentrates at a cost of 10d. per ton, it was found that all the gold obtained from these concentrates was a clear gain, minus working expenses, because the cyanide works would not have given any more gold if these 600 tons had remained in the tailings, and this small improvement gives a monthly profit of £1,700.

The arrangement of the Frue vanners at the Wemmer mine is shown in Plate XX.

Missing Page

During August, 1894, the results were as follows :—

| | | | |
|---|---|---|---|
| Crown Reef Mine | 435 tons | 577·70 oz. | 1·32 oz. per ton. |
| Glencairn Mine | 14·25 ,, | 96·10 ,, | 6·70 ,, ,, |
| Langlaagte Estate | 360 ,, | 1037·0 ,, | 2·88 ,, ,, |
| Langlaagte Block B | 152 ,, | 262·10 ,, | 1·72 ,, ,, |
| Robinson Mine | | 1160·00 ,, | |
| Simmer & Jack Mine | | 143·50 ,, | |
| Rand Central Works | 300 ,, | 949·11 ,, | 6·33 ,, ,, |
| ,, ,, | 52 ,, | 125·00 ,, | 2·40 ,, ,, |
| Robinson Works | | 2371·00 ,, | |
| Simmer & Jack Works | | 324·10 ,, | |
| Total | | 7045·60 ozs. | |

The Robinson and the Simmer and Jack companies buy concentrates from other mines, and, therefore, the second figures represent custom work. The Rand Central works only purchase or do custom work. At some of the mills, especially in those erected in earlier days, the tailings are lifted up by wheels; at the Jumpers are two large tailing wheels, 40 ft. in diameter, constructed of timber. These lift the tailings into a series of launders, from which they are distributed on to the vanners, which are placed in an independent building (see Plate XV., facing p. 426, and Fig. 188, p. 614). In this building fifty vanners are at work.

In Plate XXI. is shown the battery at the Wemmer mine, with its fine tailing wheel, which lifts the tailings into the launder, placed on trestles, whence the pulp flows by gravitation to the cyanide works. As will be seen from the illustration, the bearings of the wheel rest on strong pillars of masonry.

The tailings at the Jumpers, before they go on to the vanners, assay 5 dwts., and contain on an average 3 per cent. of iron pyrites, and 40 per cent. (or 2 dwts.) are taken out by the vanners. The concentrates are taken out nice and clean and sold to the Rand Central works. They assay 3 to 4 oz., and it costs £2 per ton to save each ton of concentrates. In spite of the high charges paid for treating the concentrates at the

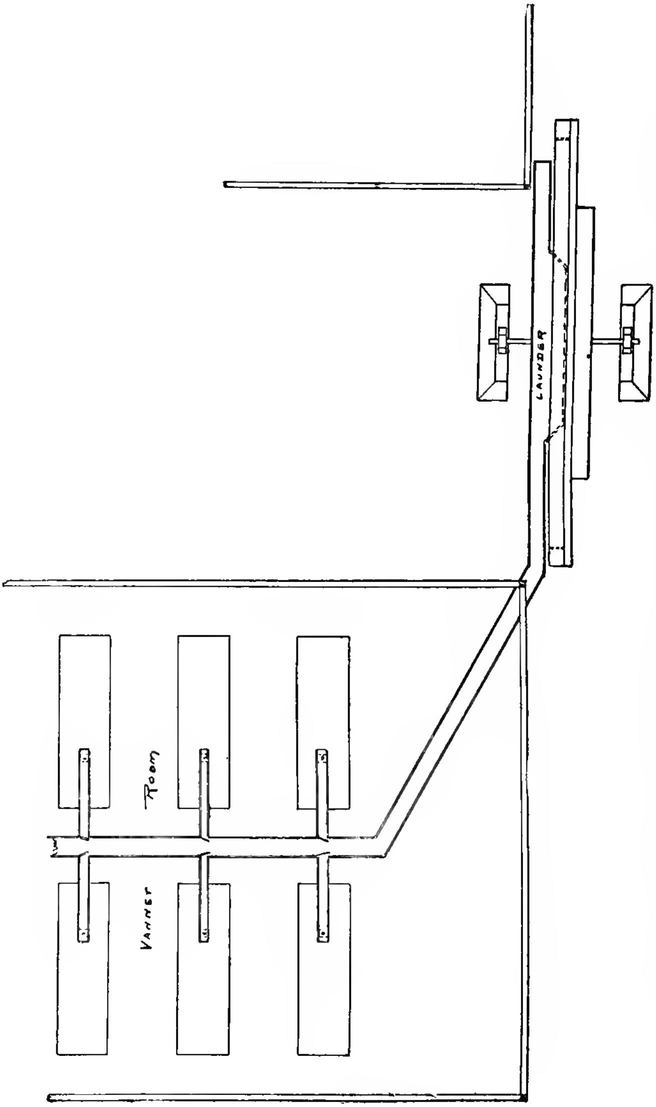


Fig. 188.—Plan of Jumpers Mine Vanner Room, showing Method of Distributing Tailings to the Vanners.

Missing Page

chlorination works, and the cost of concentration, a very handsome profit is made every month, and also a considerable quantity of gold is scraped from the vanner plates and amalgam savers every month, which goes far towards making up the profit.

At the Robinson it costs £1 0s. 10d. to get each ton of concentrates.

At the Langlaagte Estate mine, during the six months ending 30th June, 1893, 107,257 tons of ore were treated, giving 700 tons concentrates, or 0·65 per cent. of the tonnage crushed, at the following cost for working:—

| | £ | s. | d. |
|---|---|---|---|
| Frue Vanners at 4,704d. per ton . . | 2,102 | 2 | 5 |
| Pumping expenses and maintenance . | 150 | 0 | 0 |
| Depreciation . . . . . . | 600 | 0 | 0 |
| | 2,852 | 2 | 5 |
| Cartage and chlorination at £5 per ton (700 tons) . . . . . . | 3,500 | 0 | 0 |
| Total . . . | £6,352 | 2 | 5 |

| | £ | s. | d. |
|---|---|---|---|
| Mining concentrates per ton . . | 4 | 1 | 5 |
| Treatment of concentrates . . . | 5 | 0 | 0 |
| Total . . . | £9 | 1 | 5 |
| Value of concentrates per ton . . | 13 | 10 | 0 |
| Less cost . . . . . . . . | 9 | 1 | 5 |
| Profit per ton of concentrates . | £4 | 8 | 7 |

For the six months ending 30th of June, 1894, the following returns were made from the Witwatersrand gold fields:—

| | | |
|---|---|---|
| Number of tons milled (tons of 2,000 lbs.) | | 1,370,240 |
| Number of stamps running . . . | | 2,565 |
| Yield of gold from mills by amalgamation | oz. | $638,850^{14}$ |
| ,, concentrates . . | ,, | $36,642^{15}$ |
| ,, tailings . . . . | ,, | $270,912^{17}$ |
| Total yield . | oz. | $973,737^{19}$ |
| Value of yield . . . . . | | £3,355,486 |

At the Village Main Reef and Champ d'Or Deep, Mr. J. S. Curtis has introducd the Scoular table, which is a bumping table with side discharge. This machine does very good work, and certainly offers the advantages of being cheap and requiring little care and attention to keep it in running order.

**The Deep Level Crown Deep Mine.**—There can be no question of the immense values which the so-called Deep Level properties represent, because it is almost impossible to predict now what improvements will be made in the next ten or twenty years in the methods of mining. To give an idea of the extent of the operations required to open and develop one of these undertakings, I will give some items regarding the Crown Deep Mine.

This mine has 191 claims (150 by 400 ft.) about 4,800 ft. on the strike of the reef and 2,400 ft. on the dip, or the north and south line. The north line of the property, is on an average 1,500 ft. from the outcrop of the Main Reef series. The ground is being opened by means of two four-compartment shafts, very similar to those we find on the Comstock Lode. These shafts are 20 by 6 ft. in the clear. The east shaft is only 235 ft. from the north line of the company's ground, and the west shaft, 2,000 ft. distant, is 800 ft. from the north line.

Two, and perhaps three of the banket beds will be worked by means of these shafts. When the second bed is reached the vertical shaft will be turned on the incline as shown in Fig. 189. The west shaft will cut the reef at a depth of 1,400 ft., but at 900 ft. a cross-cut north will be driven to intersect the reef, which will be 670 ft. long, and connections will be made with drives from the east shaft, where the reef has been intersected at a depth of 910 ft. The work is done by compressed air drills, the cost of sinking has been £26 per ft., and the average rate of progress has been 66 ft. per month. The cross-cuts are driven by means of air drills at the rate of about 30 ft. per week. It is intended to develop 500,000 tons of ore before crushing operations begin, and a 200-head battery will be erected with a monthly capacity of 27,000 tons.

At the Geldenhuis Deep, Henry Nourse Deep, Roodepart

Deep, Village Main Reef, and other important deep level properties, developments are being pushed ahead rapidly, and within three years about 1,000 stamps will be at work on several of these properties, which will largely contribute to the output.

**Results of Concentration.**—The introduction of classification will allow of coarser crushing than at present, and the duty of each stamp will be increased with smaller amounts of slimes. The most essential point to be considered is the cost of manipulation, and it has to be found out if the present method, although not the most perfect, may not after all from a commercial standpoint be the best, and if a direct high percentage of extraction would be consistent with the higher cost of manipulation.

At a great many mines blanket strakes, 12 to 16 ft. in length, are placed below the copper plates. The blankets are washed every two hours into wooden boxes. There are three sets of blanket strakes on each table, and, while washing one set, the pulp is defleted over the other two, causing no interruption to the working of the battery. At the George, Goch, and some other mines corduroy was used with success instead of blankets. At some mills the blanketings are ground and amalgamated in Wheeler pans (Jumpers, Simmer and Jack), or in Berdan pans (Meyer and Charlton), or in amalgamating barrels (Stanhope).

When running on oxidized ores, a so-called black sand settled on the copper-plates, and adhered so firmly that the stamps had to be hung up, and the plates cleaned by brushing the black sands down and collecting them. A great deal of amalgam was brushed down with the sands, and these are treated in the amalgamating pans. Of course, now the mines have reached the pyritic zone, blanket strakes are getting out of date, and Frue vanners take their place.

The cost of obtaining clean and rich concentrates by means of Frue vanners seems to be about £1 to £2 per ton saved. I believe that it will eventually resolve itself into the preparation of a middling product, assaying from 1 to 2 oz., which will be treated by cyanide, and instead of treating thousands of

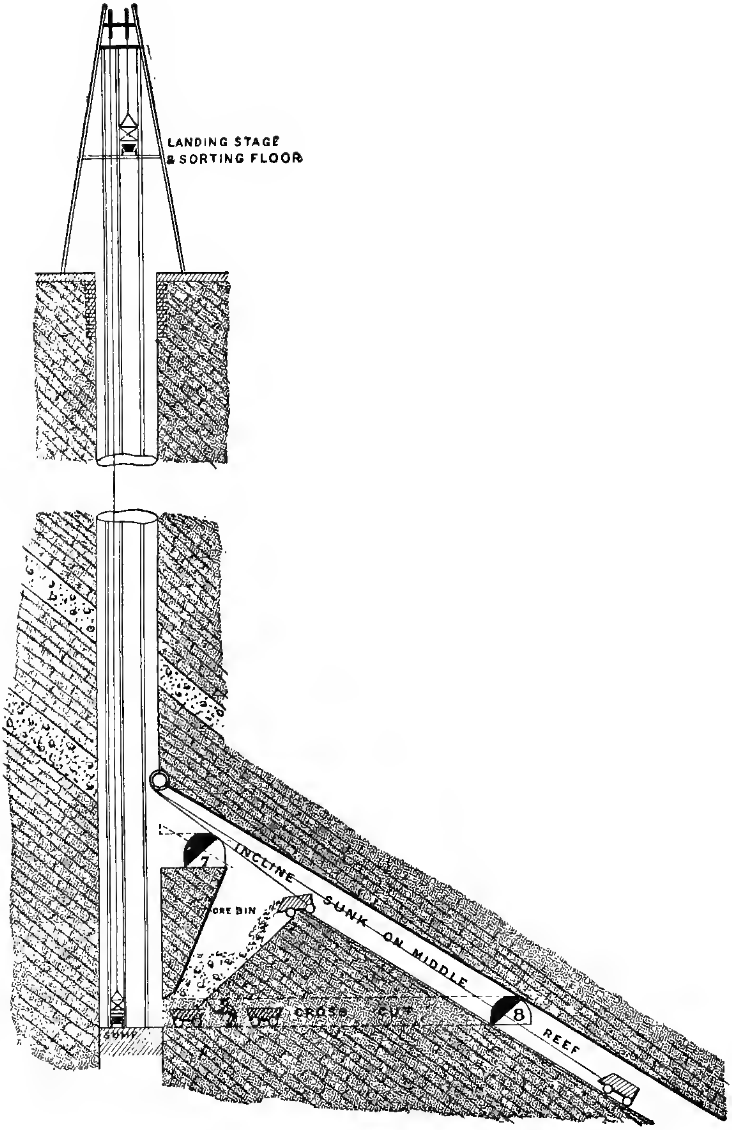


Fig. 189.—Deep Level Shaft in cross-section, showing Vertical Shaft continued as an Incline Shaft. Scale 1:500.

tons of tailings per month at every mine, the quantity will be reduced to, say, 20 or 30 per cent. of their weight, and effect a great economy in the working of the cyanide plant. The question is now being studied and will take time and money to work out. Of course the slime problem has to be considered separately, as it is hardly possible that any concentrator will deal with them successfully.

**Treatment of Slimes.**—According to the latest information received from Johannisberg, Mr. Bettel seems to have solved the slimes problem, to which I have already referred. Mr. Butters has, in one of his reports, stated that the gold annually lost in slimes amounts to £1,344,000.

Mr. Bettel uses some cheap chemical, by means of which he precipitates the slimes from the Primrose battery (100 stamps) in a reservoir 30 feet square by 10 feet deep. The precipitation of the fine suspended particles takes place very rapidly. The cost of the chemical used per ton of slimes is ½d., and in the leaching vats it is said to prevent the decomposition of cyanide. Having precipitated the slimes, he next deals with them in their sludgy condition by pumping them up to the vats by means of a centrifugal pump, sufficient water being added, of course, to keep the mass in a semi-liquid condition suitable for pumping to an upper launder, where it is mixed with a weak cyanide solution, and passes into the vats. He employs a ·03 per cent. solution, and the gold is dissolved in the course of a few minutes.

From the vats after settlement, the solution is syphoned off and passed through the extractor boxes. After syphoning off the cyanide solution, a water wash is given, the sludge being agitated by compressed air instead of agitators, as suggested by me in the subsequent chapters. Mr. Bettel has found out by his experiments that gold in slimes is rapidly dissolved in weak cyanide solutions, and the same has been discovered by Mr. Ottokar Hoffman in dealing with silver ores in his trough lixiviation system with hypo-solutions. A vat holding 10 tons is treated in 24 hours.

We are told now that Mr. Bettel, in the precipitation of the gold from the weak solution, obtains results before unheard of in the precipitation by zinc from weak cyanide solutions, and consequently he has solved the problem which in the subsequent chapter I mentioned had not been solved, when I left Johannisberg, on 1st October, 1894. An ordinary zinc box is used, but the secret lies in keeping the zinc shavings clean by the addition of a chemical agent causing stronger voltaic action. (I may here ask—as a voltaic action causes the precipitation of the gold on zinc—why not apply an electric current at once, as in the Siemens and Halske process?)

It is found that to obtain good results in the precipitation box, the zinc has to be washed every few days. This certainly would be a very weak point in the process, even if mechanical means were employed to effect the washing of the zinc.

From 12 charges of slimes (at 10 tons each), assaying 2·58 dwts., he has obtained an extraction of 86·3 per cent., leaving in the residues 0·35 dwts. The total expense of treatment is about 3s. per ton.

It affords me much pleasure to be able to record this advance in treatment, and to congratulate Mr. Bettel on his success.

**Crushing the Ore.**—The general mode of treatment of the Rand ores is as follows:—The ore as it comes from most of the mines is dumped at the shaft over grizzlies, by which the fine (going direct into ore-bins) is separated from the coarse. A Kaffir turns on a sprinkler and washes the ore, which permits the conglomerate to be distinguished from the quartzite, which is sorted out and goes to the waste dump. The conglomerate lumps are fed into a Blake-Marsden or Gates crusher, and the broken material falls into the ore-bins. At some mines, as at the Robinson, the ore is sorted on revolving tables. This sorting and breaking of the ore at the head-gear is to be recommended, as it prevents a lot of waste to be crushed in the battery, and reduces the cost of erection of the battery building and grade by having the breakers outside the mill.

**Grizzlies.**—The grizzly is in use at every mine, and consists of a series of iron bars, set apart at regular distances, according to the fineness to which it is desired to separate the ore —from ¾, say, to 1¼ in.—the finer the better.

The fine, after passing through the bars, goes into a shoot, which generally delivers it into the same hopper into which the stone, broken by the rock-breaker, discharges, and this arrangement is the best, as all the ore gets mixed, and is evenly distributed in the battery. It was found that, by having separate bins for the fine, uneven results were obtained in the battery, as the fine ore is richer than the coarse lumps. The grizzly should be set at an angle of 45 to 55 degrees; the clayey and sticky ores require the steepest angle. Steel bars are preferable to iron bars, and they are generally 1 in. on the face and 2½ to 3 in. deep; and the lower face is usually ¼ in. narrower, which prevents choking. They are generally 8 to 10 ft. in length.

**Crown Reef Mine.**—Plates XXII. and XXIII. show the head-gear, including pit-head, crusher, and ore-bins, at the incline shaft of this mine.

The battery is located some distance from the shaft, and the broken rock is hauled by means of an electric tramline, the car of which is shown in Plate XXII. The overhead system is in use, the current being collected by a trolley-arm from a wire running above the centre of the track, and, after passing through the motor, being returned to the generators by the rails themselves. Care is taken to avoid any bad contacts between the different section of rails, and an auxiliary copper conductor is run in the ground alongside them, being connected at short intervals to every piece of rail. The locomotives themselves are two in number, and are equipped with starting and regulating switches and brakes. The motors, which are 20 horse-power, have double steel helical gearing to the axles, and capable of running at 10 miles an hour. The total weight of the locomotive is about 10,000 lbs., and each is capable of taking about 25 empty trucks from the mill to the mine at a time.

Where vertical shafts are in use, the arrangement is as shown on Plate XXIV., representing the old vertical shaft of the Robinson mine. From the rock-breaker house, the ore is trammed to the battery and dumped into storage hoppers, at the mouths of which are placed automatic feeders, in order to regulate the supply of ore to the stamp.

**Ore Sorting Floors.**—Within the last two years many companies recognised that a large portion of the rock put through the battery was waste, especially in those mines where the reefs are narrow, and that the average value of the ore crushed could be raised by sorting out the waste. The system, I believe, was first introduced at the Ferreira mine, and it has since been followed by nearly every company engaged in working thin, but rich, veins of ore. At the Ferreira mine, where the ore is hoisted by skips and dumped into a small hopper, the same is run into tipping-trucks running along a grizzly 25 ft. long, with ore-bins underneath, the coarser material is picked out and trammed to the rock-breaker, and the sandstone goes to the waste dump. The rock-breakers are generally large enough to break in 10 hours sufficient stone to keep the battery going 24 hours.

A Blake-Marsden crusher, having a mouth 15 in. long by 9 in. wide at top, easily crushes sufficient ore for 20 stamps, crushing 4 tons per stamp per day of 24 hours, the crusher running 10 hours per day.

Of late, many mines crushing large quantities of stone, and having wide reefs, have adopted the Gates rock breaker (the Comet), described in a previous chapter (page 140).

At the Langlaagte Estate the crushing plant consists now of three No. 6 Gates crushers, in place of the eight 15 by 9 in. Blake-Marsden type. Here the whole body of the Main and South Reef, together about 12 feet wide, without any attempt at sorting, is passed through these three crushers.

The tipping trucks are loaded at the shoots in the mine, hoisted to the surface, and dumped into hoppers above the crushers without screening over grizzlies.

Missing Page

Missing Page

The sorting of the ore is not costly at the Princess Mine, the same is estimated at 6d. per ton, as the Kaffirs get very expert at the work. A No. 5 Gates crusher requires 40 indicated horse-power, and will do the same work as three 15 by 9 in. Blake-Marsden crushers, requiring 30 horse-power.

I would point out to those laying out a crushing plant that, by having only one crusher, there is always a great risk in case of a break-down that the whole battery may have to stop, which would not be the case if there are duplicate machines. As a No. 5 Gates crusher, costing £700, has capacity enough to keep 60 heavy stamps going, it is not judicious to put in a duplicate when three Blake-Marsden crushers can be bought for about the same price. With companies who do not mind this initial outlay I dare say the Gates crusher will prove the most advantageous.

**The Langlaagte 160-Stamp Battery.**—This battery crushes 22,000 tons of ore monthly, the capacity of each Gates crusher being large—over 300 tons per day, as the machines do not work on Sunday. A photographic view of the mill in operation is given in the frontispiece to this volume, and a sectional drawing in Plate XXV.

The power for driving the crushers is supplied electrically from the mill engine. The ore on being crushed falls direct into tipping-trucks which, when pushed outside the building, are immediately picked up by an endless chain run off the mill engine, which carries the trucks up an incline tramway, the empties being returned in a similar way. The cost of tramming and breaking the ore does not exceed 4d. per ton. There are three dynamos, which are placed in a separate room adjoining the mill, two of 56 horse-power and one of 25 horse-power, and they serve to drive the crushers, to light the surface to supply the motor at the cyanide works.

The 160-stamp battery (Fraser and Chalmers), and the 96 Frue vanners, are driven by a compound engine, surface condensing, with 4 ft. stroke, the diameters of the high and low pressure cylinders being 22 in. and 32 in., working with 21 in. of vacuum and 100 lbs. steam pressure. Before the pulp passes to the Frue vanners, it goes through small hydraulic separators to eliminate a portion of the slimes.

**City and Suburban Battery.**—This consists of 80 stamps (Fraser and Chalmers). In the boiler-house are 9 boilers, giving a total horse-power of 780. The mill engine is vertical type, triple expansion, condensing, with cylinders measuring respectively 20, 31, and 48 in. diameter, with 4 ft. stroke. Alongside the engine is a large air compressor.

The transmission is made by ropes. Three large dynamos, equal to 370 ampères, 115 volts, which operate the pumps at the water reservoir, which has a capacity of 45 million gallons, where a double set of three-throw pumps are employed with 14-in. plungers to deliver the water to the battery, which are driven by Electric motors. These are two Edison-Hopkinson motors of 140 ampères and 215 volts, which drive the three-throw pumps, capable of delivering 80,000 gals. per hour, each under a head of 120 ft. and a distance of 2,000 ft. through 12-in. pipes. Special switch-boards are in connection with the generators and at the pumping stations, which place the whole of the machinery under the control of the engine-driver in the mill engine-house. The pumps mostly used in conjunction with electric motors are the treble-barrel pumps, with externally packed trunk plungers, working in gun-metal glands, the throw-crank of which is driven through a counter shaft and double helical steel gearing, by an Elwell-Parker-Morton, mounted on the same bed-frame as the pumps.

There are two Green's economisers, each worked by a small engine. The tailings are raised by a 37-ft. elevator wheel, and pass to the cyanide works to undergo the direct treatment.

**Paarl Central Battery.**—40 stamps erected near the head-gear, which is 56 ft. in height. This has the usual grizzlies, bins, and sorting floors, where about 20 per cent. of waste rock is picked out, and the ore itself thrown into the hopper of a No. 5 Gates crusher, whence it passes into the trucks and is run to the feeders. The power is transmitted to the crusher

Missing Page

from the engine-room by a wire rope. In the engine-room is a corliss of 95 horse-power, which is now running at 115 indicated horse-power. This drives the battery, dynamos, and tailing wheel. In the same room is a 60 horse-power Globe engine, 2 Elwell Parker dynamos, and a Hirnant air-compressor capable of driving 10 drills. In the boiler-house adjoining are four 65 horse-power Fraser & Chalmers' multitubular boilers, with an overhead flue and a Green's economiser with 120 tubes. There are also three additional boilers of 40 horse-power each, to be used in any break-down. The steam is carried to the engine-room at the shaft and to the workshops. The hauling is done by a 16 horse-power double drum engine, and the 10-in. plunger-pump is driven by a 12 horse-power engine. There is an additional engine for surface haulage.

The workshops, which are 160 by 35 ft., consist of blacksmiths', fitters', and carpenters' shops. The Bradbury drill-sharpener is used, and it can sharpen 1,000 drills per shift, against 300 to 400 by hand labour. The fitters' and carpenters' shops are well supplied with machine tools, driven by a 12 horse-power compound engine. There is a compound to hold 500 Kaffirs, a retort house, assay office, manager's house, dispensary, workmen's quarters, and stables.

The water supply is drawn from a dam, which has 16 ft. of water at the wall, and is about ⅓ of a mile long by a ¼ broad. The 3-throw pump is capable of delivering 16,000 gals. per hour, and is driven by electricity, the motor being an Elwell-Parker and the pressure 110 volts, delivering the water through 1,800 ft. of 8-in. piping. The voltage of the two generators is 220, part of the current being utilised for the lamps (300 in number) both above and underground.

In the amalgamator-room is an amalgamating barrel for the treatment of the mercury from the troughs. The percentage of gold from amalgamation is 61½ per cent. The tailings, which average 5·40 dwts., are raised by a 36-ft. wheel, and pass an automatic sampler and go to the cyanide works.

At many mines the continuous rope system has been introduced for the transmission of power from the engine, and it presents many advantages as compared with leather belting, the prime cost being about one-fourth of the latter, and the cost of maintenance much lower.

**Champ d'Or Deep Level Battery.**—One of the best batteries erected on the gold fields is a new 40-stamp mill of the Sandycroft foundry, of Hawarden, Chester. The guide-block and fingers for lifting the stamps are placed behind the stems. Challenge feeders are used, and the mortar-boxes are of the latest design. The tailings from each five stamps pass first over two gilt-edge concentrators, made in the United States, and the tailings from these pass over two Scoular tables, and are then elevated by a Raaf bucket-wheel to the intermediate settler, fitted with Butters' distributor.

The stamps weigh 950, and the battery is calculated to crush 4,000 tons monthly. The rock-breaking is carried out in an adjoining building by means of two No. 3 Gates crushers. The ore is lifted to the top of the building by means of an American lift, and dumped on grizzlies. After being broken in the crushers, the ore drops into trucks, which are hoisted up again to the level of the ore-bins in the battery.

In the boiler-room are four Easton and Anderson boilers, and two Reunert and Lenz boilers, having a total capacity of 360 horse-power. These are provided with two Green's economisers. A 120 horse-power Tangye engine, compound condensing, drives the battery, hoist to breaker, and 2 dynamos for pumping, each furnishing a pressure of 550 volts. A large air-compressor, capable of supplying 10 drills, draws its steam supply from these boilers.

**Water Power.**—While in the Lydenburg district, I had occasion to work a battery which was driven by water power by means of a turbine wheel. The mill was of Sandycroft make, and the 15-stamp iron frame, shown on Plate V., was erected in 1889. During 1890 I added the other 15-stamp wooden frame, see Plate II., and since then many improvements have been made by the makers in the construction of their batteries.

Water power, where available, effects a great economy in the working of a mill, but in South Africa only few localities are favoured with a sufficient supply.

Where a high head of water can be obtained, a high efficiency of the power developed can be utilised in specially constructed pressure wheels. There are several varieties of these wheels, some with the buckets cast with the wheel, some with the buckets cast separately. The principle of these wheels is that the higher the head the greater the power they can communicate, or the more machinery they can drive with the same amount of water. The quantity of water used is in the inverse ratio to the fall.

Where high pressure is obtainable, the Pelton and the Knight water-wheels are to be recommended, which do an enormous amount of work, as they give 85 to 90 per cent. efficiency. These wheels are made from 6 in. in diameter to a size that will give 1,000 horse-power. Further particulars, and data as to the efficiency of the wheel, can be obtained from the makers of milling machinery.

**Inside Amalgamation.**—It has not been settled yet if inside amalgamation offers any advantages. The percentage of gold caught by amalgamation ranges from 54 to 70 per cent. Now that the cyanide process is in successful operation, attention is devoted to a high chemical extraction rather than to perfect amalgamation. Even in very pyritic ores—say, ores containing 3 to 4 per cent.—from 63 to 70 per cent. is recovered by amalgamation. Some advocate amalgamation inside the mortar-boxes, others do not, and maintain that the mercury in the boxes is floured by the action of the stamps when pyrites are present, and that an escape of amalgam is the result. In the United States, the general adoption of inside amalgamation is a proof that it is an advantage, but how far it benefits Rand ores has never been definitely proven. At the Robinson and Crown Reef mines, extensive comparative experiments have been made, and as inside amalgamation is adopted there, one may assume that its advantages have been demonstrated.

I have not been able to get many figures as to the loss of mercury. At the Geldenhuis Estate 80-stamp mill, after crushing 98,141 tons, the loss amounted to 1,226½ lbs.

**Duty of Stamps.**—The "duty" of the stamps is very large on an average, as is seen from the following figures:—

| | | | | |
|---|---|---|---|---|
| At the | City and Suburban | the duty per stamp | per day is | 4·38 tons. |
| ,, | Crown Reef | ,, | ,, | 4·95 ,, |
| ,, | Ferreira | ,, | ,, | 3·69 ,, |
| ,, | Geldenhuis Estate | ,, | ,, | 4·20 ,, |
| ,, | George and May | ,, | ,, | 4·61 ,, |
| ,, | Henry Nourse | ,, | ,, | 2·90 ,, |
| ,, | Langlaagte Estate | ,, | ,, | 4·38 ,, |
| ,, | Robinson | ,, | ,, | 4·59 ,, |

The quantity crushed is in proportion to the weight of stamps, height of drop, speed, size of screen, nature of the ore, height of discharge, and width of mortar.

The drop of the stamp ranges from 85 to 95 per minute and height of drop from 7½ to 9 in.

With double-armed cams the speed must not be great enough to bring the cam into collision with the falling tappet, which means that the stamp must be given sufficient time to finish its drop before being lifted up again by the cam. When the cam strikes the descending tappet, the shoe boss or tappet is often dislodged, and breakage is the result.

# CHAPTER XXII.

## *MILL SITE AND BATTERY—RECENT APPLIANCES.*

MILL SITE AND BATTERY—Framing of the Battery—Homestake Mortar—Shoes and Dies—Cams, Cam-Shafts, Stems, Stamp-Heads—Blanton's Cam—Tappets—Automatic Tailing Sampler—Mercury Troughs—The Batea.

**Mill Site and Battery.**—In laying out the plan for a mill a locality should be chosen where there is plenty of grade, so as to admit of an economical construction, and the passing of the rock through the mill without undue handling; allowing also ample fall for the tailings and the intervening cyanide works. The topographical features of the Witwatersrand offer these favourable conditions only in a minority of cases, and the result was that at the commencement of mining operations the batteries were erected in the proximity of the water or the shafts, whereby some costly constructive errors were committed.

Much has been altered within the last three years, and in the erection of the new batteries all due regard has been given in the selection of mill sites, and the process of milling has been made almost automatic, thereby reducing the cost of milling the ore.

There being no running streams on these fields, the water necessary for metallurgical purposes is collected in reservoirs, after passing through the tailing and slime-pits, where ample time is given to it to clear itself of nearly all sediment, and it is then pumped back to the battery. Every mine pumps daily a certain quantity of mine water, and during the rainy season—October to April—there are heavy showers, when precautionary measures have to be taken to preserve the dams from bursting, as the rainfall is very large.

As the reservoirs are located quite a distance from the batteries, large pumps are placed near the dams, and pipes sometimes a mile or more in length have to be laid. Many of the pumps are actuated by electric motors, which are the most economical to work.

At the Crown Reef mine is a duplicate plant, each pump being capable of pumping 60,000 gals. per hour, with a 200 ft. lift, at a distance of 3,500 ft. The total pumping capacity is 120,000 gals. per hour. These pumps are driven by two Elwell-Parker motors, each giving 100 actual horse-power. Each pump consists of two cylinders 18 by 36 in., and the shafting is so arranged that either motor can drive either pump.

**Framing of the Battery.**—*Stonework.*—All foundations and retaining walls to be built of large stone, properly bonded, and well laid in cement mortar, composed of ten parts good, clear sand, two parts good quality of lime, and one best Portland cement, special care being taken to keep all dirt or clayey material excluded; all exposed faces of retaining walls to be well pointed up and finished with the same material.

*Ore-bins*, for large batteries to be made of 12 by 14 in. timbers, laid flatwise; for small ones, 8 by 8 in.

Foundation posts to be made of 14 by 14 in. timbers, sills, posts, and caps for ore-bins proper to be made of 12 by 12 in. timbers; the posts to be boxed into the sills and caps 1 in. Braces for incline bottom to be made of 10 by 12 in. timbers, or 8 by 8 in. All planking to be 3 in. thick and lined throughout with 1 in. boards to break joints over the planks.

The ore-bins receive the ore from the grizzlies and crushers directly over them, if the breakers are inside the mill building, and discharge it through chutes into the hoppers of the feeders. They are triangular, with one vertical side, facing the battery and reaching down to the cam floor.

Just above the latter are the openings (one for each feeder) through which the ore passes downward into the chutes, terminating in the hoppers of the feeders. The quantity of the discharge is regulated by a sliding door. In a double mill, the inclined bottoms of the two bins diverge, leaving an open space between them which has the shape of an inverted **V**.

*Battery Frame.*—Mud sills for heavy batteries to be made of 14 by 16 in. sugar pine, or good yellow pine free from sap; to be well bedded in concrete, which must be put on to the clean bed rock. Line sills to be made of 12 by 16 in. and 20 by 16 in. sugar pine, or yellow pine of good quality, to be bolted down to the mudsills.

*Mortar blocks* to be made of two or four pieces in double row, wide enough to fill space between the line sills and battery posts. All to be sized and well fitted.

The timbers for mortar blocks are to be accurately fitted together and secured with 1-in. bolts. Blocks to be sized and finished above the floors.

*Battery posts* to be made of 12 by 24 in. good quality pine timber, to be dressed all over, and bolted down to the line sills with 1-in. joint bolts, the large posts to be made with double tenon on the bottom. The braces are made also of 12 by 12 in. timber and fitted into the line sills, the whole being secured by long bolts passing through the battery post and sill.

The placing of the mortar blocks requires great care and attention. Rectangular pits 10 ft. or more are dug or blasted out if in rock, and sufficiently wide and long to leave a space about 24 in. around the block. The bottom is then carefully levelled, and some sand tamped down or laid with grouting in cement.

On this are placed two layers of 2 in. plank, spiked crosswise to each other, and this wooden floor must be perfectly horizontal. The top of the mortar block is planed off and must be carefully levelled before placing the mortars.

The space around the mortar block is then carefully filled, and tamped with rock and tailings up to the level of the mud sills. When the top of the mortar block has been planed off and levelled, a sheet of rubber cloth, ¼ in. thick, is placed over it and the mortar put in place. Very often mortar blocks are built up of 3 in. planks. The cam shaft is to be set 4¾ in. from the centre to the centre of the stems.

The details of construction here given will be found illustrated in several of the Plates already given.

The stamps used for fine crushing are generally made up into batteries of 5 heads, as seen in the frontispiece, showing a battery erected by Messrs. Fraser and Chalmers (see p. 623).

Each battery is driven separately from a pulley on the main driving-shaft, so that it is not necessary to stop the whole mill for repairs to a single battery, or when cleaning the plates. The tendency is to have heavy stamps, and all the lighter stamps are being replaced by a heavier type of 1,050 lbs. weight. The screens are usually 900 mesh per sq. inch of steel wire; the copper plates usually 10 ft. long by 5 ft. wide.

The water necessary to form the pulp flows into the mortar-box at the rate of 1,500 to 3,000 gals. for every ton of rock crushed. From ¾ to 1½ miner's inches per battery should be provided. A miner's inch of water in twenty-four hours is equal to about 16,800 gallons.

Among the self-feeders in use, Hendy's Challenge commends itself, as the machine is so constructed as to have the ore supplied by a "carrier," and the superiority is marked when feeding wet and sticky ore, which adheres to the chute leading to the mortar. The capacity of the battery is increased by feeding low, depth of discharge 2 to 3 in., and feeding steadily, because if large quantities are fed at a time, the ore is piled up to a height that reduces the fall of the stamps, and forms a cushion of ore on the dies that impairs the efficiency of the impact of the shoe. When a mortar gets packed with ore, feeding should be stopped till the same is cleared, and if this precaution is not taken, the screens generally break.

**Homestake Mortar.**—This style of mortar (Figs. 190, 191) is now used in most mills. The discharge opening is 48½ in. long and 21¾ in. high. The frame is inclined outwards about 10 degrees from the vertical. On the short sides of the discharge opening are grooves to receive the chuck-block, screen-frame, and curtain, which are held in place by keys and sockets. The chuck-block is also fastened at the bottom by two horizontal keys, supported by lugs on the outside lip of the mortar below the discharge. The chuck-block is used to fasten a narrow strip of amalgamated copper plate, where inside amalgamation is used. When rear copper plates are used, the same are fastened underneath the feed-shoot, which protects them. These back plates are held in place by screws.

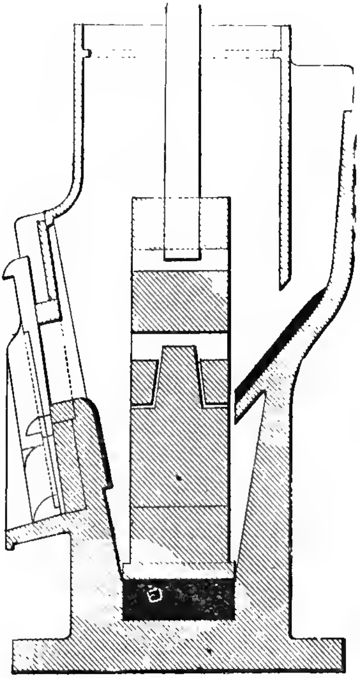


Fig. 190.—Homestake Mortar Box.

Another improvement consists in having bottom liners, which are placed under the dies, forming a protection to the bottom of the mortars; and a liner is also put over that part of the feed shoot upon which the ore drops. The sides of the mortar-box most exposed to the scouring action of the pulp are also protected by a set of steel plates.

In stamping, the discharge is kept as low as practicable, and as the dies wear down the depth of discharge increases; therefore, to equalise the discharge, there are for every mortar several chuck-blocks of different depths, and as the dies wear down usually 5 in., it is usual to have four blocks, whose heights diminish by 1 in.

The introduction of cast-iron plates in the bottom of the mortar-boxes has increased the life of the dies; these plates,

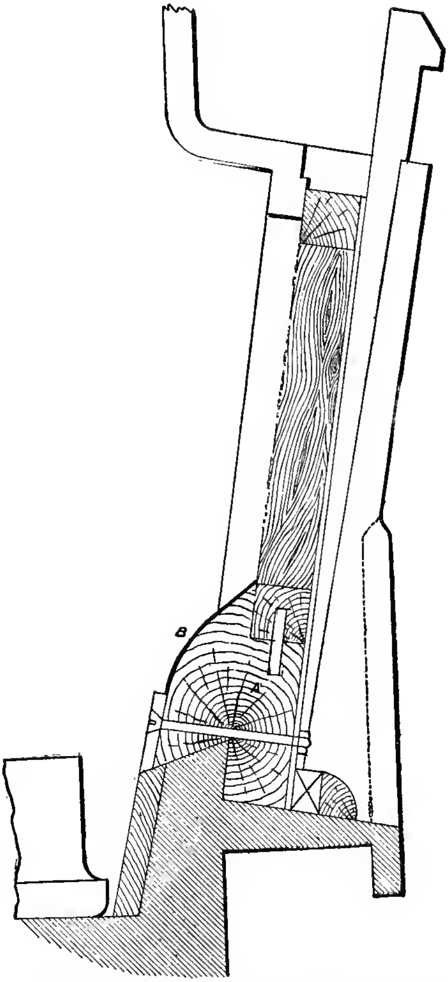


FIG. 191.—HOMESTAKE MORTAR CHUCK BLOCK AND SCREEN.
A, Chuck Block. B, Front Copperplate.

two in number to each battery, fit snugly in the mortar, and they serve a double purpose, as they decrease the height of the discharge.

The screen frame is made of wood, and there is a seating provided for it in front of the mortar-box, the frame being held in place by means of two long steel keys at either end, and one or two short wedges at the bottom. The screen frame must fit well, and there must be no leakage, so that all the pulp shall escape through the screen; and for this reason, when the screen is tacked to the frame, strips of thick flannel or blanket are tacked over it. Duplicate screen-frames should always be ready at hand in case of breakage. Screens are usually 10 to 12 in. wide, and it is customary, when the lower portion, exposed to the splash, wears out, to turn them upside down, and, of course, the screen-frame must be constructed accordingly.

**Shoes and Dies.**—These are now (1895) universally made of steel—manganese steel being largely used—and great competition exists between the German and English makers. At the Robinson mine the shoes are made 2 in. longer, so as to prolong their wear. Steel shoes made for stamps of 850 to 950 lbs. weigh from 145 to 155 lbs., measuring about 9 in. in diameter, by 7½ to 8 in. long. The neck is from 4½ to 5 in. long.

The steel dies weigh from 110 to 125 lbs., and measure 9 in. in diameter, by 4½ in. in height; with a rectangular foot-plate 10½ by 9¼ in., by 1½ in. thick.

Dies wear less rapidly than the shoes, as they are protected by the thickness of the broken rock, which covers them to the depth of from 1½ to 3 in. But while the actual wear of dies is less than that of the shoes, the life of the dies is shorter than that of the shoes, owing to the fact that the shoes have several inches of greater length of wearing part than the dies. To obtain the maximum crushing capacity of the battery, the dies must be kept as high, with reference to the lower edge of the screens, as is compatible with the safety of the screens. To prevent pounding of iron, it is necessary to preserve more or less uniformity in the level of the dies. Should one die in the battery project much above the others, little or no pulp would remain upon it, and the shoe would consequently drop upon the naked die.

The wear and tear in shoes and dies on the Witwatersrand may be put down at ½ lb. per ton of ore crushed. The shoes are kept at work till they are worn down to 1 in.

**Cams, Cam-shafts, Stems, Stamp-heads.**—These parts of a battery should last for years. Occasionally a stem breaks, and can be swedged or planed down, and an additional length welded on. Stems are reversible, as both ends fit into the socket of the stamp-head.

| | |
|---|---|
| The stem of a 950 lb. stamp weighs about . | 420 lbs. |
| The stamp-head about . . . . . | 275 ,, |
| Length of stem . . . . . . | 11 to 12 ft. |
| Diameter . . . . . . . | 3¾ in. |

The length of the stamp-heads is 17 in., and they are generally made of cast steel.

The cam-shaft is made either to drive ten or five stamps; the same is usually 5 to 5½ in. in diameter, made of cast steel or the best hammered scrap iron.

The order of drop of the stamps is usually as follows:—

| | 1 | 2 | 3 | 4 | 5 |
|---|---|---|---|---|---|
| | ● | ● | ● | ● | ● |
| | 1st. | 3rd. | 5th. | 4th. | 2nd. |
| Or | 1st. | 3rd. | 5th. | 2nd. | 4th. |
| ,, | 1st. | 4th. | 2nd. | 5th. | 3rd. |
| ,, | 1st. | 5th. | 2nd. | 4th. | 3rd. |

**Blanton's Cam.**—The Blanton patent cam was first introduced in the gold mills of the El Callao Mining Company, Venezuela, and, after proving entirely satisfactory by several months' trial, were adopted as the standard design throughout the mills. I am indebted to Messrs. Fraser and Chalmers for the subjoined particulars concerning it.

All millmen have experienced the disadvantages connected with the ordinary fastening of cams on the cam shafts in stamp mills. These inconveniences are as follows:—The keys have to be firmly driven after accurate fitting, and in a contracted space; and in case of a breakage of cam, or wearing out of the same, it is a tedious business to drive back the keys, take off in some cases a number of the cams, and replace them in a proper position with sufficiently firm keying to prevent any working loose.

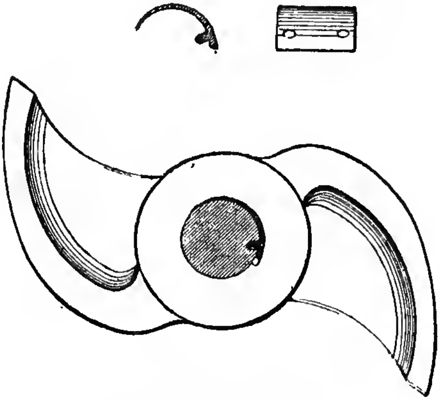


FIG. 192.—BLANTON'S CAM.

With inexperienced hands (such as are often employed in foreign countries), the reckless use of a hammer in driving the keys out or in frequently causes injury to the shaft and key, and occasionally leads to the breaking of a cam. Again, in the case of replacing a single cam it is necessary that the key-way in the new cam shall be cut in such a position on the hub as to give the necessary order of drop to the cam in respect to the other cams on the same shaft. The consequence of this is, that the extra cams for repairs are usually shipped to the mines without key-ways being cut, and the cutting of a satisfactory key-way in a steel cam (as now almost universally used) is in itself a troublesome business.

In the Blanton's improved cam, key-ways and keys, in the ordinary sense, are dispensed with. A reference to the illustration will show that the cam is fastened to the shaft by means of a circular taper wedge, which is prevented from slipping on the shaft by two short pins dropping into two holes bored in the cam shaft in the process of manufacture of the latter. The order of the drop of the stamp is fixed on the cam shaft by the position of these small holes, which determine the position of the wedge on the circumference of the shaft. It follows from this arrangement that every cam is of exactly the same shape, whatever the order in which it may occur on the cam shaft; and further, that no key-way or other fitting has to be done at the mine, but all cams are interchangeable (except so far as being right or left-hand). It has been found in actual working practice that this cam fastening is perfectly effectual, and furthermore that there is no tendency to shift laterally, owing to the powerful binding action in the natural running of the cam and direction of strain on the taper key. The cam is placed in position by simply putting the wedge into place and slipping the cam over the same in its loosest position, and then turning the cam slightly until a grip is taken, after which the starting of the cam shaft itself will complete the tightening operation. For loosening a cam to replace with a new one, or to slightly alter its position on the key if it should have shifted in the first start of the cam shaft, it is only necessary to tap the point of the cam back with a hammer, when it freely loosens from the wedge and can be slipped off without difficulty.

The great simplicity of this fastening allows of the complete change of five or ten cams on the cam shaft in a very short time, and without injury to any portion of the machinery by the driving of keys. All good millmen appreciate the advantages of limiting the time of stoppages in mills, and the facilitating of repairs by the inexperienced hands employed in many mining camps.

**Tappets.**—The revolving cam, besides lifting the tappets, imparts to it a rotary motion, which, to some extent, is communicated to the stamp in dropping. This produces a grinding effect, which assists in crushing the ore. The rotation of the stamp performs a far more important office in equalising and, consequently, diminishing the wear of the shoes and dies. When there is but little grease on the tappet it makes a complete revolution in from four to eight lifts by the cam. In falling, of course, but a small part of the rotary motion imparted to the stamps in rising is retained. When there is much grease on the tappets or cam, or when the tappets have so worn that the face of the cam strikes a grooved, instead of a level face on the tappet, the rotary motion is greatly impaired. Tappets last for several years, from four to five years being their usual life. Sometimes they are broken by being too tightly keyed. When their faces are worn they are planed down. Tappets are reversible, so that when one face has been worn as far as possible the other face is placed downwards. Tappets are usually of steel, and weigh about 125 lbs. when 950 lbs. stamps are used.

**Automatic Tailing Sampler.**—The tailings, as they are carried away by the stream in the launder, D (see Fig. 193), may be submitted automatically to regular sampling by means of a very simple arrangement, which is housed in and kept under lock and key.

The sampler is placed between the battery and the cyanide works, and consists of a wooden frame, A, connected by a lever with the sans box, E. At the lower end of the frame, A, is fastened a piece of 2½ in. gas-pipe, B, with a longitudinal slit on the upper surface, as shown in the drawing. The sans trough, E, which is pivoted, is filled with water by means of the pipe, F, and when full tips over, and in so doing pulls the pipe, B, into a horizontal position, and the horizontal slit faces the stream of tailings which enter the pipe for a few seconds, and an average sample goes into the bucket, C. Now the flow at F can be so regulated as to fill E every ten, fifteen, or twenty minutes, and this is about the most practical way of obtaining a good average. The sampler requires no attendance, and any carpenter on the spot can build it.

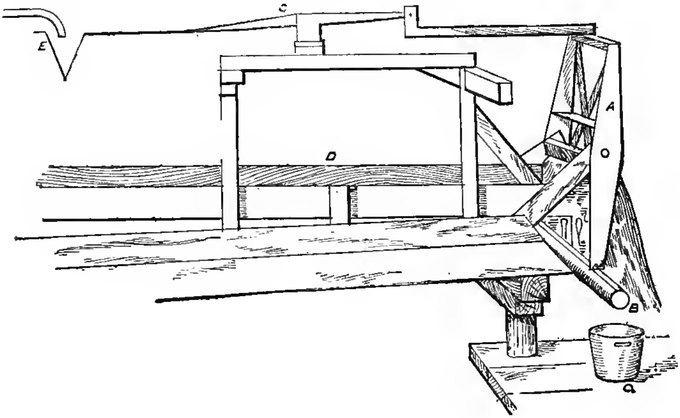


FIG. 193.—AUTOMATIC SAMPLER.

**Mercury Troughs.**—I have mentioned that in ordinary battery amalgamation the pulverized material flows in a comparatively large body of water over the smooth, slightly inclined, copper plates, and that we depend upon the higher specific gravity of the gold to settle on the amalgamated copper plates, to be arrested on them and collected in the shape of a hard amalgam. It will be readily understood that in the comparatively rapid current many fine particles of gold will have a tendency to float away, and to avoid this it is useful to put obstructions in the path of the current, so as to cause the solid material to settle for a short period—a period infinitesimally small, but still long enough to give the solid material a chance to rest, and offer an opportunity for a further separation by gravity.

This object is very easily effected by putting two, three, or four troughs between the plates, as shown in Fig. 194. These troughs have a partition board, called the splash-board, which reaches within a short distance of the surface of the mercury. As the pulp flows into the trough, the pulp and water strike against this board and the sands fall on top of the mercury in the trough, and the water, so to say, bubbles back and then scours the sands between the narrow slit out over the lip of the trough; and it is during this scouring action that a certain amount of fine gold has a chance to come into contact with the mercury, and remains arrested in the trough. Care must be taken not to allow the traps to get choked with sand, but the outflow must be regular. It is a mistaken idea that the sands have to be forced through the quicksilver, as the trap is there to collect simply whatever mercury and amalgam escapes from the battery, or what mercury flows down the plates when they are silvered, and to stop any light gold by gravitation.

When setting the tables, look out that they are set true cross-ways, because if they are higher on one side than on the other the distribution of the sands will not be uniform, and the sands will pack and bank up on the plates. The plates at all times must be clear of sands, and the pulp must flow off regularly with the current of water.

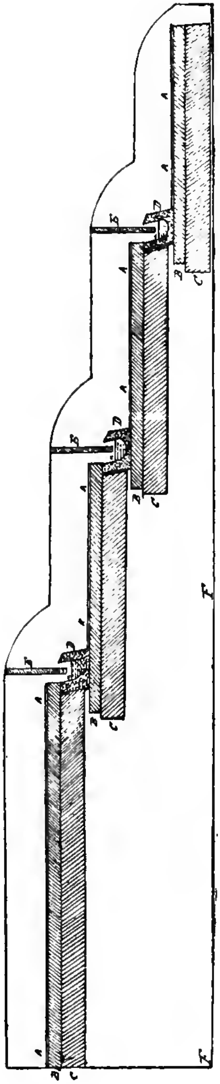


FIG. 194. TABLE AND MERCURY TROUGHS. Scale ¼ in. = 1 ft.

In many batteries lip plates are used, which are set at an angle against the splash, and a considerable portion of the gold attaches to this plate, which is cleared once in twenty-four hours. I can see no advantage in this arrangement. The best arrangement is that shown in the battery illustrated in the frontispiece to this volume, where the plates are the full width of the mortar, and so arranged that their inclination can be adjusted to any desired grade. They should fit very snugly under the lip of the mortar.

The length of the tables varies and depends on the available room. The largest portion of the gold amalgam is always at the head of the table, and where the table is broken into steps, the gold is caught at the steps where the splash from the upper plate strikes the lower one. With coarse gold the table need not be long, but with fine gold the same can be 15 and even 20 ft. in length. Great attention must be paid to the grade, so that no sand should accumulate on them. The vibration set up by the falling stamps causes a pulsation of the water flowing over the plates, similar, in a way, to the action of a jig. This assists the work of gravitation.

In cleaning up, before taking out the screen a little mercury is sprinkled on to the tables, to catch any free gold that may escape. The stamps are hung up and the feed water turned off. The chuck-block is removed and the amalgam plates taken off. The apron is covered with a blanket and the sands in the box cleaned out. The back copper plates are taken out by removing the keys and washers and lifting the plates carefully up off the studs, and are scraped by means of a flat blunt chisel about 1½ in. wide; if the amalgam is soft it is removed by rubbers made out of pieces of belting.

It is claimed in some mills that 50 per cent. of the gold won by amalgamation is saved by the inside plates. The clean-up of the amalgam from the table is proceeded with. A stream of water is turned on to wash away all sands, and the amalgamators proceed to clean-up with rubbers, commencing at the bottom end of the table and scraping it gradually upwards towards the top. The amalgam, which lies thickest just near the lip of the mortar, is scraped off with a flat blunt chisel. The whole of the amalgam is collected in enamelled basins. It takes half an hour to clean-up one mortar, and the inside plates are cleaned-up every two days. The amalgam on the apron plate should be kept soft by sprinkling quicksilver over it, but not so soft as to have globules running away.

Fig. 195.—Batea for Gold Mill. Elevation.

**The Batea.**—At the Robinson mine, Captain Mein has introduced, for the cleaning-up of the quicksilver and amalgam, a Californian invention called the batea. It consists of a round cast-iron shallow pan, about 4 ft. in diameter, and about 4 in. deep, with a rounded bottom and a plug in the centre.

As will be seen from the accompanying illustrations (Figs. 195, 196), the pan is supported at one end on a roller, and suspended at the other by a couple of rods of light round iron, this mode of suspension allowing it great freedom of motion.

At the suspended end there is a small vertical crank driving a pin, which forms part of the pan itself, at a rapid speed, so that the pan receives a gyratory motion — not unlike that used in panning up amalgam in the hand pan— which serves admirably to collect the quicksilver and amalgam in the bottom of the shallow cast-iron pan, the lighter materials finding their way to the sides.

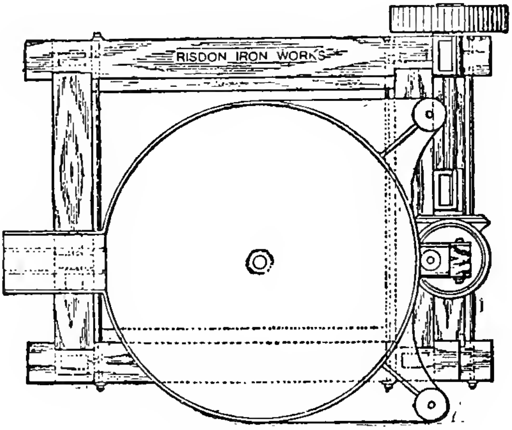


Fig. 196.—Batea for Gold Mill. Transverse Section.

A force of about one horse-power is required to drive the batea.

# APPENDIX A.*

*CAUSES OF FAILURE IN GOLD-MINING.—MR. WILLIAM TOPLEY* (1887) *ON THE FUTURE PRODUCTION OF GOLD.—THE OUTLOOK (DECEMBER,* 1894) *IN THE SOUTH-AFRICAN GOLD FIELDS.*

**Causes of Failure in Gold Mining.**—A few remarks upon this subject will not be out of place.

(1.) It is notoriously true of mining property generally, and not only of gold mines, that instead of dealing with such undertakings as we do with other business enterprises, they are too often made the object of stock speculations, in which "corners" are made by the unscrupulous, and the venturesome and unwary are again and again caught, as they ever will be, by their more vigilant and astute adversaries. These "stock speculations" are in plain language gambling, and a continuation of such gambling—with its resulting losses to many and gains to a few—leads to periods of depression in the mining market, during which time many promising mines have to suspend operations for lack of capital to develop them. Gold mines are then generally at a great discount. But all this has, it is needless to say, no necessary connection with legitimate mining, which should in no sense suffer disparagement from practices so detrimental to its best interests and so foreign to its objects. Still, many failures of legitimate undertakings are directly due to over-speculation.

---

* The first two sections of Appendix A., which formed the concluding chapter of earlier editions, are reprinted here as still of interest; but it will be noted as regards the paper by Mr. Topley (since deceased) that the question of the future production of gold has now to be reconsidered in view of more recent developments.

(2.) The cases where large sums of money have been spent on extensive and expensive machinery before the investors have definitely ascertained that they have a veritable gold mine to deal with, can be counted by scores or hundreds, perhaps even by thousands. It is a safe rule to develop your mine first, before a penny is spent on mills, batteries, and other appliances for treatment of the ores.

(3.) Adopting wrong processes for the treatment of the produce is another fruitful cause of loss. It should never be forgotten that the fact of a certain process working well for the ores of one mine is no guarantee that it will do the same for others. Remember that a particular process is not necessarily suitable for every ore, though it may happen in individual cases that the ore is found adapted to a process selected more or less fortuitously.

(4.) Too small a percentage of the precious metal saved after heavy expenditure for extracting and raising ores is another cause of failure. This result may fall within the category indicated in the last paragraph; or it may be owing to excessive outlay in works and appliances, which have been estimated on too large a scale; or it may be owing to incompetent or defective management.

(5.) Again, investors may be misled as to the value and prospects of a gold mining property by high assays of samples. *The assay of picked samples never tells the value of a gold mine.* To determine the *average value* of the ores throughout a vein is a difficult and complicated matter, which is not to be determined by single assays, and many failures in promising speculations may be traced to high preliminary assays and low returns after the mills have been erected.

These cautions may seem to many to afford them very little help or guidance in particular cases in which they may desire to determine for themselves what is, or is not, a prudent investment. But no rules which would be sufficient for that purpose can be laid down. Those familiar with mining processes can only point out the special risks attaching to enterprises, the conduct of which is beyond the knowledge and control of the great body of investors who find the means of working them, leaving it to the investors themselves to shape their course in venturing upon undertakings which they cannot personally control.

**The Future Production of Gold.**—In a paper read before the British Association at Manchester, in 1887, Mr. William Topley, F.G.S., Assoc. Inst. C.E. (of the Geological Survey of England and Wales), dealt very carefully and exhaustively with the important question of the future production of gold, treating it with regard both to the geological distribution of auriferous ores and minerals, and to the actual yield of gold from the various sources of supply in former years. The whole paper (which deals in like manner with the future production of silver), may be commended to the consideration of all who are interested in the production of the precious metals; but I am glad to avail myself of Mr. Topley's permission to quote here the subjoined passages,* in which he summed up the conclusions at which he had arrived, so far as regards the production of gold:—

"In taking a general review of the goldfields likely in the near future to yield the most constant supply, it is evident that an important place must be given to Russia. With a very slight fall in the produce of Australia and of the United States, Russia would again take her old place at the head of the gold-producing countries. With its enormous areas of placer gold only partially worked, and its Siberian veins untouched, a steady yield of gold may be anticipated for many years to come.

---

* From the Report (1887) of the British Association for the Advancement of Science. Mr. Topley's paper was prepared at the request of the Section of Economics, and was ordered to be printed *in extenso* by the General Committee.

"The United States and Australasia have of late years been running very closely together, Australia being slightly in excess. In the former there is now a slight tendency to rise in yield.* A permanent rise cannot safely be anticipated; a more steady yield than in past years is all that can be hoped for: and this it seems likely may be the case, largely due to quartz mining. The rapid fall in the gold produce of the United States from 1877 to 1883 was chiefly due to the decrease of silver mining in the Comstock district, about forty per cent. of the value here being gold. If we deduct the silver gold, we see that the fall from 1877 was a very gradual one. The vast placer deposits of California, now in great part sealed by repressive legislation, will be to some extent again worked either by drift mining or by hydraulicking, with provision for the retention of the débris. Australasia shows a gradual steadying of the produce, neither placer nor quartz mining varying much from 1880 to 1885.

" Of the newer goldfields the first place should probably be given to Venezuela, &c. The wealth of this country in gold quartz is well established; but we may perhaps expect for a time a greater development of alluvial mining.

"South Africa is generally looked upon with favour as a source from whence our future supply of gold may in part be drawn. Without doubt there are here rich lodes, and it would be strange if this country were destitute of rich placers; though of this there is as yet but little evidence. From these sources mines may possibly be worked at a profit which will give a steady yield of gold; but there is as yet no evidence that the yield will be sufficient in amount to materially influence the world's production.

" As regards India the prospect is still less hopeful. That large quantities of gold were raised here by the native princes in times preceding the British rule is tolerably certain; but it is probable that this large production was spread over long

---

* Since this was written, the United States' statistics for 1886 have appeared. The yield of gold for the last four years is stated as follows: 1883, $30,000,000; 1884, $50,800,000; 1885, $31,801,000; 1886, $35,000,000. —W. T.

periods of time, and certainly it was raised under conditions—of forced labour, &c.—which are not now applicable.

"It is unlikely that India will ever contribute to the world's stock sufficient native gold to materially influence the total production. A far more important point is the amount of gold hoarded in India, and the probability or otherwise of that being some day set free. Most estimates concerning gold are ludicrously vague, but on the question of the amount hoarded vagueness is unavoidable. It is known that £130,000,000 of gold has been taken into India since 1835; practically none of this is in circulation (silver being the standard and the coinage of India). How much was hoarded in the centuries preceding 1835 no one can say. If it only equals the amount hoarded since, we have £260,000,000, or nearly thirteen times the world's present annual production. The original source of this gold and the ways by which it reached India would be an interesting subject for inquiry. Since 1851 it is the gold of the world, mainly sent through England; but in the long past times it was probably in part of native production, in part the gold of Europe, sent over the old trade routes in return for the manufactured articles of India. It is supposed that at least as much silver is hoarded in India as gold. If so the value of silver and gold hoarded in India since 1835 nearly equals in value one-third of the total amount of gold and silver coin now in circulation in the world.

"Famines set free some of this gold, and we may perhaps anticipate that the diffusion of Western ideas will free more, but it is unlikely that gold will come from this source in sufficient quantities to influence the annual production of the world.

"British Columbia may possibly increase its yield. Other countries (as yet little known), as Equatorial Africa, Borneo, and North China, may add somewhat to the world's stock. A steady though comparatively small supply may be looked for from the treatment of silver ores and from the auriferous ores of other metals. In all parts of the world an increased supply is assured by improved methods of mining, milling, and metallurgy; this will be obtained by an actual increase from ore now worked, and also from ores of lower grade, made profitable by the improved methods; whilst tailings, &c., of former times can in some cases be profitably worked over again.

"But for all practical purposes the chief sources will probably continue to be the goldfields of the United States, of Australasia, and of Russia, aided by the development of the goldfields of South America. Everything points to a steady production from the three areas first-named, and to an increased yield from the last.

"In all metal-mining there is far too much gambling and wild speculation, and it is doubtful if, taking metal-mining all round, the value raised equals the expenses. But gold-mining cannot be ranked with ordinary mining. There is a glamour about gold which blinds men to ordinary prudential considerations. The wildest schemes meet with willing supporters, and money is always forthcoming to develop the poorest mines and to keep them going upon the most shadowy of hopes.

"Enormous fortunes have been made in gold-mining by a few lucky speculators. But for one who has been thus fortunate there are scores who have lost heavily.

"The facts which we have been considering as to the probable future of gold-mining warrant us in believing that the industry will gradually make for itself a sounder and more honest position. But there must ever be great uncertainty, and therefore a wide field for speculation and for dishonest dealing.

"If a steady and undiminished production of gold is essential for the well-being of the world, perhaps what we have most to dread is a sudden influx of common-sense and prudence in the investing public; for this would at once close a great number of mines, and might considerably diminish the world's production. But probably this contingency is sufficiently remote to be safely left out of consideration."

**The Outlook in the South African Gold Fields.**—Having so recently returned from South Africa, I may be allowed to add here a few words as to the prospects of the numerous gold-mining enterprises which have been set on foot in that region, and with many of which I have had personal acquaintance during the last three or four years.

From a rough, though careful, calculation of the amount of the present value (as at December, 1894) of the capital funds of the several joint-stock companies engaged in gold mining in the Witwatersrand fields—taking, in most cases, the prices quoted for shares in London at the date mentioned—I estimate the capital sum on which dividends have to be earned at between sixty-one and sixty-two millions sterling. The companies which have been brought into the calculation may be classified as follows:—

| | £ |
|---|---|
| (1) Dividend-paying mines, having a present capitalised value of | 26,725,000 |
| (2) Mines which have paid dividends, ,, ,, | 1,072,000 |
| (3) Non-dividend-paying mines, ,, ,, | 13,311,000 |
| Besides the enterprises included under the above heads are— | |
| (4) The Deep Level Companies, having a present capitalised value of . . . . . . . . . . . . | 15,000,000 |
| (5) The Consolidated Gold Fields Company, having a present capitalised value of . . . . . . . . . . | 5,000,000 |
| Grand Total . . . | £61,308,000 |

The dividend-paying mines (in which category I have reckoned 29 companies) will most likely distribute for 1894 dividends amounting to £1,500,000 sterling, which is hardly 6 per cent. on their present capitalised value of nearly £27,000,000; but it may be fairly presumed that the dividends on many of these properties will increase, as steps are being taken to increase their productive capacity.

The total gold production of the Witwatersrand for 1894 will amount to 2,000,000 ozs., of the value of £7,000,000 sterling, and one-fifth of this amount will be paid in dividends. As stated above, about £14,500,000 sterling represents the present capital of mines not now paying dividends, these mines being about 50 in number. Many of these will, no doubt, be paying dividends in the course of 1895, and others later on. If these 50 mines are also to pay a 6 per cent. dividend, they will have to earn £870,000, which means that the gold production of the fields will have to be raised by £2,500,000 sterling.

In regard to the Deep Levels (including the Rand Mines, the Gold Fields Deep, and various other companies) representing £15,000,000 of capital value, £1,500,000 will have to be produced to pay 6 per cent. dividend, and we arrive at a total gold production of £14,000,000 per annum required to make our mines yield a return of 6 per cent. This production, I am satisfied, will be reached in course of time; but as the mines must sooner or later be exhausted, provision will have to be made for a reserve fund, and as the working expenses are brought down a larger margin of profit will remain for the creation of such a fund.

It is impossible at the present time to say what, a few years hence, the proportion of profits will be on the gold produced. I am taking it all round at 25 per cent. as things now are; but considering that, with improvements in methods, and with approved appliances, working expenses are reduced year by year, and remembering that further improvements may confidently be anticipated, the percentage of profits should rise annually; and when the annual production of gold shall have reached twelve to fourteen millions of pounds, the proportionate profit may by that time be raised to 35 or even 40 per cent. This would mean a dividend of 8 to 9 per cent. on the present capitalised value, instead of 6 per cent., as calculated above.

It is an important consideration that about three-fourths of the present gold production comes from the 29 dividend-paying mines, and consequently a large increase of production will be necessary to put the large number of non-dividend paying mines on a paying basis. My belief is that the gold fields, at the end of three years from the present time (December, 1894), will be producing £12,000,000 sterling per annum, and will distribute in dividends £3,000,000.

The question of the "life" of the mines is of the greatest importance, and can be put down in individual cases at from 15 to 25 years.

A very important section of the Witwatersrand gold fields, which will contribute largely in the future to the output of gold, is the East Rand, extending from Knights over a stretch of five miles of country, where important developments are being made, and where in the near future a considerable amount of capital may be expected to be attracted.

To those who desire to pursue further the questions I have raised in these brief paragraphs, the report by Mr. Hamilton Smith, which is set out in the ensuing Appendix B., may be commended for their careful consideration.

# APPENDIX B.

## *RECENT DEVELOPMENTS IN THE WITWATERSRAND GOLD FIELDS.*

An important report upon this subject, presented by the eminent expert, Mr. Hamilton Smith, has recently been made public in the columns of the *Times*,* and is here set out in full, as given in that journal:—

"A report by me upon these gold fields was published in *The Times* of January 17th, 1893, based upon an examination of the Rand in 1892. I have lately revisited the Transvaal, remaining there from August to December, 1894, and in this communication I propose to state in what degree my original conjectures have been verified by the actual work of mining and development during the past two years, concluding with some general observations and a review of the financial results obtained in operating the mines of the Rand district up to the end of 1894.

"In the former report my belief was stated that in a few years the gold product from the Rand would increase to a value of over £10,000,000 per annum, and that a total yield of £325,000,000 in gold could be reasonably expected from this one district, this future product being chiefly dependent upon four things, viz.:—

"'First, Will the 'Main Reef series' continue to great depths?'

---

* Feb. 9, 1895.

"'Second, If they do, what will be their general inclination or dip, and to what vertical depth will it be practicable to work them?'

"'Third, What amount of gold will be found with increased depths?'

"'Fourth, On how large a scale can mining operations be conducted?'

"Since 1892 work has been carried on more vigorously than ever before in about 60 mines owning claims at and near the surface outcrop of the main reef series. From them 5,000,000 tons of ore have been extracted in the years 1893 and 1894, and, as a matter of course, the ore produced comes from deeper workings month by month; also, in a number of these mines developing shafts and drifts have been sunk and driven on the reefs at considerable distances below the working stopes. During the same period many vertical bore-holes have been put down by diamond drills to the dip of the series, one of them having reached the great depth of nearly 2,500 feet; several working shafts have been sunk upon 'deep-level' properties, finding the reefs of the series at vertical depths of from 600 feet to 1,000 feet, and from these shafts a considerable amount of drifting has been done. The *data* at hand for determining probably correct answers to the first two of the above queries are, therefore, now much fuller than they were in 1892.

"As stated in my former report, there were three theories generally held in regard to the dip of these reefs. Some experts believed that the Rand quartzites and their enclosed beds of 'banket' (conglomerates), these beds being now universally called 'reefs,' were truly conformable, but that great 'step' faults would occur, abruptly lifting up the reefs, and, perhaps, bringing fresh outcrops of them to the surface.

"The most general opinion, however, was that regardless of the dips shown by the upper rocks at the surface, the inclination of the reefs would steadily flatten with increasing depths, so that in a not very great distance from their outcrop, they would become nearly horizontal.

"The third theory was that the beds were conformable, that no large uplifting faults were probable for a distance of three or four miles from the outcrop of the series, and that hence the depth of the reefs below the surface at any particular point could be pretty closely determined by knowing the dip of the strata on the surface. Assuming either of the first two suppositions to be true, the depth at which the reefs would be found by vertical bore-holes or shafts located a mile or two from the outcrop would be much less than would be the case were the third supposition the correct one.

"In my judgment, from all the indications afforded by mining and development work up to date, the third theory appears to be much the most probable, and several of the leading experts of the district now agree with me in this opinion. The surface dip of the strata varies at different sections; generally being quite steep at the outcrop of the series, and then flattening to inclinations of from 40° to 20° for a distance of a couple of miles; in other sections the dip is as gradual as 23° at the outcrop, becoming as steep as 45° at a distance of a mile. My general conclusion is that at a horizontal distance of three miles from their outcrop, the reefs of the main series are probably 10,000 feet, or about two miles, beneath the surface, and at a distance of two miles their depth as a rule will be not quite one and a half mile.

"The vertical depth to which these mines can be worked with profit will chiefly depend upon the value of the ore, the amount of capital required for plant and development, the quantity of water to be pumped, and the temperature of the ground. With the abundant supply of coal in the Rand the cost of hoisting the ore from great depths will not be a very important item.

"Roughly speaking, to equip a mine for working on a large scale at a depth of 3,000 feet, the first cost for plant and development, before any returns can be expected, will be £600,000. This is a large sum, and before such an expenditure is determined upon the investor ought to be well assured as to the probable value of the ore and the costs of working. None of the mines on the Rand have thus far encountered large inflows of water. Most of the water comes from springs near the surface, so that during the dry season the cost of pumping is but slight. No one can speak with certainty as to the amount of water which will be found at serious depths like 3,000 feet, but from present indications the chances are that the inflow will not be so large even at that depth as to add very considerably to the cost of mining.

"In regard to increase in the heat of the rocks with increased depth, I was able during my visit to make some careful determinations of the temperature of the water in the deep bore-hole before spoken of. This bore-hole is in comparatively dry ground, so that only a trifle of water flows from it. The water in it, therefore, is practically quiescent, and represents accurately the temperature of the surrounding walls of rock. These determinations show a temperature of 67·2° F. at a depth of 200 feet, increasing in a regular manner to 95·3° at a depth of 2,494 feet; this indicates a temperature of about 100° at a depth of 3,000 feet. Supposing the rocks to have this degree of heat, when a mine is opened up the current of cooler air from the surface passing through the workings will reduce their temperature from 5° to 10°; such a heat will add somewhat to the mining costs, but not very greatly. At the depth of 3,500 feet the high temperature will probably cause a serious addition to the mining costs. I must, however, state that the above determinations are not final, as the thermometers used must be tested again at the Kew Observatory; should it be necessary to apply a notable correction, its amount will be given hereafter.

"Summing up all these facts, I think with present conditions the conclusion is warranted that with most of the mines a vertical working depth of but little over 3,000 feet can be assumed as the limit with which they can be operated at a profit, while with richer or thicker ore, such as is now found in a few of the outcrop mines, a limit of something over 3,500 feet in depth seems reasonable. Should, however, the price of skilled labour and the cost of supplies decline in time to a European level, still deeper limits would be possible.

"The third and most important query is as to the continuance of the gold with increased depths. The results of boring, and the developments shown by the shafts and drifts on several deep-level properties indicate that the gold contents of the ore are about the same as in the outcrop mines above; but, as these explorations thus far cover only a comparatively limited extent of area, I think a much safer test is to compare the yield obtained in the last year (1894) from the producing mines with their yield up to the middle of 1892. For the purpose of this comparison I will take the mines extending from the United Langlaagte to the May Consolidated inclusive, embracing a length of about 11 miles along the outcrop, and which have furnished to date 71 per cent. of the total output of the Rand. These mines are now extracting ore from an average depth of about 400 feet on the depth of the reefs, while in 1892, as stated in the former report, their average depth on the dip was about 160 feet. Taking the average returns from such a long extent of mining ground as 11 miles, with a depth increased more than twice, should afford one fairly safe *data* to determine whether or not future changes are probable.

"Up to August 1, 1892, there had been taken from these particular mines about 3,000,000 tons of ore, which had yielded 12½ dwt. of gold bullion per ton, after making some allowance for gold remaining in tailings unworked. Since then these mines have produced about 4,000,000 tons of ore, of which a little less than 2,000,000 tons were taken out in 1894. The yield per ton in 1894 was 13$\frac{2}{10}$ dwt. of gold bullion per ton, thus apparently showing a slight increase in yield; but in the estimate of 1892 I now see that I somewhat under-estimated the yield from the tailings, and my present estimate is that these mines for the 7,000,000 tons of ore mined from 1887 to the end of 1894 have yielded (including tailings, but not including slimes, which have not yet been treated on a commercial scale) at the rate of 13$\frac{3}{10}$ dwt. per ton. I therefore conclude that there is no sign whatever up to the present time of any change in the grade of the ore. I may mention that in this comparison I have taken into account the sorting—throwing out the poor pieces of rock before milling—which has lately been practised in several mines, and on the other hand, the greater thickness of reef matter extracted from a few of the mines.

"As all present indications show that the average values of the reefs remain unchanged to a vertical depth of 1000 feet—say 1750 feet upon their dip—it is not a very daring expectation to count upon ore of about the same grade to a limit of 3000 feet vertical. It is of interest to note that up to the present time, with hardly a notable exception, the value of the ore remains nearly constant in each of the several mines—that is to say, the lowest situated ore in the best mines continues rich, while with the poorer mines the ore still continues low grade. The yield of the ore in these mines along this stretch of eleven miles varies from 8 dwt. to 27 dwt. per ton, and I was greatly surprised during my late visit to find that this variation remains so nearly constant with increased depths. I cannot but think that this will alter when much greater depths are attained, so that in years to come better ore will be found under mines which are now poor, and poorer ore under mines which are now high grade.

"The fourth and last query has been solved by the formation of a number of deep-level mining companies, each one owning from 130 to several hundred mining claims (a mining claim is a little less than 1½ English acre). Properties of so large an area warrant the sinking of deep shafts, and the consequent heavy expenditure of first capital cost.

"In the report of 1892 I estimated for the length of 11 miles the average thickness of the ore to be worked was 5 feet; I should now estimate it to be 6 feet, but I find in many mines that a good deal of poorer ore has been left standing, so that the average yield from this thickness of 6 feet when it is all mined will be less than the 13 dwt. before given. My opinion, though, as given in 1892 of the quantity of gold to be extracted remains unchanged, the greater thickness compensating for the smaller yield per ton. In 1894 the value of the Rand gold bullion was £7,000,000, and this without any increase from the new deep-level mines; these latter will become fairly productive in 1897, so for that year a product of fully £10,000,000 can be fairly expected. Judging from present appearances, the *maximum* product of the Rand will be reached about the end of this century, when it will probably exceed £12,500,000 per annum.

"In addition to the yield which may be expected from the main reef series, I think in a few years a considerable quantity of gold will be produced from other reefs, especially from what is called the 'black reef.' This reef, with perhaps one exception, thus far appears to be what in mining parlance is called 'spotted,' the ore varying greatly in value in the distance of a few feet. Very likely more money will be lost than made in working this deposit, but the gold from it may in time add appreciably to the bullion output of the district.

"From the foregoing statement it is evident that the chances are far greater now than they were in 1892 of my conjectures of that date being realised, and to-day nearly every one conversant with the Rand considers them as being considerably under the mark. The Rand for 1894 with its product of £7,000,000 stands third in the world, the United States still remaining first with its greatly-increased output of over

£9,000,000, and Australasia (Australia, New Zealand, and Tasmania) being probably second with a product of about £8,000,000. In 1849 the world's product of gold was about £6,000,000, which increased to something over £30,000,000 in 1853, owing to the discovery and working of the rich placers of California and Australia; from 1853 the yield steadily declined until in 1883 it had fallen to less than £20,000,000. Since 1887 the yield has advanced by leaps and bounds, the increase being chiefly due to the new discoveries in South Africa, until for 1894 the product has most probably amounted to fully 8,600,000 ounces of fine gold, worth over £36,500,000, an output certainly much greater than that for any previous year in the history of the world. In 1853 it was evident that the great yield from both California and Australia would be short lived, whereas the probabilities now are that this great product of £36,500,000 will be fully maintained for quite a number of years to come; and yet, in spite of this fact and this belief, the prices of commodities generally in use, such as wheat, cotton, wool, sugar, iron, copper, &c., are now lower than they have been for the past 100 years.

"It has been generally accepted that one of the principal causes of the rise in the price of standard articles from 1849 to 1860 was due to the influx of gold from California and Australia. Will the same rise in values measured by the ounce of gold take place in the coming five years? This is a question of vast importance to all of us, from the richest capitalist to the poorest labourer.

"I see that at least one authority of position seems to be of the opinion that the probable rise in prices due to this great flood of gold will have the effect of so increasing the cost of mining and reduction that many mines in the Rand will be compelled to suspend work. This, I think, is an altogether erroneous view, for should general prices recover to their level of ten or fifteen years ago, this additional cost would be fully compensated for by the increased economies which year by year will be carried into effect in operating the Rand mines; so, unless a mountain of gold should be discovered somewhere or other, the Rand will in every probability continue to increase its yield for at least five or six years to come.

"Admitting that the foregoing anticipations of such a large gold product from the Rand will be realised, it does not necessarily follow that the profits to mine-owners as a class will be correspondingly great. In most mining districts the brilliant success of a few rich mines has had the effect of inducing men to expend large sums in exploiting neighbouring mines, with disastrous financial results, so that in the long run the aggregate losses have often been greater than the aggregate profits. The Rand deposits are, however, so much more regular than has been the case with previous gold deposits or lodes that it is almost certain that in the next 20 years or more the profits from working them will much exceed the losses. Still, the investor should remember that even on the Rand there are mines and mines, some good and some poor, and that, in order to avoid losses, careful discrimination in his ventures will be necessary. The excellence of these mines is not due to their exceeding richness, but to large continuous bodies of ore of moderate grade. In order to work them profitably first-rate mechanical plants are necessary, and they must be skilfully and economically managed. I think it is safe to say that there are now on the Rand not more than three or four mines which would yield any considerable profit were their management radically bad; hence the investor should in advance inform himself as to this important point. From the regularity which has thus far characterized the reefs it is now not very difficult to make a pretty fair guess as to the length of life which each mine will have; hence one should determine first the rate of interest at which he is willing to risk his money, and then calculate what the rate of amortization will be to protect his capital. As to expected profits past results now afford a pretty fair guide, but the prudent investor should not forget that nearly every mining man—I do not claim to be an exception--is over sanguine when counting upon future mining profits.

"From the Rand mines since their commencement in 1887 up to December 31, 1894, I estimate that 10,110,000 tons of ore (2,000lb. each) have been extracted, yielding 6,544,584 ounces of gold bullion, worth about 69s. per ounce, and thus having a gross value of about £22,600,000. The dividends paid by the producing mines during the same period amount to £4,484,541, but of this sum I consider nearly £200,000 was unquestionably not fair mining profit, so that £4,300,000 can be assumed as having been the net return, or 19 per cent. of the output. For the year 1894 there was milled 2,827,365 tons, coming from fifty producing mines, yielding 2,024,162 ounces of bullion, worth about £6,980,000, and the dividends declared by them for the year amount to £1,406,266, being 20 per cent. of the output. The market value of these fifty mines on January 19, 1895, taking middle quotations, was £33,000,000.

"During 1894 several of the leading mines expended a considerable portion of their earnings in paying for property and in the construction of new works, and quite a number of the fifty mines referred to will in 1895 be able to increase their output, and in all probability their dividends also. On the other hand, several of these mines were operated at an actual loss in 1894. From 1887 to 1894 inclusive the profits made in actually working the mines of the district have, I think, been much more than the losses, although in the years 1887-1890 large sums were expended in developing mines, many of which thus far appear to be worthless. As a very rough guess I should say the losses from 1887 to the present time amount to £2,000,000, while the dividends distributed have been twice that sum. Since 1892 the proportion of losses to profits has been much smaller.

"In quite a number of deep-level properties the work of development is now being vigorously pushed forward by means of vertical shafts and drifts from them. These new workings will be so extensive that they will before long add very greatly to our present knowledge of the Rand deposits, so that both from an economical and technical point of view the years 1895 and 1896 promise to be the most interesting in the history of the district. I hope, with your permission, at some future date to trace these new developments, and to point out the final conclusions which can be drawn from them."

# APPENDIX C.

## *STATISTICAL NOTES AND MEMORANDA.*

For convenience of reference in case of similar undertakings, I append some statistics of working expenditures and the like at various mines in the Witwatersrand gold-fields, for which I am indebted to the officials of the several undertakings.

### Crown Reef Mine.

For twelve months ending 1st June, 1894 :—

| | £ | s. | d. | £ | s. | d. | |
|---|---|---|---|---|---|---|---|
| Mining expenses | 84,686 | 0 | 1 | 0 | 13 | 7·068 | per ton. |
| Transport | 2,066 | 11 | 3 | 0 | 0 | 3·979 | ,, |
| Milling | 20,552 | 11 | 4 | 0 | 3 | 3·575 | ,, |
| Cyanide | 31,698 | 5 | 10 | 0 | 5 | 1·036 | ,, |
| General charges | 16,602 | 9 | 10 | 0 | 2 | 7·969 | ,, |
| Maintenance | 16,879 | 18 | 10 | 0 | 2 | 8·503 | ,, |
| Mine development redemption | 6,094 | 0 | 2 | 0 | 0 | 11·734 | ,, |
| Depreciation | 24,962 | 12 | 2 | 0 | 4 | 0·067 | ,, |
| Total operating costs | £203,542 | 9 | 6 | £1 | 12 | 7·931 | ,, |
| Yield from mills | 187,563 | 1 | 4 | 1 | 10 | 1·164 | per ton. |
| Yield from tailings | 94,145 | 7 | 10 | 0 | 15 | 1·282 | ,, |
| Yield from concentrates | 4,087 | 9 | 8 | 0 | 0 | 7·870 | ,, |
| | £285,795 | 18 | 10 | £2 | 5 | 10·316 | ,, |
| Less expended | 203,542 | 9 | 6 | 1 | 12 | 7·931 | ,, |
| Year's profit | £82,253 | 9 | 4 | £0 | 13 | 2·385 | ,, |

#### DETAILS OF MINING EXPENSES.

| | £ | s. | d. | |
|---|---|---|---|---|
| General costs | 0 | 0 | 8·052 | per ton. |
| Banking, pumping, winding, tramming, and stone-crushers | 0 | 2 | 7·716 | ,, |
| Stoping | 0 | 6 | 6·543 | ,, |
| Drifting, rising, and sinking | 0 | 2 | 8·346 | ,, |
| Timbering | 0 | 0 | 10·935 | ,, |
| Plate-laying | 0 | 0 | 2·171 | ,, |
| Total | £0 | 13 | 7·763 | ,, |

## The Robinson Mine.

The output for 1893 of this undertaking—the premier mine on the Witwatersrand—at which were crushed with 60 stamps 94,842 tons of ore, was as follows:—

| | | | Per ton crushed. | | | Value. | | |
|---|---|---|---|---|---|---|---|---|
| | Oz. | Dwts. | Oz. | Dwts. | Grs. | £ | s. | d. |
| From mill . . . | 104·222 | 18 | 1 | 1 | 23·48 | 377,116 | 15 | 0 |
| From 2,714 tons concentrates treated by chlorination . . . | 10·659 | 18 | 0 | 2 | 5·95 | 43,902 | 16 | 2 |
| From 55,200 tons tailings treated by cyanide . | 17·921 | 4 | 0 | 3 | 18·70 | 57,944 | 10 | 4 |
| Total . . . | 132·804 | 4 | 1 | 8 | 0·13 | £478,964 | 1 | 6 |

### The Robinson Mine: Cost of Mining and Milling (exclusive of Mine Development) for 1893.

| | | | | | | | Cost per ton. | | |
|---|---|---|---|---|---|---|---|---|---|
| | £ | s. | d. | £ | s. | d. | £ | s. | d. |
| *Mining*, wages, and material, mine maintenance | 64,760 | 7 | 0 | | | | | | |
| Repairs to hauling, pumps and mine plant . . | 3,088 | 5 | 10 | 67,848 | 12 | 10 | 0 | 14 | 0·91 |
| *Milling*, assaying, retorting, smelting . . | 14,154 | 4 | 6 | | | | | | |
| *Mill maintenance*, repairs to mill and pumping station . . . | 4,287 | 2 | 8 | 18,441 | 7 | 2 | 0 | 3 | 10·66 |
| *General maintenance* of dams, buildings, &c. . | | | | 2,169 | 18 | 5 | 0 | 0 | 5·49 |
| General expenses . . | | | | 14,465 | 11 | 11 | 0 | 3 | 0·61 |
| Total . . . . | | | | £102,925 | 10 | 4 | £1 | 1 | 8·45 |

## THE ROBINSON MINE: COST OF MILLING IN DETAIL.

| | | | | | | | Cost per ton of ore milled. | |
|---|---|---|---|---|---|---|---|---|
| *Rock Breakers*— | £ | s. | d. | £ | s. | d. | s. | d. |
| White and native labour and food | | | | 360 | 0 | 0 | 0 | 0·91 |
| Crusher jaws, 7,945 lbs. | 132 | 10 | 11 | | | | | |
| Power . . . . . . | 581 | 7 | 0 | | | | | |
| Stores (oil waste, &c.) . . | 213 | 9 | 2 | | | | | |
| | | | | 927 | 7 | 1 | 0 | 2·35 |
| Transport of ore to mill . . | | | | 165 | 9 | 3 | 0 | 0·42 |
| *Milling*— | | | | | | | | |
| White labour . . . . . | 3,557 | 1 | 9 | | | | | |
| Native labour and food . | 727 | 13 | 0 | | | | | |
| | | | | 4,284 | 14 | 9 | 0 | 10·84 |
| *Supplies*— | | | | | | | | |
| Mercury, 3,650 lbs. = ·56 oz. per oz. of gold recovered . | 507 | 13 | 11 | | | | | |
| Heads . . . . . . | 206 | 9 | 0 | | | | | |
| Shoes, 38,696 lbs. . . . | 708 | 15 | 4 | | | | | |
| Dies, 19,560 lbs. . . . | 360 | 13 | 1 | | | | | |
| Stems . . . . . . | 153 | 0 | 0 | | | | | |
| Tappets . . . | 40 | 19 | 0 | | | | | |
| Cams . . . . . | 33 | 6 | 0 | | | | | |
| Sieving . . . . . . | 253 | 2 | 2 | | | | | |
| Stores for battery and engine . | 345 | 18 | 4 | | | | | |
| | | | | 2,609 | 16 | 10 | 0 | 6·60 |
| *Fuel* (mill engine) . . . | | | | 2,875 | 10 | 5 | 0 | 7·27 |
| *Water*— | | | | | | | | |
| Paid Ferreira Co. during joint pumping arrangement . . | 1,243 | 19 | 0 | | | | | |
| Own pumping station:— | | | | | | | | |
| White labour . . | 286 | 0 | 0 | | | | | |
| Native labour and food . . | 58 | 0 | 0 | | | | | |
| Fuel . . . | 804 | 10 | 6 | | | | | |
| Stores . . . | 121 | 8 | 0 | | | | | |
| | | | | 2,513 | 17 | 6 | 0 | 6·36 |
| *Mill Maintenance*— | | | | | | | | |
| White and native labour and food . . . . | 777 | 1 | 0 | | | | | |
| Stores . . . . | 234 | 7 | 0 | | | | | |
| Proportion of workshops expenses . . . . | 48 | 2 | 6 | | | | | |
| Cut timber . . . | 45 | 11 | 0 | | | | | |
| Foundry castings . . . | 320 | 3 | 4 | | | | | |
| | | | | 1,425 | 4 | 10 | 0 | 3·61 |

THE ROBINSON MINE: COST OF MILLING IN DETAIL (*Continued*).

| | | | | | | | Cost per ton of ore milled. | |
|---|---|---|---|---|---|---|---|---|
| | £ | s. | d. | £ | s. | d. | s. | d. |
| *Concentrating—* | | | | | | | | |
| (Cost per ton of concentrates saved £1 0s. 10·05d.) | | | | | | | | |
| White labour | 1,170 | 0 | 0 | | | | | |
| Native labour and food | 412 | 0 | 0 | | | | | |
| | | | | 1,582 | 0 | 0 | 0 | 4·00 |
| Power and water supply | | | | 124 | 0 | 0 | 0 | 0·32 |
| Vanner rollers | 87 | 9 | 0 | | | | | |
| Vanner belting | 457 | 0 | 0 | | | | | |
| Stores | 120 | 1 | 2 | | | | | |
| | | | | 664 | 10 | 2 | 0 | 1·68 |
| *Maintenance—* | | | | | | | | |
| White and native labour and food | 194 | 2 | 6 | | | | | |
| Stores | 156 | 0 | 0 | | | | | |
| Proportion of workshops expenses | 11 | 18 | 0 | | | | | |
| Cut timber | 30 | 0 | 0 | | | | | |
| Foundry castings | 27 | 12 | 10 | | | | | |
| | | | | 419 | 13 | 4 | 0 | 0·06 |
| *Assaying and smelting* (proportion)— | | | | | | | | |
| White and native labour and food | | | | 351 | 0 | 0 | 0 | 0·89 |
| Stores (chemicals, coke, fuel, &c.) | | | | 138 | 3 | 0 | 0 | 0·35 |
| | | | | 18,441 | 7 | 2 | 3 | 10·66 |

## Ferreira Mine.

Returns for June, 1894:—

| | |
|---|---|
| Tons mined | 7,499 |
| Tons sorted | 8,679 |
| Tons milled | 3,821 |
| Per cent. sorted | 49·04 |
| Tons treated by cyanide | 3,692 |

| | | | | |
|---|---|---|---|---|
| Value per ton won by amalgamation | 22 | dwts. | 9·25 | grs. |
| Value per ton won by concentration | 5 | ,, | 19·855 | |
| Value won by cyanide per ton of ore milled | 5 | ,, | 0·56 | |
| Total value recovered per ton of ore milled | 33 | ,, | 15·662 | ,, |

## Jumpers Mine.

During three months ending June 30th, 406 tons of concentrates, yielding 3,085 oz. 19 dwts. 14 grs. fine gold, or 6·214 dwts. per ton of ore crushed, have been sold. The profit, after treatment, amounts to £10,100 7s. 5d., equal to £24 17s. 6·95d. per ton.

## Simmer and Jack Mine.

| | | | |
|---|---|---|---|
| The ore assays . . . . | | 13 dwts. | 5 grs. |
| Blanketings . . . | 1 oz. | 5 „ | |
| Tailings . . . . | | 3 „ | 18 „ |

At this mine the blanketings are amalgamated in Wheeler-pans, and the tailings from pans are run into six round buddles, and the headings from the buddles are treated in a chlorination plant erected close to the battery.

## Cost of Milling at Various Mines.

| | | s. | d. |
|---|---|---|---|
| Jubilee Gold Mining Co. . . | 35 stamps | 5 | 0·36 |
| Meyer and Charlton . . . | 50 „ | 3 | 6·5 |
| City and Suburban . . | 50 „ | 3 | 11·4 |
| Robinson Mine . . . . . | 70 „ | 4 | 0·13 |
| Jumpers Mine . . . . . | 100 „ | 6 | 1·48 |
| Crown Reef Mine . . . . | 120 „ | 3 | 3·57 |
| Simmer & Jack . . . . . | 100 „ | 5 | 8·59 |
| New Primrose . . . . . | 100 „ | 2 | 6 |
| Princess Mine . . . . . | 30 „ | 3 | 6·5 |
| United Roodepart . . | 30 „ | 4 | 3·0 |
| Worcester Estate and Gold Mining Co. . . . . . . . | 20 „ | 6 | 7·2 |
| Ferreira Gold Mining Co. . . | 20 „ | 6 | 7·68 |
| May Consolidated . . . . | 50 „ | 5 | 3·69 |
| Geldenhuis Estate . . | 80 „ | 5 | 0 |
| New Aurora West . . . . | 40 „ | 3 | 6 |
| Henry Nourse . . . . . | 25 „ | 5 | 5·36 |

## TABULATED STATEMENT (VARIOUS MINES) FROM MONTHLY REPORTS FOR OCTOBER, 1894.*

| | Tons Mined and Milled. | Tailings Treated. | Cost of Mining. | Cost of Milling. | Maintenance. | General Charges. | Development Redemption. | Total Cost per Ton. | Value per Ton. | Tailings. Value per Ton. | Cost per Ton. | Total Profit. | Capital Expenditure. | Remarks. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | s. d. | s. d. | s. d. | s. d. | s. d. | s. d. | s. d. | s. d. | s. d. | £ | £ | |
| George Goch | 5,340 | 7,348 | 10 9·93 | 4 3·56 | 1 3·24 | 0 8·38 | 5 0 | 22 1·11 | 22 0·17 | 12 4·53 | 3 2·51 | 3,405 | 3,919 | |
| Princess | 3,111 | 2,943 | 19 10 | 3 4·3 | 3 5·8 | 2 1·9 | 8 4 | 37 2 | 28 10 | 7 8·2 | 4 8·6 | 625 | 1,705 | Loss. |
| United Main Reef | 3.740 | 7,196 | 21 5·76 | 3 0·26 | 3 3·8 | 2 2·19 | — | 30 0 | 37 8·72 | 10 9·47 | 4 1·21 | 3,850 | 2,058 | |
| Jumpers | 9,272 | — | 8 9·49 | 5 5·51 | 2 11·87 | 0 10·38 | 5 0 | 23 8·85 | 33 10 | — | — | 4,675 | 4,616 | |
| Metropolitan | 3,455 | — | 11 7·21 | 3 7·84 | 2 0·69 | 1 4·82 | 4 0 | 22 5·56 | 28 9·41 | — | — | 1,102 | 2,928 | |
| Robinson | 9,703 | not stated | 13 11·39 | 3 7·44 | 0 10·97 | 2 3·12 | 8 1·04 | 32 11·51 | 63 2·4 | — | — | 27,855 | 3,940 | |
| New Kleinfontein | 4,579 | 4 160 | 16 3·99 | 3 9·85 | 0 10·84 | 1 7·56 | — | 23 0·24 | 29 9 | 6 10·7 | 3 2·98 | 2,312 | 2,232 | |
| Meyer & Charlton | 3,999 | 3,220 | 9 10·54 | 3 6·58 | 2 0·39 | 3 6·59 | 3 0 | 25 8·39 | 51 8·29 | 13 4·7 | 3 8·30 | 5,237 | 4,323 | |
| Simmer & Jack | 8,315 | — | 15 7 | 8 1 | — | 1 5·5 | 3 9·5 | 28 11 | 27 9 | — | — | 2,955 | 9,681 | Tailings sold. |
| New Primrose | 13,370 | 9,730 | — | — | — | — | — | 19 9 | — | — | 6 7·0 | 10,500 | — | |
| Chimes | 4,248 | — | — | — | — | — | — | 26 11+ | — | — | — | 3,305 | 871 | +Cyaniding included. |
| Crown Reef | 17,602 | 20,272 | 16 0·46 | 3 0·15 | — | 2 4·04 | 1 3·28 | 22 7·95 | 25 6·74 | 10 5·5 | 5 6 | 10,754 | 3,066 | |
| Henry Nourse | 2,259 | 1,800 | 26 8·45 | 7 8·61 | — | — | 8 0 | 42 5·06 | 59 6 | 23 6·6 | 8 6·21 | 2,996 | 8,659 | |
| May Consolidated | 6,545 | — | 12 4·84 | 5 3·42 | — | — | 2 1·56 | 19 9·82 | 22 9 | — | — | 1,190 | 3,636 | |

---

* From the *South African Mining Journal* of 24th November, 1894.

Missing Page

# INDEX.

---

PRINTED BY J. S. VIRTUE AND CO., LIMITED, CITY ROAD, LONDON.

# New and Standard Works

ON

# MINING AND METALLURGY,

PUBLISHED BY

CROSBY LOCKWOOD & SON,

7, STATIONERS' HALL COURT, LONDON, E.C.

---

Fourth Edition, Enlarged, 8vo, 700 pp. with 200 Illustrations in text and 25 Folding Plates and Working Drawings. 16s., cloth.

THE

## METALLURGY OF GOLD:

*A Practical Treatise on the Metallurgical Treatment of Gold-Bearing Ores, including the processes of Concentration, Chlorination, and Extraction by Cyanide, and the Assaying, Melting, and Refining of gold.*

BY M. EISSLER,

MINING ENGINEER; A.I.M.E.; MEMBER OF THE INSTITUTE OF MINING AND METALLURGY.

"The whole process of gold milling, from the breaking of the quartz to the assay of the bullion, is described in clear and ordorly narrative and with much, but not too much, fulness of detail."—*Saturday Review.*

---

*ALSO BY THE SAME AUTHOR. Being a reprint of Chaps. XI.—XV. of the above.*

Large crown 8vo. Price 7s. 6d., cloth.

THE

## CYANIDE PROCESS FOR EXTRACTION OF GOLD.

And its Practical Application on the Witwatersrand Gold Fields in South Africa. By M. EISSLER, M.E., Mem. Inst. Mining and Metallurgy, Author of "The Metallurgy of Gold," &c. With Diagrams and Working Drawings.

"This book is just what was needed to acquaint mining men with the actual working of a process, which is not only the most popular, but is, as a general rule, the most successful for the extraction of gold from tailings."—*Mining Journal.*

---

*ALSO BY THE SAME AUTHOR.*

Second Edition, 360 pp. and 180 Illustrations, Crown 8vo. Price 10s. 6d., cloth.

THE

## METALLURGY OF SILVER:

*A Practical Treatise on the Amalgamation, Roasting, and Lixiviation of Silver Ores, including the Assaying, Melting, and Refining of Silver Bullion*

'A technical work which, we are convinced, will supply a long-felt want amongst practical men, and at the same time be of value to students, and others indirectly connected with the industries."—*Mining Journal.*

Revised Edition, Fcap. 8vo. Price 7s. 6d. leather.

THE

# MINER'S HANDBOOK,

*A Handy Book of Reference on the Subjects of*

MINERAL DEPOSITS, MINING OPERATIONS,
ORE-DRESSING, ETC.

*For the Use of Students and Others interested in Mining Operations.*

COMPILED BY JOHN MILNE, F.R.S.,

PROFESSOR OF MINING IN THE IMPERIAL UNIVERSITY OF JAPAN.

"Professor MILNE's handbook is sure to be received with favour by all connected with mining, and will be extremely popular among students."—*Athenæum.*

"It is a veritable miniature compendium of mining, which is likely to find a place not only on the shelves, but also in the luggage of most mining engineers."—*Nature.*

---

Fcap. 8vo, 400 pp. Price 9s. leather.

A POCKET-BOOK FOR

# MINERS AND METALLURGISTS

*COMPRISING RULES, FORMULÆ, TABLES, AND NOTES FOR USE IN FIELD AND OFFICE WORK.*

BY F. DANVERS POWER, M.E., F.G.S.

"Mr. POWER has succeeded in producing a pocket-book which certainly deserves to become the *vade-mecum.*"—*Mechanical World.*

"A marvellous compendium which every miner who desires to do work rapidly and well should hasten to buy."—*Redruth Times.*

---

Third Edition, Revised, Small Crown 8vo. Price 2s. 6d. cloth.

NOTES AND FORMULÆ FOR

# MINING STUDENTS.

BY JOHN HERMAN MERIVALE, M.A.,

CERTIFICATED COLLIERY MANAGER, AND PROFESSOR OF MINING IN THE DURHAM COLLEGE OF SCIENCE, NEWCASTLE-UPON-TYNE.

"Invaluable to anyone who is working up for an examination on mining subjects."—*Iron and Coal Trades Review.*

"A vast amount of technical matter of the utmost value to mining engineers."—*Schoolmaster.*

Crown 8vo, 580 pages, with upwards of 300 Illustrations. 12s. 6d. cloth.

## MACHINERY FOR METALLIFEROUS MINES.

*A PRACTICAL TREATISE FOR MINING ENGINEERS, METALLURGISTS AND MANAGERS OF MINES.*

By E. HENRY DAVIES, M.E., F.G.S.

"Mr. Davies, in this handsome volume, has done the advanced student and the manager of mines good service. Almost every kind of machinery in actual use is carefully described, and the woodcuts and plates are good."—*Athenæum.*

"From cover to cover the work exhibits all the same characteristics which excite the confidence and attract the attention of the student as he peruses the first page. The work may safely be recommended. By its publication the literature connected with the industry will be enriched, and the reputation of its author enhanced."—*Mining Journal.*

Fifth Edition, Crown 8vo. 524 pages, with 150 Illustrations. Price 12s. 6d. cloth.

*A TREATISE ON*

## METALLIFEROUS MINERALS AND MINING.

By D. C. DAVIES, M.E., F.G.S.

FIFTH EDITION REVISED, AND MUCH ENLARGED BY THE AUTHOR'S SON,

E. HENRY DAVIES, M.E., F.G.S.

"The most exhaustive and the most practically useful work we have seen. Neither the practical miner nor the general reader interested in mines can have a better book."—*Mining Journal.*

"As a history of the present state of mining throughout the world, this book supplies an actual want, for no such information has hitherto been brought together within such limited space."—*Athenæum.*

By the same Author. With Illustrations, Crown 8vo. Price 12s. 6d. cloth.

*A TREATISE ON*

## EARTHY AND OTHER MINERALS AND MINING.

THIRD EDITION, REVISED AND MUCH ENLARGED, BY THE AUTHOR'S SON,

E. HENRY DAVIES, M.E., F.G.S.

"We do not remember to have met with any English work on mining matters that contains the same amount of information packed in equally convenient form."—*Academy.*

"We should be inclined to rank it as among the very best of handy technical and trade manuals."—*British Quarterly Review.*

By the same Author. Third Edition. Fcap. 8vo. With numerous Illustrations and Folding Plates. 3s. cloth.

## SLATE AND SLATE QUARRYING,

### SCIENTIFIC, PRACTICAL, AND COMMERCIAL.

"One of the best and best-balanced treatises on a special subject that we have met with."—*Engineer.*

"Mr. Davies has written a useful and practical handbook on an important industry, with all the conditions of which he appears familiar."—*Engineering.*

Large crown 8vo, 446 pages, with numerous Illustrations. Price 16s. cloth.

# THE IRON ORES OF GREAT BRITAIN AND IRELAND:

*Their Mode of Occurrence, Age, and Origin, and the Methods of Searching for and Working them.*

WITH A NOTICE OF SOME OF THE IRON ORES OF SPAIN.

BY J. D. KENDALL, F.G.S., MINING ENGINEER.

"A useful and solid contribution to professional literature."—*Engineer.*

"The book is one which will be welcomed by mining engineers as a very handy book of reference. The author is a careful observer, he has visited almost all the localities mentioned by him, and his descriptions are clear and trustworthy."—*Athenæum.*

"To all requiring a manual on British iron-ore occurrence and extraction we recommend Mr. Kendall's work."—*Iron.*

---

Seventh Edition, revised and enlarged. Price 4s. cloth.

*A RUDIMENTARY TREATISE ON*

# COAL AND COAL MINING.

BY

THE LATE SIR WARINGTON W. SMYTH, M.A., F.R.S.

"Every portion of the volume appears to have been prepared with much care, and as an outline is given of every known coalfield in this and other countries, as well as of the two principal methods of working, the book will interest a very large number of readers."—*Mining Journal.*

---

Sixth Edition, revised and enlarged. Price 5s. 6d. cloth.

*A TREATISE ON THE*

# METALLURGY OF IRON

CONTAINING OUTLINES OF THE

*History of Iron Manufacture, Methods of Assay, and Analyses of Iron Ores, Processes of Manufacture of Iron and Steel, etc.*

BY H. BAUERMAN, F.G.S.

"Invaluable as a text-book for the student and practical metallurgist."—*Colliery Guardian.*

"The new edition brings the record down to the latest date, and is of sound practical value."—*Mining Journal.*

Sixth Edition, thoroughly Revised and much enlarged. Small crown 8vo, price 3s. 6d. cloth; or 4s. 6d. leather, pocket-book form with tuck.

## THE
## PROSPECTOR'S HANDBOOK:
### A GUIDE FOR THE PROSPECTOR AND TRAVELLER IN SEARCH OF MINERALS.

BY J. W. ANDERSON, M.A. (Camb.), F.R.G.S.,
AUTHOR OF "FIJI AND NEW CALEDONIA," ETC.

"Will supply a much-felt want, especially among Colonists, in whose way are so often thrown many mineralogical specimens, the value of which it is difficult for any one not a specialist to determine."—*Engineer.*

"Toilers who explore the trodden or untrodden tracks on the face of the globe will find much that is useful to them in this book."—*Athenæum.*

"How to find commercial minerals, and how to identify them when they are found, are the leading points to which attention is directed. The author has managed to pack as much practical detail into his pages as would supply material for a book three times its size."—*Mining Journal.*

---

Second Edition, revised, Super Royal 8vo, upwards of 950 pages, with 230 Illustrations. Price £2 2s. cloth.

## BRITISH MINING:
### *A TREATISE ON THE HISTORY, DISCOVERY, PRACTICAL DEVELOPMENT, AND FUTURE PROSPECTS OF*
### METALLIFEROUS MINES
### IN THE
### UNITED KINGDOM.

BY ROBERT HUNT, F.R.S.
LATE KEEPER OF MINING RECORDS; EDITOR OF URE'S DICTIONARY OF ARTS, MANUFACTURES, AND MINES.

"A sound, business-like collection of interesting facts, put together in a manner that will make them generally attractive, and cause many to comprehend that mining is by no means the lottery which some have pretended, but an industry which, if honestly and energetically carried on, is not only capable of being made remunerative to those engaged in it, but beneficial to the entire community. . . . . . The amount of information which Mr. Hunt has brought together is enormous, whilst his position and experience have enabled him to preserve most of the wheat without the chaff."—*Mining Journal.*

"An exhaustive work, which we have little doubt will come to be regarded as a standard work on the subject with which it deals. . . . . In the bulky volume before us there have been collected together a vast number of interesting facts and much useful information, which, while being of most intimate concern to those engaged in mining pursuits, cannot fail to be appreciated by a wider circle."—*Iron.*

*Established* 60 *Years.*

# THE MINING JOURNAL,

## *RAILWAY AND COMMERCIAL GAZETTE.*

A complete record of Mining, Metallurgical, Financial, Industrial, and Engineering Progress.

**PUBLISHED EVERY SATURDAY. PRICE 6d.**

**SUBSCRIPTION: Inland, £1 4s. per annum. Abroad, £1 8s. per annum.**

***The Mining Journal*** circulates all over the World, amongst Miners, Engineers, Manufacturers, Capitalists and Investors.

**Specimen Copy and Advertisement Scale on Application.**

**ADVERTISEMENT SCALE.**—Prospectuses £12 12s. per column, £30 per page; Auctions, Sales, Wanted Situations, &c., 8d. per line, minimum charge, 4s.; Companies' and Legal Announcements, 9d. per line, minimum charge, 7s. 6d.; Special and Displayed Advertisements, not included in the foregoing, inserted on arranged terms.

**Offices: 18, FINCH LANE, LONDON, E.C.**

Telegraphic and Cable Address: "TUTWORK, LONDON." Codes used: "A.B.C.," "MOREING'S," & "UNIVERSAL."

☞ The International Exhibition of Mining and Metallurgy, Crystal Palace, 1890, the Institution of Mining and Metallurgy, London, and the Mining Section of the London Chamber of Commerce, were initiated and established by the efforts of ***The Mining Journal.***

---

Just Published, Medium 8vo, about 700 pages, strongly bound for Office and Desk use. Price £1 5s. cloth.

A NEW EDITION OF THE

# COLLIERY MANAGER'S HANDBOOK:

*A Comprehensive Treatise on the Laying-out and Working of Collieries, designed as a Book of Reference for Colliery Managers, and for the use of Coal-Mining Students preparing for First-Class Certificates.*

BY CALEB PAMELY,

MINING ENGINEER AND SURVEYOR; MEMBER N. E. INSTITUTE OF MINING ENGINEERS.

*With nearly 500 Plans, Diagrams and other Illustrations.*

Third Edition, Revised, with Additions.

"Mr. PAMELY's work is eminently suited to its purpose, being clear, interesting, exhaustive, rich in detail, and giving descriptions of the latest machines in every department. . . . A mining engineer could scarcely go wrong who followed this work."—*Colliery Guardian.*

"Contains a great deal of information which managers and students will find of use to them. The excellent index will enable them to lay their hands at once on any part they desire to consult."—*Nature.*

"This is the most complete 'all-round' work on coal-mining published in the English language. . . . No library of coal-mining books is complete without it."—*Colliery Engineer* (Scranton, Pa., U.S.).

**LONDON: CROSBY LOCKWOOD & SON, 7, Stationers' Hall Court, E.C.**

MINE DRAINAGE: a Practical Treatise on Direct-acting Underground Steam Pumping Machinery. By Stephen Michell. With numerous Wood Engravings and Folding Plates. 8vo, 15s. cloth.

THE MINERAL SURVEYOR AND VALUER'S COMPLETE GUIDE. Comprising a Treatise on Improved Mining Surveying and the Valuation of Mining Properties, with new Traverse Tables. By W. Lintern, Mining Engineer. Third Edition, with an Appendix on Magnetic and Angular Surveying. With Four Plates. Fcap. 8vo. 3s. 6d. cloth.

THE RUDIMENTS OF MINERALOGY: a Concise View of the Properties of Minerals. By A. Ramsay, F.G.S. Fourth Edition, Revised. With Woodcuts and Steel Plates. Price 4s. cloth.

RUDIMENTARY TREATISE ON GEOLOGY: Physical and Historical. Partly based on Major-General Portlock's "Rudiments of Geology." By Ralph Tate, A.L.S., F.G.S., &c. Price 5s. cloth.

SUBTERRANEOUS SURVEYING, with and without the Magnetic Needle. By Thomas Fenwick, Surveyor of Mines, and Thomas Baker, C.E. Illustrated. Price 3s. cloth.

A MANUAL OF MINING TOOLS. For the Use of Mine Managers, Agents, Students, &c. By William Morgans. Price 3s. cloth.

ATLAS OF ENGRAVINGS to Illustrate the above, containing 235 Illustrations, drawn to scale. 4to. Price 4s. 6d. cloth.

A FIRST BOOK OF MINING AND QUARRYING, for Primary Schools and Self-Instruction. By J. H. Collins, F.G.S. Price 1s. 6d. cloth.

THE BLOWPIPE IN CHEMISTRY, MINERALOGY, AND GEOLOGY. Containing all known Methods of Anhydrous Analysis, Working Examples, and Instructions for Making Apparatus. By W. A. Ross, Lieut.-Col. R.A., F.G.S., &c. Illustrated. Second Edition, revised and enlarged. Crown 8vo. Price 5s. cloth.

STATIONARY ENGINE DRIVING. A Practical Manual for Engineers in charge of Stationary Engines. By Michael Reynolds. Fourth Edition. Crown 8vo. Price 4s. 6d. cloth.

PRACTICAL TUNNELLING. By Frederick W. Simms, F.G.S., M.Inst.C.E. Third Edition, Revised and Extended, by D. Kinnear Clark, M.Inst.C.E. Imperial 8vo, with 21 Folding Plates and numerous Wood Engravings. Price 30s. cloth.

THE COAL AND IRON INDUSTRIES OF THE UNITED KINGDOM. By Richard Meade, Assistant Keeper of Mining Records. With Maps. 8vo. Price £1 8s. cloth.