RECENT COPPER SMELTING

EDITED BY

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PREFACE

Copper smelting practice is now in a period of rapid development similar to that upon which cyanide practice entered a decade ago. This compilation of recent articles on copper metallurgy will, it is hoped, serve to present in convenient form the views of well known engineers on current practice from Chile to Alaska and from Russia to Japan.

THOMAS T. READ,
Editor.

New York, May 5, 1914.

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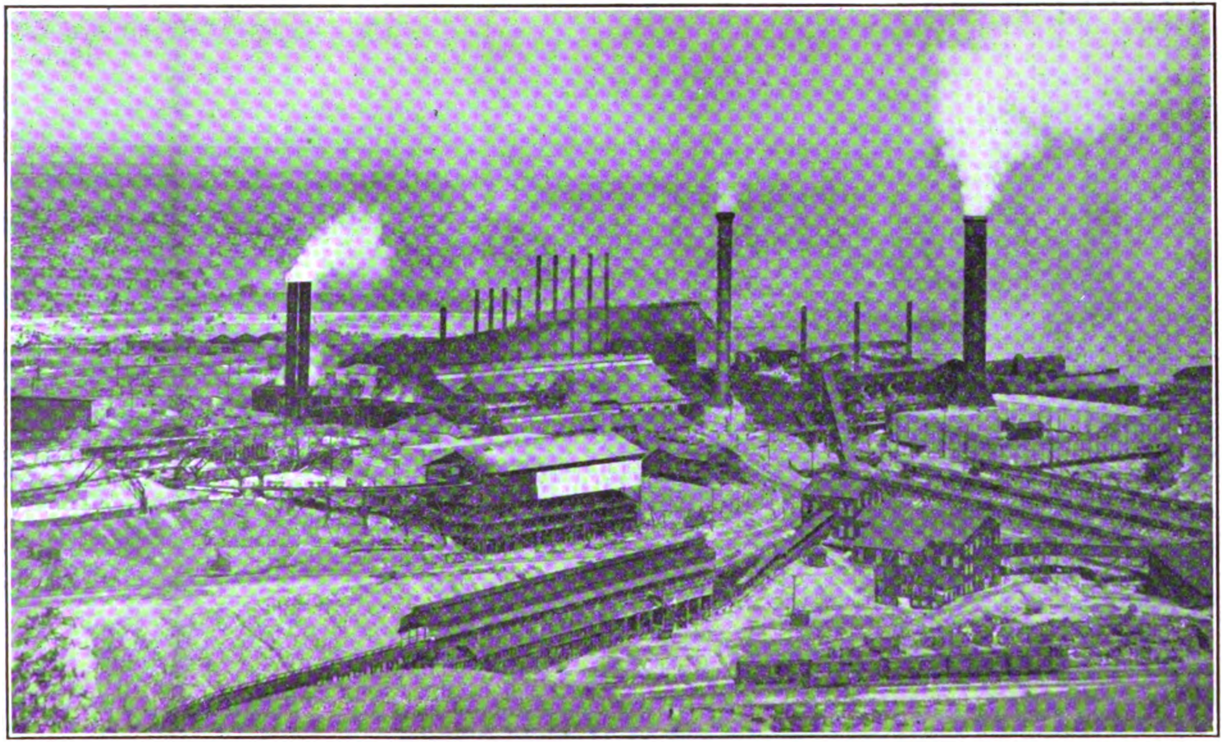


General View of Smelting Plant, Greene Cananea Copper Co.

EDITORIAL INTRODUCTION

The articles which make up this volume have, with some exceptions, appeared in the pages of the *Mining and Scientific Press* during the past four years. Several papers presented before the American Institute of Mining Engineers have, because of their importance, and with the kind permission of the Institute, been included. Two others are of earlier date than 1910. The essential features of the results of John Hollway's classic experiments, made in 1878, are told in his own words, showing how that early worker, in the course of an attempt to demonstrate that the heat of oxidation of pyrite ores is sufficient to maintain fusion during smelting operations, outlined the essential features of the production of copper from matte in the converter, without apparently recognizing the full significance of his work. The article by Lewis T. Wright is included for a variety of reasons. It is a brilliantly clear and concise statement by one of the leaders in the development of this phase of copper metallurgy, and it also serves, when read in conjunction with the articles on reverberatory smelting, to illustrate the great changes in the metallurgy of copper which a few years have brought about.

A decade ago it was clearly recognized by everyone that the growing use of copper for industrial purposes would make necessary a great increase in its production. It was equally evident that no more great bodies of comparatively rich sulphides or native copper, such until then as had been the mainstay of the industry, were likely to be found. Universal attention was directed toward the large deposits of 'cupriferous pyrite,' as they were generally termed. Meanwhile a group of metallurgists, L. T. Wright, W. H. Freeland, F. R. Carpenter, Robert Sticht, Herbert Lang, and many others, had been busy developing the technique of the metallurgical operation of cheaply smelting ore consisting largely of pyrite, marcasite, or pyrrhotite. Nearly everyone regarded pyrite smelting without coke as the metallurgy *par excellence* of the immediate future.

Meanwhile an apparently small discovery had been made. 'Cupriferous pyrite' proved to be a figment of the imagination, the ores so designated being actually pyrite with chalcopyrite or other copper sulphides disseminated throughout them. It was further found that copper sulphides not infrequently occur disseminated throughout rock masses, chiefly igneous intrusives or the adjacent rocks, so as to form large bodies of low-grade ore. A second group of engineers, chief among whom was D. C. Jackling, saw the new possibilities which this opened. Cheap smelting of such material, silicious rock containing disseminated sulphides corresponding to a copper content of 25 to 50 pounds of copper per ton of ore, was out of the question, but if it could be cheaply mined and cheaply concentrated many tons into one, making a reasonable saving of the copper present, low-cost smelting would not be essential.

Thus two sharply contrasted methods of treatment are involved: on the one hand large or small-scale mining, the production of the ore in lump form, and the smelting of the whole mass, utilizing the heat of the sulphur present in order to economize in the use of fuel; and, on the other hand, large-scale mining, cheap crushing to small size, concentration (more than twenty tons being concentrated into one in some cases), roasting of the sulphide concentrate, and smelting in reverberatory furnaces. The success of the latter method hinges on the percentage of the copper present which can be recovered by concentration. As is well known, careful large-scale experiment demonstrated that two-thirds of the copper present could be recovered by wet concentration. On the basis of a cost of 42 cents per ton for mining, 28 cents for transportation, and 41 cents for milling, and the concentration of twenty tons into one, the whole operation can be conducted at a profit, even with material containing only 30 pounds of copper per ton. On this basis the relatively higher cost per ton of material smelted in the reverberatory furnace becomes of minor import in the total cost.

Several other factors contribute toward the increased and increasing favor with which reverberatory smelting is regarded. Elsewhere in this volume, E. P. Mathewson discusses with authority the causes which have led to this growth. Another cause, to which my attention has been drawn by R. L. Lloyd, is the improvement in roasting practice. When the control of the temperature and air-supply during roasting is faulty, a certain amount of ferric oxide and of ferrite of iron is produced, and additional reactions in the laboratory of the reverberatory are made necessary, thus decreasing the speed of smelting as compared with the treatment of material in which all the iron is present in the ferrous state. As a matter of practical operation, the use of liquid petroleum or powdered coal has cut down the fuel cost, and the use of boilers in the exit flues, by recovering the he[illegible] escaping gases and returning it to the power-plant as [illegible]de a further important reduction. The reverberatory is a convenient place for disposing of the increased quantity of slag made by converters since the addition of silicious ore to the matte charge became general practice. Above all, it handles without difficulty the finely divided concentrate which is the necessary product of the method of treatment referred to above, and similarly disposes of the dust accumulations yielded by other departments of the plant, such as the roasters. The most recent phase of development is the use of flotation to recover the copper content of the ore. This makes even finer crushing necessary and accentuates the advantages of the reverberatory for the treatment of material in a fine state of comminution.

Blast-furnace smelting of lump ore has many advantages, which it can scarcely be necessary to mention, but is subject to the limitation that even under the most favorable conditions a considerable proportion of the ore will be of smaller size than desired. This may be sent directly to the blast-furnace, with the result that the speed of smelting is decreased, which amounts to increasing the overhead costs, and a large quantity of flue-dust is produced. This can be caught in dust-chambers, but if put back into the blast-furnace without further treatment is merely blown out again in a sort of perpetual cycle. It may be agglomerated in brick presses using a suitable binder, agglomerated by the use of converter slag, or sintered by one of the various processes. All these operations add to the total cost and somewhat complicate the work of the plant. With the exception of the latest methods of sintering, they are all unsatisfactory, and the tendency of present practice all seems to be toward the extension of reverberatory smelting. In large plants coarse material is smelted in the blast-furnace, while reverberatories take care of the blast-furnace flue-dust and the fine ore or concentrate. Where the amount of finely divided material is too small to justify the construction and operation of a reverberatory, the sintering of fine ore and flue-dust is an efficient way of meeting the problem. But where, as in the plant recently built at Hayden and the one now building at Globe, all the material to be smelted is finely divided, the reverberatory alone, coupled with the present very efficient roasting technique, serves to meet the situation.

The short space of time covered by the articles which make up this volume bridges over three sets of problems in the metallurgy of copper. I have already discussed the readjustment of smelting problems which grew out of the exploitation of the disseminated copper ores. But, as always in human endeavor, the solving of any important problem merely leads up to the recognition of another. The financial profit resulting from the exploitation of the disseminated ores does not serve to obscure the fact that one-third of the copper content escapes recovery. In addition, large ore reserves of oxidized copper minerals are known to exist in various places, and these are not amenable to profitable treatment by any present method. The use of wet methods for the recovery of copper from its ores has been proposed ever since early times, and they have been successfully practised in a few places where local conditions favor their use. Generally speaking, however, they are complicated and require a nice adjustment of mechanical equipment to chemical reactions. Except upon a large scale, they are, for general practice, an economic impossibility, but the readjustment of modern conditions seems to be tending toward a state of affairs where wet methods become feasible. The number of processes proposed is legion, but leaching with sulphuric acid is the rule in present work. The precipitation of the copper from the pregnant solution is a vital problem. It can be done by the use of metallic iron, or by electrolysis, but the difficulty is to attain a low cost. Both methods are about to be put into operation on a large scale, and in a year or two our knowledge will be more complete than at present. Sulphuric acid, either for leaching or for other methods of chemical attack, seems likely to remain the chief reagent, for the sulphur content of the fume escaping from smelter stacks is everywhere a problem, and its utilization for the manufacture of sulphuric acid, some of which may be used in the treatment of oxidized or roasted sulphide ore, is likely to be one of the lines along which the copper metallurgy of tomorrow may develop.

THOMAS T. READ.

New York, May 5, 1914.

Recent Copper Smelting

HANDLING FURNACE CHARGES AT CANANEA

By Morris Jesup Elsing

(May 4, 1912.)

The methods of handling furnace charges at the smelting plant of the Greene Cananea Copper Co., Cananea, Sonora, Mexico, involve many unusual features. Under the management of Arthur S. Dwight, with the assistance of E. H. Messiter, consulting engineer for the Robins Conveying Belt Co., a series of belt conveyors and mechanical devices were designed and constructed for mixing the blast-furnace charge and delivering it to the furnace. Several parts of the original construction have been replaced because of difficulties found in operating, and the installation is now working with great success. The diagrammatic 'flow-sheet', Fig. 2, illustrates the following description, and shows the details of capacity. The 'flow-sheet' of the sampling works is also included.

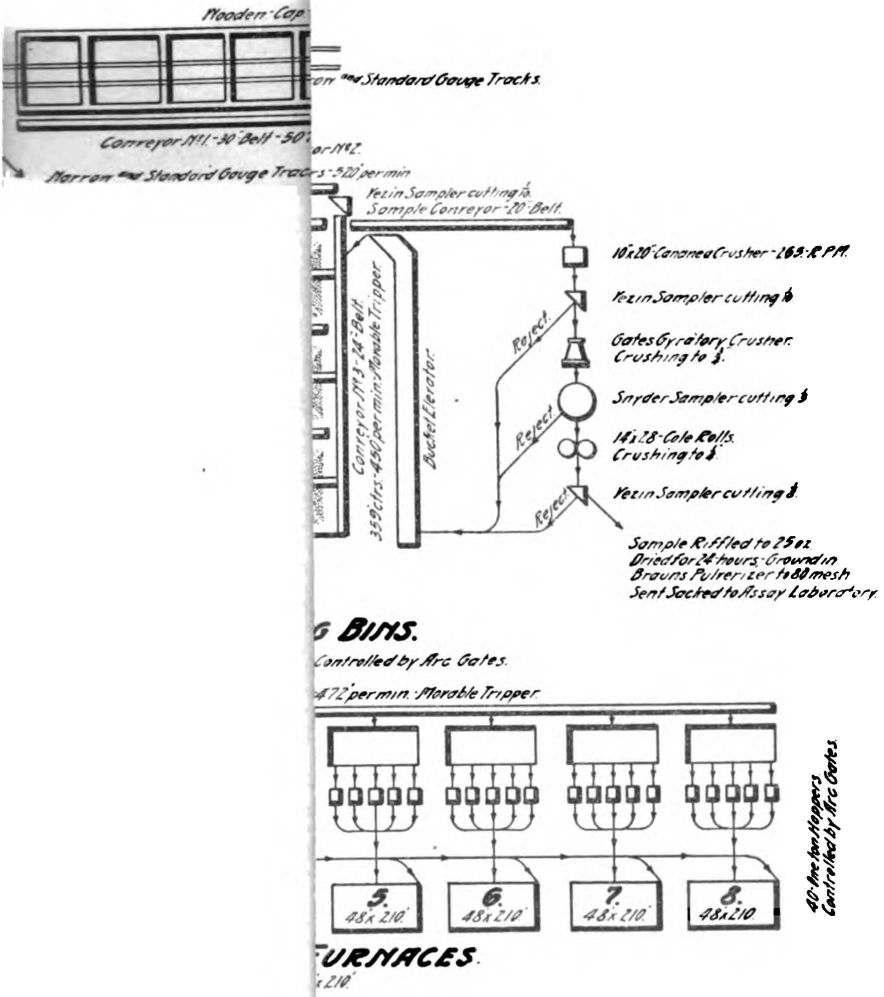
In outline the plant consists of a series of eleven receiving bins, constructed of wood, for ore and secondaries, and eight steel bins for concentrate and picked ore. These bins are filled by trains from the mines and concentrator, after each car has been weighed on a Fairbanks scale. The bins are equipped with automatic feeding devices that deliver to a belt conveyor, which transports the material to the sampling works, where it is automatically sampled.

The feeding device attached to the wooden bins consists of a steel table suspended by four rods at the bottom of the bins. This steel table is given a reciprocating movement by a connecting-rod attached to an eccentric placed at the back of the bins. This eccentric is attached to a shaft, which extends the entire length of the wooden bins. Each of these bins has four of these feeding devices. The ore feeds into a movable steel hopper, which breaks its fall upon the belt. The picked ore from the steel bins is fed upon the belt by a shaker feed-car, which is a hopper mounted on a carriage astride of the conveyor. This hopper has a table below its discharge opening, which is given a reciprocating motion, feeding the material on the belt at the desired rate, being controlled by certain adjustments.

The concentrate is fed upon the belt from the bottom of the bins through a steel chute which is provided with an arc gate. In order to assist the steady feeding of this wet material, an attendant is stationed at the chute with a compressed-air barring device. This consists of a piece of 1-in. pipe, about 6 ft. long, to which is attached a short piece of tapered pipe with a ¼-in. opening in the end. A compressed-air hose and valve is attached to the other end of the 6-ft. length. When compressed air is turned on, this device becomes an efficient means of barring down material that would otherwise 'hang up.' The different classes of ores, such as heavy sulphide ore, limestone ore, and the silicious ores are kept in separate bins, as well as the by-products that are smelted in the blast-furnace.

From the sampling works the material is conveyed to one of three parallel belts, which are supported on steel bridges about 20 ft. above a concrete floor (see Fig. 3). By means of Robins Conveying Belt Co.'s automatic trippers, a long narrow pile or bed is built up in successive layers. The tripper is a carriage astride of the conveyor belt, through which the belt passes. The direction of travel is determined by certain fixed brakes and traction wheels, which are reversed on reaching the end of the conveyor. In this way a continuous stream of ore is spread on the floor below the traveling tripper. The receiving bins, containing known tonnages of material, of which the approximate analysis is already known, are in turn emptied on the bed. From the sample taken as the ore passes through the mill, the exact amount of each constituent of the charge is determined, and as the bed nears completion, such materials are added as are required to make it the self-fluxing mixture that is desired.

Reclamation.—The reclamation of this bed of self-fluxing material is accomplished by a device known as the reclaimer, which was especially designed for this work. It consists of a traveling steel bridge or framework, which spans the entire width of the bed, is mounted on car wheels, and rides on rails. Under the forward side of the bridge is suspended a scraper conveyor, operating on a nearly horizontal steel bottom plate. The flights are inclined to the axis of the conveyor, and are provided with plough-shaped extensions, which draw on to the bottom plate all material within reach, and carry it along to a conveyor paralleling the bed, and placed in a trench a little below the elevation of the floor of the bed. A triangular harrow covering the entire cross-section of the bed, and given an inclination of approximately the angle of repose


Wooden Cap
and Standard Gauge Tracks.
Conveyor No. 1 - 30" Belt - 50
or No. 2.
Narrow and Standard Gauge Tracks - 520' per min.
Vezin Sampler cutting
Sample Conveyor - 20" Belt.
10 x 20" Cananea Crusher - 263 - R.P.M.
Vezin Sampler cutting
Gates Gyratory Crusher.
Crushing to
Snyder Sampler cutting
14 x 28 - Cole Rolls.
Crushing to
Vezin Sampler cutting
Reject.
Reject.
Reject.
Conveyor No. 3 - 24" Belt.
359 ctrs - 450' per min. Movable Tripper.
Bucket Elevator.
Sample Riffled to 25 oz.
Dried for 24 hours; Ground in
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Sent Sacked to Assay Laboratory.
G BINS.
Controlled by Arc Gates.
472' per min. Movable Tripper.
40 - Arc-ton Hoppers.
Controlled by Arc Gates.
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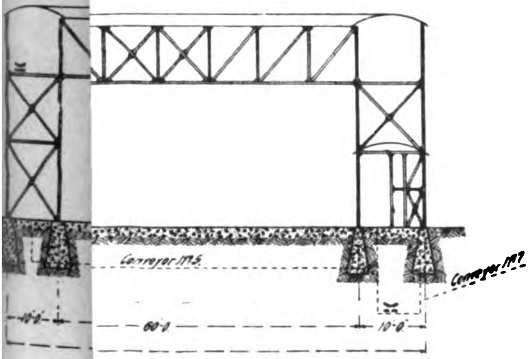


NY

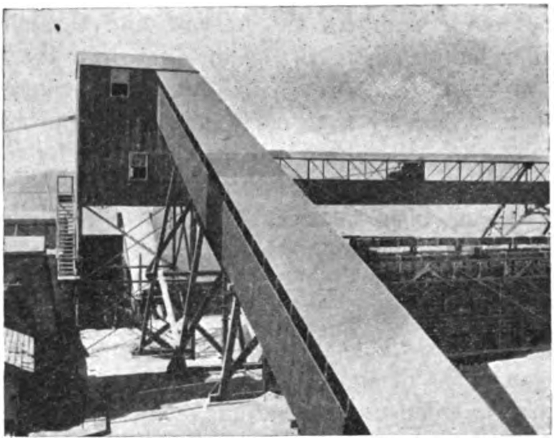
of the material, is mounted on the front of the machine. The harrow is given a reciprocating motion cross-wise to a pile. This continuously dislodges the material from all parts of the face of the bed, and causes it to roll down to the conveyor below, by which it is carried away as described.

An electric motor operates the harrow and scraper conveyor, while a smaller motor advances the machine into the pile at any desired rate, or runs it back to the end of the bed, where it is transferred to the next bed by a transfer car. The thoroughly mixed charge then passes to the other belt conveyors, being elevated and finally discharged directly into narrow steel storage bins of 75 tons capacity above each blast-furnace. These bins feed steel pockets, containing about one ton each, which are partly emptied into the furnace, and partly on the feeding floor by the simple operation of a chute and deflecting gate. The coke is brought to the furnace in wheelbarrows. The whole system of conveying and reclaiming is controlled by a series of signals so that each operator of any part is notified when to commence operations or of any stopping or overload, thus unifying the whole system.

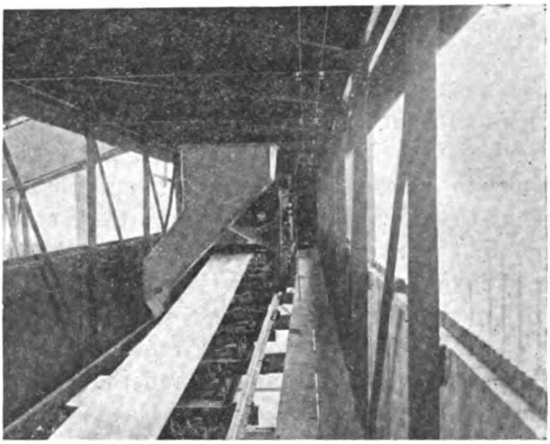
Advantages.—Among the advantages due to this method of mixing and delivering the furnace charge the reduction of the cost of smelting is the most important. This reduction of cost is two-fold and may be divided into mechanical cost and metallurgical cost. The mechanical cost has been reduced by the elimination, so far as possible, of manual labor, and its replacement by the use of the mechanical devices described. The metallurgical reduction is due to the uniformity in the chemical composition and physical character of the charge, and to advance information on these subjects.

It is evident that a given amount of ore will not often make an even number of layers on the bed, and, therefore, there will be an overlapping of layers. When the bed is completed it contains about 9000 tons or about 40,000 lb. per linear foot. With each overlap, roughly 50 lb. per foot has been added, and the result is that any particular foot may receive 1/800 more or less of a certain flux or ore than some other foot. This overlap will not constantly occur at any one place in the bed, but will be compensating. It will apply to all of the different materials added to the bed and the final outcome will be only infinitesimal variations in the analysis of any transverse section of the bed. The thorough mixing of the charge by the reclaiming machine is self-evident. Its advance into the bed is approximately an inch per minute, so that only a small quantity is being taken from every part of the cross-section. The inclination of the harrow being adjustable, makes it possible to set it at an angle so that material being agitated by it will roll down from every part of the pile and be combined on the bottom plate.

Because of the elimination of variations in the composition of the charge, it is possible to operate the furnace with a low factor of safety. In nearly every other method of charging there is a constant variation in conditions; that is, at one time the slag being produced is more acid, and at another more basic than desired. Due to this constant change, it becomes necessary not only to add fluxes to counteract these differences and restore equilibrium, but to use



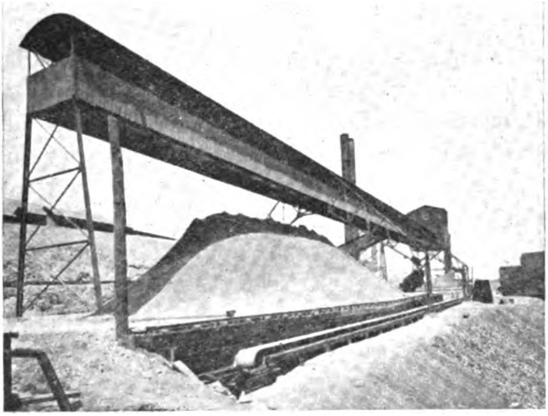
CONVEYOR LEADING TO BEDDING CONVEYOR.



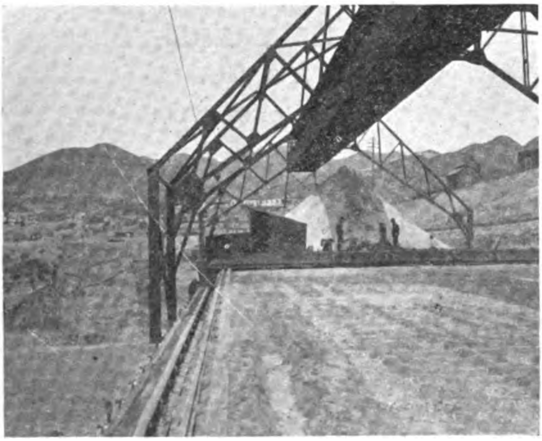
BEDDING TRIPPER.

FIG. 3. PLANT FOR HANDLING ORE.

an excessive amount of coke, and the outcome is that costs increase. With this careful mixture of charge, however, the analysis of the slag obtained can be predicted with extreme accuracy. With a given mixture the slag will show little variation from day to day, and the costs are brought down to a minimum. Other conditions being equal, accretions can in part be traced more or less directly to variation in the composition of the charge.



STACKING AND RECLAIMING CONVEYORS.



RECLAIMING SYSTEM, CONVEYOR No. 7 AT THE LEFT.

FIG. 4. MECHANICAL PLANT FOR HANDLING CONCENTRATE (Stephens-Adamson construction).

To summarize, with an exact knowledge of the composition and physical character of the furnace charge it has been possible to increase the speed of smelting, to reduce excessive accretions in the furnace, to reduce the amount of costly fluxes, coke, and air-blast required, and it has been possible also to increase the percentage of fine in the charge.

Roaster Bedding Plant.—A system somewhat similar to that in use for the blast-furnace charge has recently been installed by the Stephens-Adamson Manufacturing Co. for handling the concentrate that is roasted, and then sent to the reverberatory furnaces. The concentrate is emptied from the standard railroad cars that come from the mill into especially designed bins. From the bins it is transported by belt conveyors to a bridge below which a bed is formed by a traveling tripper. The reclamation of the bedded concentrate is accomplished by means of a transverse belt conveyor, mounted upon a movable carriage. The roaster charge is then shoveled upon this transverse belt, which discharges into a longitudinal belt, and is conveyed to the roaster bins where it is fed to the roasters as required.

THE WEDGE MECHANICAL FURNACE

By L. S. Austin

(December 28, 1912.)

The Wedge calcining furnace, of the same general type as the McDougall and Herreshoff, has been before the public for some years, and its general appearance is familar to metallurgists. It differs from the Herreshoff principally in having a large central shaft. The furnace is built in sizes from 12 to 22½ ft. diameter, and with three or more hearths and dryer hearth. It may be arranged with a fire-box or not, according to the nature of the ore to be roasted. Fig. 5 is a seven and Fig. 6 a five-hearth furnace, the latter having a fire-box.

The ore is fed at the periphery of the furnace upon the top or dryer-hearth, and is mechanically rabbled across it, entering the upper hearth at the centre of the furnace. By thus utilizing the top of the furnace for drying and pre-heating, the entering ore begins burning on the upper hearth almost at once, and need not wait until dried out. In fact, the heat used in drying out an ore containing 7% moisture may be computed as equivalent to raising the temperature of the dried-out ore to 250°C., at which a self-burning sulphide will begin to ignite.

The central shaft is 4 to 5 ft. diam., and is lined outside with fire-tile, so that it is not so hot inside but that workmen may enter at any time and remove the breech-block of any arm that is worn out.

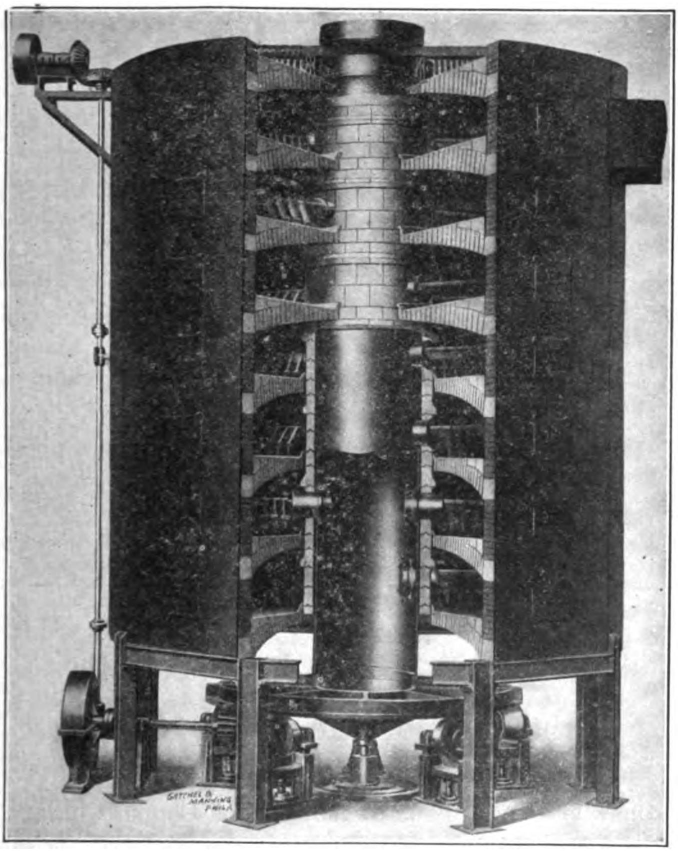


FIG. 5. SEVEN-HEARTH WEDGE MECHANICAL FURNACE.

Workmen on the outside withdraw the arm through one of the doors and insert a new one, the men inside again replacing the breech-block. Each arm, whether air or water-cooled, has its own supply and discharge-pipe, and hence independent regulation and replacement. The weight of the central shaft and attachments is not borne on a step, but on roller-bearings.

The furnace is built with flat firebrick hearths, since accretions are more easily removed from the flat hard surface, which is better adapted for uniform rabbling. Fig. 6, the fire-box furnace, has an external fireplace, so that it can be used for a dead roast or for a chloridizing or sulphatizing roast. The passages or drop-holes. from hearth to hearth are left of ample area, so that the velocity of the upward-passing gases is low, and hence less dust is carried away by the air currents.

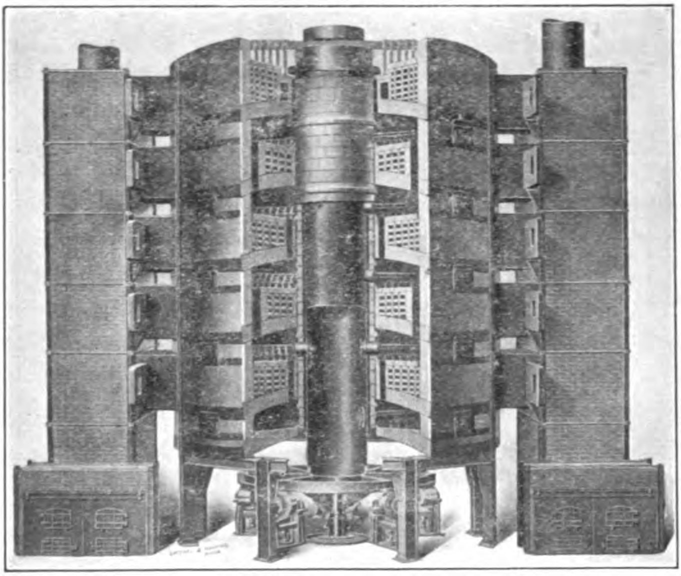


FIG. 6. FIVE-HEARTH WEDGE MECHANICAL FURNACE.

THE HERRESHOFF ROASTING FURNACE

(November 1, 1913.)

In the new smelting plants of the Arizona Copper Co., at Clifton, and the Calumet & Arizona Copper Co., at Douglas, Arizona, the roasting of concentrate preparatory to smelting in reverberatory furnaces is done in 6-hearth Herreshoff roasters. As the Herreshoff in its original form was a modification of the McDougall roaster to meet the conditions required in the roasting of pyrite to yield SO_2 for the manufacture of sulphuric acid, its use in copper smelting practice is of great interest. Eight 21½-ft. roasters have been built for the Arizona Copper Co. and 12 of the same size at the Calumet & Arizona plant. The most notable features of these are: the use of air-cooled rabbles; the return of the air thus heated to the lowest hearth, thereby increasing the thermal efficiency of the roaster and allowing the roasting, without the use of fuel, of concentrate lower in sulphur than is the usual practice; and the use of rabbles so designed that a single casting will operate in any position on the rabble arm, thus doing away with the necessity for keeping a variety of teeth in stock.

The accompanying illustration, Fig. 7, shows the construction

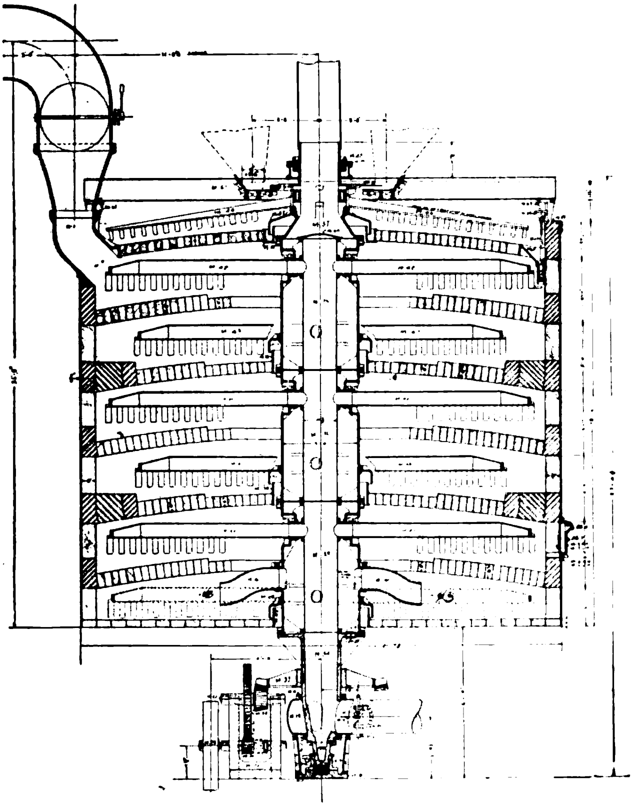


Fig. 7. The Herreshoff Furnace.

of the furnace. The outer shell is made of ⅝-in. steel plate three courses high, the outer diameter being 21 ft. 7¼ in. The horizontal seams are 3-in. lap, riveted with ¾-in. rivets, with 5-in. pitch. The vertical seams are butt joint, triple riveted, with butt-plates, 5/16 by 16½ in., inside and out. This shell is 19 ft. 5 in. high and is lined with 8-in. fire brick. There are six superimposed hearths, the top of the furnace being also used as a drying hearth. Each hearth is made of a single arch of 6 or 8-in. fire brick (as shown in illustration), the spring of the arch being 10½ in. The hearths for outside discharge have 54 openings. 18 by 14 in. each, around the periphery; those with inside discharge have an 8-ft. diameter circular opening at the centre. The central opening on the other hearths is merely large enough to permit rotation of the central shaft, which is 42 in. diameter. Each hearth is provided with six doors on the outside, except the topmost, which has but four. Each door is 24 by 16 in. and is fitted so as to make a tight joint.

The central shaft is made in three sections, each approximately 6 ft. long. These are fastened together with tongue-and-groove joints, and bolt heads and nuts are protected by pockets which can be filled with refractory material. The rabble arms are similarly connected with the central shaft. Within the shaft is an 18-in. steel tube, bolted to nozzle outlets at each arm. Air under pressure is forced through this tube, passes to the outer end of the rabble arm, and back to the central shaft. The rabble arms are connected in parallel so that each arm receives its due quota of cool air. The central web which causes the air to pass to the end of the arm before returning also greatly adds to the strength of the arm. The heated air is taken off at the top and brought back through channels in the outside walls to the lower hearth. The heat which it has absorbed is thus transferred to the lowest hearth and by decreasing the cooling effect of the air admitted for combustion allows the roasting without extraneous heat of material much lower in sulphur than would otherwise be the case. On the other hand, material with a given sulphur content will yield more heat and will therefore roast faster than without heat conservation. The rabble teeth are so cast that the lug at the top which fits against the arm is at different distances from the centre line on the right and left side. When a uniform series of teeth are placed upon the rabble arm this causes each tooth to assume a different angle, according to its distance from the centre, and the increased work which the inner teeth have to do is thus automatically compensated. The outermost rabble is kept in place by a single drop-pin. This is of special design to provide for the feed from the hearth above, as shown.

The ore is fed upon the upper drying hearth from hoppers. Just beneath each hopper is a steel plate and the ore is caused to feed by a rotating finger which passes between the hopper and the plate. The ore thus drawn out falls upon the top of the furnace, and is worked to the periphery by a set of rabbles. These are simply dropped into place on the rabble arm and can be removed or put in place without stopping the furnace, thus permitting easy changes in the rate of feed. This is also controlled by cast iron sleeves at the end of the feed hoppers. By raising or lowering these, the rate of feed may be altered as desired. The dry ore passes down to the first hearth through a series of hoppers at the outer edge which are provided with adjustable plugs to prevent the escape of gases. The space between the top of the furnace and the central shaft is closed with a sand seal, while on the lower hearths the calcine serves as a seal. The gases escape from the furnace through two exits, having a combined area of 18.8 sq. ft. The flues which convey the gases to the stack are so arranged that the flue-dust is automatically returned to the lower hearth, thus eliminating the labor cost of handling flue dust. The rabble arms are driven at one revolution per minute by a 72-in. bevel gear. The pulleys and gearing are so mounted on an A-frame that they can be removed for repairs without disturbing the rest of the furnace. The equipment for the roasters at these two plants was furnished by the Pacific Foundry Co., and they were erected under the supervision of C. H. Repath and A. G. MacGregor, who designed the plants.

The largest installation of Herreshoff roasters now at work is at the Edgewater, New Jersey, plant of the General Chemical Co., where eighteen 16-ft. furnaces are at work roasting Spanish pyrite for the production of SO_2 for acid manufacture. The operating conditions are scarcely comparable with the roasting of copper concentrate, since in this case it is necessary to secure a gas high in SO_2 and a calcine low in sulphur rather than the larger tonnage desired in copper-smelting practice. Thus a 16-ft. roaster handles only 5 to 6 tons per day, reducing the sulphur content from 46% to ½%. The ore is crushed to pass 4 mesh before roasting, and the calcine is briquetted for use in iron smelting. Operating under these conditions, it has been found that only one arm has broken in the past six months with 18 roasters in use. A crew of six men per shift is sufficient for this number of furnaces, including the wheeling of the pyrite from the storage bins to the feed hoppers of the furnaces. Broken rabbles can be replaced in a few minutes, but in the case of a broken rabble-arm a delay of 4 to 7 days is the result, since it is necessary to allow the furnace to cool slowly to the temperature at which a man can enter to loosen the bolts. Some experiments have been made in the developing of a detachable arm, but it has been found that delays from the breaking of rabble-arms are so infrequent that the advantage thus gained is practically neutralized by the increase in construction cost and the difficulty of obtaining an air-tight joint. The total cost of upkeep on these roasters as used at Edgewater, amounts to 10c. per ton roasted. This amount may be expected to vary with the quality of gas, the temperature, tonnage, time of year, and other factors.

INCREASING ROASTER EFFICIENCY AT GREAT FALLS

By Frank R. Corwin and Selden S. Rodgers

I. Introduction.

*Since McDougall roasters were first constructed at the Great Falls smelter of the Anaconda Copper Mining Co., their capacity has been more than doubled. During the first decade the tonnage treated remained practically unchanged. Experimental work carried out in 1906 showed that increasing the draft would increase the capacity. In 1909 the roasters were connected to the new flue system, and the stronger draft increased the capacity. The change in draft conditions made it necessary to operate differently, and systematic experimental work was started to adapt the furnaces to the new conditions. This extended over several years, and resulted in largely increasing the capacity, decreasing the percentage of flue dust, and in bringing about other changes and improvements tending to raise the efficiency of the McDougall roasters, with a corresponding reduction in operating expenses, without building any new roasters or enlarging the old ones.

The first McDougall roaster at Great Falls was started on June 15, 1898. It had eight hearths, was lined with fire brick, and had an air-cooled central shaft, but no provision was made for cooling the rabble arms, which became distorted. To draw the heated gases away from the hearths a pipe was connected to alternate hearths and to the flue. Dampers were provided so that the gases could be drawn from any of the hearths.

This first furnace not proving altogether a success, it was reconstructed as the Evans-Klepetko type, having 6 hearths and water-cooled central shaft and rabble arms. Except for a few minor changes, such as substituting solid arms for the water-cooled arms on the first hearth, the removal of the branch pipes inside of the rabble arms to distribute the cooling water, and changes in design of the rakes, rake frames, and rabble arms, they still remain essentially the same as when built in 1899.

The composition of the concentrate when the roasters were first started was approximately as follows: Cu, 13.3%; SiO_2, 24.2%; Fe, 25.6%; Al_2O_3, 4.6%; Zn, 0.9%; S, 30.0%.

When first built, the McDougall furnaces had a rated capacity of 40 tons of wet concentrate containing about 35% sulphur, roasted down to 7% sulphur, per furnace day. In practice, however, the

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*Abstract of a paper presented before the Amer. Inst. Min. Eng., August 1913.

capacity of the furnaces averaged 35 tons of cupriferous material per furnace day. The first experimental work was carried out by J. H. Klepinger to determine whether an increase in the draft would increase the capacity. A No. 8 Sturtevant exhauster was connected to the top of the furnace, as shown in the accompanying sketch, (Fig. 8), the gas being discharged into the main McDougall cross flue. When the exhauster was in use the slides in the regular draft necks were closed. When for any reason the exhauster was stopped these slides were drawn out, allowing the furnace to operate under natural draft. Several tests were made on No. 2 furnace, first working under natural draft, and later with the exhauster in operation. During the tests a careful record was kept of the weight of material charged to the furnace, samples were taken of the concentrate charged and calcine produced, and observations taken

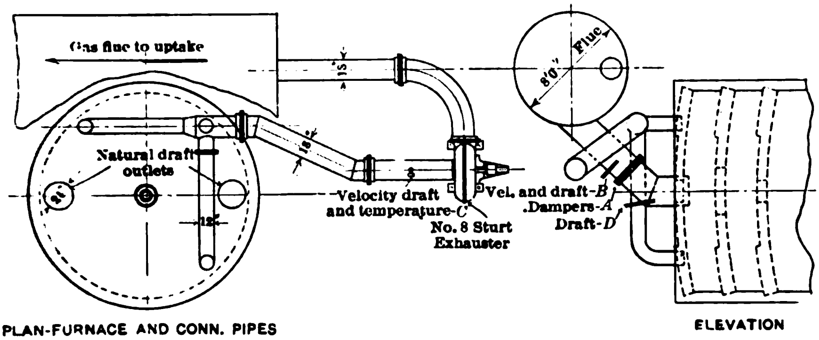


FIG. 8.—CONNECTIONS AT NO. 2 MCDOUGALL FOR INDUCED DRAFT.

Summary of Data from Tests on No. 2 McDougall Furnace *

	Natural Draft.		Draft Induced by No. 8 Sturtevant Exhauster.		
	Test No. 1.	Test No. 2.	Test No. 4.	Test No. 5.	Test No. 6.
Test Period, 1906.	June 16-19.	June 20-22.	July 6-12.	July 13-28.	July 29-August 2.
Concentrate per 24 hr., wet weight, tons	30.382	42.885	39.803	43.014	45.858
Moisture in concentrate, per cent	7.2	7.4	6.9	6.9	7.3
Concentrate per 24 hr., dry weight, tons	28.19	39.71	37.06	40.05	42.51
Sulphur in concentrate charged, per cent	35.4	35.8	37.0	37.4	38.1
Sulphur in calcine produced, per cent	5.3	11.6	8.8	9.6	11.5
Speed of exhauster, r. p. m.	.....	.....	1,450	1,454	1,503
Average temperature, exit gases, °F.	567	557	518	512	523
Average volume, cu. ft. per min.	9,380	8,910	11,570	11,160	10,800

*Test No. 3 has been discarded.

to determine the draft, and the temperature and volume of the discharged gases. A brief summary of part of the principal data obtained, taken from Mr. Klepinger's report, is tabulated herewith.

Increasing the tonnage of concentrate treated from 30.38 to 42.88 tons on natural draft increased the percentage of sulphur in the calcine from 5.3 to 11.6%, whereas with induced draft (in Test No. 5) 43.01 tons of concentrate was treated and calcine assaying 9.6% sulphur was obtained. An idea of the amount by which the capacity was increased was obtained by comparing the work of McDougall furnace No. 2 in Test No. 5 with the average work of all the others during the same period. It should be noted that furnace No. 2 was treating concentrate only, whereas the feed in the other furnaces, as was the regular practice at that time, was 92.6% concentrate (wet weight), 4.8% first-class ore screenings, 1.0% of tailing, and 1.6% dried slime. There would, therefore, probably be a slight advantage in favor of the test furnace.

Average.	No. 2 Furnace.	Average of all Other Furnaces.
	(Induced Draft.)	(Natural Draft.)
Material treated, wet weight, tons per 24 hr.	43	33.8
Material treated, dry weight, tons per 24 hr.	40	31.5
Copper in calcine, per cent	10.5	9.9
Sulphur in concentrate, per cent	37.4	37.9
Sulphur in calcine, per cent	9.6	9.0

According to this above comparison the induced draft caused a 27% increase in capacity and a 22% increase in amount of gas discharged.

In June, 1909, the new stack and flue system having been finished, the McDougall furnaces were started under the increased draft. The draft in inches of water at the McDougall dust chamber was 0.255 in. with the old flue system and 0.9 in. with the new. At the flue necks from McDougall furnaces to cross flue, it was 0.139 in. with the old and 0.93 in. with the new. The first effect of the stronger draft was to put out all the furnaces. The amount of heat generated by the oxidation of the sulphur remained practically unchanged; whereas, more air was drawn through the furnaces, more heat lost in the escaping gases, and consequently the furnaces became cold and went out. As the foremen and furnacemen became familiar with operating the furnaces under the new conditions, the feed was gradually increased and more use made of the available draft. The increased capacity predicted by Mr. Klepinger's experimental work was more than realized, as is shown by the following figures, taken from the monthly smelter reports:

Data on McDougall Department from Monthly Smelter Reports

Month.	Tons Treated per Furnace Day.						Total Tons per Square Foot of Hearth Area.	S in Calcine, Per Cent.
	Concentrate.	Screenings.	Dried Slime.	Flue Dust.	Limestone.	Total.		
August, 1908....	28.7	...	.....	...	...	28.7	0.032	9.1
September, 1908..	43.4	...	.....	0.1	0.1	43.6	0.046	9.1
October, 1908....	41.4	...	0.1	0.9	...	42.4	0.045	8.9
November, 1908..	36.2	...	.....	0.3	...	36.5	0.039	7.7
December, 1908..	37.8	...	(Copper Precip. 0.3)	0.1	...	38.2	0.041	8.1
January, 1909...	33.1	...	.....	...	...	33.1	0.036	8.8
February, 1909...	35.4	...	.....	...	...	35.4	0.038	8.2
March, 1909.....	35.1	...	0.1	...	...	35.2	0.038	8.4
April, 1909......	34.3	...	0.2	...	0.01	34.5	0.037	9.2
May, 1909.......	3.33	...	.....	...	...	33.3	0.036	9.1
Average	35.9	...	0.2	0.1	...	36.1	0.039	8.7
July, 1909.......	49.7	...	.....	...	0.5	50.2	0.053	9.7
August, 1909....	40.8	...	0.7	...	0.4	41.9	0.046	8.6
September, 1909..	41.6	...	2.1	0.1	...	43.8	0.047	8.3
October, 1909....	40.2	...	2.4	...	0.3	42.9	0.046	8.2
November, 1909..	37.3	...	2.0	...	0.2	39.5	0.042	7.7
December, 1909..	Smelt	er do	wn on	accou	nt of	railro	ad strike	
January, 1910....	49.1	...	.....	...	0.5	49.6	0.053	8.2
February, 1910...	51.9	...	.....	...	1.7	53.6	0.057	8.7
March, 1910.....	40.5	...	.....	...	2.3	42.8	0.046	7.6
April, 1910......	44.8	1.2	.....	...	1.2	47.2	0.050	7.9
Average	44.0	0.1	0.8	...	0.8	45.7	0.049	8.3

The new smelter stack and flue system had thus brought about an increase in capacity from 36.1 tons to 45.7 tons per furnace day. Treating more than 50 tons per furnace day, however, increased the sulphur in the calcine an undesirable amount and caused the furnaces to crust up heavily. It was hoped that by a systematic study of the conditions these difficulties could be overcome. A test was carried out under the direction of Arthur Crowfoot, May 3 to 7, 1910. Four furnaces were selected for the test work, No. 1 and 3 operating as a unit and No. 18 and 19 operating as a unit. The essential differences between the two units were as follows:

1. The central drop hole connecting the third hearth with the fourth hearth in furnaces 1 and 3 measured approximately 20 in. from the shaft to the edge of the drop-hole, while in furnaces 18 and 19 this opening measured approximately 14 inches.
2. The rabble arms in furnaces 1 and 3 made one revolution in 50 seconds, while the rabble arms in furnaces 18 and 19 made one revolution in 37 seconds.

A fractional part of the total data obtained from the test is tabulated below.

Tonnage Test on McDougall Furnaces

(A). Average results obtained from No. 1 *and* 3 *Furnace Unit*

Date of Test, 1910.	May 3.	May 4.	May 5.	May 6.	May 7.
Nominal tons wet conct. to be fed per 24 hr.	55	60	65	70	**75**
Total tons wet conct. fed per 24 hr.	52.72	59.05	64.54	69.17	**59.18**
Actual rate of feed, tons per 24 hr.	58.80	66.13	71.85	80.18	**94.54**
Hours furnace in operation	21.51	21.30	21.71	20.70	**15.79**
Hours furnace down	2.49	2.7	2.29	3.30	**8.21**
Per cent of 24 hr. furnace down	10.37	11.25	9.54	13.75	**34.20**
Draft reading on flue necks, inches water	1.28	1.26	1.23	1.3	**1.36**
Concentrate Fed.					
Moisture, per cent	7.5	8.0	6.7	7.6	**7.1**
Cu, per cent	8.80	7.55	8.20	8.4	**8.45**
S, per cent	36.0	32.8	34.6	34.4	**35.8**
Calcine Produced.					
Cu, per cent	10.55	9.75	10.35	10.1	**10.25**
S, per cent	8.2	9.2	11.3	12.2	**12.6**

(B). Average results obtained from No. 18 *and* 19 *Furnace Unit*

Nominal tons wet conct. to be fed per 24 hr.	55	60	65	70	75
Total tons wet conct. fed per 24 hr.	52.45	58.31	60.91	59.41	63.01
Actual rate of feed, tons per 24 hr.	64.20	72.72	71.40	74.16	79.84
Hours furnace in operation	19.58	19.24	20.49	19.25	18.92
Hours furnace down	4.42	4.76	3.51	4.75	5.08
Per cent of 24 hr. furnace down	18.42	19.83	14.60	19.79	21.17
Draft reading on flue necks, inches water	1.16	1.09	1.09	1.12	1.16
Concentrate Fed.					
Moisture, per cent	8.4	6.6	6.6	7.1	7.1
Cu, per cent	6.40	8.10	8.25	8.65	8.0
S, per cent	32.8	33.4	34.8	35.0	35.4
Calcine Produced.					
Cu, per cent	10.35	9.65	9.85	9.95	10.1
S, per cent	8.4	7.8	8.8	9.6	10.4

The most interesting fact clearly brought out by this test was that increasing the feed to the furnaces did not increase the percentage of sulphur in the calcine as much as expected. The test showed that on increasing the feed of concentrate more sulphur was burned per minute and per square foot of hearth area, and the furnaces ran much hotter. In fact, the main difficulties experienced were on account of too high a temperature being developed. The heat generated caused the drop-holes to become built up, and heavy accretions to form on the roofs.

The 18–19 furnace unit with the 14-in. third hearth central drop lost on an average about 7% more time than the 1–3 furnace unit with the larger third hearth central drop-hole. The reason for this was that the smaller hole, on account of the higher velocity of the gases and the concentration of heat at this point, caused heavy accretions to build up on the roof. Practically all of the additional time lost by the 18–19 furnace unit was due to barring accretions from the roof of the third hearth. The observations showed that in treating high tonnages, increased speed was advantageous, for the reason that it kept shallower beds on the furnace hearths, and there was less banking up of material in front of the rabble arms.

The data obtained from this test outlined the way for future work. It was necessary to have the furnaces run cooler, which could be accomplished either by drawing more air through the furnaces, and consequently discharging more heat in the waste gases, or else by decreasing the sulphur content of the feed. Also, for treating high tonnages, the test indicated that the larger size of drop-hole and increased rate of speed were in all probability desirable.

John A. Church, Jr., was then detailed to make a special investigation and acting on the information obtained from Mr. Crowfoot's test, furnace No. 19 was operated on a high tonnage, but the fuel value of the feed was lowered by mixing screenings and limestone with the concentrate. The feed treated was made up of 20 tons of concentrate and 6 tons of foreign material per shift. By 'foreign material' is meant any constituent of the feed lower in fuel value than the concentrate, such as slime or first-class ore screenings. Shifts are all 8 hr. The furnace had a small third hearth drop-hole, measuring 14 in. from the shaft to the edge of the drop-hole, and ran at high speed, the rabble arms making one revolution in 38 sec. The draft was regulated by dampers in the flue necks, and by closing all doors except one on the sixth hearth. The furnace handled the 20 tons of concentrate per shift satisfactorily and the foreign material mixed with the concentrate kept the temperature of the furnace from becoming too high. It was found, however, that the work of keeping the roof clean above the third hearth drop-hole was excessive for normal running. On May 17, furnace No. 18 was equipped with an enlarged third hearth drop-hole and started for parallel comparison with furnace No. 19. Both furnaces had two of the six drop-holes covered on the second hearth, and two closed on the fourth hearth. Two doors were left open on the sixth hearth.

A few days of operation showed that No. 18 ran considerably cooler and crusted up less than No. 19. Also No. 18 would not take as large a tonnage of foreign material without becoming too cold. Although by means of feeding foreign material along with the concentrate the temperature of No. 19 furnace as a whole could be maintained at about the right point, yet on account of the concentration of heat at the third hearth drop-hole and the high velocity of the gases at this point, the work of keeping the roof above the third hearth clean still continued to be excessive.

While enlarging the third hearth drop-hole of a furnace greatly reduces the work of keeping the furnace clean, it also has the undesirable effect of making the calcine run higher in sulphur. A year or two later a means was found of retaining the smaller drop-hole, and yet avoiding excessive incrusting on the third hearth roof. Two extra drop-holes were cut in the third hearth close to the edge of the centre drop-hole. The inner rakes on the third hearth rabble arms were set so as to push the material away from the centre, and therefore all material passing from the third to the fourth hearths was forced to pass through the two extra drop-holes. This meant dropping a large bulk of material four times in every revolution of the central shaft, instead of having the material showering down continuously over the edge of the centre drop-hole. It reduced the amount of incrusting on the third hearth roof and localized what incrustations formed over the extra drop-holes, making the work much easier for the furnacemen. The 14-in. centre drop-hole combined with the two extra drop-holes is the present practice for the third hearth. It is mentioned here because, although during the work in the spring of 1910 to increase the tonnage per furnace it was found that the 20-in. was an improvement over the 14-in. third hearth drop-hole, later work as noted above, has made it desirable to again change back to the smaller drop-hole.

While No. 19 was down for repairs and to have the drop-hole enlarged, observations were kept up on No. 18. More doors were gradually opened on the sixth hearth until finally the furnace was run with all four of the sixth hearth doors open. (The doors on the upper hearths are always kept closed except when they must be opened for cleaning, or pulling lumps out of the furnace.) The opening of each successive door brought about an improvement in the condition of the furnace, showing that previously not enough oxygen had been supplied. Similarly, the dampers in the flue necks were opened wider, improving the work of the furnace, until in time the full available draft of the lateral flue was being used, about 1.2 in. of water.

The chief difficulties experienced with No. 18 were due to the sensitiveness of the furnace. Changes in the quality of the feed, irregular feeding of foreign material, and other adverse conditions would often cool the furnace too far, necessitating the use of the oil machine to restore a normal working temperature. The first improvement in maintaining the furnace at a steady temperature was made by closing one additional second hearth drop-hole. This made three second hearth drop-holes closed in all. At first it was thought that the increased amount of heat obtained by decreasing the second hearth drop-hole area was due to increasing the frictional resistance to the passage of gas through the furnace, allowing less air to enter the furnace and less heat to be carried away in the waste gases. The improvement which had been made in the work of the furnace, however, by opening more doors on the sixth hearth and opening wide the dampers in the flue necks, showed that this was not the case. The drop-hole area on the second hearth, even with three drop-holes covered, was large enough not to throttle to any appreciable extent the gases traveling through the furnace. The effect, therefore, of closing one extra drop-hole was to concentrate the ascending hot gases, which had previously been passing through four drop-holes, into three, thus intensifying the heat at these three points. Near the outer edge of the second hearth is where the concentrate should first begin to burn, and a certain temperature is required for its ignition. As No. 18 had a tendency to run cold, closing an extra second hearth drop-hole, therefore, benefited the furnace by helping to secure proper ignition for the incoming concentrate. Decreasing the second hearth drop-hole area probably also helped by lessening the amount of heat lost by radiation from the third hearth.

Another improvement made in maintaining furnace No. 18 at a good working temperature was increasing the amount of concentrate fed. The feed of concentrate was raised first to 22 and later to 24 tons per shift. With the higher tonnage the furnace developed more heat and it was found advisable to remove one of the second hearth drop-hole covers, again returning to the condition of four open and two closed. On June 2, No. 19 was again started, the third hearth drop-hole having been enlarged to 20 in. The furnace was operated in every respect similar to No. 18. It was found that for some unexplained reason No. 19 normally ran hotter than No. 18, and as the tonnage of concentrate fed was increased to 24 tons per shift it was found that No. 19 ran best with all of the second hearth drop-holes uncovered. No. 18 throughout the testing work was found to give the best results with two of these drop-holes covered. It will be seen, therefore, that the proper second hearth drop-hole area is a variable quantity, depending on the temperature conditions of a furnace, quantity of feed treated, and so on.

On June 17 it was decided to run furnace No. 22 against No. 18 and 19 to determine the relative advantage of high and low speed, all other conditions being the same. Accordingly a course of brick was knocked out around the third hearth drop-hole of No. 22, and instructions were given to gradually raise the tonnage of concentrate to 24 tons per shift. No. 18 and 19 continued to revolve once in 38 sec. and No. 22 was left unchanged at one revolution in 53 sec. The operation of all three furnaces during about a 10-day period showed that the high speed furnaces gave less trouble from material banking up in front of the arms and in general produced calcine assaying lower in sulphur. A discussion of the relative advantages of high-speed, from the report of Mr. Church, is presented herewith.

For high tonnages the increased speed gave better results, in that the depth of bed on the hearths was kept lower and there was less tendency for the feed to bank up in front of the arms. Every revolution of the central shaft moves a constant amount of material at any one point on the hearth over a radial distance equal to the width of two furrows. If the speed is increased and the depth of bed remains constant, every revolution moves the same amount of material the same distance as before, but in a given time the greater number of revolutions will move a greater tonnage. If, however, the speed is increased and the tonnage remains constant, every revolution will move a less quantity of material than before, that is, the depth of bed will be reduced.

Increasing the speed means not only a decrease in the depth of bed, but also means that the bed is stirred more frequently. Every passage of a rabble arm, by stirring the bed, presents a freshly exposed surface to the action of the oxygen-bearing gases. The best speed is that which exposes a fresh surface just as rapid oxidation of the sulphur is beginning to die on the old surface. The time required for this rapid oxidation of the surface to cease varies with the character of the material, the amount of oxygen in the gases, the temperature of the bed, temperature of the gases, and many other factors. In general, however, it may be said that the period during which the surface of the bed undergoes rapid oxidation is shorter for the fifth and sixth hearths than for the third and fourth hearths. Up to the point at which a surface is buried while still undergoing rapid oxidation of its sulphur, increasing the speed tends to increase the rate of elimination of sulphur.

On the third floor, the speed of one revolution in 38 sec. undoubtedly made the stirring too rapid. On the fourth floor observations indicated that the increased speed was probably about right, while for the fifth and sixth hearths a still higher speed might have been used to advantage.

The effect on the roaster department, as the improved methods of operation developed from the experimental work were gradually applied to the regular daily work of the furnaces, is shown by the figures tabulated below. These figures are taken from the monthly smelter reports.

Average Tonnage Treated per Furnace Day

Month.		Concentrate.	First-Class Ore Screenings.	Miscellaneous.	Total Cupreous Material.	Limestone.	Total.	Total Tons per Sq. Ft. of Hearth Area.	Per Cent S in Calcines.
April,	1910..	44.8	1.2	....	46.0	1.2	47.2	0.050	7.9
May,	1910..	46.9	3.9	0.1	50.9	2.6	53.5	0.057	7.6
June,	1910..	47.2	3.6	0.6	51.4	3.6	55.0	0.059	8.0
July,	1910..	55.2	6.7	0.1	62.0	4.6	66.6	0.071	9.0
August,	1910..	52.4	4.0	....	56.4	3.9	60.3	0.064	8.2
September,	1910..	67.5	3.9	....	71.4	6.5	77.9	6.083	9.8
October,	1910..	68.1	5.3	....	73.4	8.2	81.6	0.088	8.6
November,	1910..	73.8	4.6	....	78.4	9.8	88.2	0.094	8.0
December,	1910..	73.1	4.2	....	77.3	11.0	88.3	0.094	8.4
January,	1911...	67.6	2.4	....	70.0	11.9	81.9	0.088	8.4
February,	1911..	70.5	4.0	....	74.5	14.3	88.8	0.095	10.6
March,	1911..	75.1	0.3	....	75.4	10.3	85.7	0.092	11.4

Before the experimental work was started, the reasons for not treating more than about 14 tons of concentrate per furnace per shift were that increasing the tonnage raised the percentage of sulphur in the calcine, caused the furnaces to become too hot, especially locally on the roof above the 14-in. third hearth drop-hole, and made the furnaces crust up heavily, necessitating an excessive amount of cleaning. The problem of treating an increased tonnage therefore involved the following factors:

(*a*). Supplying sufficient oxygen and stirring the hearths frequently enough to burn the increased amount of sulphur.

(*b*). Regulating the heat from the increased amount of sulphur burned per square foot of hearth area so that the furnaces would not become too hot.

(*c*). Regulating the drop-hole area so as to avoid too great a concentration of heat and too high a velocity of the gases through the drop-holes, thus preventing the furnaces from building up too heavy incrustations, particularly on the roof above the third hearth centre drop-hole.

The specific changes made, through which the treating of higher tonnages became possible, were as follows:

(*a*). The furnaces were operated under the full available draft (about 1.1 in. of water), with no dampers in the flue necks and with four doors open on the sixth hearth, so as to draw enough air through the furnace to supply the necessary amount of oxygen.

(*b*). The speed of revolution of the central shafts was increased from one revolution in 53 sec. to one in 45 sec. on the first battery (four furnaces) and to one revolution in 38 sec. on all the other batteries (18) furnaces. This increase of speed had two effects:

1. It permitted an increase in tonnage without increase in the depth of bed, thus helping the furnaces mechanically.

2. It stirred the beds more frequently and aided in the mor-

rapid oxidation of sulphur by bringing the sulphur into contact with sufficient oxygen. In a given time, to obtain the same degree of calcination, a furnace treating a higher tonnage has to eliminate a larger quantity of sulphur.

(*c*). First-class ore screenings and limestone were mixed and fed with the increased tonnage of concentrate to reduce and regulate the temperature of the furnaces.

(*d*). The area of the third hearth centre drop-hole was increased. This reduced the amount of heat which had formerly been concentrated at this point and decreased the velocity of gases through this opening, resulting in permitting the use of a higher tonnage without an excessive amount of crusting on the third hearth roof.

(*e*). The second hearth drop-hole area was regulated so as to obtain sufficient heat at the edge of the second hearth to ignite the incoming concentrate.

A short time later it became desirable to increase the roaster output to supply the feed for an additional reverberatory furnace. The furnaces were already handling practically the entire output of fine concentrate, and it was necessary to treat a larger amount of first-class ore screenings, of which a sufficient supply could be obtained.

In the previous work, the ore screenings had been used as a cooling agent to keep down the temperature of the furnaces. The problem now, instead of running the McDougalls so as to treat the best possible tonnage of concentrate, using screenings only as a cooling agent, was to treat the largest possible tonnage of screenings by utilizing the available heat in the most efficient way. The means used were as follows:

1. Operation of the furnaces on a high tonnage of concentrate.
2. Proper regulation of the second hearth drop-hole area.
3. Withdrawal of limestone from the feed, its place being taken by screenings.

1. Screenings being low in fuel value and having a cooling effect, as previously stated, it was necessary to keep the furnace as hot as possible. Increasing the feed of concentrate meant burning more pounds of sulphur per square foot of hearth area per minute and developing more heat. The concentrate feed to each individual furnace was therefore maintained as high as possible without unduly raising the percentage of sulphur in the calcine.

2. In order to secure proper ignition for the feed, the second hearth drop-hole area was regulated so as to obtain as high a temperature at the outer edge of the second hearth as possible without too much throttling of the draft.

3. Limestone having no fuel value, it was found that its place could be taken by a greater quantity of screenings, the screenings carrying an average of from 18 to 20% sulphur. The construction of the McDougall furnaces was therefore changed so that on two furnaces in each battery limestone could be fed directly to the outer edge of the sixth hearth. All the limestone treated was fed to the sixth hearth of these furnaces.

In addition to the possibility of treating a larger tonnage of screenings, there were a number of other advantages in favor of this arrangement. One advantage was that the amount of limestone mixed in with the calcine was not limited by the temperature conditions of the furnaces, for feeding limestone directly to the edge of the sixth hearth had very little cooling effect. Another advantage was that its removal from the main body of the furnace lessened the danger of breaking rakes by the wedging of large fragments under the blades. Furthermore, there had been no advantage in sending the limestone through the furnaces. As the temperature in the furnaces was not great enough to burn limestone to lime, it underwent no chemical changes. By sending it directly to the outer edge of the sixth hearth it was warmed as much as if it had passed entirely through the furnaces and was nearly as well mixed with the calcine.

A test to determine the maximum percentage of screenings which could be treated was started in December 1910. The results are of unusual interest as showing the wide range in sulphur content of the material which can be successfully roasted in a McDougall furnace.

At the time of starting the test, about 8820 lb. of screenings could be treated per furnace per shift, mixed with about 50,000 lb. of concentrate. The ratio was therefore about 25 tons of concentrate (wet weight) to 4.41 tons of screenings per shift, or 85% of concentrate to 15% of screenings. The screenings were fed intermittently during a shift, and, as they always had a cooling effect, they were used only when the furnaces were running hot. This led the furnacemen to believe that the continuous treatment of screenings alone would be impossible, and in carrying out the test it was necessary to increase the percentage of screenings by slow degrees. The successful solution of the problem depended on keeping the furnace sufficiently hot. Furnace No. 16 was used for the test work, and whenever necessary the temperature of the furnace was raised by means of either or both compressed air or coal and oil.

The advantage of compressed air in keeping a McDougall furnace hot was discovered by noticing the effect produced by air blown into the furnace through an oil machine after the oil in the machine had been exhausted. It was observe dthat the jet of air on the fourth hearth caused more rapid oxidation of the sulphur, developing more heat and increasing the temperature of the furnace. The increase in rapidity of the oxidation was caused by the air impinging on the hearth, bringing the calcine into intimate contact with fresh oxygen.

In the test to determine the maximum percentage of ore screenings which could be used in the feed to a McDougall furnace, furnace No. 16 was first equipped for compressed air with three ¼-in. pipes inserted through the observation holes of the fourth hearth doors. It was found that air blown through these pipes aided greatly in increasing the temperature of the furnace, and many times when the furnace showed signs of becoming cold a recovery was effected by means of compressed air without the use of oil or coal. One disadvantage, however, was that there was an increased amount of incrusting on the rabble arms, due to sparks produced by the air blowing on the bed. A large amount of this incrusting was done away with, and better results obtained from the air, by replacing the ¼-in. pipes by ¾-in. pipes. This change made it possible to slightly decrease the velocity of the air entering the furnace, at the same time increasing the volume. Thus the amount of sparking was reduced and the hearth area affected by the air increased.

A further improvement was made in the method of introducing the compressed air into the furnace before the completion of the test. On the fourth hearth, three ¾-in. pipes each 4½ ft. long were inserted through holes drilled in the shell and brickwork of the furnace. These pipes were run toward the centre of the furnace along the fourth hearth roof, leaving just enough room for the rabble arms to pass beneath them. Just outside of the shell of the furnace the pipes were each connected to a rubber hose delivering compressed air from the converter air main. Inside of the furnace the pipes each had two series of ¼-in. holes drilled in them, extending from the brick wall of the furnace to the ends of the pipes. The holes were drilled 1½ in. apart, and the two series were each ½ in. apart, directing the air at nearly right angles down on the bed. These pipes caused the compressed air to affect a large area of the bed, and because the air was directed against the bed at nearly right angles, sparking was practically eliminated. Pipes were also used to introduce air to the fifth hearth, the arrangement being exactly the same as for the fourth. It should be remarked here that, although the equipment described above was found highly satisfactory during the short period of the test, it would not be suitable as a permanent installation, for the reason that the pipes are liable to become bent by the heat and interfere with the arms, and also because they make it difficult to clean the roofs of the hearths where they are placed.

The experiment was also tried of blowing air in on the third hearth, but the results were unsatisfactory. The temperature of the third hearth is too variable, and unless the air is blown against a hot bed it does not increase the rate of oxidation of the sulphur, having simply a cooling effect.

As the percentage of ore screenings in the feed to No. 16 was increased, the furnace could not be kept hot enough by the use of air only, and it became necessary to use a small amount of slack coal. This coal was fed through a different hopper from the screenings and concentrate, a fairly even feed being obtained by poking the coal through the hopper a little at a time and allowing it to mix with the screenings and concentrate on the first hearth. Whenever the furnace received a slight set-back, oil was also used to keep it hot, the oil being blown in on the fourth hearth in the usual manner.

On night shift of February 7, 1911, ore screenings alone were first treated in the furnace and from then until Feb. 12, the furnace was maintained on this feed. Coal and air were used continuously to keep the furnace hot, and oil when necessary. The ore screenings contain only 17.4% S as compared with the 33.4% S of the concentrate.

Record of McDougall Furnace No. 16 During Test on Treatment of Ore Screenings

Date, 1911.	Tons of Material Charged.			Per Cent of Feed.		Fuel Used.		Fuel per Ton of Feed.		S in Calcines.
	Concentrates.	Screenings.	Total.	Concentrates.	Screenings.	Coal.	Oil.	Coal.	Oil.	
						Pounds.	Gals.	Pounds.	Gals.	Per Cnt.
January 25...	65.16	17.75	82.91	78.59	21.41	.....	..	.....	...No	sample
January 26...	48.71	26.97	75.68	64.36	35.64	3,425	63	45.26	0.83	8.9
January 27...	45.68	31.40	77.08	59.26	40.74	3,500	74	45.41	0.96	12.8
January 28...	34.64	34.42	69.06	50.16	49.84	4.100	43	59.37	0.62	7.2
January 29...	30.51	36.11	66.62	45.80	54.20	3,050	35	45.78	0.52	5.6
January 30...	24.37	40.15	64.52	37.77	62.23	2,025	35	31.39	0.54	6.0
January 31...	24.39	46.64	71.03	34.34	65.66	3,400	41	47.87	0.58	5.7
February 1...	18.30	48.21	66.51	27.51	72.49	4,350	56	65.40	0.84	6.7
February 2...	14.21	66.95	81.16	17.51	82.49	10,605	17	130.67	0.21	6.6
February 3...										
February 4...	No screenings used from Feb. 3 to 6, inclusive.									
February 5...										
February 6...										
February 7...	7.80	56.32	64.12	12.16	87.84	9,255	24	144.34	0.37	7.3
February 8...	....	74.13	74.13	....	100	13,745	8	185.42	0.11	8.0
February 9...	....	74.35	74.35	....	100	11,430	46	153.73	0.62	6.2
February 10...	....	69.80	69.80	....	100	10,260	47	146.99	0.67	6.5
February 11...	....	65.80	65.80	....	100	8,335	24	126.67	0.36	5.6
February 12...	....	66.52	66.52	....	100	7,955	5	119.59	0.07	4.9

On the completion of the test to determine the maximum percentage of ore screenings, experimental work was again resumed on increasing the tonnage. McDougall furnace No. 16, which had just been used on the screenings test, was used also for the tonnage test. The object in view was to treat in this furnace the highest tonnage consistent with good calcination. As already stated, this furnace was equipped with pipes for blowing air on the beds of the fourth and fifth hearths. In the screenings test the air had been used to increase the rate of oxidation, in order to increase the temperature of the furnace. In the high tonnage test compressed air was used to increase the rate of oxidation, but not to make the furnace run hotter. With the high tonnage of concentrate, and therefore the large amount of sulphur burned per square foot of hearth area, the furnace ran so hot that it was always necessary to feed screenings along with the concentrate to cool the furnace and prevent the material on the third hearth from fusing. The object of increasing the rate of oxidation was not, therefore, to raise the temperature of the furnace, but to improve the degree of calcination. The speed of revolution of the central shaft was not changed, remaining at one revolution in 38 seconds.

The high-tonnage test was from Feb. 13 to Mar. 3, 1911. The maximum tonnage treated in any one 24-hr. period, 105.08 tons, was put through the furnace on Feb. 23. Below is given a record of the furnace for the three shifts of Feb. 23:

Record of McDougall Furnace No. 16, Feb. 23, 1913

Shift.	Tons Material Charged.		Total Feed—Tons	Per Cent of Feed.		Fuel.		Sulphur in Calcine. Per Cent.
	Concentrate.	Screenings.		Concentrate.	Screenings.	Coal. Pounds.	Oil. Gallons.	
Day	32.48	1.60	34.08	95.31	4.69	...	...	12.2
Afternoon	32.14	3.16	35.30	91.05	8.95	...	...	13.1
Night	32.47	3.23	35.70	90.95	9.05	...	...	11.6
Total 24 hr...	97.09	7.99	105.08	....	....	...	...	...

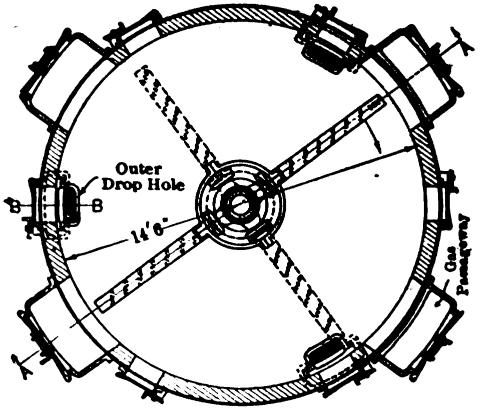
Although it was found that on some days around 100 tons per 24 hr. could be treated, producing calcine assaying about 11.5% sulphur, the test showed that this could not be done continuously on account of trouble with the driving machinery. Sometimes the feed would become so heavy on the first hearth that the machinery would refuse to carry it, and it would be necessary to rabble the material by hand upon the second hearth. Breakages of the driving machinery were of frequent occurrence, necessitating shut-downs for repairs. During these shut-downs the furnace would lose not only the tonnage which would have been treated had it been possible to continue the running of the furnace, but also, on account of the cooling off of the furnace during the shut-down, it would lose tonnage on account of the necessity of reducing the feed on starting up, until again reaching a normal working temperature.

The heavy load on the driving machinery was the result of increasing the tonnage treated without increasing the speed of revolution of the central shaft. As explained earlier, increased tonnage with constant speed means deeper beds on all the hearths. The test showed that with the central shaft making one revolution in 38 sec. 100 tons per 24 hr. could not be treated continuously in the furnace, on account of overloading the driving machinery. Increasing the speed of the central shaft was not tried and might or might not have caused too frequent stirring of the bed. Whether or not this tonnage could have been treated successfully by means of increased speed is still an open question.

The experimental work was undertaken to reduce the percentage of the flue dust made. The flue dust produced by a McDougall furnace is caused by the material dropping from hearth to hearth through the rising current of gases. With a strong draft small particles of partly roasted concentrate are picked up by the gases and carried along into the flue. Similarly, on the lower floors the incrustations around drop-holes are the result of the material falling through the rising gases. Small particles of calcine are picked up by the draft, and as during the fall through the gases they undergo very rapid oxidation and become partly fused, when they are thrown up against the roof they stick there, forming accretions. The experimental work to reduce the amount of flue dust was therefore based on eliminating the fall of material through the upward current of gases.

The first tests were made with the method invented and patented by Repath and Marcy. This method provides drop-holes for the passage of the material from hearth to hearth, which are sealed by the material itself against the gases, and also provides separate passageways from hearth to hearth for the gases. A furnace fulfilling these requirements was designed and furnace No. 17 was remodeled to conform with the design. This furnace will be referred to as the Repath-Marcy furnace. The construction of the furnace is shown in the accompanying sketch, Fig. 9.

The Repath-Marcy furnace was started March 15, 1911. At first there were a number of mechanical difficulties encountered which interfered with the regular discharge of the material from hearth



PLAN OF HEARTHS NOS 2 AND 4

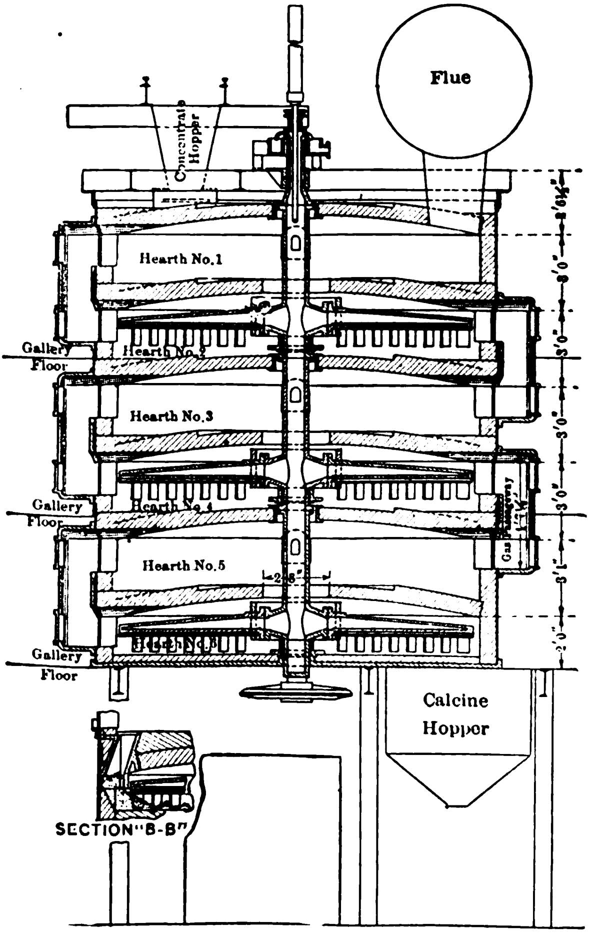


FIG. 9. REPATH AND MARCY METHOD FOR DUST PREVENTION.

to hearth. At times the drop-holes would allow the material to run through too fast and would become open for the passage of gases. At other times they would not allow the material to pass through fast enough and would become choked. Another difficulty met with was the excessive loss of heat by radiation from the gas passageways. The furnace was therefore shut down after a few days running, the defects in the drop-hole construction were remedied, and the gas passageways lined with brick. On starting up again after making these changes, with the furnace running satisfactorily mechanically and as nearly as possible in accordance with the ideas of the designers, it was found impossible to operate the furnace without the use of coal and oil. Different conditions of tonnage and draft were tried, but the result was always the same: The furnace would not run without fuel. Even when the fourth hearth was kept hot with coal and oil the calcine produced contained 18 to 20% sulphur. The test showed, therefore, that the furnace as first constructed was not a success.

From the above it will be seen that the furnace was unsatisfactory on account of its reduced power to eliminate sulphur. Not enough sulphur was burned to either keep the furnace hot or produce desirable calcine. The reason for the low elimination of sulphur was that one of the principal means of oxidizing the sulphur in the Evans-Kleptko type of McDougall furnace—namely, the showering down of material through the rising current of hot oxygen-bearing gases—had been removed without anything else being substituted to take its place. In the regular type of McDougall furnace a large amount of sulphur is burned and a considerable proportion of the total heat is generated during the fall of material from hearth to hearth. During the fall, each particle is exposed on all sides to the action of oxygen and oxidation goes on much more rapidly than when the particle is lying on the surface of the hearth. Eliminating this fall therefore reduces the amount of flue dust made by the furnace at the expense of oxidizing power.

To make sure, however, that the reduced amount of oxidization was not caused by an insufficient amount of air being drawn through the furnace, a test was carried out under the direction of Mr. Crowfoot to determine the relative amount of gas being discharged by the Repath-Marcy furnace in comparison with two of the regular McDougall furnaces, No. 18 and 1. The test showed that there was about 87% as much gas by weight being discharged from the Repath-Marcy furnace as from furnace No. 18, and about 81% as much as from furnace No. 1. The lesser amount of gas was probably caused by a greater frictional resistance to the passage of air through the furnace in comparison with the regular McDougall type. The gas passageways, after being lined with brick, had a smaller area than the drop-holes in the ordinary McDougall furnace. Also, the path of the gases through the Repath-Marcy furnace was longer. Inasmuch, however, as the Repath-Marcy furnace was treating less than two-thirds as much material as the other furnaces, it was evident that sufficient oxygen was being drawn through the furnace for the tonnage treated, and that the decrease in power to oxidize sulphur was caused by eliminating the fall of material from hearth to hearth through the rising current of hot gases.

Observations on the McDougall furnaces had tended to show that most of the dust carried into the flue was produced in the fall through the centre drop-hole from the first hearth to the second, and that a few of the sparks formed in the fall through the side drop-holes from the second hearth to the third were also carried into the flue. The test work having indicated that, as then constructed, a certain amount of showering down of material through the rising gases was necessary in order to obtain a good elimination of sulphur, it was decided to modify the design of the Repath-Marcy furnace. The furnace was accordingly shut down and the following changes made: Between the fourth and fifth, and the fifth and sixth hearths the sealed drop-holes and separate gas passageways were eliminated, and the construction was changed back to that of the regular McDougall furnace, allowing the material traveling from the fourth to the fifth and the fifth to the sixth hearths to fall through the rising current of gases as before. Also, pipes were installed to blow compressed air on the fourth and fifth hearths, the equipment being exactly the same as was used in the test to roast screenings alone.

After the above changes had been made there was no trouble experienced in running the furnace on a feed not exceeding 20 tons per shift, although to maintain the furnace at a working temperature it was necessary to use compressed air continuously. Whenever the air was shut off for a while a cooling action would soon be observed on the third hearth, and in 2½ to 3 hr. the calcine would run high in sulphur. Also, it was found impossible to increase the feed over 20 tons per shift without cooling the furnace and producing high sulphur calcine. The furnace was operated in this way for about a month (from May 16 to June 15, 1911), treating on an average 18 tons of concentrate, wet weight, per shift and producing calcine averaging 7.8% sulphur.

During the regular operation of this furnace it was desired to compare it with the Evans-Klepetko type of McDougall furnace, to determine whether or not any improvement had been made in reducing the amount of flue dust produced. The test figures showed that the modified Repath-Marcy furnace as constructed at the Great Falls smelter made flue dust amounting to 10% as compared to 17.6% for the Evans-Klepetko type. But the furnace as arranged had a much reduced capacity and could not be operated without the continuous use of compressed air, and it was not thought advisable to adopt this type of furnace for regular work.

It should be remarked in passing that the figures for flue dust given include all of the flue dust actually passing out of the furnaces. Part of this dust settles in the McDougall cross flues, and from there passes through hoppers and chutes to the calcine hoppers below. Part settles in the main McDougall flue and is drawn off into cars by the calcine trammers. Practically all of the remainder is recovered in the uptake and crosstake flues and the main dust chamber. Although the McDougall furnaces produce a large amount of flue dust, therefore, a considerable proportion of the dust is recovered at about the same cost as is involved in handling calcine.

A new test was then started on a furnace equipped for dust prevention designed by George S. Crouse. This furnace was similar to the modified Repath-Marcy furnace in that it was arranged to carry the material from the first hearth to the second and from the second to the third without dropping it through the rising gases, but the method of doing this was entirely different. In the Repath-Marcy furnace drop-holes were provided for the descending material and separate gas passageways for the rising gases. In the Crouse furnace the gases traveled upward through the drop-holes exactly the same as in the Evans-Klepetko type of McDougall furnace. Concentrate passing from the first to the second hearth was discharged through an extra drop-hole, which, by an ingenious arrangement, was sealed against the rising gases only during the brief period when material was falling through it. From the second to the third hearth the concentrate fell through the regular side drop-holes, these drop-holes being open for the rising gases except when concentrate was being discharged through them.

A brief description of the equipment of the Crouse furnace is as follows: The extra first hearth drop-hole measured 7 by 8 inches in plan and was placed about 4 in. from the edge of the regular centre drop-hole. All the rakes on the first hearth rabble arms pushed the concentrate toward the centre of the furnace except the two inner rakes, one on each arm. These rakes pushed the material away from the centre. The concentrate, therfore, instead of showering continuously over the edge of the first hearth centre drop-hole, as in the regular type of McDougall furnace, was forced to pass through the extra 7 by 8 in. drop-hole, falling twice in every complete revolution of the central shaft. The rabble arms on the first, second, and third hearths were built so as to be always in line one above the other. On the second hearth rabble arms, toward the centre of the furnace, were attached two vessels open at the top and bottom, extending from the bed of the second hearth up to nearly flush with the roof.' When one of the first hearth rabble arms would be dropping concentrate through the extra 7 by 8 in. drop-hole, the receiving vessel on one of the second hearth rabble arms would be directly below the drop-hole, sealing it against the rising gases. Similarly, a receiving vessel was attached near the outer end of each of the third hearth rabble arms, sealing the regular second hearth side drop-holes while material was falling through them.

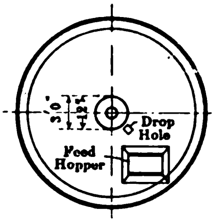
In dropping the material from the third to the fourth hearth, two extra drop-holes were used. This drop-hole construction for the third hearth was referred to earlier in this report in connection with the question of enlarging the third hearth centre drop-hole from 14 to 20 in. The extra third hearth drop-holes were not peculiar to the Crouse furnace, being in use in several of the other furnaces at that time. They were not at any time sealed against the rising gases, but by dropping a large bulk of material four times in every revolution of the central shaft, instead of having it showering continuously over the edges of the centre drop-hole, the amount of incrustation formed on the third hearth roof was greatly reduced. Also, the incrustations formed were easier to remove, because they were localized over the two extra drop-holes. These holes permitted the use of a 14-in. third hearth centre drop-hole.

One further feature in the equipment of the Crouse furnace was the use of 'spark catchers' on the ends of the fourth hearth rabble arms. These 'spark catchers' were plates extending horizontally before and behind the ends of the arms. The object of the plates was to catch the sparks formed while the rabble arms were pushing material through the fourth hearth side drop-holes, thus preventing the building up of incrustations on the roofs. The plates could be cleaned through one door, and could be cleaned more easily than the roof above the drop-holes. The drop-hole arrangement and general equipment of the Crouse furnace is shown in the two accompanying sketches, *A* and *B* of Fig. 10.

The Crouse furnace, then, eliminated the dropping of concentrate through the rising gases between the first and second and the second and third hearths without providing any separate gas passageways. One result of this was that the Crouse furnace was much more compact than the modified Repath-Marcy furnace. At this point it may be remarked that the dropping of material through the rising gas current may be eliminated between the first and second and the second and third hearths with much less injury to the capacity of the furnace than would be produced by eliminating this drop between the lower hearths. For example, in falling through the rising gases from the first to the second hearth, the concentrate may be dried and heated to a certain extent, but no sulphur is eliminated and no heat generated. In dropping through the gas current between the lower floors, a certain amount of sulphur is eliminated from the calcine and heat is evolved. Dispensing with this drop between the lower floors would therefore not only raise the percentage of sulphur in the calcine, but would also take away part of the means on which the furnace depends for the supply of heat to keep it running. In a furnace where the heat is all produced from the oxidation of the sulphur without the help of any outside fuel, any factor tending to cut down the total amount of heat produced is a serious detriment to the furnace.

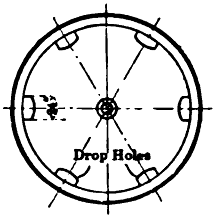
McDougall furnace No. 3 was remodeled to conform with Mr. Crouse's design, and was first started on Dec. 18, 1911. From the very beginning this furnace demonstrated that it would run satisfactorily without the use of coal, oil, or compressed air and would produce calcine assaying the same percentage in sulphur as the average of the other furnaces, although at first it was necessary to feed it a slightly lower tonnage than the average of the other furnaces. A number of minor changes were made in the equipment from time to time, such as rebuilding the second hearth drop-holes to make them come down more nearly to the tops of the receiving vessels on the third hearth rabble arms, and adding two extensions to the third hearth receiving vessels. In general, these changes were designed to seal the drop-holes more securely against the ascending gases during the period of the falling of the material from hearth to hearth.

The first test to compare the Crouse furnace with one of the regular McDougall furnaces as to the relative amounts of flue dust produced, was carried out in April, 1912. The results of this test showed that the Crouse furnace was producing considerably less flue dust, but at the expense of a certain reduction in calcining capacity. With both furnaces being fed practically the same ton-



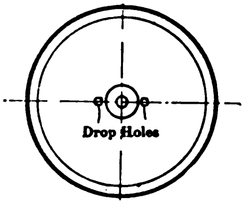
Hearth No. 1.

The centre drop hole is now used as a gas passageway only, and extra drop hole for concentrates.



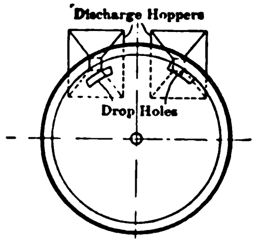
Hearths No. 2 and 4.

Usually one or more of these side drop holes are covered to damp the air current.



Hearth No. 3.

The two extra drop holes through which the calcines fall to the hearth below, localize and reduce incrustations.



Hearth No. 6.

The discharge drop holes are 8 in. wide and 28 in. long on outer arc.

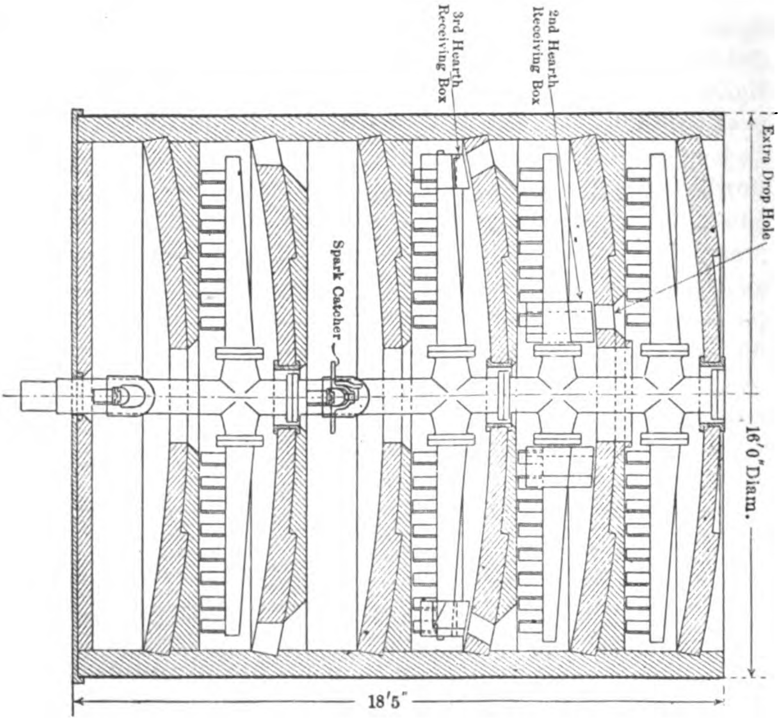


Fig. 10.—Crouse Equipment for Dust Prevention in McDougall Furnace.

nage of concentrate, 64.5 tons, wet weight, per 24 hr., the Crouse furnace produced calcine assaying 14.2% sulphur, as against 10.5% in the calcine from the regular McDougall furnace. Temperature readings taken to compare the Crouse furnace with the regular McDougall furnaces gave the following results:

Temperature in °F. on Hearths of McDougalls.

	Hearth No. 1.	Hearth No. 2.	Hearth No. 3.	Hearth No. 4.	Hearth No. 5.	Hearth No. 6.
Average of six regular McDougall furnaces ..	429	1,017	1,142	1,216	1,162	1,106
Crouse furnace	560	880	960	1,100	1,135	1,150

From the above figures it will be seen that the Crouse furnace ran considerably cooler than the regular furnaces, as would be inferred from the fact that in the comparative test, treating the same feed as the regular furnace with which it was compared, the Crouse furnace produced calcine higher in sulphur; that is, it burned a lesser amount of sulphur per minute and per square foot of hearth area.

As a result of this test the Crouse furnace was shut down and the following changes were made:

Centre drop-hole increased in size from 11 to 14 in. from shaft to edge of drop-hole. Side drop-holes enlarged by chipping out brick between edge of drop-holes and shell of furnace to approximately 12 by 15 by 21.5 in. Centre drop-hole rebuilt, making edge of 14 in. from shaft, and two extra calcine drop-holes 6 by 7 in. Side drop-holes enlarged from approximately 5 by 9 by 20 in. to approximately 10 by 12.5 by 25 in. Brick chipped out between edge of drop-hole and shell of furnace.

These changes increased the drop-hole area, making it possible for a larger volume of air to be drawn through the furnace. Also, the increase was made as far as possible by chipping out the brick between the outer edges of the side drop-holes and the shell of the furnace, thus increasing the drop-hole area with as little reduction in hearth area as possible. In the regular McDougall furnaces the standard size of drop-hole for the second and fourth hearths is 5 by 9 by 24 in. In comparison with the regular McDougall furnaces, therefore, the Crouse furnace, after the above changes had been made, had a larger second and fourth hearth drop-hole area.

When the Crouse furnace was again started it was found that the changes in construction had brought about a considerable increase in calcining capacity, and that it would handle satisfactorily as large a tonnage as the average of the other furnaces. A comparative test was made between the Crouse furnace and McDougall furnace

No. 1, lasting for eight days during the period from May 21 to May 29, 1912. Data obtained from this test are tabulated below:

Summary of Test Data.

Test period: 12 *m.*, *May* 24, *to* 9 *a. m.*, *May* 29, 1912.

Average.	Furnace No. 1.	Crouse Furnace. (No. 3.)
Concentrate treated per 24 hr. (wet weight), tons	67.157	68.493
Moisture in concentrate treated, %	8.0	7.9
Concentrate treated per 24 hr. (dry weight), tons	61.78	63.079
Calcine produced per 24 hr., tons	39.372	42.706
Sulphur in calcine, %	10.3	9.6
Total products, excluding flue dust, per 24 hr., tons	40.531	44.334
Dry weight of feed recovered in above products, %	65.6	70.3
Copper fed recovered in above products, %	78.6	87.0
Sulphur fed eliminated from above products, %	81.3	81.2
Calculated.		
Flue dust per 24 hr., lb	21,049	13,422
Flue dust per dry ton fed, lb	341	213
Flue dust, per cent of dry weight of charge	17.0	10.7

From the above figures it will be seen that the Crouse furnace as finally constructed was a complete success. It had a slightly greater calcining capacity than the regular furnace with which it was compared, and was a big improvement over the regular furnace in that it converted a larger percentage of the feed into furnace products and made less flue dust. Enlarging the drop-hole area on the regular furnace to correspond with the drop-hole area of the Crouse furnace might or might not have given the regular furnace a slightly greater calcining capacity, but the Crouse furnace would still have had the big advantage in its favor of making less flue dust. It may also be remarked here that although on account of the rebuilding of the smelter, and possible changes in the McDougall plant, the old McDougall furnaces have not been remodeled to conform with the Crouse design, the original Crouse furnace, No. 3, has continued in successful operation up to the present date.

McDougall furnace No. 18, with which the modified Repath-Marcy furnace was compared, in the test figures shows a greater calcining capacity than McDougall furnace No. 1, with which the Crouse furnace was compared. These two McDougall furnaces, No. 18 and 1, are exactly similar in construction, and with the same character of feed have practically equal calcining capacities. A change in the grade of concentrate between the dates of the two tests accounts for the larger tonnage treated by McDougall furnace No. 18.

A comparatively recent change made in the construction of the

McDougall furnaces, which has tended to decrease the operating cost of the department, has been the substitution of reinforced concrete hearths in place of the old brick hearths. The brick hearths were 9 in. thick, 14 ft. 6 in. diameter, and were made of regular size bricks laid in regular courses. Due to the expansion and contraction caused by changes in temperature when the furnaces were started and shut down, these bricks would often become loosened and would fall out around the drop-holes. Also, bricks would become loosened by barring heavy incrustations from the roofs, and holes would occasionally be formed in the hearths. The falling out of bricks from either of the two above-mentioned causes would decrease the area of the hearths and would allow material to fall through to the hearth below before being sufficiently calcined. On this account furnaces would sometimes have to be shut down for repairs after running for only a few days. The hearths would generally last a great deal longer than this, but it was never possible to tell when it would be necessary to close down for repairs.

To avoid the above disadvantages of brick hearths, the substitution of concrete hearths was suggested by Mr. Corwin. As an experiment, the third and fourth brick hearths were removed from McDougall furnace No. 6 and concrete hearths substituted in their place. The third hearth was selected because it undergoes probably the greatest variation in temperature during regular operation, and the fourth hearth because it is probably subjected to the greatest variation in temperature on starting up and shutting down the furnace. The material from which the concrete for the hearths was made was portland cement, 1 part; tailing, 2 parts; crushed slag, 4 parts.

The hearths were constructed by first building wooden forms in the furnace and then putting the concrete in on the forms. These hearths were 9 in. thick where they joined the shell of the furnace, decreasing to 6 in. in thickness near the centre shaft. The concrete was reinforced by two series of concentric iron rings tied together by radiating iron rods. One series of concentric rings and radiating rods was buried in the concrete about 1½ in. above the bottom of the hearths, the other about 1½ in. below the top. The inner one of the concentric iron rings in each series was made of ½-in. iron rod, the two ends fastened together, and the outer rings of ⅜-in. iron rod. The radiating iron rods were all ¼-in. thickness, 24 in each series. The method of reinforcing the concrete in shown in detail in Fig. 11.

After putting in the third and fourth hearths of concrete, McDougall furnace No. 6 was started on July 24, 1912. In November the furnace was shut down and the second and fifth hearths changed from brick to concrete, the furnace being started again on Nov. 23, 1912. From November, 1912, to May, 1913, the furnace has been in continuous operation except for short shut-downs to repair the machinery, etc. The hearths are still in as good condition as at first, showing no cracks or wear of any kind. The heavy incrustations that form on the roofs of the hearths are removed more easily, as they do not adhere as tightly to the smooth surface of the concrete as to the rough surface of the brick hearths. The hearths give every appearance of being practically indestructible, and the indications are that the McDougall furnaces, when constructed

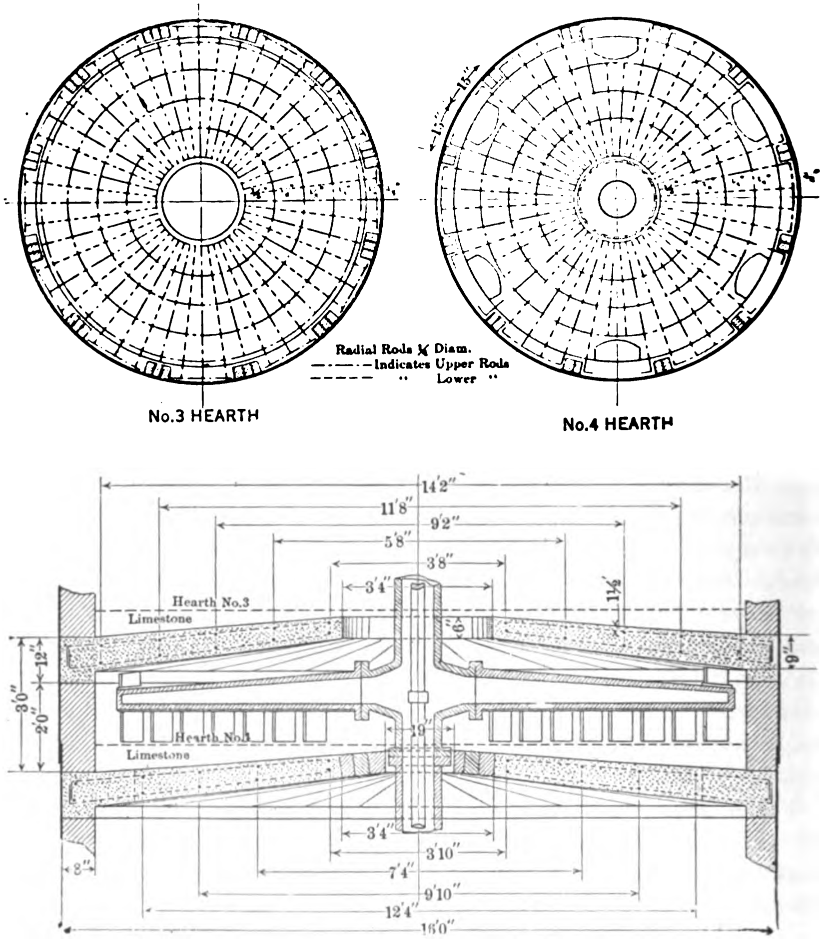


Fig. 11.—Concrete Construction of Hearth for McDougall Furnace.

throughout with concrete hearths will run practically independent of masonry repairs.

Some figures are given showing the reduction in labor required for operating the department. The figures on labor do not include foremen, trammers, oilers, samplers, and so on, as the number of men employed in such positions is practically independent of the capacity of the furnaces. The figures include only the occupations directly connected with the running of the furnaces where a decrease in the number of furnaces in operation means a decrease in the number of men required.

	April, 1910.	April, 1913.
Number of days McDougall department in operation.	30	30
Total tons cupriferous material treated during month	17,778.4	18,369.4
Number of furnace days	386.9	238.4
Average number of furnaces in operation per day	12.9	7.95
Tons cupriferous material treated per furnace day	46.0	77.1
Average per cent S in calcines	7.9	8.9
Partial List of Labor used in Operating the McDougall Department		
Total number of furnacemen during month	270	180
Average number of furnacemen per 24 hr.	9	6
Average number of furnacemen per 100 tons cupriferous material treated	1.52	0.93
Total number of feeders during month	105	80⅞
Average number of feeders per 24 hr.	3½	2⅔
Average number of feeders per 100 tons cupriferous material treated	0.59	0.44
Total number of laborers pulling lumps during month	90	....
Average number of laborers pulling lumps per 24 hr.	3	....
Average number of laborers pulling lumps per 100 tons cupriferous material treated	0.51	....

From the above figures it will be seen that the McDougall department treated a larger tonnage of cupriferous material in April, 1913, with an average of 7.9 furnaces operating per day, than in April, 1910, with an average of 12.9 furnaces operating per day. Due to the fewer furnaces running in April, 1913, there was a saving in labor over April, 1910, of three furnacemen per 24 hr., three laborers pulling lumps per 24 hr., and 5/6 feeder per 24 hour.

SMELTING IN SHAFT FURNACES AT GREAT ALTITUDES

By Vincente Pazos y Sacio

(July 26, 1913.)

*The numerous failures made in Peru in attempting to smelt raw copper ores seem to result from a lack of knowledge of the behavior of shaft furnaces at the great altitudes at which they are there generally worked. The following data, deduced from observation and proved by numerical results, perhaps may be found useful by those who have to manage furnaces in such places.

The solution of the following problem will give insight on the vital point.

Problem.—What will be the temperature of a water-jacket furnace in Cerro de Pasco, where the atmospheric pressure is 8.75 lb. per sq. in.,[1] when the same furnace at sea-level (15 lb.) gives 1365° Centigrade?

By the laws of combustion,

$$\begin{array}{c} C + O_2 = CO_2, \text{ developing 8100 calories} \\ 1 \text{ kg.} + 11.56 \text{ kg.} = 12.56 \text{ kg.} \end{array} \qquad (1)$$

or 1 kg. of carbon combining with the oxygen of 11.56 kg. of air gives 12.56 kg. of heated gases which produce 8100 kilogram-calories.

Assuming that the coke contains 10% ash; the carbon in each kilogram of fuel will be 0.9; and equation (1) will reduce to:

$$\begin{array}{c} C + O_2 = CO_2 = 7290 \text{ kg-cal.} \\ 0.9 \text{ kg.} + 10.4 \text{ kg.} = 11.3 \text{ kg.} \end{array} \qquad (2)$$

If the above reaction did not produce heat, the volume of the gases at 0°C. and 76 cm. (15 lb.) pressure, would fill:

$$10.4 \div 1.3 = 8 \text{ cu.m.} \qquad (3)$$

But the reaction generates a great quantity of heat, which it may be supposed raises the temperature to 1365° Centigrade.

This temperature has been assumed because it gives an even expansion of the gases, and it is near the minimum at which raw copper ore can be practically smelted.

Since air at 8.75 lb. pressure is lighter than at 15 lb., and as the temperature depends on the relative weights of carbon and air combined in a given time, to burn 1 kg. of ash 10% coke, which at the coast requires 8 cu.m. of air, at Cerro will need at the same initial temperature:

$$8 \times (15 \div 8.75) = 13.71 \text{ cu.m.} \qquad (4)$$

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*Excerpt from the *School of Mines Quarterly*, July 1913.

[1] This is the pressure at the American smelter ne de Pasco.

It is seen that if a furnace is designed to burn, at sea-level, a certain quantity of coke, to fit this to burn the same quantity at Cerro, the capacity of the blowers, air conduits, and the area of the tuyeres must be increased about 70%. If these changes are not made, but the blowers are only run faster, the blast-pressure will increase, introducing evils of which I will treat later, and at the same time the blowing engine will be forced to do a great deal of useless work, increasing leakage at a rate which will surprise men who have had no experience in operating furnaces at such high altitudes.

The problem of the volume of air, which now looks so simple, was solved on a scientific basis about 30 years ago. Before this the greater number of those who tried smelting in the Cordilleras used too little, although some went to the other extreme and used too much air; but none could tell in a scientific way the amount that should be blown. As up to that time results of shaft-furnace smelting had been unsatisfactory, it became an axiom that "smelting in shaft furnaces is impossible in the high Cordilleras of Peru." Engineers of almost all nations had tried it and failed. Percy, in the older editions of his 'Metallurgy,' treats of this subject.

The question now arises, does the same amount of carbon burned in the same time at Cerro de Pasco afford the same temperature as at sea-level? If one cubic metre of the eight (equation 3) is taken, formed by the combustion of 1 kg. of 10% ash coke,

At 0°C. the gases will fill 1 cubic metres.
" 273° " " 2 "
" 546° " " 3 "
" 1365° " " 6 "

Thus the assumed temperature of the furnace expands the gases to six times their volume at 0° and 15 lb. pressure.

If this is plotted (see Fig. 12) taking the origin of coördinates at O, which represents absolute zero, and making Oy, Ox, represent respectively the absolute volumes and pressures, then the point t_{15} can be made to represent the temperature 1365°C., since the line Ov_{15} represents the absolute volume of 6 cu.m., and the line Op_{15} the absolute pressure of 15 lb. at sea-level. The suffixes of p, v, and t, in the diagram, represent the pressures in pounds to the inch, to which different pressures, volumes, and temperatures correspond. To find the relative values of temperature, pressure, and volume at any other point, pass a curve of expansion through the point t_{15}.

Since a furnace made for sea-level work can by increasing the volume of the blast, be made to burn the same quantity of coke at Cerro, the rate of burning is the same. As the heat of combustion inside a furnace continually replaces the heat spent, and as the number of heat units, no matter at what temperature, will always be constant, the curve must be an isothermal, no matter how much the atmospheric pressure may vary. In other words, what is desired is to find the quantity of coke necessary to keep a fixed temperature of 1365° at a pressure which varies from 15 to 8.75 pounds.

An isothermal is an equilateral hyperbola

$$p\,v = P\,V = \text{constant}.$$

In this case $P \div 1$ atmosphere = 15 lb. per square inch = 10,333 kg. per square metre.

$V = 6$ cu.m., which is the starting volume at sea-level pressure and at 1365°C. temperature. From the nature of the curve of expansion, it is quite easy to find as many points as wanted. Since $p\,v = P\,V$, $v = P\,V \div p$.

The work done is represented in the diagram by the area bounded by the lines $(t_{15} \ldots v_{15})$, $(v_{15} \ldots v_{8 \cdot 75})$, $(v_{8 \cdot 75} \ldots t_{8 \cdot 75})$, and $(t_{8 \cdot 75} \ldots t_{15})$. which represents a given volume displaced by a certain varying pressure. From the nature of the curve, this area is the same as that enclosed by the lines $(p_{8 \cdot 75} \ldots p_{15})$, $(p_{15} \ldots t_{15})$, $(t_{15} \ldots t_{8 \cdot 75})$. and $(t_{8 \cdot 75} \ldots p_{8 \cdot 75})$, which represents a varying volume displaced between two pressures.

As the first area is the more natural representation of the expansion of the gases, between two given pressures, it can be evaluated by arithmetic. Of course, this valuation, although good enough for practical purposes, is only a rough approximation, for the hyberbolic arc $(t_{15} \ldots t_{8 \cdot 75})$ is considered as a straight line.

Thus 1 cu.m. of the gases of combustion, in expanding from 15 lb. to 8.75 lb. pressure, is found to have done 36,260.16 kg-m. of work. As there were 8 cu.m. of gas (Eq. 3) formed in the combustion of 1 kg. of 10% ash coke, the total for 1 kg. of coke will be.

$$36{,}260.16 \times 8 = 290{,}081.12 \text{ kg-m.}$$

As 425 kg-m. = 1 kg-cal., the loss of heat will be as follows:

$$290{,}081.12 \div 425 = 682.54 \text{ kg-cal.}$$

If closer work is wanted, then the true value for the curve has to be used.

The efficiency of the furnace at the tuyere zone, and under a pressure of 15 lb., is found to be 60%, equal to 1365° Centigrade.

At Cerro, under a pressure of 8.75 lb., owing to the heat lost by the expansion of the gases, the same furnace, with the same weight of air, which means 70% more volume blown in the same time, and with the same weight of fuel, will give only 52.1% efficiency, equal to 1184°C. Thus the furnace at Cerro has lost about 13% calorific capacity, as compared with the same furnace at sea-level. This is

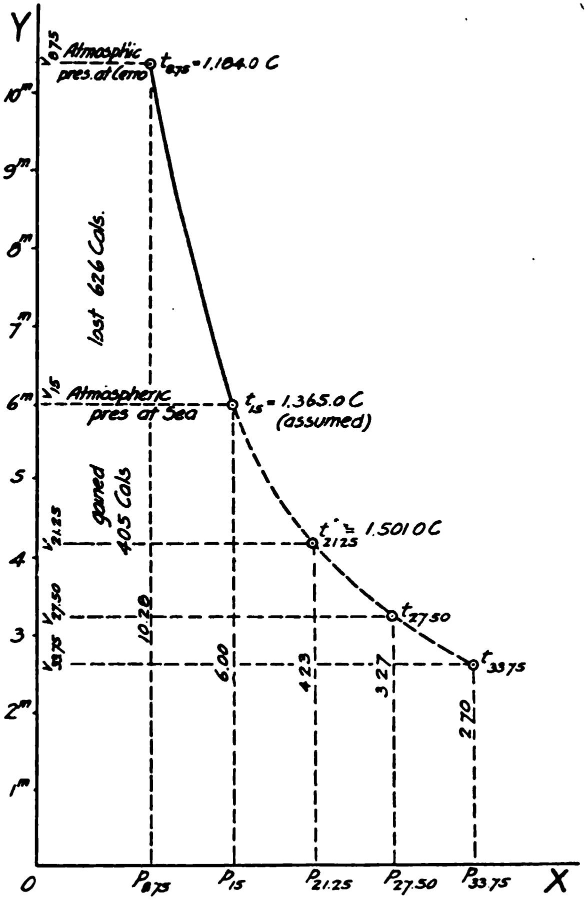

FIG. 12. PRESSURE-VOLUME DIAGRAM.

on the assumption that the air is perfectly dry. To obtain closer results, it will be necessary to make an approximation to bring the respective temperatures nearer to their true value.

At sea-level, owing to the greater pressure and temperature, the air, especially in tropical latitudes, contains a quantity of water vapor which cannot be disregarded; while on the high altitudes of the Andes, the low pressure and temperature make atmospheric moisture very small. At sea-level, the air will be blown at a mean temperature of 25°C., while at Cerro the mean will be 6°, varying during the day from 1 to 11°. Not taking into account the initial temperature of the air at the two places, one can estimate the moisture which, at the coast, has to be heated to the required temperature of 1365° Centigrade.

It is computed that the diminished pressure at Cerro is equivalent to a lowering of 20° in temperature; and the moisture in the air will correspond to that of -14°C., at 76 cm. pressure. From Renault's tables it is found that at this temperature 1 kg. air contains 1.7 gm. water vapor; then the water in the air required to burn 1 kg. of coke at Cerro will 17.68 gm., or nearly 9% of that at the coast.

Thus nearly 9/10 of the heat units used at sea-level, to heat the water of the air blown into the furnace, is saved at the higher attitudes. As the number of heat units at the coast was 112.7 kg-cal., at Cerro these will be reduced to only 10.1 kg-cal., leaving 102.6 which is saved, thus increasing the efficiency of the furnace. Allowing 2.6 calories for the heat necessary to bring the water vapor from 6° to 25°, there will be, in round numbers, 100 calories at Cerro. The corrected efficiency will be 53½%, as compared with an efficiency of 60% at the coast. In other words, if the furnace at the coast smelts 100 tons per day, the same furnace at Cerro, under the best conditions, will smelt only 89; for the fuel has lost 11% of its calorific capacity. At this temperature only copper ores that have been roasted, and of which the slags are very fusible, can be smelted.

The next question is regarding the effect of a high-pressure blast. Suppose that the blast is 2.19 lb., and assume this pressure is necessary to blow the required weight of air, 10.4 kg. (Eq. 2).

If 1 cu.m. of air at 8.75 lb. is taken and an outside pressure of 2.19 lb. (¼ the atmospheric pressure), applied, the cubic metre will be compressed to 0.75 its original volume; this compression generates heat, but as it is gradual, and as the bustle-pipe which connects a blower to a furnace is generally long, the heat generated by this compression is lost through the thin iron shell of the pipe, and when the air reaches the furnace it has the same temperature as the surrounding atmosphere. When the air passes the tuyeres and expands again, practically to atmospheric pressure, for the ore column of the copper furnaces is so low that this factor can be disregarded, the 0.75 cu.m. will not fill its original volume. The heat lost in the compression has reduced its elasticity. In order to get 1 cu.m. in the furnace, one must take 1 metre raised to the 1.4 power, and for any volume v inside the furnace, one requires $v^{1.4}$ from the outside, as expressed in the equation for adiabatic expansion.

Equation (4) gives 13.71 cu.m. of air necessary to burn 1 kg. coke at Cerro. The pressure 2.19 lb., which is ¼ the atmospheric pressure, will give: $13.71 \times \frac{3}{4} = 10.28$ cu.m. air at an absolute pressure of 10.94 lb. Expanding back these 10.28 cu.m. from a pressure of 10.94 (pressure at Cerro, 8.75 + pressure of blast, 2.19) to a pressure of 8.75, the volume required is 11.78 cu.m. of air.

The work of the gases in this expansion is 29,181.3 kg-m.; and the heat units, $29{,}181.3 \div 425 = 68.6$ kg-cal., which makes the relative efficiency 52.56% = 1195.5°C. Two pounds of blast additional at these high altitudes will lower the temperature of the furnace about 20 degrees.

At sea-level the furnace has given as sensible heat 4539.4 cal., some of which is lost in the escaping gases. If the gases escape at 500°. 22.8% is wasted and 39.4% is utilized. Then it follows that

2750.6 cal.	vanished as radiation, etc.	37.8%
1662.8 cal.	wasted in escaping gases	22.8
2876.6 cal.	avail. for melting (efficiency)	39.4
7290.0 cal.		100.0

To find how much ore this furnace will smelt, and to make the calculation simpler, suppose that only slag is produced. Assuming the average copper slag to melt at 1150°, requiring for this 316 cal., and 90 cal. more to fuse it, the slag at 1220°, which is the probable temperature in the crucible, will take up 419 cal. per kilogram; but each kilogram produces around 43.4 cal. heat of formation.

If x represents the number of kilograms of slag formed, then:

$43.4x =$ heat of formation of x kg. of slag.

$2876.6 =$ heat available for making x kg. of slag.

$2876.6 + 43.4x =$ heat at hand for formation of slag.

$419x =$ heat wanted for formation of slag.

Equating:

$43.4x + 2876.6 = 419x$, whence $x = 7.6$ kg. slag formed.

Smelting matte, the proportion of coke to ore will be more advantageous; and if the proportion is 3 of matte to 7 of slag, 1 kg. of coke will smelt 10 kg. of ore.

Perhaps this treatment of the formation of the slag may be thought arbitrary by many readers; but a moment's reflection will show that the slag and the gases of combustion cannot have the same temperature. As soon as the slag, in this case, rises to 1150° and gets the 90 cal. necessary for its liquation, it drops into the crucible, and whatever excess of temperature it obtains above this is due to the transfar of heat from a hotter to a cooler body. From the nature of the furnace, once the slag is in the crucible, the ratio of temperature between slag and hot gases must be more or less fixed, since the constant tapping of the slag does not give enough time for equalizing the temperatures.

In estimating the efficiency of the furnace in the higher altitude, it may be assumed that the radiated heat is the same (37.8 per cent).

In the course of this article it has been seen that 100 cal. is gained by the furnace in the higher altitude, by saving the heating of the water vapor; but there is a loss of 626.1 cal. due to the expansion of the gases; thus the net loss is 526.1 cal., or 7.2%. Keeping the same relation between the wasted and available heat, the heat distribution is as follows:

2750.6 cal.	vanished as radiated heat	37.7%
526.1 cal.	vanished in expanding gases	7.2
1466.2 cal.	lost in waste gases	20.1
2544.2 cal.	avail. for melting (efficiency)	34.9
7287.1		99.9

Equating as before:

$$43.4x + 2544.2 = 419x; \text{ whence } x = 6.7 \text{ kg. slag.}$$

By roasting and careful fluxing, more fusible slags can be made, thereby increasing the output of the furnace; but this is expensive, and only rich ores can stand it. This is what is done by successful smelters; the ores are carefully prepared before going into the furnace; and to have a safe margin of heat, the quantity of coke, as well as of air blown in, is somewhat increased, for it is seen that the temperature of the furnace has fallen from 1365^{b} to 1215°, which is not sufficient for the regular operation of smelting copper ores.

It would be tedious to enter further into these details, but the calculations are sufficient to show that, even with the necessary amount of air, smelting is much more difficult at the Andean altitudes than at sea-level; and the man who at these altitudes wishes to work with open eyes must not forget the laws of the expansion of gases.

To apply the same process of reasoning to pyritic smelting, assume an ore having 70.84% FeS and 29.16% SiO_2.

To make this case as nearly as possible like the one already examined (using coke), consider 1 kg. of fuel, which in this case is the ore itself, and suppose it is converted into slag. From the chemical equation,

$$2FeS + 6O + SiO_2 = (2FeO)SiO_2 + 2SO_2$$

From thermodynamics it can be deduced that a total of 931 cal. is available to run the furnace.

In coke smelting, the furnace gave a certain distribution of heat, which will be more or less the same in pyritic smelting; this is:

Per cent.		Calories.
37.8	vanishing heat	352
22.8	wasted heat	212
39.4	efficiency or melting capacity	367
100.0		931

The slag of the equation will heat up and melt with an expenditure of 395 cal. per kilogram, and most likely, at the crucible of the furnace, will have consumed 406 cal.; thus, 406 × 0.87 = 353 cal., is the heat necessary in this case to melt 0.8712 kg. As there are 367 cal. in the furnace, there is an excess of heat above what is necessary to run the furnace.

The heat produced is almost equally divided between the Fe which gives the solid product FeO, and the S, which gives the gaseous SO_2. The O necessary to burn these elements is 0.3864 kg., which represents 1.288 kg. additional N, giving 1.6752 kg. of air, or 1.2886 cubic metres.

The principal data are now available to find the heat losses of this furnace working at Cerro de Pasco, at an atmospheric pressure of 8.75 lb. to the inch. The item lacking is the relative radiation between the two pressures; but as it cannot be had, its value will have to be approximated.

The gases and the solid products of combustion, while gradually losing their temperature in doing the work of heating and melting the slags, will consume, as has been seen, 62.2% of the total heat, or 577 cal. If the temperature to which this 577 cal. will raise its own products is computed, and the heat lost by the gases in expanding from 15 lb. to 8.75 lb. is deducted, there will result the sensible or temperature-making heat which the furnace gives at Cerro.

It can be computed that the fuel constituents of the ore, if not allowed to lose their heat in melting the slag, would, at sea-level pressure, raise their own temperature to 2325°C. The work of expansion is 52,720 kg-m. per cubic metre of gas formed. But as there are 1.20 m., the total work will be 63,264 kg-m. Reducing work to heat, 63,264 ÷ 425 = 148.8 kg-cal. Therefore the sensible heat has been reduced by 148.8 cal., and the temperature must be lowered 25.6 per cent.

Making a new distribution of the heat:

Per cent.		Calories.
37.8	vanishing heat of radiation and convection	352.0
14.9	vanishing heat expansion	142.0
16.9	wasted heat in gases	158.3
29.3	melting heat (efficiency)	278.7
98.9		931.0

This distribution of the heat is rather arbitrary, for the vanishing heat of radiation is kept the same in both cases. Decreased temperature seems to demand decreased radiation; but, on the other hand, the decrease in atmospheric pressure increases radiation. Not having data on this point, I have thought convenient to let increased radiation balance the decreased temperature; and if the results are not absolutely true, they will, at least, be relatively so. At any rate, ordinary practice confirms that this assumption cannot be far out of the way.

From the new distribution it is seen that at the higher altitude the smelting heat has dropped from 366 cal. to 278.7; or, reducing to kilograms of slag, from 420 to 312 cal. With this heat no pyritic copper slag can be melted and made to run.

To secure the heat that would be available at sea-level, so as to be able to smelt, add about 4% coke with the corresponding amount of air to burn it; but this addition tends to make poor mattes, for at high temperatures oxygen has a very strong affinity for carbon, as is proved by the bessemer converter, where 2 to 3% carbon is burned without any great oxidation of the iron through which the air blast is made to pass. To counteract the reducing action of the coke, a great excess of air has to be blown; this excess and the consequent increase of the pressure of the blast, cools the furnace, and to remedy this, more coke has to be added. As a result, I have never seen in Peru a shaft furnace trying to do oxidizing smelting and consuming less than 10 to 15% of coke. At Cerro de Pasco, copper ore containing from 6 to 8% Cu is reduced to matte of 40 to 45% with an expenditure of from 10 to 12% coke. At Yauly, the same quality of ore was concentrated to 28 or 30% with a minimum of 12%; and in the smaller furnaces through the country, the concentration is only 2 or 3 to 1, with an expenditure of 15% coke and an immense amount of air blown in. In fact, oxidizing smelting in Peru today is nothing but pot roasting carried to fusion at the expense of fuel. This is but natural since the altitude reduces furnace temperatures, as has been seen, by 11 to 25%, according to the fuel, quantity of air, and intensity necessary for the operation.

In view of all that precedes, I believe that all efforts to do oxidizing smelting in the Cordilleras will be useless so long as they are conducted at atmospheric pressure. But if artificial pressure were put on the furnace, and the fuel were forced to burn at higher pressure, not only oxidizing but real pyritic smelting could be done in the Cordilleras as easily as at lower altitudes. If this were aided by a hot blast, then even ores that at present are thought unfitted for this work could be profitably reduced directly, perhaps with the addition of some coke inside the furnace.

It is true that heavier blowing engines would be required, but although these would have to do more intense work, it is likely that, on the whole, not much more work would be done. At present, the quantity of hot unburned air that escapes up the furnace, due to the small ore columns which are indispensable while working on the present system, must represent a large amount of fuel and power, which, if saved, would be enough to run the heavier machines, while a much cheaper and abundant product would be obtained than at present.

The diagram gives the theoretical temperatures of a furnace at −6.25 lb. and +6.25 lb., on the assumption that at sea-level the temperature is 1365°C. From this it will be seen that ores which cannot be smelted at an absolute pressure of 8.75 lb. will be easily smelted at an absolute pressure of 21.25 lb., with the same amount of coke and air used in both cases. In pyritic smelting, on account of the greater quantity of air and the higher temperature necessary for the work, the difference will be still greater in favor of the higher pressure, as has already been seen.

There are several ways of increasing the temperature of a furnace:

1. Reducing to a minimum the heavy losses of radiation and conduction; those planning to use a shaft furnace in the Cordilleras should not overlook this point. The crucible and the jackets, where the loss is the greatest, should receive special attention.

2. The direct use of fuel inside the furnace is the readiest and easiest way to increase temperature, but in copper matting, every pound of carbonaceous fuel inside the furnace tends to make poor matte; and poor matte is nothing but a reduction of the output of the furnace.

3. The use of hot blast. The well known properties of hot blast make it unnecessarry to point out the advantages of its use at high altitudes where the heat losses are so great.

4. The use of back pressure on the furnace, in such a way that the fuel is forced to be burned at the pressure desired for its most economical working.

The combination of all these conditions favorable to high temperature in a single and easily worked furnace would be the ideal for high altitudes.

PROBLEMS IN MODERN COPPER SMELTING

By S. E. Bretherton

(February 10, 1912.)

*As copper is the most interesting of the base metals, its importance in the commercial world being augmented by the wonderful science of electricity, I will discuss its metallurgy; only attempting to describe modern copper smelting, which is more important for advanced students. When I say 'modern,' I mean within the last thirty years. Less than thirty years ago such mines as the United Verde of Arizona, the copper mines of northern California, and others. were considered of little value because the ore was sulphide and considered refractory. Such properties were examined by several noted engineers and rejected. even at a comparatively low valuation. Today such mines have become the basis for the cheapest and best metallurgical work in the world.

For several years the Dominion Copper Co. of Globe, Arizona, hauled a trainload per day of sulphide ore all the way from Bisbee to Globe, Arizona, several hundred miles, in order to get the sulphide to make copper matte for converting. Not only this: it enabled them to make a much cleaner slag than is possible when smelting directly to black copper. I can remember the time when at Clifton and Globe, Arizona, Santa Rita, New Mexico, and other places, rich oxidized copper ore was smelted to black copper, reducing a comparatively small tonnage in the furnace, using 20 to 24% fuel, and throwing a slag over the dump containing 2 to 3% copper, worth $6 to $9 per ton. Today the average slag thrown over the dump will not exceed, I think, 60c. per ton in copper, and the fuel used in the blast-furnace from 1 to 10%, depending on the sulphur contained in the ore and the character of the slag to be made. Briefly, the advantages of smelting sulphide copper ores are as follows:

1. Greater capacity for the same sized blast-furnace.
2. Less danger of 'freeze-ups,' a term generally used by the smelter-men when the furnace is lost by incrustations forming from any cause.
3. Smaller fuel consumption.
4. Cleaner slag.

I might add the saving of roasting the ore. When smelting direct to black copper, it was necessary to roast any rich sulphide ore. For some time after the system of smelting to copper matte came

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*An address delivered before the Mining Association of the University of California, January 24.

into general use, it was thought necessary to roast most of the sulphides, and more fuel was used than at present, but as the method was improved, first with the aid of hot blast, and then cold blast under higher pressure, the percentage of fuel was continually reduced until, in some cases, smelting in the blast-furnace to a low-grade matte was accomplished without any fuel.

I have just referred to 1 to 10% fuel in the charge in matte smelting, depending on the percentage of sulphur and character of the slag to be made. If the percentage of sulphur is low, it is necessary to use more fuel to furnish the required heat and save the sulphur for matte making. Again, if it is necessary to make a slag low in iron and high in magnesia, alumina, barium, or zinc oxide, a much greater heat is required to keep it fluid. A slag chiefly containing iron base requires less heat than when the iron is replaced with an alkaline base (manganese is generally classed as iron for fluxing purposes), but it is necessary to study the specific gravity of the slag, in order to get a good matte separation, and at times in running customs plants I have had so little iron, and so much zinc and barium or perhaps magnesia and alumina, that it required more than the usual amount of heat to keep the slag hot enough to run properly. Too much iron base means a fluid slag, but too heavy for a good matte separation; too little iron and too high a percentage of alkaline bases, especially magnesium and barium, requires more heat, and is also too viscous, holding matte or shots of metal suspended mechanically. Slag containing too much silica and alumina will do the same thing, but the most treacherous slag of all is one which contains too much zinc oxide or barium, especially if they are together.

Alumina, like magnesia, is not objectionable in limited quantities. In iron smelting, alumina is classed as a base, but in ordinary copper or lead smelting I consider it safer to class it as an acid. I find it acts so for me, no doubt, for the reason that in the shallow lead or copper furnace a much lower temperature and less percentage of fuel is used than in the deeper iron blast-furnaces. In lead smelting the metallurgist has very little margin to work on when figuring his slag; that is, he cannot depend on getting any iron from unroasted sulphides; if he did, the volatilization of lead would be too great, causing serious losses, and he must be careful to watch his silica so as not to form silicate of lead in the slag. The copper metallurgist does not fear the volatilization of copper or the formation of silicate of copper; in fact, a silicious slag for copper smelting does cleaner work and is more suitable for fire concentration. But if he depends on getting all his iron for slag-making purposes from raw sulphides, as I have frequently had to do, it is necessary, after figuring the charge, to see that his instructions are carried out on the feed floor, for too much fuel, too deep burden, or too small a volume of blast might mean a freeze-up, due to lack of iron in the slag. The shape of the furnace, the size and number of the tuyeres, the way the furnace is fed, all have some bearing on this question, as well as the amount of the blast used. The size and character of the ore particles has also to be considered.

Fine ore is very objectionable in blast-furnace work (here comes in the question of the proper method of crushing and sampling). Reverberatory smelting practice has the advantage over the blast-furnace in smelting fine ore, but the ore has to be all crushed fine and a preliminary roast given it. I have also found that very little zinc can be volatilized in the reverberatory furnace, whereas I have volatilized over 49% of it in the ordinary blast-furnace, which is a great advantage when the ore contains more zinc than the slag can carry. You have doubtless read articles describing the necessity of roasting matte before re-smelting in order to get a matte richer in copper for either the converter or for marketing direct. I remember an article written several years ago by one of our prominent metallurgists, claiming that the re-smelting of matte in the blast-furnace without roasting did not increase its copper content by concentration. My experience for several years has been, both in Arizona and California, to ship matte containing 30 to 40% from ore in Arizona, averaging about 1½% copper, and in California 3%. This means a concentration of 20 and 13 tons of ore to 1 ton of matte, respectively, requiring a re-smelting of this matte with silicious ore, a re-smelting which was made more difficult on account of having a poor quality of silicious ore on hand to work with.

This roughly outlines the principles of copper matte smelting down to where the converter is used. To make this clearer, it will be necessary to describe each particular feature I have taken up in detail. Some of you may be called upon at some time in the future to decide as to the advisability of erecting a small smelter for a partly developed mine, perhaps some distance from a railroad, the owners having read about 1½% copper in 'porphyries' paying dividends where the ore is first concentrated and then smelted in the reverberatory or blast-furnace, or about where 2½ to 3% copper sulphide is made to pay dividends by direct smelting to matte and then converted to blister copper containing the silver and gold which is shipped to Eastern refiners for further separation.

It may be well here to put in a word of caution. Converting the matte to blister copper means a saving in freight on waste, such as iron and other sulphides contained in the matte, and also a better price for the copper of 1 to 1½c. per pound, but it is not practicable to operate a converting plant in connection with a small matting plant, so that unless the ore contains much precious metal to help the copper carry the expenses, and conditions are unusually favorable, it is often better to ship the crude ore and put in no reduction plant at all. It is true that some of our largest and best paying copper properties of today, such as the United Verde, and Copper Queen of Arizona, the copper smelters of Montana, and others, which could be mentioned, were started as copper matting plants.

To illustrate this point: Some time ago I was called upon to examine a property 45 miles from the railroad. The owners felt sure they had a good property which could be made to pay handsome profits with less than 3% copper. The cost of marketing copper is greater than is generally realized, especially in the form of matte. Matte containing 40% copper is the most profitable to make, on account of slag losses, and is the most suitable for converting plants, where it is blown to blister copper. A 40% copper matte made from a 3% ore would equal a concentration of 14 tons of ore into 1 ton of matte, allowing for about 5% loss of copper in smelting. The expense on this ton of matte would be, estimating copper at 15c. per pound: Wagon haul as back freight, 42 miles, not less than $6; shipping to Salt Lake City (the nearest market), railroad freight, $5.25; copper refining deduction, 2¾c. per pound, $22; copper deduction, wet assay, 1.3%, $3.90. This amounts to $37.15 per ton of matte, which, divided by 14 tons, equals $2.65 per ton of ore. Estimating mining and all development expenses at least $2.50 per ton of ore; general and all office expenses at 35c. per ton; smelting so far from a railroad, $3 per ton, at the very least; made a total of $8.50 per ton, even on the basis of a comparatively large plant. Divide this by 60 lb. of copper, less 5% for loss, or 57 lb., the cost would be 14.93c. per pound, in round numbers 15c. Usually copper ore contains precious metals to assist the copper in carrying the expense, but in this particular case the gold and silver content was scarcely worth considering.

The method of concentrating matte to metallic copper in the converter was the greatest advance in copper metallurgy within the last thirty years. The refining of copper by electricity can perhaps be classed as the second most important advance. The practice of converting copper matte has been constantly improved since being first introduced in this country at Butte, Montana. The use of silicious ore for lining, instead of barren quartz and clay; larger converters; machinery for tamping; and, last but not least, the long-wished for method of adding silicious ore to the converters, as required, is now being accomplished successfully, by using a basic lining and adding the silicious ore instead of depending entirely upon the lining to furnish the silica required for combining with the irôn in the matte as the sulphur is oxidized and driven off.

The great success and continued improvement in converting practice has had considerable bearing on blast-furnace and reverberatory smelting practice. To some extent the same principles are applied in the much higher blast pressure and larger volume of blast now used. The higher pressure and deep shaft furnaces for copper matting can be used where ores are comparatively clean, but my experience has been in the smelting of ore containing high percentages of zinc that it was necessary for us to carry a very low burden above the tuyeres, with an extremely hot top, otherwise the incrustations would form so quickly on the walls of the furnace that the furnace would have to shut down in order to be barred off. The shape of the furnace has considerable bearing on this question also. For this reason I have planned a blast-furnace, especially designed for the smelting of refractory ore containing zinc, shown in the accompanying illustration, Fig. 13.

The extremely large top of this furnace is to allow to fall back on the charge, the dust which results from the fume passing through this increased area and decreasing its speed just before entering the smokestack. This was found to be a great advantage. The dust, which is mechanically thrown out by high blast-pressure and usually goes over into the dust chamber, was found to be very much lessened by the use of this enlarged top. There is a 6-in. space or shell surrounding this top, which is for blast-heating purposes. This feature was added only at the request of those for whom I was designing the furnace, as I do not consider the results obtained justify the expense. By this arrangement the space required for feed doors is cut out, the heating surface is not very great; then again, the interior of this shell, exposed to the hot fumes, quickly becomes coated with dust, which is a poor conductor of heat. The greatest advantage of this air chamber above the feed floor would be the benefit of the cooling effect to the men working on the feed floor. Of course, this can also be obtained by brickwork.

In the body of the furnace below the feed floor, it is only 7 ft. from the centre of the tuyeres to where it is possible to carry the top of the charge, which is much lower than the ordinary type of blast-furnace, but this furnace was especially designed for the smelt-

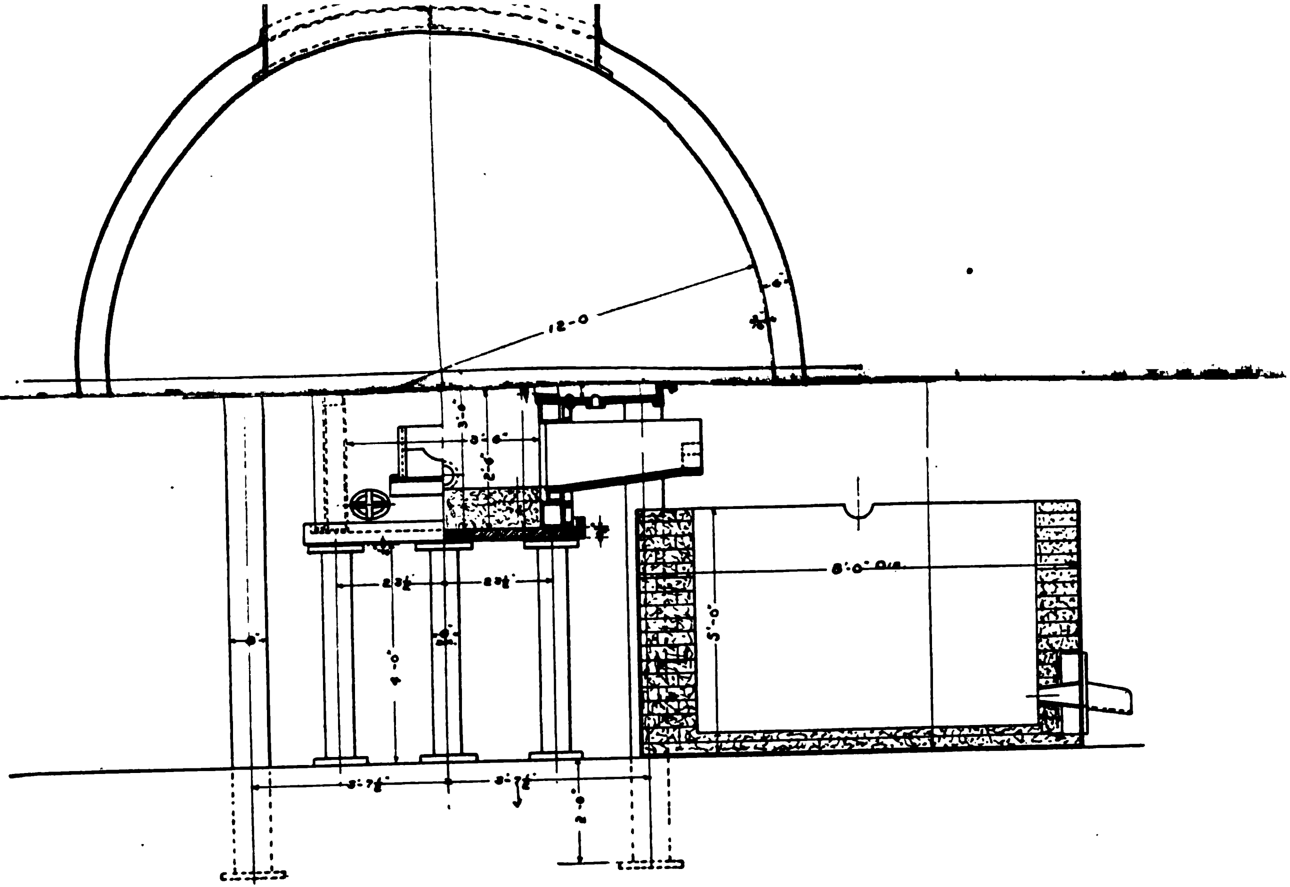

FIG. 13.—BRETHERTON'S DESIGN FOR BLAST-FURNACE

ing of ore containing zinc. The bosh, starting a few inches above the tuyeres, only extends up a distance of 2 ft.; from there to the top of the furnace the walls are perpendicular. I found this a great advantage, for the reason that the smelting zone belongs between the tuyeres and for a few feet above them. By allowing the weight of the charge to be carried by the boshes it keeps the pressure off the crucible and allows freer circulation of the blast between the tuyeres, at the same time holding the burden more compactly above. The following, showing the shape of one of the first blast-furnaces I was requested to use in copper smelting, illustrates my reasons for this, and while it was very provoking at the time, the experience taught me a valuable lesson.

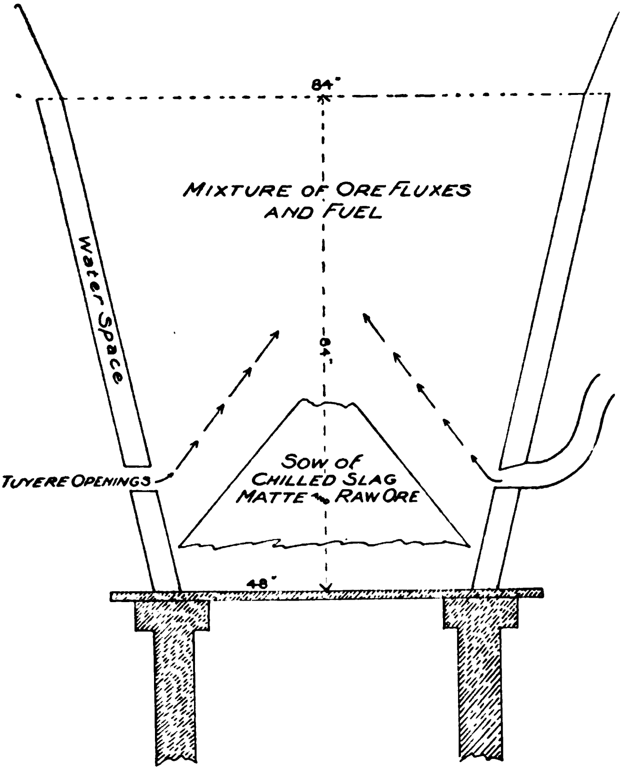

FIG. 14. CROSS-SECTION OF A POORLY DESIGNED BLAST-FURNACE.

The total height of the furnace from crucible to feed floor was about 9 ft.; the diameter at the top was 9 ft. and 4 ft. at the tuyeres, with a continuous slope from the top to the bottom. The ore was silicious copper ore with iron and zinc sulphides, containing some zinc and as much iron as we cared to use. After operating for a few hours, the smelting zone was all at the top of the furnace, slag and matte in the crucible, too cold to run, and a cone or sow formed, as shown in the sketch, Fig. 14. This was due to the blast passing up the sides of the furnace instead of reaching the centre of the charge. It followed the lines of least resistance, blowing the heat with it and leaving an unfused centre between the tuyeres above the crucible, which kept growing larger until the furnace was lost. What little smelting was done the last few hours the furnace was kept alive was between the centre and the walls, the slag dropping into the crucible near the outer rim, which we found to be the case in cleaning out the furnace. This furnace was afterward abandoned and has never been used since, to my knowledge. The man who designed this furnace had had some practical experience, and I was very much surprised at his making such a blunder.

Some metallurgists advocate very wide blast-furnaces, even as much as 56 in., but this is only possible where the ore is coarse, having had the finer material removed by screening, and a considerable depth of burden is carried above the tuyeres. There is a limit to the coarseness of the ore as well; more especially in smelting ore containing zinc in shallow furnaces. As zinc sulphide is one of the most infusible ores, it is necessary to oxidize it so as to volatilize the sulphur and form zinc oxide. For this reason it is evident that in shallow furnaces the centre of any large lump of ore would pass into the crucible where oxidation ceases, and there accumulate until the furnace is lost, unless a good foreman stops the blast, takes out the breast, draws off all such accumulations, replaces the breast, and resumes operations. It will be evident to those of you who have had some practical experience that the blast entering at the side of the furnace will have a tendency to ascend while passing through the charge to the centre; so that if the furnace is too wide or the charge too fine, what is commonly known as a 'sow' will form in the centre of the furnace in the shape of a cone. This will mean the loss of the furnace in time, as it forces the blast with the heat away from the crucible and too much to the top of the furnace. Before this becomes serious there are generally one or more large blow-holes formed in the charge above the tuyeres, through which the blast escapes like wind forced up a chimney. It is the prevention of these blow-holes by proper feeding, which requires some of the feeder's skill.

Much zinc is always objectionable, but it can be handled to better advantage in copper smelting than in lead smelting; in fact, zinc-blende seems to give less trouble in a blast-furnace with hot-blast than when roasting in a reverberatory furnace, as it requires much more heat to liberate its sulphur than when roasting pyrite; and it must be that with hot-blast the oxygen, not having to unite with fuel, has a better chance to combine with the sulphur where such an intense heat exists as in the blast-furnace. Some metallurgists claim that no copper is necessary, and an iron matte will save the precious metals present. Mixtures containing much less iron and more silica can be smelted to better advantage than those I have named; but it would require too long to discuss the question of the different combining powers of each base with silica, to form singulo-silicates for lead smelting or bi-silicates for copper smelting.

The first heat required is for heating the air-blast up to the temperature where the oxygen will combine with either the carbon of the coke or the sulphur contained in pyrite; then the heat necessary for melting ores and fluxes so that they will combine to form the proper silicates for fluid and clean slag is produced by the oxidizing of the fuel in the charge by the oxygen in the blast. If cold blast is used, the free oxygen going in with it is required to oxidize the extra fuel required to heat the blast, thus leaving little for the sulphur. If more cold blast is used so as to get still more free oxygen, it drives the heat still farther away from the tuyere openings into the furnace, and reduces the smelting area of the furnace in that proportion, driving the heat higher up in the furnace and burning the fuel and smelting the ore so near the top of the furnaces that any metals volatilized have no chance to get caught. It keeps the fuel burning so high above the tuyeres that it leaves very little for the blast to encounter as it enters, unless a large quantity is used. When the cold blast enters the furnace and encounters hot charge without fuel mixed with it to generate heat, a crusted furnace soon results, starting at the tuyere nozzle and reducing the capacity of the furnace until it is closed. This results when too much cold blast is used, or too little fuel and sulphur are used with the cold blast. By the use of the hot blast this trouble is greatly decreased and the hotter the blast is the better it is, up to a point where all the heat necessary for smelting can be produced by the combination of the oxygen in the blast with easily oxidized elements in the ore, such as sulphur and arsenic. When these elements are not in sufficient quantity to produce by oxidation the heat required without making too high a concentration; that is, when there is not enough sulphur to make matte for the regular clean working of the furnace, then sufficient fuel should be used in order to save the necessary sulphur for making a proper grade of matte.

BLAST-FURNACE SMELTING WITH CRUDE OIL

(Discussion, February 8, 1913.)

The Editor:

Sir—Your timely editorial on the use of crude oil in blast-furnace smelting brings to my mind some additional facts pertinent to the inquiry. A great many attempts have been made in this direction, naturally enough, considering the obvious nature of the proceeding and the probable advantages which it should have under the conditions prevailing in this state. Where petroleum is so abundant and cheap it is most evident that its successful application in smelting would be attended with great advantages. In regions like California, where coke, the customary fuel in blast-furnace smelting, commands an excessive price, and where on the contrary the liquid fuel is extremely cheap, there can be no question of the desirability of employing the latter could means be found of rendering its employment perfectly feasible. As has often been remarked, the heat units are there, and only the details of its use remain to be settled.

I have been at much pains to procure information as to the various attempts that have been made to employ oil in smelting in the blast-furnace; attempts which began much farther back than most people are aware. I have seen an illustration of an antique and crude form of cupola devoted to melting pig iron which was published fully 30 years ago, exhibiting the pipe connections by which petroleum was to be fed into the furnace—evidently in connection with the blast. It entered at the tuyeres, and it may be observed that while provided with means for atomizing the oil, the combustion was intended to take place wholly within the shaft. Of the propriety of thus burning the liquid in the melting zone I will speak hereafter. The next instance of the use of oil in this style of work that I shall instance, was the proposal of W. L. Austin, the celebrated metallurgist, the father, I may say, of pyrite smelting, to employ oil jets in the tuyeres of his furnace, not especially for the heat thus to be gained, but for the special purpose of preventing the formation of deposits of the magnetic oxide of iron, by which he was troubled at that time. I think Mr. Austin took out patents for such application of petroleum, but I am not aware that his invention came into practical use.

Subsequent to Mr. Austin's proposal Mr. Nesmith of the Colorado Iron Works at Denver invented a method of heating the blast of smelting furnaces by burning petroleum within the air main. His apparatus consisted virtually in an enlargement of the main, affording space for the development of the flame of one or more jets, the gaseous products of which, along with the unconsumed air, went on into the furnace. This involved, of course, the provision of refractory linings in the main and its connections. By this means the blast was effectively heated, but the method was criticised on account of the vitiation of the air, which lost a portion of its oxygen, taking up instead a corresponding amount of CO_2 and water vapor as the result of the combustion of the oil. I was under the impression at that time and later that the loss of active oxygen thus brought about was of a serious nature; but reflection convinces me that the proportion of oxygen thus sacrificed is likely to be too small to affect the smelting process noticeably, and I regret that I expressed antagonistic views thereon. It is easily capable of demonstration that the blast may be even highly heated without losing more than a fraction of the total oxygen, and there is also the consideration that if, as is usual, the blast of air proceeds unimpaired into the shaft it still requires to be heated at the expense of fuel, and of its own oxygen. It is now difficult to see why the preliminary heating which it would undergo by either Nesmith's or Austin's method would prove in any way destructive or prejudicial to the smelting process. Another important consideration here supervenes: the preliminary heating can be done with greater economy than the subsequent heating; since the fuel expended in the Nesmith (and probably in the Austin) device would be thoroughly burned, the end products being CO_2 and water; while the coke upon which we otherwise rely for the inter-furnace reactions, is largely converted into carbon monoxide. The reader will appreciate this according to his knowledge of the reactions in furnace metallurgy. What would be gained by such an application of oil would evidently rest upon the saving in cost of oil versus coke, added to that gained by the more perfect combustion of the former. Some rough calculations may make the possible savings quite perceptible and important. A pound of the ordinary coke sold on this coast sets free approximately 13,000 heat units when completely burned, —that is, to carbon dioxide. A pound of oil produces nearly or quite 20,000, when burned equally thoroughly. Oil is therefore worth fully one-half more than an equal weight of coke under the same conditions as to perfection of combustion. In California, crude oil may generally be had for $1 per barrel, more or less, depending on the cost of transportation. Allowing six barrels to the ton, oil costs about one-third as much as coke, which under similar circumstances may cost $12 per short ton, or $18 for a ton and a half, the equivalent in heating power of one ton of oil. Oil, therefore, whenever its use is permissible, costs only one-third as much as coke. This computation would doubtless hold good wherever the matter is one merely of heat units, irrespective of the different behavior of the two fuels within the furnace. If the mere melting of a substance be the end sought, as of pig iron in the foundry cupola, or that of the materials within the open-hearth or copper reverberatory, there is no doubt that the monetary consideration holds. In the high iron, as well as in ordinary lead and copper-furnaces, where deoxidation by means of carbon is necessary, there would seem to be little likelihood of oil supplanting coke in spite of its greater heating power and less cost. However, there is much to be learned on this subject.

Concerning the complete combustion of crude oil within the tuyeres or before reaching them, experience has taught something of great significance. I have found by experience, and I dare say others have observed the same, that ample free and unencumbered space is essential for the proper burning of liquid fuel. If the space be restricted, or if the jet, atomized or otherwise, is directed against the materials to be melted, the oil is only partly consumed, and much carbon (soot) is set free, giving a very dense smoke, and in many cases a deposit of heavy difficultly combustible pitch is produced which, passing below the zone of fusion, produces some ill effects. Even if the materials within the shaft are wholly incandescent and the combustion lively, as befits the smelting region, there is a waste of oil and the production of no great heat. The lesson learned from this is that the introduction of the liquid fuel directly into the furnace is not a workable method. It is true that in the lower part of the shaft of a pyritic furnace, as commonly run, there is much empty space, through which the currents of gas and air play comparatively freely; but if into a furnace thus working we proceed to inject oil, the immediate result is to lower the zone of fusion, by melting away the partly or wholly solidified matters (the 'noses', etc.) which support the charge above, permitting the whole mass to slide down, obliterating the cavities upon which the perfect admixture of the oil and air depends. When this takes place their blending becomes imperfect and the heavy smoke of unconsumed carbon is again produced, and the rate of smelting diminished. Such was the result of experiments in oil-burning when applied to lead-smelting some years since. The men in charge of that test declared that while the oil effected some apparent saving in the cost of fuel, the advantage was more than compensated by the diminished rate of driving and the attempt was given up. It appears then, in general terms, that oil cannot be worked in a close charge, and I believe that success is only possible when the liquid is applied to the preliminary heating of blast, which brings us back to the question so long argued in connection with pyritic smelting, as to the relative advantages of the heated and the unheated blast. This question, by the tacit decision of the metallurgical world, has been decided in favor of the unheated blast; but the possibility of employing so desirable and cheap a fuel as petroleum adds another chapter to the discussion. The question in its broad aspect is, can we substitute cheap oil for dear coke in some measure, and are there other advantages in the use of oil other than its superior heating power?

There is another advantage, of no mean significance in the use of oil, which I have discovered, and which I desire to call to the attention of practical smelting men. I have observed the strongest tendency on the part of the oil jet to prevent the customary obstructions about the mouth of the tuyeres. The heat bestowed upon the blast is sufficient to immediately liquefy the slag which otherwise would gather there and freeze into a nose, or partly close the exit of the wind stream. Everyone who has handled blast-furnaces recognizes the tendency to choke up the tuyere openings as a serious drawback in ordinary work, and will hail with delight any practicable means of obviating it. The result of the highly heated blast, or perhaps I had better say, the presence of the burning jet at that particular spot, is very favorable to the proper running of the furnace, lessening labor, and by reason of the unimpeded flow of the air-blast relieving the blowing machinery of much unprofitable work. I have used a fan in this connection, and I am convinced that it will deliver fully 50% more air when the oil jets are burning in the tuyeres than when delivering cold air, entirely because the tuyeres are kept free from obstructions. This is an advantage of great moment when large smelting capacity is sought.

Wallace Dow, of Dow Engine Co., of Alameda, has constructed a melting cupola of his own invention, in which oil is the sole fuel. It has proved an unqualified success, melting with regularity the pig iron for the numerous castings manufactured by that company and doing it with an economy unapproached by coke-burning apparatus. I believe he saves fully three-fourths the cost of fuel by the substitution of petroleum for coke, and there is the greatest probability that through further experience the economy will prove still more decided. Not only is the melting done with dispatch, but the bath of metal may be kept fluid and in proper condition for pouring for many hours; in fact there is no reason why it may not remain molten for weeks, since the jets are so arranged in part as to play on the surface of the metal. By this means it becomes possible to cast at any desired time. The problem before Mr. Dow differs from that which confronts the copper furnace manager, since in foundry practice it is necessary to supply all the heat of melting by the combustion of the oil alone, while in copper matting we get a great proportion of it from the reactions between the ore and the blast inside the furnace. In the one case the products of combustion must enter the furnace at a very high temperature—perhaps as great as 3000° F.; while in the other the blast, to carry out all the desired objects, need be no higher as a general thing than 1000° F. Indeed it is generally conceded that with a blast of that temperature no fuel whatever need be used. It is necessary, therefore, to burn but little fuel for the preliminary heating, probably not more than enough to take up one-seventh of the oxygen of the air. In matting stacks there are required on the average about 1¾ tons of blast to each ton of material charged. To raise this weight of air to 1000° F. would require about 875,000 heat units, which is produced by 43.75 lb. oil of 20,000 heat units per pound. This is in the event that combustion is perfect and that there is no loss of heat in the operation. This is in the neighborhood of 7 gallons of fuel. I perceive that Thomas Kiddie has successfully used oil in this kind of smelting, and that he records a consumption of one barrel of oil to three tons of charge, or 14 gallons per ton. Probably Mr. Kiddie's results are more to be relied on than any merely theoretical calculations, although his published experiments related to only a short run. Mr. Dow has not as yet accumulated the facts necessary for precise statements of results, but since he has put his furnace into regular operation, it taking the place of two coke-burning cupolas, he will be able shortly to speak with authority on a number of phases on which there are at present only theoretical considerations for our guide.

Fully realizing the desirability of completely burning the fuel before its entrance into the cupola, Mr. Dow has provided his furnace with two heating chambers, one on each side of the shaft at the bottom, in which the atomized oil is brought into intimate mixture with the incoming blast, and from which the hot gases depart through broad apertures over the sole of the cupola. The heat is necessarily very intense, and its effect upon the firebrick of which the structure is composed is very marked. Better fire-resisting materials than the commercial bricks may be necessary for this purpose. Water-jacketing might be resorted to, but for a furnace which is out of commission the greater part of each 24 hours it seems hardly necessary, differing thus from lead and copper smelting.

For my part I have taken out patents upon a form of construction which lends itself very well to the new requirements. My stack being supported on hollow columns and an encircling windbox, I have modified these to admit of the combustion of the oil inside of them, directing the highly heated blast to the interior of the stack through tuyeres lined with refractory materials instead of the iron pipes commonly employed. Each supporting column conveys and heats the blast, being lined with firebrick and provided with an efficient burner, by which the oil is injected downward so as to meet and be consumed by the incoming blast, which rises from subterranean pipes connected with the blower. By this means I still retain the feature for which my furnaces have been noted, that of no visible air connections. As I am about to put in operation one of these furnaces I shall take pleasure in later communicating to your readers the economic results of my work.

HERBERT LANG.

Oakland, California, January 15, 1913.

OIL-BURNING IN FURNACES

(Duscussion, July 12, 1913.)

The Editor:

Sir—In my letter on 'Blast-Furnace Smelting with Crude Oil,' published in your issue of February 8 last, I endeavored to emphasize the necessity of thoroughly consuming the oil before the entrance of the mixture of hot gases into the furnace. That such emphasis was needed is shown in a variety of ways. I have heard subsequently of two different attempts to utilize oil as fuel in cupola smelting, which came to naught, apparently by a neglect of this principle. Observation of oil-burning boiler furnaces, so common now, will suffice to show the necessity of a thorough preliminary combustion. The conditions essential to complete combustion—which, by the way, never takes place under steam boilers—are these: There must be enough, but not too much air; the space within which the combustion takes place must remain at the highest possible temperature; the space must be ample; and its form must be such that there be no interference with or between the gaseous currents. The ideal form would be tubular. The best construction would be cylindrical, being composed, let us say, of a sheet-iron tube lined with firebrick of good thickness, and placed horizontally. The diameter should conform to the amount of oil to be consumed, and its length should be such that the gases would be thoroughly burned, and that no smoke, the sign of faulty combustion, should issue from the far end. The temperature at the front (where the air and oil enter) will necessarily be less than at the back, but should be kept as high as possible, by cutting off the direct radiation of heat outwardly by means of baffle-plates. It is contrary to common sense to permit of openings at any point wherein the interior can be seen. The fire-chamber is, of course, connected to the furnace proper, and when thus connected to a blast-furnace would constitute a tuyere. Several of them would be provided for each furnace, spaced regularly about its periphery, and pointing radially inward, or having some inclination downward, according to the fancy of the designer. They must be contracted at the inner end, to increase the velocity of the entering gases and assume the customary effect.

The application of oil-burning to the steam boiler is not scientifically made. The old-fashioned coal or wood-burning fire-box is retained and made to do a duty to which it is not well adapted. The interior form of the fire-box is about the worst that could well be devised. Its faults for oil-burning are, first, its many rectangles, which produce conflicting gas currents and prevent the proper mixing which is essential to perfect combustion. Then the large useless space at the front, which cannot be kept hot; but chiefly the presence of the immense bulging boiler immediately over the fire at its incipiency, which effectually keeps down the temperature, which, instead of reaching 3000° or more, as it should, can barely surpass the red heat. I do not mean that all the space is at this low temperature, but that it does not surpass the red heat as an average.

When a liquid or gaseous hydrocarbon burns, the tendency is for the hydrogen to be consumed first, as the carbon, requiring a higher temperature, may escape combustion in part, passing off in solid particles and making a black smoke. A cold object introduced into the path of the burning gases abstracts heat and produces the smoky effect. The bulky boiler, relatively cool, is responsible for the great volume of smoke and soot which often defiles the atmosphere in the neighborhood of steam plants. The remedy is obvious: it is to burn the fuel out of contact with the boiler, and to heat the latter by contact with the fully burned and very hot gases. Smoke is so unnecessary that it is quite a wonder that legislative interference has not been invoked effectively; a proper 'blue-sky' law should be placed on our statute books.

It is in reverberatory smelting that oil-burning has reached its ideal phase. In that furnace the whole interior forms practically a great combustion chamber, kept at a high temperature throughout its entire length (exceeding in some cases 100 ft.), into one end of which the air and atomized oil are introduced, while the fully oxidized gases are withdrawn from the other. Being surrounded by brick-work, cunningly arranged to prevent the escape of heat by radiation and conduction, the whole interior remains at a dazzling heat, the conditions being perfect for the development of the flame and the complete combustion of the fuel. This condition of affairs should be imitated so far as practicable in the other applications of petroleum as fuel, even if it involves the construction of special combustion chambers. Such constructions are in the way of being realized. Several manufacturers of oil-burning boilers manifest, in their catalogues at least, a tendency to prolong their combustion chambers outwardly, or by extending an arch over the internal combustion space, which comes to the same end, to give the fuel a chance to develop its flame unhampered by the presence of the cold boiler surface.

HERBERT LANG.

Oakland, California, June 6, 1913.

PYRITE SMELTING WITHOUT COKE

By LEWIS T. WRIGHT

(February 24, April 7, and September 29, 1906.)

From prehistoric times, copper has been extracted from cupriferous pyrite by a fire treatment consisting of alternate roasting and smelting with carbonaceous fuel and fluxes. Each roasting oxidized a portion of the sulphur and iron and each smelting removed, as iron slag from the copper sulphide, the iron oxidized in the previous roasting process; and these alternating operations were continued until a nearly pure copper sulphide was left; such copper sulphide by melting and oxidation finally became the refined metallic copper which was the aim of these combined processes. The treatment of the ore therefore was one of elimination, in stages, of sulphur and iron, and finally the sulphur; it was a gradual scraping away of the husk and chaff to find the valuable grain.

The successive oxidations and smeltings were carried on with the aid of carbonaceous fuel, which, by its reducing action, is counteractive to an oxidizing process.

Let us review for a moment the so-called English method of smelting cupriferous pyrite: 1. An ore (say 5%) is roasted in a reverberatory furnace with the aid of 15% in weight of coal.

2. The roasted ore is then smelted with the necessary flux,

thereby cutting out, as slag, the iron oxidized in No. 1. This smelting is carried on with the aid of 40 to 45% coal and leaves a 35% copper matte, or coarse metal.

3. The 35% matte is roasted in a reverberatory with the aid of 40 to 45% coal.
4. The partly oxidized 35% matte from No. 3 is now smelted with the aid of 50% coal, again cutting out, as slag, the iron previously oxidized; this leaves a 75% matte.
5. This 75% matte (white metal) is melted in a reverberatory furnace and by a succession of oxidations and reductions, in which the oxidations prevail, is now freed of its sulphur, leaving 'blister' copper. This is done with the aid of 60% coal.
6. The blister copper is refined by a succession of partial oxidations and reductions to fine copper.

The practice of large-scale copper smelting in the United States quickly led to the use of blast-furnaces and converters, and by 1895 the old cumbersome, laborious, and fuel-consuming method had been largely abandoned in favor of more rapid (and more economical) treatment. The coarse sulphide ore was often roasted in heaps with a little cordwood and the fine ore was roasted in mechanical furnaces, especially if in the form of concentrate, with a not inconsiderable proportion of fuel. The smelting of the roasted ore in the blast-furnaces was effected with a quantity of coke equal to about 10% of the weight of the charge, producing a 40 to 45% matte, which was brought up to blister copper in the converters, without the expense of further fuel, except that required by the blowing engines. The sulphide fine, after roasting, was briquetted and smelted in blast-furnaces or smelted direct as a hot calcine in reverberatories. This last method offers advantages where suitable fuel can be obtained at a low cost and where the ore is fine.

At Keswick, California, frequent attempts were made to smelt the sulphide ore (which was pyrite with some chalcopyrite) with hot blast but at first without increase of economy. The coarse ore was, therefore, roasted in heaps or kilns and a smelting charge of this average composition was made for the water-jacket blast-furnaces:

Calcined ore	49.5%
Cupriferous pyrite	12.3%
Raw matte	8.1%
Slag	7.6%
Fluxes	22.5%
Coke	9.3% of total burden
Grade of matte produced	42.0%
Rate of running	5.49 tons per sq. ft. per 24 hr.
Blast	22 oz.

This charge at first ran with extreme difficulty, the furnaces chronically and promptly freezing up. The difficulty was wrongly attributed at the time to the presence of a small amount of zinc in the ore, and it was recommended by the metallurgists then directing the smelting operations to crush the ore, to roast it in mechanical furnaces, and to smelt the product in reverberatory furnaces. An adoption of this recommendation would have entailed the reconstruction of the smelter. When I took charge, in the beginning of 1897, I discovered that the cause of the difficulty was due to the top of the charge being held too low in the shaft of the furnace and on carrying the charge a little higher the trouble immediately disappeared.

The normal charge was calcined ore only and the sulphide in the above average represents occasions when by especially good roasting of the coarse ore the sulphur ran below that required and sulphide ore was added to the charge to bring the matte down to the normal grade. In the reverse manner, if the coarse sulphide had not been so well roasted, the grade of matte fell below the normal and had to be re-treated; and thus raw matte appears in the average charge. The furnaces ran with remarkable regularity and ease; but the heap roasting was an unsatisfactory feature. The fine sulphide had to be roasted in mechanical furnaces and briquetted. Occasionally when the supply of calcined ore ran short, all-pyrite charges were run; but always with the production of a low-grade matte that required two or three re-smeltings to be brought to the normal grade, of 40 to 45%. No attempt was made at that time to reduce the coke on the pyrite charges and the operation was, therefore, not economical; it seemed that, owing to the inability to secure a reasonable concentration, the all-pyrite smelting could not become an economical practice. Later, the roasting of the coarse sulphide ore was entirely abandoned and the fine sulphide was roasted perfectly and a mixture of coarse sulphide (pyrite) and well roasted fine sulphide enabled a 40 to 45% matte to be produced economically. Other attempts to run all-pyrite charges, this time again with the aid of hot blast, did not succeed, owing to the old difficulty of low concentration; and when the coke was reduced to about 3% the furnaces quickly closed up with obstinate accretions. The practice of running all-pyrite charges was abandoned at that time, chiefly owing to the unsatisfactory oxidation of the charge. Many mistakes, as afterward discovered, had been made in the make-up of the charge; among other evils, the furnaces were overloaded with slag.

The first attempt at all-pyrite smelting failed because it was not then realized that the coke then being used could be reduced and that, being counter-active to oxidation, its presence prevented the required concentration of the pyrite into high-grade matte. With 9 to 10% coke on a charge of pyrite, a 12 to 15% copper matte ran through the furnaces like water. Being intensely hot and liquid, the ganister and fire-brick lining was eaten away and the hot, low-grade matte was frequently bursting out from the furnaces, producing explosions. The amount of air in the blast was only sufficient for the combustion of the coke.

When the first serious attempts were made to run all-pyrite charges with hot blast, with only the amount of coke absolutely necessary to keep the matte and slag running, a charge was made up, of which the following are two examples:

	Per cent.	Per cent.
Cupriferous pyrite	58.7	60.0
Raw matte	4.3	3.2
Fluxes	16.6	10.0
Slag	20.4	26.8
Coke in charge	2.0	3.5
Copper in matte	18.7	18.7

The runs were of short duration because in from 5 to 10 days the obstinate accretions threatened to close up the throat of the furnace entirely and, to prevent a freeze-up, the charge was modified.

It must be admitted frankly that the above mentioned attempts at all-pyrite smelting were purely empirical and in no sense founded upon scientific reasoning. Had the experiments been varied as regards the proportion of inert heat-absorbing slag, reducing this to a minimum, no doubt it would have been then and there demonstrated that such charges would run entirely without coke and the concentration would thereby have been somewhat better. But those were not our scientific days of pyrite smelting.

However, the advantage of hot blast in the case of a composite charge of coarse sulphide and well calcined fines was abundantly proved, and hot blast was adopted for all furnaces. It was immediately seen that the hot blast, other conditions being equal, enabled the furnaces to run one-fifth faster with the use of one-fourth less coke and produced 25% higher grade matte. This was the result of being able to take off from the charge so much de-oxidizing carbon.

The composite charge of sulphide and well roasted fines was then smelted with hot blast in a two-stage process. The first-stage furnaces carried the following average charge:

	Per cent.
Calcined ore	26.1
Cupriferous pyrite	35.5
Flue dust	9.7
Slag	14.5
Fluxes	14.2
Coke in charge	4.0 of total burden
Copper in matte	31.0

This matte was re-smelted with some sulphide and calcined fine ore in a charge, of which the following is typical:

	Per cent.
Calcined ore	31.4
Cupriferous pyrite	8.7
Raw matte	26.7
Slag	13.3
Flux	19.9
Coke in charge	5.3 of total burden
Copper in matte	53.5

The final average charge of all the furnaces, both first and second stage, was now:

Calcined ore	28.8%
Cupriferous pyrite	24.9%
Raw matte and flue dust	14.7%
Fluxes	17.3%
Slag	14.3%
Coke in charge	5.2% of total burden
Copper in matte	53.5%
Rate of running	8 tons per sq. ft. of furnace are per 24 hr.
Blast	28½ oz.

It will be seen that a notable economy of coke had resulted. The make-up of the charge on all furnaces was one that produced the smoothest running. Long campaigns were the rule. The rate of driving had increased. This was partly due to the increased blast pressure and partly to its increased temperature.

During the attainment of these results I had been endeavoring to secure some information respecting the necessary and minimum heat requirements of a copper-smelting furnace and to ascertain the minimum amount of pyrite that must be oxidized in a given time in such a furnace to furnish the heat requirements. It was evident that the amount of air required was necessarily more than that which we are in the habit of calling the 'theoretical', meaning that which would be required to furnish the amount of oxygen indicated by the equation FeS_2 plus 5O equals FeO plus $2SO_2$, because that reaction would cease when the tension of the SO_2 reached a degree much inferior to that indicated by the equation. Finally I became convinced that the calorific value of pyrite as a fuel was sufficient for its self-smelting; again I made attempts to run pyrite charges without carbon fuel, but invariably failed because of the incredulity of my staff who were firmly convinced from past failures that it was impossible.

Finally, to convince those operating the furnaces and to induce them to effect the required result of their own volition, it was decided to order it to be done and forbid the use of any coke in the charge except that needed for starting the furnace, it being understood that if the furnaces froze up the general manager would hold himself responsible, and that the crew could have the fun of digging them out.

As a result it was quickly demonstrated that pyrite could be smelted without coke and monthly furnace runs with an average consumption frequently of only 0.1% coke and lower, which coke was only required to start the furnace, became the practice with this charge:

Cupriferous pyrite	74.3%
Flue dust	9.0%
Slag	2.8%
Fluxes	13.9%
Coke in charge	0.32% of total burden
Copper in matte	26.8%
Rate of running	8.75 tons per sq. ft. furnace area

If it had been then convenient to convert this grade of matte direct to blister copper the smelting might have easily stopped there. The matte resulting from this charge was, however, all resmelted in a charge of the following composition:

	Per cent.
Calcined ore	8.8
Raw matte	19.4
Cupriferous pyrite	31.1
Flue dust	6.4
Slag	18.4
Fluxes	15.9
Coke in charge	4.3 of total burden
Copper in matte	41.8

The average of all furnaces was:

Calcined ore	4.3%
Cupriferous pyrite	51.5%
Raw matte	10.6%
Flue dust	6.3%
Slag	11.9%
Fluxes	15.4%
Coke in charge	2.7% of total burden
Copper in matte	42.0%
Rate of running	8.83 tons per sq. ft. furnace area
Blast	32.2 oz.

The smelting of pyrite to 27% matte had therefore been finally carried out in blast-furnaces on a large scale without the aid of coke. So far, attempts to carry it along, without the use of heated blast, have failed. It will be noted that the furnaces run very fast; that is apparently one of the conditions of success. The amount of fuel used in heating the blast is equal to about 1 to 1¼% coke.

It now having been fully demonstrated that the furnaces would run without coke with a sulphide charge of the right composition, efforts were made to force the concentration so as to produce a 35% matte in the first stage and to convert this to blister copper direct without the intervention of a second smelting stage, and to smelt the fine sulphide direct without preliminary calcination and bricking, which latter process is very expensive.

This object was achieved, but the one and only smelting stage had then to clean the converter slags and work up all the fine sulphide ore, the flue-dust and the furnace cleanings; therefore the average furnace charge of sulphide ore became somewhat diluted to:

	Per cent.
Cupriferous pyrite, including fine	65.6%
Flue dust	8.0%
Slag and furnace cleanings	12.1%
Fluxes	14.3%
Coke in charge	1.8% of total burden
Copper in matte	33.7%
Rate of running	8.75 tons per sq. ft. furnace area

Though the furnaces would often run without coke, the slightly heavier burden of inert material and lessened amount of sulphide ore on the charge seems to necessitate the occasional use of a trifling amount of coke. However, the operation, being reduced to one stage of smelting, is much simplified, and the lower grade of the matte adds but little to the cost of the converting department, and there is the advantage that the whole of the slag is discarded at the lowest tenor of copper, and that the whole of the fine sulphide ore is smelted without previous calcination and bricking.

The two great difficulties in the practice of pyrite smelting are the obstinate and rapidly forming accretions, tending to close up the furnace, and the low-grade matte produced. In reviewing any average of charges it is always seen that the consumption of coke varies inversely as the percentage of sulphide ore. Broadly, the addition of each 25% sulphide ore in the charge with hot blast is equal to a reduction of about 2.5% coke. The calorific value of carbon is a little more than four times that of FeS, burned to

FeO and SO_2, and yet it requires ten times more pyrite than coke to keep up the smelting temperature in the blast-furnace. The following average results of large quantities smelted in several furnaces will illustrate the relative practical values of pyrite and coke in smelting:

HOT BLAST.

Percentage of pyrite on charge.	Percentage of coke.	Percentage of copper in matte.
24.9	5.2	53.5
32.1	5.1	52.4
45.7	3.4	46.3
51.5	2.7	42.0
58.5	2.3	59.6
74.3	0.3	26.8

The calorific value of one unit weight of carbon burning to CO_2 is 8.08 calories; that of a unit of pyrite, or FeS_2, burning to FeO and $2SO_2$ I have calculated as 1.976 from a statement that has been made by Schnabel* that the heat of combustion of FeS_2 oxidized to Fe_2O_3 and $2SO_2$ is 2.253. This result appears to be high if that of FeS to FeO and SO_2 is correctly given in the text-book, at 1.29 calories (113.6 calories per molecule).

Pyrite as fuel has disadvantages. Its sulphur burns in a fractional manner or in three stages. On first heating, one-third of the sulphur is lost and Fe_3S_4 is left. On further heating, FeS is formed with the loss of one-sixth of the original sulphur. The loss of one-third of the sulphur at a relatively low temperature on the top of the charge is the cause of the 'hot top' so troublesome in pyrite smelting. This sulphur burns on top of the charge, and while it adds something to the temperature of the freshly charged material it is not as efficient as if consumed lower down in the furnace or in the smelting zone. One-sixth of the sulphur is evolved between the period of the formation of the Fe_3S_4 high in the furnace and that of the matte, and must be consumed at some point lower than the first third. The FeS is now free to burn *en bloc* in the smelting zone. This long-drawn out and fractional combustion while lowering the temperature of combustion is also the cause of the troublesome scaffolding, the accretions formed to pieces and particles of the charge which, being externally semi-fused, build up from the side of a dense mass of unfused material through which the blast cannot penetrate.

FeS is stated to have a heat of combustion of 1.29 calories, which is equal to 0.95 calorie per unit weight of the original pyrite. If the

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**Lehrbuch Allge. Huttenkunde*, page 101.

pyrite charge had only to depend upon the heat of combustion of the FeS which results from the melting of pyrite, the practical result (given above) that the oxidization of an amount of FeS equal to 50% of the charge enables the smelting process to go on, by itself so to speak, can be compared with coke in the following way: Five-tenths of a unit weight of FeS_2 burning as FeS at 0.95 calorie equals 0.475 calorie per unit weight of charge. This is equal to 0.059 carbon or 0.071 coke with 83% carbon; in other words, the FeS component of pyrite consumed should, by a comparison of the respective heats of combustion, be equal to 7.1% coke on the charge. It would, therefore, appear from practice that the easily freed sulphur of the pyrite is not of much value as a smelting fuel.

The development of only a few years has revolutionized copper smelting, reducing it to an almost absolutely automatic process in which three agents are required; that is, cupriferous pyrite, flux, and heated air. With a properly balanced supply of these three agents, the oxidation and slagging off of the iron can be accomplished in two stages and with one continued heating, and in a couple of hours the copper can be separated as blister copper from cupriferous pyrite. The ancients could only have accomplished this, either empirically or by accident. They did neither. The moderns certainly should have been able to achieve it ere they did, because the results of chemistry or physics, if properly applied, would long ago have demonstrated the limits within which the process of pyrite smelting was possible.

It has already been shown how successive increments in the proportion of pyrite to the total smelting burden of a blast-furnace were attended by a corresponding decrease in the quantity of coke required as accessory fuel in the smelting process and at the same time in the grade of matte produced.

The grade of matte obtained in the smelting of calcined pyritous ores depends chiefly upon the amount of oxidized iron in the charge or, conversely, upon the proportion of iron sulphide, that is, it is determined by the amount of 'solid' oxygen in the charge or by the proportion of 'solid' oxygen to sulphide. It depends also upon the amount of gaseous oxygen left over after the carbon of the coke has been satisfied and, to some extent, upon the grade of ore; but in the case now under discussion the influence last mentioned may be neglected, because on an average it remained constant, and we are not dealing with abnormal results.

It will be interesting to consider for a moment what the oxidation by 'solid' oxygen means in practice. To burn an equivalent of FeS to Fe_3O_4 and SO_2 with the solid oxygen of Fe_2O_3, as in the old smelting and roasting processes, it is necessary to have ten equivalents of Fe_2O_3 to one of FeS_2, or twenty of oxidized iron to one of sulphide iron.[*] It was impossible to arrive at this degree of oxidation in a single heap-roasting, or in any economic reverberatory roasting of lump or fine ore in one stage; it could only be achieved in the slow oxidation of the burners at sulphuric acid works which use the purer forms of pyrite. On account of this difficulty in oxidizing so large a proportion of the iron of pyritous ores, the old processes employed so many successive roastings and smeltings.

It is only very lately that the blast-furnace has been used as an oxidizing furnace in its fullest sense, and I cannot doubt that if metallurgists will deliberately set themselves the task of pushing the use of the blast-furnace to its limits as an oxidizing method, they will attain results very different from those to which they have been accustomed; then pyrite smelting in one stage, before converting to blister copper, will become general practice.

The results already given show the relation between the grade of matte and the coke consumption. Grade of matte in such a case represents in a general way both classes of oxidation, namely, by solid oxygen as Fe_2O_3 or gaseous in the air-blast.

The following table shows the relation of coke consumption and grade of matte in the case of cold blast. Monthly averages over a long period were arranged in groups according to the grade of matte produced from the same ore and with the same general working conditions:

COLD BLAST.

Copper in matte. %	Coke to total burden. %	Number of months in average.
22.7	7.6	9
28.5	8.2	12
33.0	8.65	11
37.7	8.7	8
41.9	9.2	17
53.0	10.2	8
65.2	15.1	3

It is important also to consider the relation between the sulphur in the charge and the grade of matte produced and the coke consumption; this can be seen in the following table of results, each being the average of one month's furnace work:

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[*] Because FeS can only reduce Fe_2O_3 to the magnetic oxide of iron, Fe_3O_4. See *Eng. & Min. Jour.*, Nov. 1, 1913.

COLD BLAST.

Sulphur in charge. %	Copper in matte. %	Coke of total burden. %
24.5	20.1	6.8
22.8	19.3	7.1
19.5	24.6	8.5
17.0	26.4	7.7
13.6	41.4	10.2
7.7	60.9	16.3
6.8	69.4	15.7

When hot blast was first tried at Keswick it was quickly realized that the coke consumption could be cut down with increase of furnace capacity and, though there were some defects in the design of the hot-blast stove that caused the speedy destruction of the front iron pipes, a better design entirely remedied this trouble.

A six months' comparison of hot and cold blast brought out the following results:

	Coke to burden. %	Copper in matte. %	Tons smelted per sq. ft. per 24 hours.
Hot blast	6.83	49.2	6.91
Cold blast	9.10	40.3	5.87

But to render the comparison complete the consumption of coke with cold blast must be corrected to the same grade of matte made by the hot blast. The coke consumption so corrected is 9.9%. The actual saving with hot blast is therefore 3.07% of coke with 18% increased furnace capacity.

The hot blast brings with it an increased activity; that is one of its merits. There is still another and even more important advantage; with cold blast it was only on rare occasions that unroasted ore could be carried on the charge to make 42% matte, while with hot blast over 50% of the charge could consist of unroasted ore to make the same grade. The attitude of metallurgists in this country toward the use of hot blast in copper smelting is not on the whole cordial, because there have been some failures, perhaps due to the defects in the design of the hot-blast stove, and in other cases the saving of the coke may be unimportant; but the advantages under Keswick conditions were too apparent on its first trial to allow of its aid being rejected.

It will be useful in this discussion to consider for a moment the case of hot blast in iron smelting. Though in the latter case a much greater economy can be shown than in copper smelting, yet the cases are not chemically parallel. When Neilson conceived the idea of hot blast he must have been guided by an instinctive feeling that higher temperature would mean increased furnace activity, but he could not have foreseen all that happened. It is well known that chemical reactions are promoted enormously by increased temperature. Combustion proceeds so much more rapidly as the temperature rises, up to a certain degree of course, which we do not reach in practice, when the counter effects of dissociation are felt. Any practical man would know this. When we come to analyze the results obtained, we find that the most remarkable economy of hot blast in iron smelting was not only that which was the natural result of working at a greater speed due to a higher temperature, but was traceable to another cause that had been unsuspected.

In a case cited by Lowthian Bell† the raising of the blast temperature from 25° C. to 485° C. resulted in the furnace running 2½ times as fast, and this increased activity, which in itself is most important, was accompanied by a reduction of coke from 62.6 to 44.5 cwt. per ton of pig produced.

The cold blast required 111.18 and the hot blast 104.34 Calories‡ per unit weight of pig. The difference of 6.84 Calories is not large and would in itself not have been remarkable had that been the whole story. The faster running due to the higher temperature brought about an economy of only 2.77 Cal., due to the lesser effect of losses by radiation, cooling water, etc., when running fast; and that is about all the heat economy an examination of the analyzed heat balance-sheet will show as the physical result of the increased activity of the furnace due to hot blast. It was discovered, however, that one unit weight of coke, instead of producing 2.73 Cal. as it did if burned with cold blast, produced 3.66 Cal. with the hot blast. Here was an enormous economy. The coke was required not only for its physical effect in maintaining a working temperature, but to produce CO to react with the ferric oxide to reduce it to metallic iron, and under the conditions of cold blast the carbon was oxidized in the proportion of 6.12 units of CO to one unit of CO_2. With heated blast, the ratio fell to 3.74 units of carbon as CO to one unit as CO_2. It is this much smaller ratio of CO to CO_2 (since much improved) that accounts for the production of 34% more Calories per unit weight of the coke used in the case cited above, of hot blast in iron smelting.

In our case the carbon, instead of playing a necessary rôle, is

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†*Journal* of Society of Chemical Industry Vol. IX, p. 691.

‡The Calories used in this article are the big Calories usually spelled with a big C.; that is, 1000 grams water raised 1° C.

counter-active to pyrite smelting and must be kept out of the furnace as much as possible. It is an expensive middleman, and for the work it does in promoting production of temperature, exacts too high a commission in the shape of the oxygen of the blast.

More recently, Gayley, working with iron furnaces, has demonstrated, by large scale practice over long periods, that by drying the blast an extraordinary economy is attained, namely, nearly 20% of the coke consumption. One-third of this economy is explained by J. W. Richards to be due to the more economical ratio of carbon monoxide to carbon dioxide, that is, 1.24 with dry blast in place of 1.91 with moist blast. The immediate effects of a lower ratio between carbon monoxide and the dioxide are an increase in heat production with the same amount of coke, and a decrease in the amount of oxygen and blast required, and the losses that the moisture in this and the heating of the diluting nitrogen entails.

In the future smelting of cupriferous pyrite, carbon will play a subservient part.

I will now review what was finally done at Keswick by keeping coke off a pyrite charge and pre-heating the blast. All the ore, both fine and coarse, was smelted direct in the blast-furnace instead of four-fifths undergoing a preliminary outside roasting. For the production of 34% matte the coke was reduced from 8.7 to 0.1% and the furnace capacity was largely increased.

It would have appeared paradoxical in the days before hot blast was applied to iron smelting, to state that hot blast would result in an economy of 30% of the fuel, because it was not then known that the carbon ratio would be so favorable. In our case, we knew beforehand that taking off carbon from the charge would promote the oxidation of the pyrite by gaseous oxygen, and it was quite reasonable to believe that carbon burned outside the furnace to heat the blast would, by reason of the introduction of its heat directly into the smelting zone with the blast, be productive of a more economical and active result than by its introduction at the top of the furnace, where some portion of it is consumed before reaching the actual seat of operations. It is for that reason we found in our work, with partly calcined charges, that one unit weight of carbon burned outside the smelting furnace to heat the blast would replace three parts burned inside the furnace. In the future, by suitable design, the heated gases issuing from the top of the furnace will be made to heat the blast, and then, by forcing the oxidation, a matte of high enough tenor for converting will be produced in one operation without coke. In order to carry on this operation without coke it is evident that there must be a certain amount of oxidation of the pyrite, and it will be my object now to show in practical terms what is the import of this necessary and minimum oxidation.

From practical results, I find that when the furnaces are running without coke, the charge contains so much sulphur and so much iron as sulphide, and of this sulphur about 77.5 to 90% is eliminated as SO_2. The furnace also must run at a certain speed, say, at the rate of 8 to 11 tons of total burden per square foot of hearth-area per 24 hours.

In the self-smelting of pyrite there are two governing factors, namely, amount of iron and sulphur oxidized per unit weight of charge and the number of unit weights of charge smelted per unit of time. I propose to examine these factors and to give them a simple practical expression.

When pyrite is introduced into a smelting furnace, one-half of the sulphur might be eliminated by the heat alone, and this sulphur might rise to the top of the furnace and there burn to SO_2 with air drawn in below the hood. In practice, only a trifling amount of sulphur escapes combustion there. Thus we could convert the FeS_2 to FeS by the elimination of 50% of its sulphur, and but a trifling amount of useful work would have been done toward the concentrating of the ore into matte. It is only after 50% of the sulphur has been eliminated and the iron, as well as the sulphur, is burning, that the really useful work of concentration commences. Let us assume a sulphur elimination of 90%. That would mean an oxidation of 80% of the iron combined with the sulphur in the form of pyrite. It is found in practice that a charge containing 30 to 33% of its weight as sulphur, which sulphur is practically combined with iron as sulphide, will run without coke under proper conditions of air-supply, etc.

The sulphur elimination will vary, of course, within certain not very wide limits. If, with a charge containing 32% of sulphur as sulphide, 90% be oxidized, then 0.288 of a unit weight of the furnace charge will be eliminated as SO_2, and there should be a corresponding oxidation of 0.224 of iron which goes into the slag as FeO. We can neglect the small amounts of magnetic oxide of iron in the matte and metallic iron in the slag. It is a difficult and complicated matter to get a chemical control of the iron of the charge, part of it being oxidized iron in the converter-slag and flue-dust, and there will also be errors in the sampling of the several components of furnace charge and products. To obtain an absolutely correct balance sheet of the elements of the charge through all the operations is certainly not a simple task.

In the case of an actual charge containing 31.62% sulphur that was running satisfactorily without coke, it was found that 27.8% was lost to the solid products, and presumably in the furnace gases, as SO_2. The iron was 33.25% of the charge, but 5.86 of this amount was calculated from the analyses to be already oxidized, leaving 27.39 as the sulphide iron principally as FeS_2. The slag, allowing for the 5.86 of iron already oxidized, accounts for an oxidation of 21.8 of the iron per 100 of charge.

In almost every case examined there has been a fair agreement between the sulphur and the iron oxidized, allowing for the residuum of FeS that is left on the combustion of the FeS_2, which combustion, of course, is not entirely completed in the blast-furnace.

In the self-smelting of pyrite it is found that from 0.23 to 0.27 sulphur per unit of charge must be eliminated as SO_2 in the furnace gases and that from 0.19 to 0.25 iron must be oxidized in the furnace. In a case of partial pyrite smelting the respective amounts of sulphur in the charge and oxidized were 0.23 and 0.17; this was not sufficient for self-smelting and 4% coke was used as an auxiliary fuel.

In the discussion of such technical processes, especially during their early development, when complete scientific investigations are wanting, we require some general practical expression of requirements and efficiencies; it is something to the practical man to have these; to know, in short, that there is a working requirement of sulphur and iron as sulphide on a charge, and that of this a certain percentage must combine with the oxygen of the blast to furnish the heat for the process. I shall state the necessary sulphur component of the charge as 30 to 33% or that one-third of the charge must consist of sulphur with its corresponding amount of iron, which latter for pyrite is about 28% of the charge.

It is necessary for self-smelting that of this necessary minimum of sulphur, which naturally will vary within narrow limits under different circumstances, about 80 to 90% should be oxidized. For every pound of charge one-quarter pound of sulphur must be burned to SO_2 and one-third of a pound of sulphur as metallic sulphides should be put on the charge.

There are two other features of this smelting process that require attention and reflection: One, into which the element of time enters, is the speed of smelting. This determines the total heat production which must be sufficient after all the losses have taken place (and these will not all vary in proportion with the speed of the furnace) to discharge the matte and slag from the furnace spout at a temperature, say, about 1250°C. so as to allow of their being handled in settlers, launders, ladles, etc. The speed or rate of running of a furnace can be expressed by tons of charge smelted per 24 hr. per sq. ft. of hearth-area. In my experience in the self-smelting of pyrite from about 8 to 9 tons, running up to 11 tons on occasions, and not falling below 6.5 tons, are required. A practical expression of necessary furnace speed is therefore about 8 to 9 tons. This is much higher than the usual practice with coke-aided smelting, nine tons per 24 hr. per sq. ft. is close to 12.5 lb. per minute.

In one instance the respective amounts of iron and sulphur oxidized per unit of charge were found to be 0.22 and 0.28; therefore the weight of iron and sulphur respectively being oxidized per square foot of hearth-area per minute were:

12.5 multiplied by 0.22 equals 2.75 lb. iron
12.5 multiplied by 0.28 equals 3.50 lb. sulphur

The oxygen required for these quantities is:

2.75 multiplied by $\frac{2}{7}$ equals 0.78 lb. oxygen
3.50 multiplied by 1 equals 3.50 lb. oxygen

Total .... 4.28 lb. oxygen

The furnace results, and I am citing from an actual case, show that 4.28 lb. oxygen were being taken to convert the sulphur to SO_2 and the sulphide iron to FeO per minute per square foot of hearth-area.

It is now time to consider how much air is required to furnish this amount of oxygen, and it must be remembered that its efficiency in burning sulphides is not 100%. If pyrite burned according to the equation $2FeS_2$ plus 11O equals Fe_2O_3 plus $4SO_2$, it should give a gas containing 16% SO_2, or, if the iron burned to FeO, a gas containing 17.5% SO_2, making no allowance for the trifling amounts of SO_3 formed. I have repeatedly investigated this matter with burners in acid works, with roasting and smelting furnaces and converters, and in the laboratory; and I doubt very much whether any such gas can be obtained.

When the SO_2 reached 12% of the furnace gases the combustion of the sulphide ceases. As a result of these investigations and from practical experience I believe that 70% is about the outside efficiency of air in roasting or smelting pyrite, and I will adopt this factor of efficiency for the present and calculate the amount of air required for the 2.75 lb. iron and 3.50 lb. sulphur oxidized per min. per sq. ft. of hearth-area. One pound of oxygen is contained in 55 cu. ft. air at 72° F. At an efficiency of 70% there was required: for the above amounts of iron and sulphur: $\frac{4.28\times 55}{0.7}$ equal to 336.3 cu. ft. air. Call this 350 cu. ft. of air per square foot of hearth. A furnace of 44 sq. ft. in pyrite smelting will, under the conditions of the example cited, require 44×350 or 15,400 cu. ft. air per minute.

It has been shown in this article that for the self-smelting of pyrite, a certain quantity of iron and sulphur must be smelted per unit of hearth-area, and that, because all the iron sulphide* on the charge will not be oxidized, a certain excess, over and above the quantity required for the heat production, must be contained in the charge, and it is hoped that enough details and explanation have been given to make it clear why the man who starts out to smelt pyrite without providing a certain amount of air cannot expect to succeed in dispensing with coke or in attaining an economical degree of concentration.

The specific heats of pure substances and their latent heats of fusion have been determined in some cases with extreme accuracy and we use these values in our practice, but in our industrial practice we are not often dealing with pure substances and a small amount of impurity may cause important deviations in their physical properties. It is this application of an accurate physical determination of a highly specialized case to the more or less impure bodies encountered in practice that we call theoretical. What are called 'practical' results have, generally, more interest for practical men than those commonly termed 'theoretical,' though the latter, used in a proper manner, are useful in controlling and guiding our practice; and the technician who studies the why and the wherefore of practice gladly avails himself of the illumination and suggestion of the so-called theory. There is no gainsaying practical results. Economically they may be good or bad, but they are something that really happens, and practice will long remain unbending before the most attractive theory. The examination of our practical results by the oblique light of theory often brings details into relief that may escape the sight when viewed in the dim perspective incidental to our daily work. Standards of possible efficiency encourage the inventor to progress, and further, the control of practical results by the accurate data of science often serves to reveal gross errors in our practical determinations and indicates the direction to that highest possible efficiency after which we are striv-

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*This term is used to include both FeS_2 and the residuum of FeS left by the partial oxidation of the former.

ing. No excuse will be required for an attempt to examine the results obtained in practice by the light of science, remembering of course that the physical constants we use are merely the nearest approach reached in each case by experiment, and apply to bodies in a high state of purity.

How far is it possible, for instance, to study the comparative calorific values of coke and of iron sulphides? It should be easy; but it must be remembered that the endowed departments of science have not yet determined for us all the physical constants we need and use, and that we, in the haste of output, cannot determine for ourselves.

There are certain fundamental data—such as the heat of combustion of the iron sulphides, of sulphur, and of carbon—that we can use to explain our practice of smelting and these are stated in different kinds of calories or in British thermal units (B.t.u.)

The calorie (cal.) is often defined as that quantity of heat that is required to raise one gram of water from 0° to 1° C.; but the thermochemical equation contains a universal definition that will be found more useful for our purposes. For example, the equation

$$C + O_2 = 96{,}960 \text{ cal.}$$

states that 12 parts by weight of carbon uniting with 32 parts by weight of oxygen generate a quantity of heat that will raise 96,960 parts by weight of water from 0° to 1° C. These parts can be any weight-unit—tons or grams, pounds or kilograms, ounces or hundredweights—the only condition being that the weight-unit used shall be the same on both sides of the equation.

The big calorie (Cal.) contains 1000 small calories (cal.).

The above equation is therefore often (in round numbers) written

$$C + O_2 = 97 \text{ Cal.}$$

The big Calorie (Cal.) is the one used in this article because it is much more appropriate for technical purposes than the small one, unless it may be for stating the specific heats of substances when, because of the small quantities of heat involved, the small calorie is less cumbersome.

		Per unit-weight of sulphur.
(1)	$Fe + O = 66.4$ Cal	....
(2)	$FeS + 3O = 113$ Cal.	3.55 Cal.
(3)	$FeS_2 + 5O = 237.3$ Cal.	3.70 "
(4)	$FeS + 10Fe_2O_3 = -$*86.2 Cal.	−2.69 "
(5)	$S + O_2 = 71$ Cal.	2.22 "

*i.e. $$\begin{array}{ccccccccc} FeS & + & 10Fe_2O_1 & = & 7Fe_3O_4 & + & SO_2 & & \\ 23.8 & & 10 \times 199.4 & & 7 \times 265.8 & & 71 & = & -86.2 \end{array}$$

	Per unit-weight of carbon.
(6) $C + O_2 = 96.1$	8.08 "
(7) Coke with 83% of carbon=	6.71 "

The number above given for FeS_2 is open to some doubt, but it is the best available. By analogy with some other metallic sulphides the heat value of the sulphur of the bisulphide should be less than that of the monosulphide. By comparing the two iron sulphides on the basis of the unit weight of sulphur, we find they do not differ much and no great harm will be done if we adopt the value of 3.55 Cal. for both, for present practical purposes, until the value for FeS_2 has been confirmed; and it is interesting to note that the oxidation of ferrous sulphide by means of the solid oxygen* (as in the time-honored case of the smelting of a mixture of sulphides and roasted sulphides or what is the same thing in this sense, namely, imperfectly roasted sulphide) by equation 4 absorbs 2.66 Cal. per unit-weight of sulphur. From equations 2 and 6 we learn that one part of sulphur combined with iron is calorifically equal to 0.44 part of carbon or to 0.53 part of coke containing 83% carbon. How far is this relation approximated in practice?

In smelting mixed charges of pyrite and roasted pyrite with cold blast the coke consumption varies with the same make-up of charge, not because of variations in the composition of the charge, but according to the condition of the furnace. With hot blast the furnaces did not exhibit as much variation in this respect. To make a comparison between coke and sulphur, it is necessary therefore to obtain averages, by combining a large number of monthly furnace-runs, in order to compensate for the variable outside influences which appear to be working in one or other direction. In the following table are given results obtained by averaging several monthly runs in which the sulphur content of the charge was much the same for a set; and I hope that in this manner the fuel relation that exists in practice between the sulphur and the coke may be emphasized. All the sulphur in the unroasted ore is calculated as FeS_2 and all that in the roasted ore, slag, flue-dust, etc., as FeS. The sulphur combined with the copper is considered in the same manner as if combined with iron, and no inconvenience will be found to result from this assumption because we are concerned with that portion of the sulphur of the charge that becomes oxidized in the smelting process and not with the whole amount, and it will be appreciated that this amount consumed cannot be combined with the copper because that portion must be left for the matte. The small amount of zinc sulphide present will also not interfere with this method of examination of the fuel-effect of the combined sulphur of our charge, because its sulphur has the value of 3.57 Cal., which is about the same as that of the sulphur of FeS. In the table the results are calculated to unit-weights of sulphur and coke per 100 unit-weights of the actual average charge. The grade of the matte is also given.

RELATION BETWEEN SULPHUR AND COKE, USING COLD BLAST.

	S in FeS, %	S in FeS_2, %	Total sulphur, %.	Coke, %	Cu in matte, %.	Furnace months.
A	13.5	2.2	15.7	8.92	34.8	4
B	11.7	6.8	18.5	8.55	29.4	6
C	12.8	7.1	19.9	8.43	27.9	4
D	7.5	17.5	25.0	7.88	22.8	4

The replacement of roasted pyrite by the raw pyrite is attended in the above practice by an increase in the total sulphur, a diminution of coke, and a lower grade of matte. Of the total sulphur, 70% was oxidized to SO_2.

On comparing the extreme members of the table we find that one unit-weight of sulphur oxidized replaced 0.159 part coke. From the heats of combustion, it should have replaced 0.53 part coke containing 83% carbon if oxidized by reaction (2) and 0.4 coke if by reaction (4). Comparing results A and C in the above table, we find that a difference of one unit-weight of sulphur oxidized replaced 0.166 coke instead of 0.53 or 0.4 respectively, as indicated by theory. Comparing A and B, which are rather close together, we find that one unit of sulphur replaced 0.188 part coke. Here are practical results of great uniformity, as such evidence goes, in the fuel relation between the sulphur and coke. It must not be forgotten that the grade of the matte is progressively varying in the above tabulated results, and also that some of the oxidation of the sulphide was taking place by means of the reaction between the ferric oxide of the roasted ore and the sulphide which in equation 4 equals 2.66 Cal. per unit-weight of sulphur or 0.4 coke containing 83% carbon. Of course the unit-weight of sulphur in question was producing heat in the furnace equal to 0.53 or 0.4 coke, according to whether it was oxidized by solid or gaseous oxygen, but it did not replace the amount of carbon it should have done. Only an examination of a thermal balance sheet would reveal why it did not do so, and it may be that by nursing the coke when the increased amounts of pyrite were put on the charge, less coke might have been used, and it may be that other metallurgists using cold blast may find a more favorable practical relation between the sulphur and the coke in the

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*Oxygen chemically combined.

case of mixed charges if their attention is called to the facts above cited.

I will now give a similar fuel comparison between sulphur and coke in the case of warm blast. A recent writer on this subject has suggested that it would be more appropriate to call our heated blast warm rather than hot, and there is the more reason for agreeing with him because it does seem that a moderate heating of the blast serves all that is needed of it and that there is no need to use the high temperature employed in iron smelting and obtained with brick stoves. In the following table are given the results obtained with warm blast with three furnaces similar in design and operation. Each result is the average of six months' work with the same furnace. The three furnaces were working under the same conditions and exhibited much greater regularity in respect to coke consumption day by day than was shown by the same furnaces when previously working under cold blast.

RELATION BETWEEN SULPHUR AND COKE, USING WARM BLAST.

	Sulphur in FeS, %	S in FeS_2, %	Total sulphur, %	Coke, %	Matte, Cu, %	Blast, oz.	Rate of smelting: Tons per sq. ft. per 24 hr.
E	5.51	15.16	20.67	4.0	31.0	28.4	8.58
F	7.20	12.60	19.80	4.5	31.6	28.6	8.41
G	7.15	11.30	18.45	4.8	31.4	28.4	8.47

It will be noticed that, as in the case of cold blast, the coke consumption diminishes with addition of sulphur to the charge. Special attention was given to the coke consumption to reduce it to a minimum. The furnaces ran with great regularity and very fast, as will be seen from the average rate of smelting. It is calculated that 71% of the sulphur was oxidized to SO_2. On comparing the extremes, it is found that a difference of one unit-weight of sulphur oxidized is equal to a difference of 0.508 coke. This is certainly a surprising result, in contrast to those obtained with cold blast. These relations, which were not discovered at the time, serve to illustrate the value of warm blast, which, by increasing the temperature in the smelting zone, insures more active and regular smelting of the charge.

The above results also enable us to calculate the vanishing point of carbonaceous fuel on a pyrite charge. How much additional sulphur should replace the 4% coke used when 20.67 units of sulphur existed in 100 units of charge and 14.67 units of sulphur were oxidized, or how much sulphur must be oxidized per 100 of charge to enable the coke to be entirely eliminated. From the extremes in the above table, we calculate that an additional 7.87 units of sulphur oxidized would, if the same relation between sulphur and coke continued, be sufficient to replace all the coke That is, other things being equal, 14.67 + 7.87 or 22.54 of sulphur oxidized per 100 units of charge should enable the charge to be self-smelting. This is precisely what was later found to be possible, and often the furnaces ran with just that amount of oxidation. It is better, of course, to have more, say 25 units of sulphur per 100 of charge.

In setting forth furnace results for such a discussion as this, I find it convenient to have the average charge stated and not the official charge, which is varied from time to time; it is also cumbersome for the purpose of calculations and comparisons to have the charge stated, as it often is, in some odd number of pounds as it is given to the weighers, the foreman taking off or adding on a few pounds of the different constituents of the charge from time to time according to the working of the furnace and the class of material used, and the men perhaps using more coke than is accounted for on the charge-sheets—for is not the coke-pile handy?

Having shown the relation between carbon and sulphur as fuels in copper blast-furnace practice, I would like to draw attention to the thermo-chemical aspects of the cases cited, believing that such evidence as exists, even if but approximate in some of its details, will be instructive. Again I must refer to iron smelting in blast-furnaces because it has been extensively studied for many years by highly competent observers and the heat balance-sheets prepared by them may serve as a model for those who desire to emulate an admirable example, but in so doing I want to emphasize the fact that the two operations of iron smelting and pyrite smelting are chemically opposite processes, though similar in their main physical features. The first is a reducing process, the iron oxide being reduced to metal by the agency of carbon monoxide. And the carbon is not merely the only available fuel for the physical effects of smelting, etc., but it is necessary for the chemical work of the furnace. In the smelting of pyrite, on the contrary, we require to oxidize and not to reduce, and carbon, however useful it may be in raising temperature, remains counter-active to the chemical requirements of the problem.

For centuries copper metallurgists have been groping with the irrational use of carbon as a fuel, allowing it to impede the work of roasting by its reducing action and counter-active effect on the required oxidation. There is no doubt that by high development of the muffle principle, much of this counter-active effect might have been avoided in the reverberatory work in the old practice of smelting sulphide ores after preliminary oxidation. We have learned that sulphur will replace coke, as imagined and almost demonstrated by Hollway, whose method of first melting the sulphide in a cupola with coke and then converting it in a bessemer converter is precisely what we now practice with success, with the additional advantage that by promoting the oxidation and performing the first melting without carbonaceous fuel or its minimum, we combine two highly economical pneumatic processes and conduct both by the heat of combustion of the iron sulphides without the counter-active influence of carbon.

It is usual to state iron furnace results per unit-weight of pig iron. For the purpose of illustration, I have taken a case from James Gayley's practice as studied by Joseph W. Richards (*Trans. Amer. Inst. Min. Eng.*, p. 703, July 1905), and have arranged his figures according to a unit-weight of charge and not according to the weight of the valuable product produced by the furnace. I also use the big Calorie in preference to the small one.

IRON BLAST-FURNACE PRACTICE, WITH MOIST AIR.

Heat distribution.	Calories per unit-weight of charge.
Reduction Fe_2O_3	0.738
Reduction SiO_2	0.030
Expulsion of CO_2 from limestone	0.081
Decomposition of water in blast	0.065
Evaporation of moisture in charge	0.051
Heat in pig iron	0.112
Heat in slag	0.105
Heat in escaping gases	0.198
Radiation, etc., and undetermined	0.348
Total	1.728
HEAT DEVELOPED.	
C to CO	0.591
C to CO_2	0.959
Heat in blast	0.178
Total	1.728

By studying such a table of results we can cultivate our sense of proportion and realize at a glance what particular items cause the largest heat expenditures; we can also learn to distinguish those factors in which improvement is possible from those that are, like the reduction of the ferric oxide, inevitable to the process. We see how much heat is lost in escaping gases and in the molten iron and slag. In order to contrast this practice with pyrite smelting without coke, I have prepared the following table in order to show to what extent the two processes resemble or differ from each other.

TABLE A—PER POUND WEIGHT OF CHARGE

	Iron blast furnace.	Copper pyrite smelting.
Air used, volume	26.4 cu. ft.	33.00 cu. ft.
Air used, weight	1.9 lb.	2.43 lb.
Slag produced	0.26 "	0.55 "
Pig produced	0.45 "	....
Matte produced	....	0.23 "
Charge smelted per square foot of hearth-area per minute	8.7 "	12.5 "
Blast-pressure per square inch	15.0 "	2.0 "
Height of ore-column	70.0 ft.	6.0 ft.
Temperature of escaping gases	230°C.	800°C. (estimated)
Calories per unit-weight of charge developed or expended	1.728	1.082
Critical working temperature	1,510°C.	1,250°C. (estimated)
Temperature of blast	382°C.	222°C.

In contrasting the two processes we notice that the copper furnace takes more air, but at a much lower pressure, the last because the height of ore-column is so much less. The slag in pyrite smelting is more than double the other, but the matte is in weight but one-half as much as the pig produced by the iron furnace. The copper furnace drives about 50% faster. The temperature of the escaping gases is unfortunately very much higher in the copper furnace because of the low ore-column, the faster driving, and the uncomfortable fractional combustion of the sulphur of the pyrite. The temperature of the escaping slag and metal is higher in the iron furnace because of the high temperature of fusion cast iron, as compared to that of copper matte. The temperature of the ingoing blast is much higher in iron smelting and the total heat required is greater, because of the large amount of endo-thermic chemical work to be performed. The numbers given in Table A will serve for the preparation of a heat balance-sheet for pyrite smelting, per unit-weight of charge. The important items are as follows:

Heat in Escaping Gases.—The temperature of these was not determined, but estimated, from the color of the heated particles of flue-dust on the top of the furnace, at 800°C. It is an unfortunate feature of pyrite smelting that the furnace-top is so hot, because, as explained in a previous article, the sulphur of the charge burns in a fractional manner and partly toward the top of the furnace. The temperature of the gases being high enough to agglomerate some of the flue-dust, this, in falling back to the sides of the furnace helps in building the accretions which form the scaffold, the closing up of which eventually causes the shutting down of the furnace. The specific heat of the gases is calculated for a gas of the following composition:

SO_2	0.100
O_2	0.058
N_2	0.842
Total	1.000

On an average, it is probable that the percentage of SO_2 in the furnace gases would be less than 10%. Allowance has been made for the generally neglected increase of specific heat at increasing temperatures. The mean specific heat of a gas of the above composition was found to be 0.392 at 800°C. or 0.00049 Cal. per unit-weight of charge per 1° Centigrade.

Heat in Slags.—This has been calculated for a temperature (estimated) of 1250°C. The specific heat of the slag was found to be 0.00016 Cal. per 1°C. (between 0° and 100°). The increase in the mean specific heat was assumed to be 0.07% for each degree and the latent heat of fusion given by Voigt for 2 (FeO) SiO_2, namely, 0.085 Cal., was taken. The total heat in the slag at a temperature of 1250°C. was thus calculated to be 0.46 Cal. per unit-weight of slag; $0.55 \times 0.46 = 0.253$ Cal.

Heat in Matte.—The specific heat of this between zero and 100°C. was found to be 0.00016 Cal. The mean increase with increased temperature is estimated at 0.000064 per degree. The latent heat of fusion was calculated for a mixture of ⅓ Cu_2S and ⅔ FeS by the empirical rule. In that manner the total heat in the matte was calculated to be 0.33 Cal. per unit-weight of matte for the estimated 1250°C. or per unit-weight of charge; $0.23 \times 0.33 = 0.076$ Calorie.

Heat in Cooling Water.—This was determined and found to vary between 0.06 to 0.13 Cal. per unit-weight of charge, according to thickness of crust and rate of running of furnace. The mean loss, namely, 0.095 Cal., is taken.

The Heat in the Moisture of the Charge.—The average moisture was 3.5% and this is consequently 0.035 by 0.6 Cal. or 0.021 Cal. per unit of charge.

The decomposition of the limestone is 0.04×0.451, or 0.018 Cal. per unit of charge.

The decomposition of water (moisture) in the blast is calculated for ordinary atmospheric conditions.

Some allowance for loss of heat by radiation is necessary.

For the heat developed at the rate of 0.25 unit-weight of sulphur oxidized and adopting the heat of combustion of FeS, namely 3.55 Cal. per unit-weight of sulphur, we have,

$$0.25 \times 3.55 = 0.89 \text{ Cal.}$$

The heat of combination of $2(FeO)$ and SiO_2, I do not know. That of FeO plus SiO_2 as given by Prof. J. W. Richards ('Metallurgical Calculations'—*Electro-Chemical and Metallurgical Industry*, March, 1905) is 0.189 Cal. per unit of Fe. For the slag in question, which approximates $2(FeO)\ SiO_2$, 60% of the above value has been taken.

$$0.189 \times 0.6 \times 0.22 = 0.025 \text{ Cal.}$$

Heat in the Blast.—This is $0.303\ (222°) + 0.000027\ (222°)^2 \times 2.43 \div 1000$, to reduce to big Calories, $= 0.167$ Cal.

These calculations are made, as is usual, from a temperature base-line of 0°C., and I have added the equivalent percentage-weight of coke that would furnish the amounts of heat in question.

A summary of these items gives per unit-weight of charge:

Heat expenditure.	Calories.	Coke equivalent % Calories × 100 6.71
Escaping gases	0.392	5.84
Heat in slag	0.253	3.77
Heat in matte	0.076	1.13
Heat in cooling water	0.095	1.40
Evaporation of moisture in the charge	0.021	0.31
Decomposition of limestone	0.018	0.27
Decomposition of water in blast	0.073	1.09
Radiation and difference	0.154	2.30
Total	1.082	16.11
Heat developed.		
Combustion of sulphides	0.890	13.26
Combination of FeO and SiO_2	0.025	0.37
Heat in blast	0.167	2.48
Total	1.082	16.11

We see that the heat in the blast is only about 40% of the heat in the escaping gases. A hot-blast stove heated by these escaping gases should easily furnish the temperature required, but it would have to be so arranged as not to impede the escape of the gases to the stack or interfere with the operation of the furnace. It is obvious that the slag produced rises or falls with the oxidation of the iron sulphide and the effect of changes in these particulars can readily be estimated.

On first investigating the heat expenditure by inferential methods I was inclined to believe that the work of a pyrite furnace could be carried on with less than the above amount of heat. I had not allowed enough for the hot top, so persistent and annoying with all pyrite charges.

To give the values of the segregated items of heat-expenditure a form entirely practical, I have annexed to each the equivalent percentage weight of coke of 83% carbon burned to carbonic acid that would produce the amount of heat in question. This is obtained by dividing the calories per unit-weight of charge by 6.71, which is the calorific value of the coke (see equation 7), and multiplying by 100 to bring the result to a percentage.

The fundamental principles of thermo-chemistry, such as the conservation of matter and energy, are like those of other established sciences, beyond cavil. Certain numerical data, such as the specific heat of a substance, may be open to a little doubt, but this doubt rests generally within small limits that can be neglected in practice.

The heat in the cooling water takes an amount of heat equal to a fuel consumption of about 1.5% coke. The loss of heat in the escaping gases is serious, but even if it could be reduced one-half by discharging them at a temperature a little above that of the iron blast-furnace with its 70 ft. of ore-column, the loss of heat in that manner would equal about 2% coke. These heat-losses are inevitable. A furnace making much slag loses proportionately more heat. A study of the heat-losses of a furnace is instructive, since it is possible to calculate fairly closely what they may be for any case. To follow this path of inquiry a little further, we may ask what would be the heat requirement in terms of coke (83% carbon) for a charge consisting only of slag. This is the simplest case we can imagine. The chemical work is nothing and we have only to do with physical properties.

In the iron furnace the radiation losses are generally included with the 'difference' or 'undetermined.'

The heat in the cooling water may be equal to 2% coke.

That lost by radiation is perhaps as much. The heat in the escaping gases will hardly be much less than 4% coke. The simple heat in the melted slag will require about 6% coke. Thus, in order to melt a neutral substance like dry slag in a water-jacket furnace, an amount of heat will be required in terms of coke percentage, thus:

Heat in slag	6.0
Heat in cooling water	2.2
Heat radiated	1.5
Heat in decomposing water of blast	1.0
Heat in escaping gases	4.0
Total	14.7

According to the accepted heat of combustion of FeS, this amount of heat is equal to the combustion of 27.5% of the charge as sulphur. (The higher amount of sulphur in this case over that of pyrite is because one unit-weight of slag is produced per unit-weight of charge.

The metallurgist has to face the facts, and no amount of ingenuity on his part will enable him to produce matter and energy from nothing. There will be a minimum heat-expenditure, below which he will not be able to go in any particular operation. His concern is not to exceed this irreducible minimum.

It has been shown in these articles that the smelting of pyrite has been carried on without coke with warmed blast when the sulphur oxidized equalled from 23 to 25% by weight of the charge, and also that the heat of combustion of the oxidized sulphur supplemented by a small addition to the temperature of the blast, accounted for the necessary losses of heat from the furnace. It has been further shown that with warmed blast and when the amount of sulphur oxidized was from 13.1 to 14.9% of the charge and a small percentage of coke was employed, the fuel-relation between sulphur and coke approximated that indicated by theory; and also that the vanishing point of coke on the charge with additional sulphur oxidized, agreed with that calculated from the relation discovered in practice and very closely with that suggested by theory.

Thus, as regards quantity of heat, theoretical considerations confirm and explain the practical results. There still remains another point of view, and that is the *degree of temperature* as apart from quantity of heat. For example, we may have an infinite supply of heat at a temperature one degree lower than that of the point of fusion of copper matte and never be able to melt it. This point of view of temperature suggests two important questions; what is the working temperature of a copper-matting furnace? and what is the temperature of combustion of the iron sulphide?

The working temperature must be higher than that at which the matte and its accompanying slag solidify. The point of fusion of copper matte is about 1180°C., and that of the accompanying slag 1100°C. The working temperature must therefore be higher than 1180°C.; I have judged it to be 1250°C.

Now we have to determine the temperature of combustion of the iron sulphide. Pyrite does not burn homogeneously, and we are therefore not concerned with its temperature of combustion, but with that of the residual iron sulphide (FeS) left on the heating of the pyrite. (It may be that the iron sulphide in question may be not precisely of the exact formula, FeS, but much more complex, though closely approximating to such formula, that moot point will not affect this technical discussion.)

To calculate the temperature of combustion of FeS burning in air, we must know what excess of air is employed. I have already shown that an excess is required. Analyses of the furnace gases taken from the immediate top of the furnace charge and in such a manner that no air from above the top of the charge could contaminate the samples, gave from 5 to 10% of SO_2. Calculation shows that FeS burning to FeO and SO_2 in 50% excess of air would give a gas of the following composition: 9.7% SO_2, 7.4% O_2, and 82.9% N_2. The temperature of such a gas would be raised by the heat of combustion of sulphur (heat equation No. 2) from 0° to 1319°C. With 100% excess of air, the gases would consist of: 7.24% SO_2, 10.7% O_2, and 82% N_2, and their temperature would be 1032° Centigrade.

It will be apparent that with a range of air-excess such as is revealed by the composition of the furnace-gases, the temperature of combustion of the iron mono-sulphide is likely either to be below the melting point of the copper-matte or dangerously near it, and there need be no surprise if our attempts to smelt pyrite without coke and with cold blast failed. The temperature of combustion of the iron sulphide can easily be supplemented by raising the temperature of the blast or by using some coke; but the latter will have a counter-active effect upon the oxidation of the pyrite and this we desire to avoid.

Thus the use of warm blast is a logical sequence of a theoretical examination of the heat and the other reactions involved; and these in their turn explain our practical experience that, without coke and with cold blast, pyrite is not self-smelting.

THE PROBLEM OF FINES

(Editorial, Oct. 5, 1912.)

The treatment of finely divided material offers one of the greatest problems in metallurgical work. In ore dressing the smallest particles offer the greatest resistence to separatory processes, and in cyaniding the fine material for a long time, and even at present to a certain extent, presents great difficulties in filtering and washing, while the colloids, those curious denizens of the border land between solid and liquid, have only recently been rendered amenable to treatment. In the smelting of iron and copper ores the use of large quantities of finely divided ore in the blast-furnace is a vexing problem to the metallurgist, a problem that persists to the end of metallurgical work, for the gases escaping from the stack carry with them fine solid particles that are both a source of loss to the smelter and of annoyance to its neighbors.

It is, however, to the handling of fine ore in smelting operations that we propose to confine ourselves, referring more particularly to the smelting of copper ores. Fines have always been a problem in this work, and even in the days of the heap-roasting of lump ores the disposal of the small amount of fine ore unavoidably produced in mining was a source of worry to the metallurgist. The early conditions now seem almost like an iridescent dream to the many operators who are compelled to handle concentrate alone, or with but a small proportion of lump ore. A large and increasing part of the output of copper is derived from finely disseminated low-grade sulphide ore which must first be subjected to mechanical concentration. Smelting concentrate, practically all smaller than ⅛ inch in diameter, and unmixed with lump ore, is an irritating problem. The Gordian knot can be quite simply cut by roasting the concentrate and smelting the calcine in reverberatory furnaces, but only at a high working cost. At a plant where coke costs $10 per ton the sulphur in concentrate containing 50% of pyrite is worth $2.50 per ton as a source of heat in smelting operations, and to be obliged to roast it off at a cost of about 30 cents per ton is heart-rending. In addition, the smelting cost per ton is higher in reverberatory work, though recent advance in the use of liquid fuel has increased the tonnage smelted and lowered costs. Progress in the use of gaseous fuel is of the negative order, for at the one copper smelter in this country where it has been in use, the Great Falls plant of the Anaconda Copper Mining Company, the use of gas producers will be abandoned in the new construction now under way, and grate firing substituted. In this case the high ash of the coal used and the dust from the charge gave trouble in the regenerative checkerwork, another instance of the fines problem.

For handling fine ore in smelting, a great variety of ingenious methods have been devised. Since sulphides are readily fusible it would perhaps seem simplest to melt them and allow them to consolidate on cooling. Like most simple suggestions, this is impracticable. Melting and cooling sulphides, while retaining their sulphur content, offers annoying mechanical problems and can only be done at a prohibitive cost. But to fuse the fine ore to such a degree that it coheres sufficiently to act like lump ore in the blast-furnace is more practicable, and is the line upon which the most recent and satisfactory progress has been made. In this work the problems are essentially analogous to those of coke-making. The mixture must be fused sufficiently to make it cohere strongly enough so that it will not break up too much in handling nor crush under the weight of the burden in the blast-furnace. The resulting product should be porous in order to aid the penetration of the blast and to facilitate the reactions of smelting. The complexity of this single group of problems is too little realized. In coking, the components present vary in degree but not in kind, yet the most careful study of the mixture of coal and water fed to the ovens is required to procure a satisfactory product. In smelting, the components vary in kind as well as in degree, and the amount of study of each particular case thus required is correspondingly greater. It is not remarkable, therefore, that the attempt to use such methods has often been abandoned after a short period of unsuccessful experimentation. Smelting differs radically from other kinds of chemical engineering work in that it is essentially variable in its character. A variety of different products must be utilized, and their relative proportions vary from week to week because of conditions not under the metallurgists' control, and methods which would be adequate to solve a constant problem may fail utterly before a variable one. This is the fundamental problem in the use of agglomerative methods for handling fines, and both in sintering on grates and in pots the securing of a mixture which shall be uniform throughout, both in a single operation and during the day's work, is one of the criteria of success.

In the treatment of lead ores agglomeration by fusing in pots and on grates is carried out with marked success and has reached a higher stage of development than the agglomeration of copper ores. The conditions to be met differ with the two methods. With lead ores the endeavor is to eliminate sulphur, and the ore is commonly subjected to a light roast before sintering in pots, while with copper ores the sulphur should be preserved to as large an extent as possible. The hot zone of oxidation that passes through the mass to be sintered is easily controlled if a relatively small quantity of sulphur is present and can be allowed to oxidize nearly completely, but is rebellious where there is present a large percentage of sulphur, only a part of which can be allowed to oxidize. But this problem is essentially similar to that in coking, where excessive fusing is checked by the addition of a suitable amount of moisture, and it may eventually prove that the sintering of fine copper sulphides on grates is more easy to perform than has been supposed. The sintering of flue-dust, with or without a small quantity of coke dust, is now successfully done both at the Pittsmont and Mason Valley copper smelters, and the sintering of fine ore may follow as the next step in development.

Disposal of fine sulphides by adding them to the converter charge has often been proposed, but has not attained any marked success. Aside from the difficulties it gives rise to, it is evident that it cannot offer any universal solution of the problem, since the larger proportion of the ore must be otherwise treated.in order to produce the matte for converting. Briquetting in its various forms is as old as the industry itself, and has been a continuous source of vexation. Under certain conditions it may be applied with marked success, as at Anaconda, but the hope of escape from its annoyances is a constant lure to the metallurgist, and there is little hope of satisfactory progress in the improvement of briquetting. Special methods serve to meet special conditions, especially where labor is cheap; thus flue-dust is mixed with converter slag at Cananea, and the nodulizing method recently described in these columns is satisfactory at Shisaka, in Japan, but everywhere the hope of devising a more satisfactory method is a potent force making for progress.

SINTERING SULPHIDE ORES AT HIGH ALTITUDES

By R. L. Lloyd

(June 14, 1913.)

The results obtained by the Dwight & Lloyd sintering machines installed at the Cerro de Pasco smelter in Peru, at an altitude of 14,000 ft. above sea-level, have shown that sintering by this method at high altitudes is confronted by no more serious problems than are experienced in blast-furnace smelting at the same altitudes.

At such altitudes as 14,000 ft. there is always some difficulty in kindling a fire of any kind, and any burning material tends to go out if the conditions most favorable to combustion are not maintained. As was to be expected therefore, a sinter bed is not quite as readily ignited at Cerro de Pasco as at lower altitudes, but no serious difficulty was experienced in this regard; the ignition flame was simply made a little more intense, a condition readily attained with the crude-oil burner that was used. The ore charges treated were in all respects similar to the copper charges treated elsewhere, except that the sulphur content was higher than usual (25%). With such a sulphur tenor, considerable difficulty is generally experienced at lower altitudes, on account of the excess of the heat of combustion of the charge, but at an altitude of 14,000 ft. no difficulty at all was met. This was probably because of a tendency toward slower combustion, it being remembered that a cubic foot of air at 14,000 ft. contains only about 60% of the oxygen that a cubic foot of air contains at sea-level.

As is well known, sulphur is sometimes produced while sintering, and is carried through the fan with the gases as flowers of sulphur. At lower altitudes, if the exhaust gases are quite hot, the sulphur sometimes tends to collect in the fan, and if not cleaned out at regular intervals may take fire. No such difficulty was experienced at Cerro de Pasco, and although the sulphur appeared in the fan and was cleaned out at regular intervals, there never seemed to be any tendency to take fire, probably because of the low oxygen tenor of the atmosphere. No attempt was made at Cerro de Pasco to find the low sulphur limit for sintering, as the problem was to work the highest sulphur charges possible, and while it was proved that 25% was an entirely feasible charge at that place, it is not yet proved to be the maximum permissible.

The speed of sintering was rather surprising, and seemed to be almost if not quite equal to that at lower altitudes. This I take to be, partly at least, due to the large amount of sulphur driven off in the elemental state, as the ore contained much pyrite. It is hard to get an exact comparison of speed, however, as no such high sulphur charges are being worked elsewhere by this method. The power consumption per ton sintered was practically no greater than that used at ordinary altitudes, and the ordinary vacuum of 4 to 6 oz. per square inch was found to be entirely satisfactory.

Sintering by the Dwight & Lloyd system had been previously tried at Cerro de Pasco, in a plant erected in 1907 under an arrangement with the patentees, but designed independently. The plant was of the intermittent 'pan' type and, proving unsatisfactory, was eventually abandoned. The cause for this failure is entirely clear when the mechanical difficulties of properly placing a sinter bed on an apparatus of the pan type, and of re-establishing the best working conditions for every batch are realized. Such difficulties do not exist while practising the continuous system of sintering as is now satisfactorily employed at Cerro de Pasco and many other places. Steps have been taken to enlarge the existing plant at Cerro as rapidly as possible, so as not only to eliminate fine ore troubles, but also to get the great increase in smelting speed which invariably follows the addition of sintered material to the charge.

COPPER SMELTING IN THE ELECTRIC FURNACE

By Dorsey A. Lyon and Robert M. Keeney

*In presenting this paper the writers wish to call attention first of all to the fact that the electric furnace was not developed as a competitor of the combustion furnace, but:

1. For the purpose of doing high-temperature work which it is not possible to do in the combustion furnace; and

2. For the treatment of ores in regions where fuel is scarce and costly, and where hydro-electric power is comparatively cheap, as is the case in Chile, Canada, in certain parts of this country, and Mexico. This paper is not presented with the idea of trying to prove that the electric furnace should replace the reverberatory or the blast-furnace as used at present in the smelting of copper ores, but that it may be substituted under conditions which are not favorable to them, and where either of the two following conditions may exist:

(a) A district remote from railway transportation facilities,

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*Excerpts from 'The Smelting of Copper Ores in the Electric Furnace,' presented at the Butte meeting of the American Institute of Mining Engineers, August 1913.

where deposits, rich enough to warrant their being worked, due to the cost of transport of coke and ore, have to remain unworked, but where there is plenty of water power which can be developed at reasonable cost for the production of electric power, by the use of which for smelting the ores in an electric furnace, a concentrated product may be obtained, which can stand the necessary transportation charges.

(b) Where the transportation facilities may be good, but the cost of fuel too great to permit the use of the ordinary methods of smelting, whereas, on the other hand, hydro-electric power can be developed at a low cost, and the ores could be smelted by the electric furnace at a cost that would make their treatment possible.

With these possibilities in mind we have attempted to make a brief comparative study of the problem for the purpose of determining whether it would be possible metallurgically, and feasible commercially, to use the electric furnace in those localities where, by reason of excessive fuel costs, the cost of operating ordinary furnaces for smelting copper ores is either excessive or prohibitive.

Although the smelting of copper ores in the electric furnace has received considerable attention and more or less experimental work has been done, there are, so far as we are aware, no electric furnaces in the United States on copper ores. In Norway, however, trial smelting of copper ores with a 1000-hp. electric furnace and an estimated operating capacity of 2000 tons per year, has been conducted at the Ilen Smelting Works, Trondhjem. A consignment of 25 tons of pure copper, the first copper ever produced by electric smelting in Norway, was exported from the Bitovaria Minea at Kaafjord, near Dyngor. Some months ago it was reported that, at Christiansand, the electric smelting and refining of nickel and copper would be undertaken by the A/s Kristiansands, Nikkelroffeneringsverk.

1. **Vattier's Experiment.**—One of the best known of the experiments which have been conducted upon the smelting of copper ores in the electric furnace is that which was made by Vattier at the Livet works of the Compagnie Electrothermic Keller et Leleux, France, April 23, 1903.[1] These tests were made by Vattier for the Chilean government, which wished to secure such data as it would be possible to obtain in making experiments on a commercial scale. These tests were made before a Commission, which was composed of some of the best known metallurgists of England and Europe.

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[1] For a full account of this work see Report of the Canadian Commission, 1904.

As stated by Vattier, the main object of these experiments was to determine, if possible, whether electric heat could be substituted for the heat derived from coal, either wholly or in part. This is especially important to Chile, for it has plenty of valuable ores, and, so it is stated, an abundance of water power, but coal is costly.

The experiments were conducted on two different kinds of ore:

1. An ore the copper content of which was approximately 7%, present principally as copper pyrite. It also contained 8 to 9% of sulphur, and the gangue was made up mainly of micaceous copper oxide, together with silicates and a little carbonate of lime.

2. Low-grade copper ore from the mining regions in the vicinity of Santiago, Chile, mixed with a small proportion of manganese and lime.

The composition of the ore, matte, and slag was as follows:

	Ore, %	Matte %	Slag, %
SiO_2	23.70	0.80	27.20
Al_2O_3	4.00	0.50	5.20
CaO	7.30	....	9.90
MgO	0.33	1.40	0.39
CO_2	4.31	....	....
S	4.12	22.96	0.57
Fe	28.50	24.30	32.50
Mn	7.64	....	8.23
P	0.04	....	0.06
Cu	5.10	47.90	0.10
As	Trace	....	....

2. Wolkoff's Experiments.—In some experiments that were made by Wolkoff[2] on the smelting of copper ores, the ore experimented with was a sulphide, containing 8.20% of copper, with an acid gangue. Several experimental runs were made, but only the results from one or two of them will be mentioned in this connection. For instance, using 12 kg. of ore (25.5 lb.) and with the addition of 6.2% of hammer scale to furnish iron oxide for the silicious gangue, he obtained a thoroughly fused product, the matte containing practically the whole of the copper, the slag retaining only 0.15%, and he also states that the volatilization losses were very small.

In another experiment, he smelted 10 kg. of ore (22 lb.) with 1.25 kg. (2.74 lb.) of roasted matte, which contained 75% CuO, 8% Cu_2S, and 15% Fe_2O_3, the object being to determine the extent of the reaction, $Cu_2S + 2CuO = 2Cu_2 + SO_2$. On smelting this charge with a current of 400 amperes and 75 volts for half an hour,

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[2] *Metallurgie*, Vol. VII, pp. 99 to 102.

Wolkoff obtained 1.78 kg. of crude copper with 92% Cu, 3% Fe, and 1% S. He also obtained 10.9 kg. slag with 0.10% Cu. In this experiment 96% of the metal was extracted; the slag contained 0.60% of the total quantity of copper; therefore, as stated by Wolkoff, the copper balance is as follows:

The charge contained		1.706 kg. Cu
1.78 kg. crude copper with 92% Cu =	1.638 kg.	
10.9 kg. slag with 0.1% Cu =	0.011 kg.	
Losses	0.057 kg.	
		1.706 kg. Cu

3. **Experiments of M. Stephan.**—At the first general meeting of the recently formed 'German Metallurgical Engineers,' M. Stephan, superintendent of the Girod Electric Steel Works at Ugine, France, gave an account of some experiments made by him on the electric smelting of non-ferrous ores, one of which was copper. The follwing is an abstract of Stephan's paper:[3]

"The ore came from the Belgian Congo, being mined by a Belgian-English concern. In five different analyses given, the CuO varies from 21.01% to 5.73%; SiO_2 from 28.48% to 78.55%, with from 4 to 13% Al_2O_3; from 4 to 16% Fe_2O_3, and, besides, smaller amounts of impurities, from 2 to 7%. CoO; no nickel. * * * The moisture in the ore varied from 7% to 32%. It was not removed, in order to meet the conditions of practical operation and to carry the experiments out under most unfavorable conditions. * * * Charcoal, coke, and anthracite were used successfully as reducing agents. Charcoal would probably be the cheapest in this particular instance. * * * Electric furnaces, similar to the Girod ferro-alloy furnace type, were used and the dimensions changed in several runs within wide limits, in order to secure enough data for the construction of a large furnace for the same work. Electrodes were used suspended from the top and inserted in the bottom, also in the sides for heating by radiation, and a system was adopted of heating with a smothered arc mainly by the resistance of a thick layer of slag over the conducting metallic charge. The temperatures were measured with LeChatelier, Wanner, and Fery pyrometers.

"A slag of the composition: SiO_2 51.90%; Al_2O_3 11.31%; CaO 16.83%; MgO 13.71%; Fe_2O_3 3.55%; MnO 0.94%; CuO 0.64%; CoO 0.87% begins to melt at 1250°C.; begins to flow in a pasty condition at 1350°C.; and is liquid enough at 1400° to allow the little copper

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[3] Einiges Uber die Erzeugung von Metallen in elektrischen Ofen, von M. Stephan, *Metall und Erz*, October 1913; abstract in *Metallurgical and Chemical Engineering*, p. 22 (January 1913).

balls to settle completely. At 1550°C. the slag is liquid enough to flow off freely; 1920°C. was necessary to render the slag liquid when the highly acid ore was smelted without any fluxes. The pig copper (*Schwarzkupfer*) produced analyzed in six different runs from 65 to 95% Cu, containing from 1 to 21% Fe, and from 1 to 11% Co. The purity of the product will be higher the lower the smelting temperature. The lower the temperature the smaller will be the chance for reducing any of its impurities, but, at the same time, more copper will also be retained in the slag, since at a low temperature it is much less fluid and offers more resistance to the settling of the copper, so that the output is decreased.

"It will be a matter of commercial calculation, according to the composition of the ore and other special conditions, whether a high-quality product or a maximum output will be desirable. * * * A continuous run for several days with the same ore, aiming at slags of about the composition mentioned above, required 1000 to 1200 kw. hr. per ton (2205 lb.) of ore. * * * These figures are high, as a result of the amount of heat required to keep the very viscous and abnormal slags in liquid condition. With an easily fluxed ore, the specific power consumption was only about 500 kw. hr. The electrode consumption averaged 8 kg., or 17.6 lb., per ton of ore. It was operated with 4 amperes per sq. cm. (25.8 amperes per sq. in.). Coal for reduction was used at the rate of 25% of the copper in the charge. The best lining to withstand the severe conditions of the furnace used in this work was tamped fire-clay with 80% SiO_2 and 15% Al_2O_3."

Smelting of Sulphide Ores in the Electric Furnace

To study conditions in the smelting of sulphide ores in the electric furnace, the writers have recently conducted a series of experiments at the Bureau of Mines laboratory. The objects in view were: (1) to determine if there are any conditions which make electric smelting without air anything but a simple melting operation; (2) to note the percentage of concentration and the sulphur removal; (3) to study the possibility of condensation of the elemental sulphur as such; (4) to get general figures on power consumption with varying charges; (5) to determine gold, silver, and copper losses; and (6) to study the use of a low-copper matte as a collecting agent for gold and silver.

1. **Ores Treated.**—The ores used in the experiments were: A heavy sulphide low-grade copper ore; a gold ore; a silicious silver ore; and some roasted ore of the composition shown below. The limestone contained 63.2% CoO and 5.7% MgO.

	Pyrite. %	Nodule. %	Silicious ore. %
SiO_2	2.50	4.00	74.53
Fe	44.07	65.60	10.71
S	48.20	....	8.36
Al_2O_3	0.24	1.76	0.40
CaO	0.08	0.45	....
MgO	0.73	0.46	0.26
P	0.02	....	....
As	0.20	....	....
Cu	1.30	0.07	....
Au, oz. per ton	0.01	0.03	1.28
Ag, oz. per ton	0.14	0.05	3.20

The ores were treated in an electric furnace of the following construction, figures 15 and 16: It consists of a smelting chamber lined with fire brick or silica brick, having three carbon electrodes suspended vertically through the roof with a feeding shaft lined with

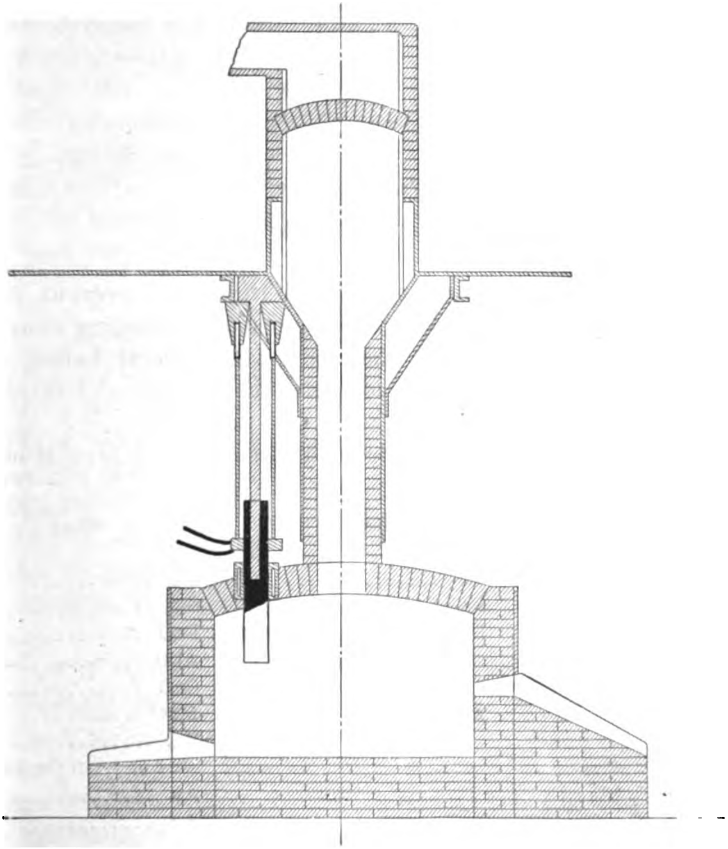

FIG. 15. ELEVATION OF ELECTRIC FURNACE.

fire brick over the crucible. This would use three-phase current at from 50 to 100 volts. The general internal dimensions of the crucible are 14 ft. 6 in. in diameter, by 9 ft. high, with a shaft 1 ft. 6 in. in diameter by 18 ft. 6 in. high. With a furnace of this type regulation of electrodes would not be necessary after regular operation was established, so that the electrical connection is made close to the roof of the furnace and regulation is by hand. There is less difficulty in handling the electrodes if they are suspended vertically; and, as the shaft is low, it is made vertical, thus eliminating the necessity of suspending them at about 55 degrees.

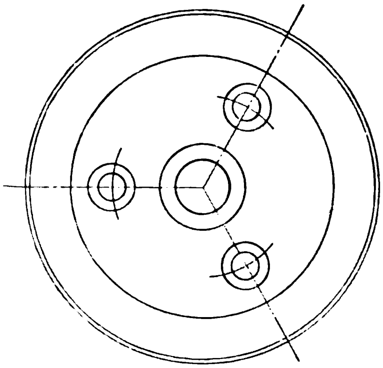

FIG. 16. PLAN OF ELECTRIC FURNACE.

The top was roofed and kept closed tightly to prevent escape of sulphur and admission of air. A condenser, consisting simply of three rectangular, horizontal chambers with several baffles in it, was attached to the furnace to condense the sulphur and catch any escaping gas.

Twenty experiments were made with these ores. A typical charge was as follows:

CHARGE	
	Lb.
Pyrite	8.80
Nodule	13.20
Silicious ore	5.76
Limestone	2.73

MATTE	
	%
Cu	1.22
Fe	64.18
S	22.38
	Oz. per ton.
Au	0.72
Ag	0.96

SLAG	
	%
Cu	0.05
SO_2	35.25
FeO	41.30
Al_2O_3	4.40
CaO	10.00
MgO	1.64
S	3.66
	Oz. per ton.
Au	0.03
Ag	0.32

2. Conclusions.—(1) Smelting in an electric furnace treating sulphide ores consists simply of melting the charge, volatilizing about 60% of the sulphur as elemental sulphur, and separating the slag and matte.

(2) The ratio of concentration is simply that possible from a separation of matte and slag, together with a reaction of oxides and sulphates upon sulphides, and the volatilization of 10% more than one atom of sulphur, but with no formation of iron oxide to enter the slag, because there is nothing present to oxidize the iron sulphide.

(3) Qualitatively, it is possible to condense some of the elemental sulphur driven off.

(4) The loss of copper, by volatilization and in the slag, is low.

(5) A matte containing about 1% copper, from a charge containing 0.30% copper, makes a good collecting agent for gold and silver, if a clean separation of slag and matte is obtained.

(6) There is very little loss by volatilization of gold in the electric furnace, but there is some loss of silver in this way. This would not probably occur in a larger furnace with closer temperature regulation.

(7) The electrode consumption in the smelting of a sulphide ore is low, and in practice need not exceed five pounds per ton of charge.

(8) In a large commercial furnace the power consumption for most ores would be about 480 kw. hr. per ton of ore, or 0.055 kw. years.

Electric Furnace v. Reverberatory Blast Furnace

So far, we have only considered the use of the electric furnace as a melting furnace in the treatment of copper ores. From what has been said, we believe that we are justified in making the following statements as to the possibility of using the electric furnace for this purpose:

(1) The melting of native copper oxide or sulphide ore of copper can be done just as efficiently in the electric furnace, and perhaps more so, than in either the reverberatory furnace or the blast furnace.

(2) The reactions desired in reverberatory smelting, or ordinary blast-furnace smelting, can be obtained in the electric furnace as well, and perhaps better, than in either of those furnaces.

(3) The loss of electrodes is small, varying from 5 to 10 lb. per ton of ore smelted, and the presence of the carbon electrode does not cause enough reduction of iron to be troublesome, or increase the consumption of electrical energy appreciably.

(4) The losses of copper, gold, and silver by volatilization and in the slag would be no greater, as shown in all the experimental work cited, than they are in reverberatory smelting or ordinary blast-furnace smelting.

(5) A matte containing as low as 1% copper can be used as collecting agent for gold and silver in the electric furnace, as well as in combustion furnaces.

(6) The comparison of costs would depend entirely upon the nature of the ore treated, and the relative cost of coal or coke, and electrical energy. In general, from the work done in the experiments, we may say that from 500 to 700 kw. hr. per ton charged would be required for smelting copper ore, depending upon the nature of the ore.

There does not seem to be any metallurgical reason why the chief objects of the pyrite smelter cannot be carried out in some electric copper blast-furnace such as Fig. 17*a* and 17*b*, as well as in a coke copper blast-furnace; namely, "to melt the great mass of SiO_2 and inert earths, to oxidize enough of the sulphides in the charge to insure a suitable matte—incidentally obtaining the heat evolved by the oxidation.". On the other hand, it would seem that the difficulties ordinarily encountered in operating a pyrite furnace might be avoided when using an electric blast-furnace, namely, the difficulties which arise when too much or too little coke is added to the charge, and moreover that it not only permits "introducing heat into the furnace, without at the same time robbing the combustion zone of oxygen," of which E. D. Peters states, "nothing would be so welcome to the furnace man as to do this," but would also permit of the heat being entirely under control and easily regulated, and thus avoid freeze-ups with their consequent vexations and costly delays. By this we do not mean that any sort of a charge could be put through the furnace and not freeze it up, an idea which seems to be quite prevalent in regard to electric furnaces, but that, if the charge be properly calculated, a very much wider variation would be permissible in the composition of the slag from that calculated, than would be the case in blast-furnace smelting.

By referring to Fig. 17*a* and 17*b*, it will be noted that the chief difference in the construction of the furnace would be in that part of it below the tuyeres—in other words, in the crucible. Because electric furnace construction has received the attention of some of the very best mechanical and electrical engineers, and because a crucible based upon the principle shown in the design in Fig. 17 is now extensively used in the iron electric reduction furnaces of Norway and Sweden, we believe that no difficulty would be experienced in this respect. Various methods and arrangements can be used for connecting up electrodes. If this be true, we have only to consider the matter of costs.

Comparison of Costs

a. General Considerations.—From what we have said, it will be evident that a comparison of costs is made simply for the purpose of giving some idea as to the outlay that would be necessary to erect and operate an electric furnace plant for the purpose of smelting copper ores.

Naturally, the first thing to be considered is whether it would be feasible to attempt to substitute electrical energy for coke, due to the cost of the former.

b. Cost of Plant.—As a matter of fact, the cost of an electric furnace plant would be the cost of a regular plant plus the electrical installation, exclusive of the generating plant, because it is assumed that it would be possible to purchase power from some power company, and, if not, the generating plant would probably be considered as a separate organization, selling power to the smelter at so much per unit. Therefore, by electrical installation we mean the cost of transformers, bus-bars, cables, instruments, etc. In order to get some idea of this cost we will assume that our furnace is to smelt 384 tons of charge per day[4] and that the composition of the charge is:

	Per cent.
Cu	5.21
SiO_2	26.41
FeO	18.60
S	11.46
Al_2O_3	4.26
CuO	17.49

This was the average composition of the entire blast-furnace charge which was smelted at the Washoe smelter, Anaconda, for several months.[5] In this plant 8.2% of coke was used. If we should attempt to smelt a charge of this nature in an electric furnace, what amount of electrical energy would be required?

c. Amount of Electrical Energy Required to Replace Coke.—Since 8.2% of coke was required at the Washoe smelter to smelt the above charge:

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[4] At the Washoe smelter, Anaconda, the furnaces were formerly 56 by 180 in. (hearth area, 70 sq. ft.) and smelted on an average a little less than 400 tons of charge daily, or 5.6 tons per square foot. From 'Principles of Copper Smelting,' Peters, p. 156.

[5] 'Practice of Copper Smelting,' Peters, p. 267.

2000 × 8.2 lb. = 164 lb. of coke, were used. This coke contained 80.24% fixed carbon; therefore, the carbon equivalent of the coke is: 164 × 80.24 =131.59 lb. If one pound of carbon be completely burned to carbon dioxide, we obtain 8100 lb. cal. Therefore, 131 × 8100 = 1,061,100 lb. cal. that would be obtained from 131 lb. of carbon contained in the 164 lb. of coke. As 1 kw. hr. is equivalent to 1897 lb. cal., it would require: $\frac{1,061,100}{1897}$ or 559 kw. hr. to replace the theoretical calorific value of the 164 lb. of coke. However, as a copper blast-furnace at its best probably does not have an efficiency of over 50%, the energy really obtained from the coke, so far as work performed in the furnace is concerned, is only 530,550 cal. and this would reprsent theoretically, $\frac{530,550}{1897}$ or 280 kw. hr. per ton. Inasmuch as the efficiency of electric furnaces of this type may be as much as 85%, and as that of an open top ferro-silicon furnace is said to be about 60%,[6] it would be fair to assume that the efficiency of an electric furnace, such as shown in Fig. 16, would be 20% greater than that of the ordinary blast-furnace, and hence we will say it is 70%. If this be true, then the theoretical 280 kw. hr. needed to smelt one ton of charge would have to be increased 1.42 times (280 × 1.42 = 397) or say in round numbers, 400 kw. hr. would be required to smelt one ton of charge. Considering that part of the heat in this case is supplied by combustion of sulphide the figures agree closely with the 480 kw. hr. calculated from the experiments on melting only. As we are to smelt 10 tons of charge per hour, 400 × 16 = 6400 kw. hr. would be the constant load on the furnace, and, as the power supply would doubtless be three-phase, this would mean that the electrical equipment should consist of:

Three 2500-kw. variable voltage transformers, to allow ample margin for sudden overloads, etc.; instruments, bus-bars, cables, etc.

As to their costs at the plant, that would, of course, depend upon the locality, and would be the factory costs plus transportation charges. To partially offset this cost, the blower capacity would not need to be so large as in the case of the regular blast-furnace, because less air is blown.

d. Cost of Smelting.—We may say that the costs of coke and electrical energy are about on a par when coke costs $7 a ton, and electrical energy 0.15c. per kw. hr., or $13 per kw. yr.—or say in the ratio of 1:1.8. Although there may be advantages in the use of an electrical blast-furnace instead of an ordinary blast-furnace,

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[6] *Revue de Métallurgie*, Vol. IX, p. 362 (1912).

these will not be taken into consideration in this connection, and we will assume that the cost of electrodes and of the electricity on the one hand, and its equivalent in heating value of coke on the other, would be the same; in other words, that the cost of electrodes and electricity would balance the cost of the amount of coke that would be necessary to do the same amount of work. We will not take into consideration the saving in blowing by reason of not having to furnish air for burning the coke.

e. **Cost of Electrodes.**—The average consumption of electrodes per ton of iron produced at Trollhättan, Sweden, is about 10 lb. Assuming 1 ton of iron is equivalent to 2 tons of charge, the consumption of electrodes per ton of charge smelted in a furnace, such as the one we have been discussing, would not be over 5 lb. per ton. If we assume that the electrodes do not cost over 6c. per pound, which allows a fair margin for transportation over their cost at the factory, the cost of electrodes would be about 30c. per ton of charge smelted. The consumption would be probably considerably less than 5 lb. per ton, as was shown in an experiment performed by the writers in which air was blown into the furnace, and in which the electrode consumption was only 2 lb.. per ton of ore treated. Such being the case, if our combined electrical energy and electrode cost is not to exceed our equivalent coke cost, with coke at \$9 a ton, the cost of electrical energy would be $\$9 \times 1.8 =$ \$16.20 a kw. yr., minus the cost of electrodes, or about \$16. Compared on the basis of calories, if the coke contains 80.24% fixed carbon, it would contain 1604.8 lb. of carbon per ton of 2000 lb., and this carbon would, if completely burned to CO_2 give a calorific value of about 13,000,000 calories. As one kilowatt year of energy is equal to 16,616,000 cal., it would require something like 1.2 tons of coke, containing 80.24% fixed carbon, to equal in calorific value the kilowatt year, but it is to be remembered that this assumes that all the carbon of the coke is completely burned to carbon dioxide, which rarely happens in practice, and so, as has been determined from actual practice, it requires from 1.5 to 1.8 tons of coke to yield the same number of calories as may be obtained from 1 kilowatt year of electrical energy.

Electric Blast-Furnace in Smelting Pyritic Ores

As previously stated, in all true pyrite smelting, fuel (usually coke) in varying amounts is added to the charge. We will assume for the sake of argument, that the ore being treated is entirely suited to pyrite smelting, and that it is not necessary to use more than 0.5% of the weight of the charge in coke. Could such an ore be treated pure as well in an electric blast-furnace as in an ordinary blast-furnace, assuming that the costs would be the same in each case. As what we have had to say in regard to the construction of the furnace and the costs of applying the process is as applicable to true pyrite as it is to semi-pyrite smelting, we will only take up the subject from the metallurgical and chemical standpoint.

In the first place let us consider the nature of the atmosphere in a pyrite furnace. As the air enters the tuyeres, practically all of it enters into immediate combination, forming SO_2 and FeO. Although some oxygen escapes combination, it is not sufficient to support combustion within the furnace, as has been shown by Sticht and others. Now let us briefly consider the changes which take place in the charge as it descends from the charging door to the fusion zone. They are few, and may be briefly stated as follows:

(1) The iron sulphide loses one-half of its sulphur as elemental sulphur, and becomes FeS.

(2) Chalcopyrite. It loses about one-fourth of its sulphur and becomes practically a mixture of Cu_2S and FeS.

(3) The silica is unchanged.

(4) The coke is for the most part consumed in the upper zone of the furnace, but not by oxygen, as no free oxygen exists in that part of the shaft, but by the O from SO_2, that is, the carbon of the coke reduces the SO_2 giving S and CO, and this CO reacts with more SO_2, forming S and CO_2. Aside from these reactions, which are of no great importance in the chemistry of the process, the chief office of the coke as pointed out by Sticht,[7] "Is apparently to heat up the sulphides and the quartz in preparation for their active oxidation deeper in the furnace." If, therefore, coke was omitted from the charge we would not have this preparation of the sulphides and the quartz as stated by Sticht. Such being the case, how detrimental to the process would this lack of preparation be?

In order to answer this question, it must be borne in mind that in true pyrite smelting, there is "no heat to spare," and that although, as pointed out by Sticht, the heat evolved by the combustion of the coke by the oxygen of sulphur dioxide is only one-third as much as it would be if the oxygen were furnished direct by the blast, and that it is given off at a point considerably higher than the focus of the furnace, it nevertheless "assists in preparing the charge for the reactions which take place in the focus of the furnace, for with 1% of coke in the charge enough heat is generated to supply one-third of the heat required to melt the entire pyrrhotite contents

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[7] 'Principles of Copper Smelting,' by E. D. Peters, p. 227 (1907).

of the charge.'' This extra heat ''doubtless furnishes just the necessary aid to bridge the operation over some critical point.'' From this we see that the office of the coke then is solely for the purpose of furnishing heat. Inasmuch as the nature of the charge in true pyrite smelting is such as to demand the oxidation and removal of a large excess of iron as silicate, and as it is necessary to cause the silica present to unite with as much FeO as possible (due to the fact that generally silica-bearing materials have to be added for slag-forming purposes, and often these are barren and hard to obtain), and as the saturation point of silica with iron oxide is greater the higher the temperature attainable in the pyrite furnace, it is quite necessary that the reactions in the zone of the furnace proceed as energetically as possible, in order to meet the requirements stated above, and to provide enough heat to keep the process going. If, therefore, pyrite smelting be attempted in an electric blast-furnace, it would be necessary to supply the heat from the electric current in such a manner as to insure there being enough heat at the focus to carry on the reactions at that point in the same manner as if coke had been added to the charge. The writers are of the opinion that it would be possible and perfectly feasible to do this in the furnace of the type shown in Fig. 17*b*. Moreover, that the use of the electric furnace would permit of a more uniform operation of the furnace than is now possible, for, with the electric current, it would be an easy matter to quickly increase or decrease the heat required for the successful operation of the furnace. In this way we could maintain the ''degree of concentration at a constant,'' which, as pointed out by Sticht, is very important, but which is extremely difficult to do. Also, we could avoid the troubles arising from the ''constant fluctuations'' which are bound to occur, and which at present demand unbroken attention and frequent changes in the charge, especially in its proportion of silica ''due to the fact that in true pyrite smelting there is no heat to spare,'' whereas, with the electric blast-furnace, it would be possible, as just stated, to increase or decrease the heat as necessary in order to meet these fluctuations.

Summary.—Summarized then we may say that the electric smelting of copper ores is nothing more than the substitution of electric heat for the heat derived from the combustion of carbon. Inasmuch as the carbon which is used either in the reverberatory furnace or in the blast-furnace plays no important part in the necessary reactions which take place in these furnaces, there is no reason, metallurgically, why electric heat may not be substituted for the heat derived from the combustion of carbon. In fact, as we have tried to point

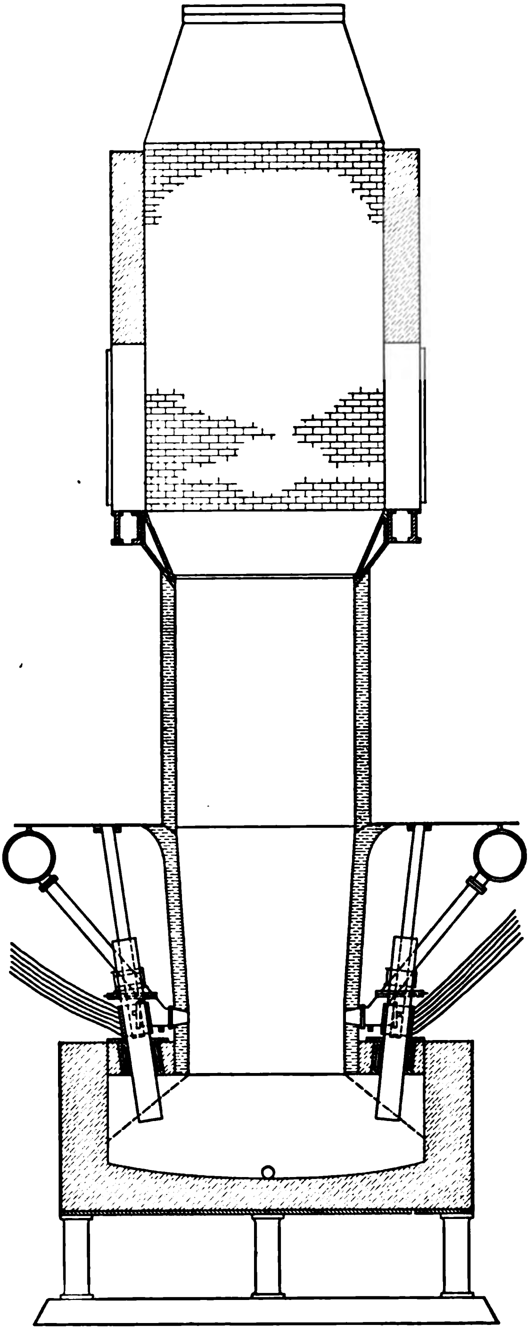


FIG. 17a. ELEVATION OF ELECTRIC BLAST-FURNACE.

out, in some cases the reactions would take place to better advantage in the neutral atmosphere of the electric furnace than in the reducing or partly reducing atmosphere of the combustion furnace. Therefore, as to whether the electric furnace would be used for the smelting of copper ores would largely depend, so far as we are able to make out at the present time, upon the relative cost of coke and electric power. As the use of the electric furnace is not advocated as a competitor of the combustion furnace, but as a substitute for it in those localities where it is not advisable because of the high cost of fuel, we see no reason why the electric furnace may not be developed as a substitute for the combustion furnace, where the conditions are such as to warrant its use, especially in the treatment of copper-bearing ores. In this connection it is to be remembered that the reason why the electric furnace was developed in the iron industry, for the reduction of iron from its ores was due to necessity. As a matter of fact the field for the electric furnace in the reduction of iron from its ores is a limited one. Perhaps the same is true as regards the possible application of the electric furnace to the treatment of copper ores, but, judging from the comparative costs as shown in the preceding pages, it would seem that the chances in favor of the electric furnace for the treatment of copper ores are greater than those for the treatment of iron ores, because there is not so great a difference in the cost of coke and electric power in copper mining districts as in iron smelting centres. Also the cost of electric power is constantly becoming less, due to improvements in gas engines and steam turbines, so that, in districts where water

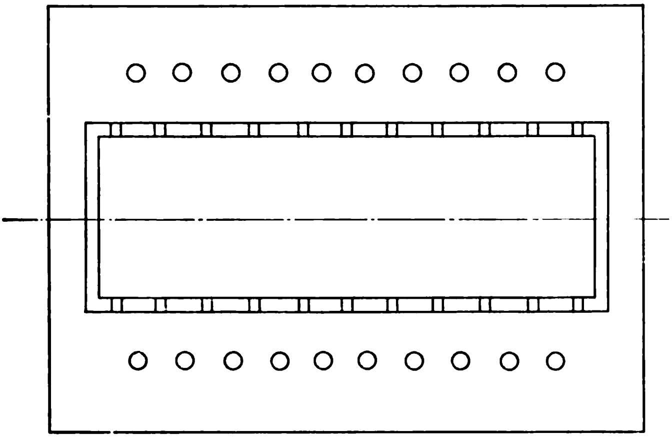


FIG. 17*b*. PLAN OF ELECTRIC BLAST-FURNACE.

power is not plentiful but cheap fuels unsuited to coking purposes are available, it may be found more advantageous to use electric heat than the heat derived from the combustion of coke.

It is sincerely hoped by the writers, although it is possible at this time to present but few facts, that the comparative study herewith presented may serve to stimulate interest in the subject of the electric smelting of copper ores, and to cause others to attempt a further investigation of it. As a result of their investigations, the writers are convinced that experimental work on a larger scale should lead to the development of an electric blast-furnace, which in some cases could be used to better advantage for smelting sulphide ores of copper than the combustion blast-furnace.

ELECTRIC SMELTING OF COPPER ORES

(Editorial, November 1, 1913.)

With the growing importance and availability of electric power, especially hydro-electric power, and the continual improvements in the design and construction of electrometallurgical apparatus, the smelting of copper, and more especially zinc, ores by electrically generated heat is a possibility which may well engage the attention of those who are concerned with metallurgical progress, as well as with the payment of present dividends. The authors of the paper presented in the current issue on the application of the electric furnace to Westen metallurgy have been engaged by the Bureau of Mines to make a detailed study of this subject, and the results of their work will be more fully set forth later in publications of the Bureau. As is both natural and right, Messrs. Lyon and Keeney have adopted an optimistic viewpoint, and in the interest of obtaining a well-rounded view of the situation, it will be well to call attention to the factors unfavorable to the immediate development of an industry in which the heat for smelting is generated from electrical energy.

In the computation of the cost of electric power for smelting, those who take an optimistic view commonly reckon that electricity may be obtained at a cost of 0.15c. per kw. hr., or about $10 per horsepower-year. As compared to the extraordinarily low cost for the production of electric power which has been attained in some parts of Europe, this may seem a liberal allowance to the advocates of electric smelting. As a matter of concrete fact, however, the cost of hydro-electric power, in the mountain districts of California, is $50 per horsepower-year to consumers of moderate amounts, and even consumers who will take a steady load of 10,000 hp. have to pay at the rate of at least $16 per horsepower-year. In his interesting paper on hydro-electric development in Montana, Mr. Max Hebgen makes the statement that the coal necessary to develop by steam power the 49,500 kw. now produced by the Montana Power Co. and its subsidiaries would cost $45 per horsepower-year. It is unfortunately probable that the use of electric energy for smelting will have to be developed under unfavorable conditions, rather than those which are most favorable, and it is only fair to state that there is every evidence that electric smelting will have to be upon the basis of a power cost which is nearer 1c. per kilowatt-hour than the figure usually cited. On the basis of a theoretical computation in which it is assumed that the thermal efficiency of a coke-fired blast-furnace is 50% and the efficiency of the electric furnace is 70%, it is deduced that coke and electric energy are on a par at $7 per ton for coke and $10 per horse-power-year for electric energy. With electric power at the more probable cost of $30, this corresponds to $21 per ton for coke, and even this figure neglects the cost of electrodes, which is estimated at 30c. per ton of charge smelted. The cautious manager hopes for the best conditions and prepares for the worst, and upon this basis it is difficult to believe that, except under unusually favorable conditions, electric smelting can be done at a cost which will permit of its wide use, for the present at least.

Other conditions than those of cost are of importance in the regular conduct of smelting work, however, and it may be pointed out that the barring down of crusts is a normal action in copper blast-furnace smelting as usually practised. This is almost incompatible with the use of refractory linings in the construction of a smelting furnace. The assumption that the thermal efficiency of an electrically heated furnace will be 20% greater than that of the ordinary copper blast-furnace will require experimental demonstration before it can gain general acceptance. It must not be overlooked that in experiments which have been conducted, the volatilization loss was 1% of the copper present when smelting native copper concentrate containing about 30% copper. In the other experiments, the volatilization loss is not given beyond the statement that it is low. It must be remembered that our knowledge of slag composition and slag losses is based upon practice in which the average temperature within the furnace is lower than is likely to be the case in electric smelting, where the heat generated is concentrated in a comparatively narrow zone between the electrodes, rather than distributed throughout the zone of combustion, as in the blast-furnace. What difference this will make in practical operation can only be determined by much experiment, in which the Bureau of Mines is properly taking the initative.

Just as smelting in the blast-furnace, utilizing only the heat of the sulphur in the charge, is theoretically possible, but in practice the use of coke to increase the tonnage handled per day is an economic advantage, so the use of electric energy will be conditioned upon the profit obtainable from its use, and profits can only be determined by introducing all working conditions in experiments upon a working scale. Considerations of space prevent fuller discussion here, but we will be glad to accord space in the 'Discussion' column for the views of our readers.

It is unquestionable that if the early experiments which painfully led up to the present converter practice and pyrite smelting had been made by an organization like the Bureau of Mines rather than under the irrelevant difficulties at Toston, Butte, and elsewhere in Montana and Colorado, progress would have been more rapid than was actually the case. Promising experiments have often been dropped because some structural feature unexpectedly failed and the management was unwilling to incur the expense of reconstruction. We congratulate the Bureau upon being both able and willing to undertake the work necessary for the development of an exact knowledge of the possibilities of electric smelting.

ELECTRIC SMELTING OF COPPER ORES

(Discussion, December 20, 1913.)

The Editor:

Sir—We have read with interest your editorial in the *Press* of November 1, on the 'Electric Smelting of Copper Ores.' Four main questions are brought up for discussion in the editorial which deserve comment: (1) the effect of power cost upon the adoption of electric smelting in the metallurgy of copper; (2) the heat efficiency of the electric furnace as compared with the blast-furnace; (3) the impossibility of using refractory linings in the construction of a copper blast-furnace, because of the necessity of frequent barring down of crusts; and (4) the volatilization loss of 1% of the copper present in smelting native copper concentrates containing about 30% copper.

The Bureau of Mines is gathering data and experimental results upon questions (1) and (2), namely, power cost for electric furnaces and heat efficiency, which it hopes to publish as soon as the required information is made complete.

In reading your statement regarding the incompatibility of the use of refractory linings in a smelting furnace where it is necessary to bar down crusts as in the copper blast-furnace, we feel that there is an inference that a water-jacketed shaft would not be used in the electric furnace as in the ordinary blast-furnace. To clear up any doubt on this question, we quote from our paper on 'The Smelting of Copper Ores in the Electric Furnace,' page 2139 of the August *Bulletin* of the American Institute of Mining Engineers, where it is stated: "To begin with, let us assume that our furnace is similar in construction to a modern copper blast-furnace, and that the upper part of the furnace, including the tuyeres, is practically identical with the same. Below the tuyeres the furnace could be constructed as shown in Fig. 17*a*. By referring to this, it will be noted that there are electrodes extending down into the crucible, the arrangement of which along the sides of the crucible, can be noted by referring to the plan of the furnace in Fig. 17*b*." The shaft of the electric shaft furnace proposed by us for copper smelting, is thus water-cooled and made of water-jackets as in the blast-furnace. This shaft is set over a crucible, as is the case in the Swedish electric pig iron furnace; but differs from the Swedish furnace in that it is rectangular rather than circular in cross-section. Only the crucible of the proposed furnace, described in our paper, is lined with refractory material, and is also covered by a roof of refractory material at the sides. The walls of this crucible could be made of water-jackets, if the electrodes were insulated from the shaft walls and crucible walls by a roof of refractory materials. There is, however, no need of adding this complication to the electric furnace, as no crusts could possibly occur that far down in the furnace. The refractory lining of the crucible would easily stand the ordinary wear on it, as has been found to be the case at Trollhättan. In the electric furnace used in ferro-alloy and calcium carbide manufacture, there is a tendency for the material to freeze on the refractory lining to such an extent that eventually the charge itself really forms the lining of the furnace. Furnaces have been run for two years without being relined. In some works, when the charge does not chill enough to form a protective lining, a water spray is played on the steel shell of the furnace, or in some cases the crucible is water-jacketed, but in most plants this practice has been discontinued on new furnaces, which are designed with the electrodes at sufficient distance from the walls to keep the material frozen on the walls of the crucible.

On page 2124 of the paper above referred to, we have stated in conclusion: (2) "The percentage of copper in the slag need not exceed 0.25, with a slag of proper composition. The loss in the slags should not exceed 0.5% of the total copper charged. Other losses should not exceed 1%, giving a total loss of 1.5% of all the copper charged." In your editorial you state that the loss of copper by volatilization was 1% of the copper present. The 1% representing "other losses" does not consist alone of a volatilization loss. It includes volatilization, oxidation, hangings of partly fused ore and dust loss, which includes both the fine material lost in mechanical handling of the fine concentrates at the furnace, and the dust blown out of the furnace by the gases. In the operation of a small furnace there is considerable mechanical loss by hanging and blowing out of fine material by the gas. In our experimental work these two factors seemed to be the chief cause of copper loss, and we found it impossible to separate these losses, because of the small working scale of the experiments.

We appreciate very much the friendly criticism expressed in your editorial, and merely wish to correct any mistaken impression which may have been given in reading our paper. We hope in a short time to present the detailed results of the experiments which have been carried out to date.

DORSEY A. LYON.
ROBERT M. KEENEY.

Pittsburgh, November 10.

[We are glad that our editorial comment has evoked such interesting additional information on the points of leading interest to the operating metallurgist. The important data and experimental results on the efficency and power cost of electric smelting which are promised will be awaited with keen interest by all.—EDITOR.]

SMELTING COPPER IN SMALL REVERBERATORY FURNACES

By EDWIN M. CLARK

(April 23, 1910.)

About October 1902, the Richardson brothers employed me to go to their mining properties, situated in the rugged mountainous district of Sahuaripa, Sonora, Mexico, about 150 miles by trail in an easterly direction from Torres, a small station on the Sonora railroad, to investigate the possibility of smelting their ores. At that time they were shipping each month, by pack-mules and burros over the above trail, something like 45 tons of first-class ore and concentrate from their mine at La Bufa, and the same amount of concentrate, with from $6000 to $10,000 worth of silver sulphide from the lixiviation plant at Santa Rosa.

The method of procedure was as follows: The ore as it came from the mine was first hand-sorted to separate the first and second-class ore from the fine which latter was passed over shaking screens so as to classify it. The coarse product was hand-jigged at the mine and the concentrate obtained was added to the first-class ore. The middling from these jigs and all the other fine and the second-class ore, amounting to about 200 tons per month, was sent by mule back, nine miles to Santa Rosa, at a cost of ₱8 per ton; where the coarser ore was crushed with rolls and concentrated on jigs and tables, the middling and fine going to stamps to be crushed fine enough to chloridize by roasting with salt in small hand-roasting furnaces. The soluble copper salts were then washed out with water and run to waste. The chloride of silver was leached by hyposulphite of soda and lime, precipitated with sulphide of lime, dried, sacked, and shipped to the railroad. This was an expensive operation. The manager saw that if he could smelt the ores at the mine it would reduce this local expense as well as save the copper they were losing: besides by concentrating the copper-silver into matte or black copper, it would make an enormous saving in the transportation expense to the railroad. I recommended reverberatory smelting, on account of its adaptability to making copper matte, using cordwood for fuel. It also required less metallurgical skill than a blast-furnace. The ores being coarse and fine concentrate would not have to be briquetted after roasting, so the zincy lead-copper concen-

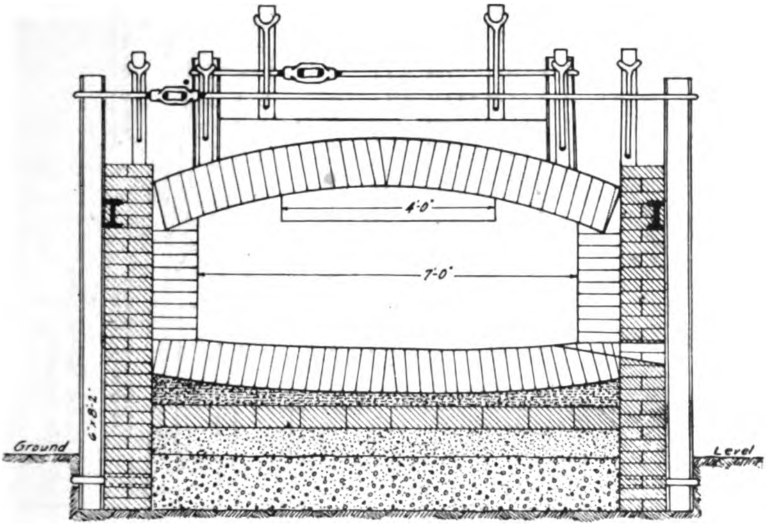


FIG. 18a. CROSS-SECTION OF SMALL REVERBERATORY FURNACE.

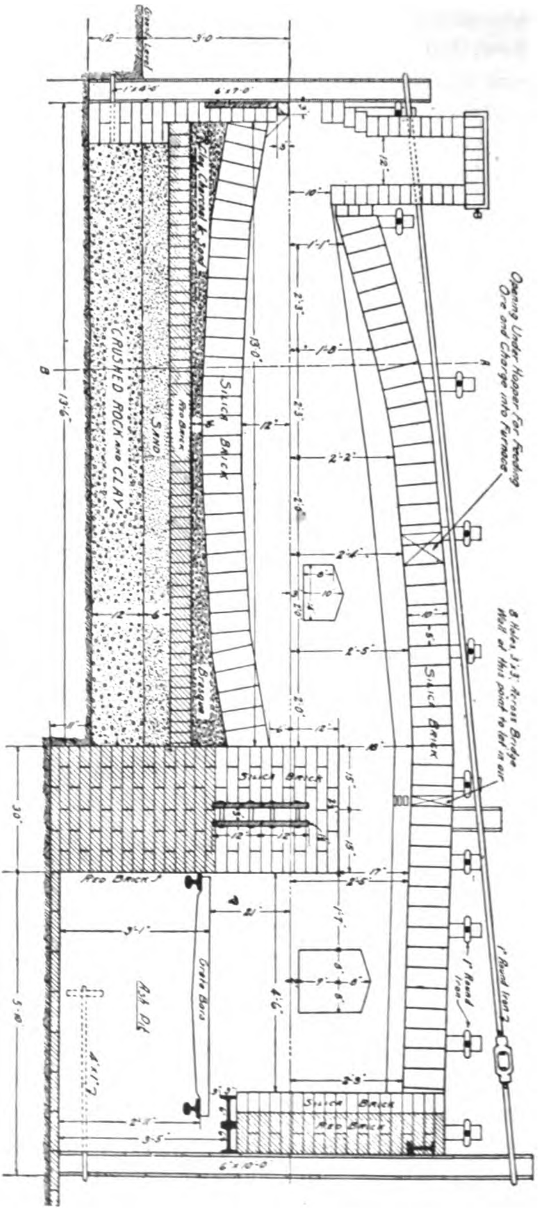
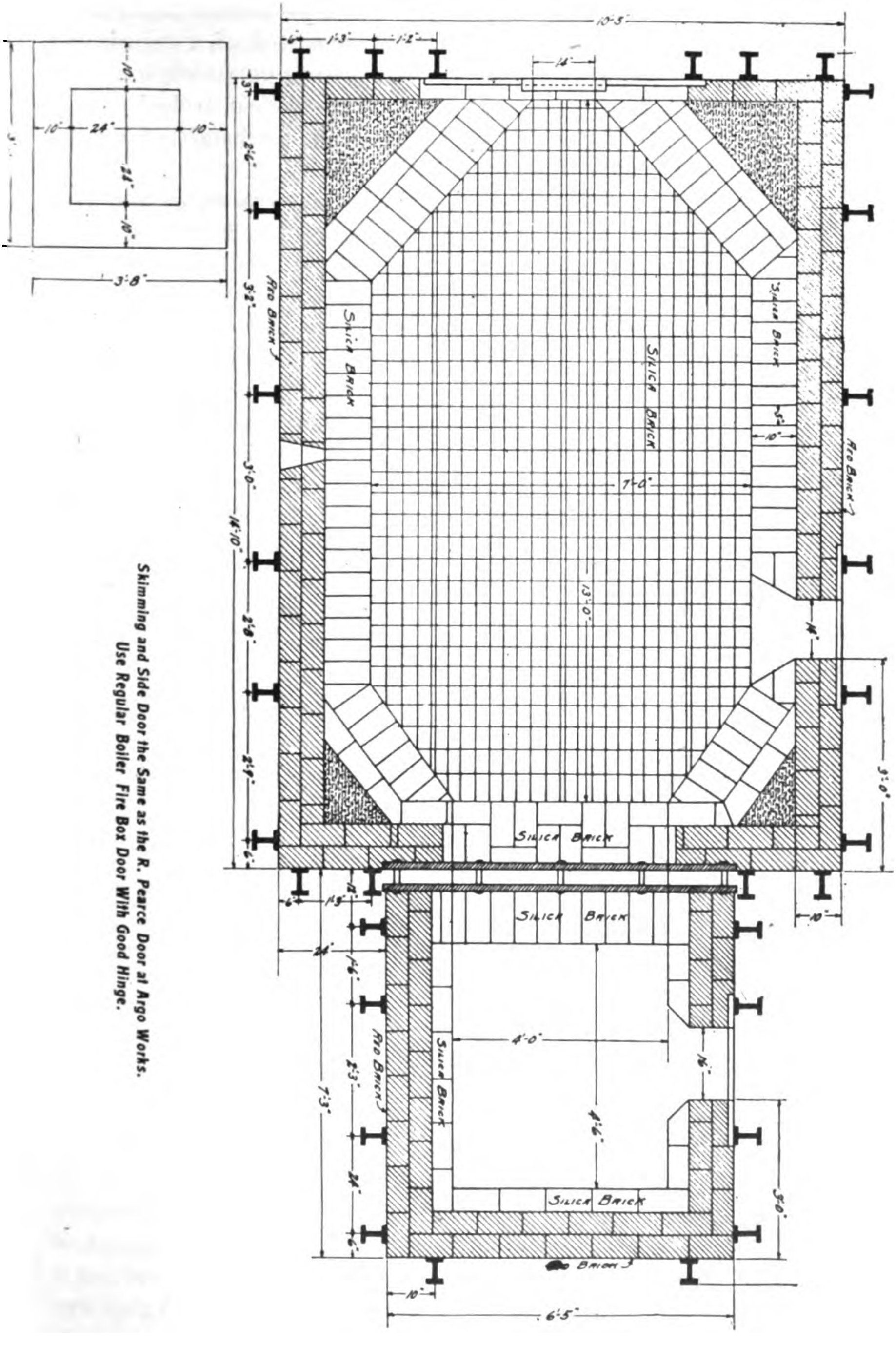


FIG. 18b. LONGITUDINAL SECTION OF FURNACE.

trate, rich in silver, would be reduced with less loss in fume and flue-dust than in a blast-furnace. I was requested to take charge of the metallurgical department and proceeded to make plans for the mill and furnaces. These consisted primarily of a 24-ton concentrator and a 14 by 50-ft. roasting furnace, built of stone and



Skimming and Side Door the Same as the R. Pearce Door at Argo Works.
Use Regular Boiler Fire Box Door With Good Hinge.

FIG. 18C. PLAN OF FURNACE.

red brick at a cost of about ₱3000, which was connected to the same flue as the smelting furnaces. The flue consisted of a trench run up the side of a hill to the top of a small butte on which was built a 20-ft. stack. The trench was 300 ft. long. When arched over it had a dimension of 4 by 6 ft. in the clear, and a height above the reverberatory furnace of about 110 feet.

The roasting furnace would desulphurize 12 tons in 24 hours, of a mixture of first-class ore and concentrate, from 23.8% down to 7%, three Mexican cords of wood (96 cu. ft.) were used per day, seven men were employed on a shift of 12 hours, including the foreman, and the *cargador* who brought up the charge and wood. A small 7 by 13-ft. reverberatory furnace, the description of which is the subject of this paper, was also built. Through the courtesy of the La Dura Milling & Mining Co. I obtained their hand brick-press, of German make, which molded a brick 2½ by 5 by 10 in., with which were made all of the common and silica bricks needed at the rate of from 1200 to 1500 per day. In searching for material to make the refractory bricks, I found an inferior fire-clay at Ocotes, some 15 miles north of the mine, which gave the following analysis:

	Per Cent.
SiO_2	68.77
Fe_2O_3	3.96
Al_2O_3	16.64
CaO	2.23
Loss by ignition	4.12
	95.72

Also a quartzite was found at La Trinidad 17 miles away, which was crushed dry at the mill, fine enough to pass a 30-mesh screen. This gave the following analysis:

	Per Cent.
SiO_2	95.15
$Al_2O_3 + Fe_2O_3$	4.15
CaO	0.60
	99.90

To make the silica bricks, 2 parts of the crushed clay were mixed dry with 5 parts of pulverized quartz (by measurement) in a mortar box, and enough water was added to make a batter of the material, of the same consistence as ordinary lime-mortar. After thoroughly mixing, it was thrown upon a canvas to dry until stiff enough to press into bricks. In the meantime other batches were made in a similar manner, so that the pressing of the bricks could continue uninterruptedly. The reason for this was to get an intimate mixture of clay and quartz. The bricks when molded were carried by boys upon a small piece of board to a level patio, where they were placed to dry, after which they were built into the centre of the kiln. The outside and arches of the kiln were built of common brick so as to protect the more fragile silica-bricks. The common red brick was made from clay and sand found near the mine. The silica-brick when burnt gave the following analysis:

	Per Cent.
SiO_2	90.10
Fe_2O_3	2.67
Al_2O_3	5.33
CaO	1.25
	99.35

These bricks were rather fragile but stood the fire well.

As will be seen by the plan of this 7 by 13-ft. hearth-furnace, it was built without a vault, starting upon a solid foundation of country rock. The outline of the furnace was built of red brick, and when high enough the tie-rods were put in place and the walls continued. A foot of crushed rock and common clay was tamped in the bottom, after being slightly moistened; upon this was placed 6 in. of clean sand, leveled off, and a layer of red brick followed. Upon this was tamped brasque, consisting of one part of inferior fire clay, one part of clean sand, and one part of fine charcoal, with sufficient moisture to make it adhere when pounded by the tamping sticks. This was scraped or cut to conform to the shape the hearth was to assume when completed. Upon this was placed the silica-brick on end. Beginning at the bridge-wall, a wedge was set in the centre of the hearth, so as to make all of the bricks incline a little, the idea being that they would not float up as quickly as if perpendicular. The brick was dipped into water, then into a batter of fine clay and quartz, of the same composition as the bricks, and set as snugly as possible until they reached the walls of the red brick on the side of the hearth, leaving no large cracks or open spaces between the bricks or rows that follow. When completed the floor must have the shape and conform to that given to the brasque beneath. The ends of the bricks where they join the outer wall of the furnace, are cut sufficiently to allow the 10-in. lining or inner wall of silica-brick to have a firm seat upon them; then the walls are completed to where the skew-backs are placed for the arch. To build the arch, I filled the body of the furnace with sand to within a few inches of the skew-backs, upon which was tamped a mixture of clay and sand a little fuller than was necessary for the forming of the arch; this was scraped to the desired form, and the brick laid upon it by beginning at the chimney end, placing the buckstaves and tie-rods as they were reached and driving the wedges just tight enough to raise the row of bricks a little above the form beneath. There were provided one hole for the feeding of the charge and eight small holes over the bridge-wall to admit air for perfect combustion. When finished the sand is withdrawn, and the hearth swept out, the doors put in place, and about ½ ton of pulverized slag placed upon the hearth around the edges. It is then gradually dried, the doors are closed and brought to the highest heat possible. Leaving the doors sealed the fire is allowed to go down until the hearth is dull red, when the first charge of one ton of sulphide ore is added and quickly smelted in. Then follow the regular charges of 1½ tons. It required about 7 days to fill the hearth with matte and it yielded about 9 tons of matte when tapped.

The calcine smelted had the following analysis:

	Per cent.	
SiO_2	31.62	
FeO	15.34	
Al_2O_3	12.52	
CaO	tr.	
PbO	9.00	
ZnO	7.40	
S	6.10	
Cu	13.10	
Ag	0.634	(0.634% = 185 oz. per ton)
	95.614	

The matte produced had the following composition:

	Per cent.	
Cu	50.5	
Pb	12.0	
Zn	3.2	
Fe	4.2	
S	25.0	
Insol. residue	2.1	
Ag	2.225	(2.225% = 661 oz. per ton)

The slag gave analysis:

	Per cent.	
SiO_2	44.10	
FeO	20.60	
Al_2O_3	16.64	
CaO	5.28	
PbO	3.40	
ZnO	9.10	
Cu	0.70	
Ag	0.026	(0.026% = 7.6 oz. per ton)

The slag shows the silver to be too high for good work, but this was due to the fact that I did a great deal of smelting with green wood, and it was almost impossible to drive the charge as rapidly as it should have been done at a time when the gases were coming off freely. With dry *mauta* or pine wood, I was able to smelt at the rate of ¾ cord of wood per ton of charge, and to skim each charge of 1½ tons every 2 hr. and 30 min. The best we attained was 2 hr., 7 min. With this small furnace I smelted over 1000 tons in four months, and the furnace was in as good condition as at starting. It could have been run for a long period with reasonable repairs. It was abandoned for a larger furnace, with a 10 by 20-ft. hearth, built in the same way, which was used intermittently for three years. I say intermittently, because it was closed 7 times one season, and 9 times another, on account of trouble in getting freighters to pack in the wood, due to rains and to the Indian war in progress at that time in the Yaqui district. I employed one American on each shift as skimmer, and two Mexicans to fire the furnace, remove the slag and ashes, and another Mexican to bring up the wood and to help put the charge into the hopper.

In regions remote from transportation, where wood is abundant and inexpensive to obtain, if the management will use nothing but well seasoned dry wood, and have sufficient ore available for a long campaign, the small reverberatory smelting furnace above described, can be made to smelt between 10 and 15 tons per day of 24 hr., with less than ¾ cord of wood per ton where one-half of the charge is hot calcine. Any intelligent person who will be careful of the material used can make a silica-brick which will stand well in such a furnace. I do not, of course, recommend wood where coal can be obtained.

RECENT COPPER SMELTING

(Editorial, May 18, 1912.)

We are fortunate enough to be able to present this week a brief account of copper smelting practice at the plant of the Compagnie du Boleo, written by a well-informed engineer. This is interesting, both because the company does not follow the common American custom of allowing engineers free access to the plant and its records, and because the working conditions are so exceptional. The slag analysis, given by our correspondent, is unusual in its composition, though not in its character. Assuming that the alumina acts entirely as a base, and that manganese is equivalent to iron (the statement of manganese as MnO_2 evidently being merely for analytical purposes), and computing the oxygen ratios, we find SiO_2, 27.55; Al_2O_3, 8.45; FeO.MnO, 2.67; CaO, 3.14; or an oxygen ratio of acid to base of 27.55 to 14.26. The slag is evidently slightly more basic than bisilicate, and since Mr. H. O. Hofman, in his admirable work upon the formation temperatures of silicates, has shown that the substitution of MnO for FeO slightly raises the temperature of slag formation, while Al_2O_3, when substituted for FeO and CaO, lowers the temperature of slag formation until three-eighths of those bases has been replaced, and then raises it rapidly, it is not remarkable that the furnaces run with a high temperature in the zone of fusion, a considerable degree of fluidity, and the rapid separation of the high specific gravity matte. Slags high in manganese are not uncommon, and Mr. J. A. Church has recorded his experience in smelting silver-lead concentrate in Arizona, with the formation of a satisfactory slag containing over 40% manganese. Our correspondent employs a loose expression common among metallurgists in speaking of slag containing a high percentage of SiO_2, as being necessarily an acid slag. Percentages depend upon weight, while the acid or basic character of the slag depends upon the molecular ratio of the acid to the basic constituents. If the basic constituents have small atomic weights the same percentage of silica which represents an acid slag would, with bases of large atomic weights, represent a very basic slag.

The use of narrow long furnaces at this plant may be contrasted with recent development of smelting practice at the Great Falls plant of the Anaconda Copper Mining Co., where Messrs. A. E. Wheeler, superintendent, and M. W. Krecji, metallurgist, under the management of Mr. C. W. Goodale, have developed furnaces 84 in. wide, the necessary blast penetration being secured, as in the iron blast-furnace, by so proportioning the cross-section of the furnace as to obtain a throttling action near the top of the shaft, thus causing the blast to spread throughout the smelting zone by pressure, rather than by depending on the initial velocity of the blast to secure penetration. The use of this design has permitted a lowering of the blast-pressure to 20 or 25 oz., with campaigns a week long without barring off crusts at the tuyeres. The wide furnace offers less radiating surface than a long one, and where it is operated so that barring down is minimized the resulting economy in heat requirement is worth securing, and construction costs are lessened.

Even more notable are the converter records which have been made at the Great Falls plant. The converters now in use are of the Anaconda type, 12 ft. in diameter and provided with 26 tuyeres, using blast at 11 or 12 pounds per square inch. These are lined with magnesia brick, and one converter has made over 7000 tons of copper without relining, while an earlier type of converter, of the same size but with only 15 tuyeres, has been in continuous operation since March 9, 1910, and has made over 12,000 tons of copper from a 28.9% copper matte. Including cold seconds, about one ton of cold material has been handled per ton of matte converted, so that this converter has treated some 60,000 tons of material with the one lining. This is a record of which the able metallurgists who have made it may well be proud, and technical men generally may share in their pride, since it is undoubtedly to a considerable degree secured by the employment of a staff of technically trained young men as investigators to study every operation and to plot records and collect data, so that each day's experience may serve as an infallible guide to the solution of the problems of tomorrow. There is no waste so regrettable as the waste of experience, and employment of trained minds is cheaper in the end than the cut and try method with structural steel, labor, fuel, and valuable ore. We hope later to be able to present details of some of the results thus secured, and we take this opportunity to congratulate the metallurgists concerned as highly upon the methods employed as upon the results obtained.

EL BOLEO SMELTING PRACTICE

By An Occasional Correspondent

(May 18, 1912.)

The chief interest attaching to the copper-smelting methods at the plant of the Compagnie du Boleo at Santa Rosalía, Baja California, results from the remarkably high percentages of silica and alumina in the charge. It is doubtful whether any other smelter in the world has continuously for long periods produced a slag averaging approximately:

	Per cent.
Silica (SiO_2)	52
Alumina (Al_2O_3)	18
Ferrous oxide (FeO)	6-8
Manganese dioxide (MnO_2)	4-6
Calcium oxide (CaO)	10-12

This unusual practice is due to the commercial necessity of dispensing with costly fluxes. Although deposits of hematite, pyrrhotite, and, to some extent, of pyrite, are not uncommon in Baja California and along the west coast of Mexico, they are generally unavailable for use at port smelters, owing to their inaccessibility, and also because of the low tenor in marketable metals characteristic of the west Mexican ores. High-grade ores in that region have never yet been developed in commercially important quantities. Hence the fluxes which are known to exist would more than eat up the value of the marketable metals in the smelting ores. At El Boleo, for example, the average copper content in the charge lies roundabout 4½%. It will readily be seen that the best California fluxes, even if sold at prices barely covering the cost of mining and loading, could not stand the charges of transportation by land and sea, plus the cost of putting through the furnace, with an ore of such grade, and the west Mexican fluxes would cost even more to move to Santa Rosalía than the ferrous ores of California.

While full details of El Boleo practice, rendering possible so brilliant an achievement as that attained by M. Michot, the manager, in reducing the FeO requirements to 6%, are not at hand, the conditions are sufficiently well known to give to a metallurgist an appreciation of the leading features.

The furnaces are of the most simple construction, measuring 12 ft. 6 in. by 3 ft. 6 in. inside, at tuyere-level. The water-jackets are 2 metres high, succeeded by 2 metres of brick shaft extending up to the charging-floor. The feed-doors are about 4 ft. high, the iron hood contracting above these to the chimney, from which latter the down-take leads off to the dust-chambers and main chimney. The furnaces are built with crucibles, and the settled matte is tapped off at intervals. It was found economically impossible to employ a forehearth on account of the rapid chilling of so acid a slag. A curious feature of the slag-discharge is that it is allowed to flow continuously from each end of the furnace through 4-in. slag-ports. It would occur to a metallurgist at once that this would lead to enrichment of the slag in copper. Actually the Boleo slags carry not less than 0.5% Cu, which is above what the majority of smelters elsewhere are willing to concede. The separation of the matte under such circumstances must be extremely rapid, else the slag would certainly carry much more than 0.5% Cu when drained from the slag-line inside the furnace. This may be explained by the high degree of fluidity imparted by the considerable quantity of manganese, at the high temperature prevailing just below the fusion-zone inside the furnace, but beyond doubt the high specific gravity of the matte contributes to this result, its copper tenor being above 60%, and often more than 65%. In passing, it is of interest to note that a premium of £2 per long ton is paid by the European refiners for matte of this grade, owing to its desirability for mixing with low-grade matte for blowing in the converter.

It will be noticed that the furnace is relatively narrow (42 in.), that the column is relatively high (13 ft. 1½ in.) and relatively long (12 ft. 6 in.). The tuyeres are 12 in number on each side, and have a diameter of 4 in. Thus conditions are favorable for a good air-penetration at moderate pressure, the water-gauge actually reading between 50 and 60 centimetres.

The furnaces are run very slowly, each of the ten in blast smelting about 250 tons of charge per day. They are operated with a cool top, which is partly due to the slow speed of smelting and partly to the fact that the charge carries from 20 to 30% of moisture, the evaporation of which lowers the temperature in the upper part of the stack. Only a small part of the moisture contained is present as combined water. The copper in the ore is almost wholly in the form of chalcocite and covellite. The coke used amounts to about 12% of the charge, and it is claimed that it is more perfectly burned to carbon dioxide than is usual in copper-smelting practice in America.

GROWTH OF REVERBERATORY SMELTING

(Editorial, July 16, 1910.)

Reverberatory smelting of copper ores is becoming more general in this country. The success attending the use of furnaces of this type already in operation at Cananea has led to the building of an additional furnace which will soon be placed in commission. The practice is the result of demonstrated economies in operation. If water-power were available the same commercial advantage would not obtain. It is mainly because the waste heat can be utilized in the development of power that the reverberatory shows marked economy as contrasted with the blast-furnace. As shown in a recent article published in these columns 0.77 barrel of oil is burned in the fire-box for every ton of charge smelted. Of this amount only 0.33 barrel is chargeable to actual smelting while 0.44 barrel represents recovery in the form of steam generated in the boilers through which the flue-gases are conducted. At Anaconda and Salt Lake a like warrant for employing the reverberatories exists. An additional advantage consists in the greater flexibility of the reverberatory in taking care of sudden changes in the composition of the charge. The blast-furnace needs to be operated with great uniformity in order to yield the best results. Though it can be operated with wide variations in the percentages of silica and bases it is not economical when sudden changes occur. This statement may be debatable, but there is no doubt that the reverberatory is less sensitive in this regard. The reverberatory will successfully smelt fine ore and flue-dust, while these materials when put into the blast-furnace produce an excessive amount of flue-dust, and herein lies another reason for the rapid growth in favor of the former type of furnace. The present situation of the custom copper smelters inclines the balance in the same direction. The quantity of concentrate offered is steadily increasing, while smelting ores are becoming scarcer. Furthermore silicious ore is not so abundant as formerly, because of the expansion of milling and other forms of local treatment. Basic ores usually make considerable fine material, and this is not adapted to the blast-furnace. Thus the reverberatory finds a large field of usefulness, in spite of the fact that it is more particularly applicable to charges of relatively high silicious content. It is easier to secure a serviceable acid than a basic fettling, and the dust from basic charges tends to corrode the arch and walls. The utilization of highly silicious fine ore has been suggested as a means of overcoming this difficulty, the plan being to introduce a certain amount of such material at the beginning of each charge so as to give the arch and walls a protective coating of acid dust. Less basic dust would then adhere, and corrosion would probably be lessened.

It is stated that the Garfield smelter near Salt Lake may soon be using reverberatory furnaces almost exclusively. An addition to this part of the plant is already being made. This is being done in anticipation of the larger deliveries of concentrate from the Utah Copper Co.'s mills that will follow the reconstruction of the mill acquired from the Boston Consolidated. At the same time it is noteworthy that other custom ores are not being offered in such abundance to the Garfield smelter as to insure a supply sufficient for its blast-furnace plant. The new copper smelter of the International Smelting & Refining Co. in Pine Canyon contains five reverberatories and no blast-furnaces. This is the first large plant to be constructed in America in which blast-furnace smelting plays no part. The reverberatories at Pine Canyon are 102 ft. long, by 19 ft. wide, and each will treat about 250 tons of charge per day. It will be noted that the length of the furnace is less than that of the old furnaces at Anaconda, and it is considered open to question whether even this length is not greater than is actually required for economic work. There is a limit of size beyond which no gain is experienced. In the opinion of some experienced metallurgists no advantage is obtained by making a reverberatory longer than 80 or 85 ft. The smelting is finished at that distance from the fire-bridge, the remainder of the hearth serving only to admit of further separation of matte from the slag, and it is doubtful whether equally good results might not be obtained in a shorter furnace. This would apparently render it possible to recover a larger amount of heat from the waste gases, and make the smelting costs proportionally less. The Pine Canyon smelter, while built primarily to treat the ores of the Highland Boy mines, will also do general custom work. This makes all the more interesting the fact that no blast-furnace smelting will be attempted. All ores received will be crushed in the sampling mill to a maximum size of 5/8 in. The crushing plant is exceedingly compact, and contains Blake crushers and rolls exclusively. The converter house contains one stand for each reverberatory, but it is doubtful whether more than two converters will be required. The furnaces and converters are all connected to a dust chamber 120 by 140 ft., and 40 ft. high, the roof being of brick held by I-beams sustained by trusses on the outside, and further supported by steel columns surrounded by hollow masonry, the latter serving partly to keep the columns cool and partly to draw air into the tunnel beneath, where the dust will be dumped from hoppers into cars. The draft will be induced by a stack 350 ft. high, 36½ ft. diameter at the base and 25 ft. at the top.

The present tendency is distinctly toward a more extended use of the reverberatory furnace, and important developments in practice are likely to follow. Modifications in size and in the lines of the furnaces are to be anticipated, all having for an object the introduction of higher economy. Conditions in this country do not favor the adoption of the Mansfeld practice with tall blast-furnaces and a closed top. By that means the waste gases may be utilized for the development of power, and the dust losses are kept within narrower limits. In America, however, ores suitable for smelting under such conditions are not abundant, and the reverberatory seems to offer a better solution of the problem where dependence upon fuel instead of water-power is necessitated.

DEVELOPMENT OF THE REVERBERATORY COPPER SMELTING FURNACE

By E. P. Mathewson

(November 9, 1912.)

Historical

*The early development of the reverberatory furnace for smelting copper ores was the work of the Welsh smelters, particularly those of Swansea. The first record of a reverberatory furnace is made by Jars, who states that in 1765 copper smelting was effected in reverberatory furnaces at Middleton-Tyas, in Yorkshire, England. The first patent, of any importance, for improvements in reverberatory furnaces was granted to Thomas Williams in 1778 for the granulation of the regulus. The next was that of William Evetts, in Sheffield, in 1812, for cooling of the fire-bridge by the admission of air. Charge hoppers above the furnaces were in common use in 1848. The size of the hearth of the furnaces, about the beginning of the nineteenth century, was commonly 11 by 8 feet.

No material development was made until the Welsh process was brought to Colorado, where Richard Pearce, as manager of the Argo works near Denver, developed the furnace to meet the requirements of a custom plant, under keen competition with lead-silver smelting plants using blast-furnaces. His improvements are outlined in Fig. 19, elaborated by permission, from E. D. Peters' 'Modern Copper Smelting,' to which the reader is referred for further details. This figure shows that the furnaces were, in 1878, 9 ft. 8 in. by 15 ft. in hearth; and in 1894 16 by 35 ft., while the capacity of the furnace had been increased from 12 tons per 24 hours to 50 tons. The next step in development was made in Butte, Montana, by the Colorado Smelting Co., this plant being at that time affiliated with the Argo works, so that Mr. Pearce's influence was apparent. This was the lengthening of the hearth to 50 ft., with consequent increase in capacity to 105 tons in 24 hours. The first furnace of this size, built from the Colorado Smelting Co.'s plans, was constructed at the Butte & Boston plant in Butte, Montana, in the year 1900. Details of Montana practice, up to the year 1902, will be found in a paper by H. O. Hofman entitled 'Notes on the Metallurgy of Copper of Montana,' *Trans* A. I. M. E., Vol. XXXIV, pp. 258-316. The 50-ft. furnace became very popular in Butte and Anaconda, and when

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*A paper presented at the Eighth International Congress of Applied Chemistry, and reprinted by permission of the author.

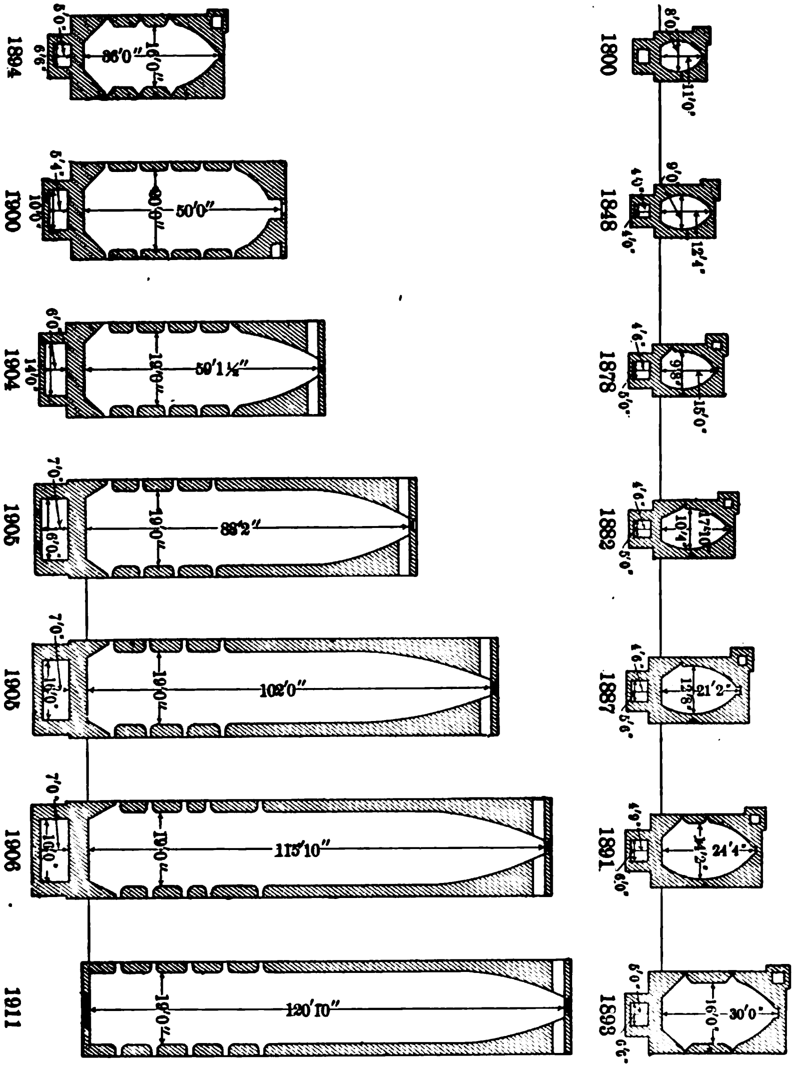

FIG. 19. STAGES IN DEVELOPMENT OF REVERBERATORIES.

the Washoe smelter was built, in 1900-1902, the 50-ft. reverberatory was adopted and 14 of these furnaces constructed.

The next improvement was the addition of limestone to the charge of the calciners so that it was thoroughly mixed into the charge before dropping the charge into the reverberatories. This apparently insignificant change resulted in greatly increasing the capacity of the furnace. The next step was the building of a furnace with a 60-ft. hearth, the results from which were so encouraging that a furnace with a hearth 85 ft. long was tried, with corresponding increase in tonnage. Then the radical step was taken at the Washoe plant of connecting two 50-ft. furnaces, making a single furnace with a hearth 102 ft. long. The saving in fuel and the increased tonnage caused the management to build a furnace with a hearth 116 ft. long. The results of these experiments are given in the following tabulation, compiled by William Wraith, superintendent of the Washoe smelter:

EFFECT OF LENGTHENING REVERBERATORY FURNACES AT THE WASHOE SMELTER

Hearth areas of furnaces, ft.	Tonnage per 24 hours.	Ratio of cupreous material to coal.	Averages.
19 by 50	121.74	2.75	For 1 year
19 by 60	190.7	3.94	For 7 months
19 by 85	234.1	4.13	For 7 months
19 by 102	264.9	4.31	For 4 years
19 by 112	267.1	4.30	For 4 years
19 by 116	270.1	4.19	For 4 years

The percentage of copper in the slags from the different furnaces averaged as follows:

Furnace.	Hearth length. Feet.	Period.	Copper. %
No. 1	50	Sept. 1903-Mar., 1904	0.50
No. 6	60	Mar.-Sept. 1904	0.44
No. 1	85	May-Dec. 1904	0.42
No. 1	102	Jan. 1906-May 1911	0.39
No. 4	112	July 1906-May 1911	0.38
No. 1	116	Jan. 1906-May 1911	0.36

The reverberatory furnaces as originally built at the Washoe smelter had a hearth area of 19 by 50 ft. After some months of operation it was decided that a longer furnace could be operated to better advantage. To determine the length best suited to the conditions one furnace was lengthened to 60 ft., another to 85 ft., another to 102 ft., another to 112 ft., and finally to 116 ft. in length. The draft at the bridge wall was from ¾ to 1 in. of water; the firebox area 7 by 16 ft. The coal used was from the Anaconda Copper Mining Co.'s mines at Diamondville, Wyoming, having the following average proximate analysis and thermal value:

Analysis: H_2O, 6.13%; volatile matter, 36.28; fixed carbon, 45.42; and ash, 12.17%.

Thermal value: Wet coal, 11,710 B.t.u.; dry coal, 12,390 B.t.u.

There was some variation in the quality of the coal, the ash ranging from 6 to 16% and the thermal value from 10,750 to 12,000 B.t.u. per pound of wet coal, or from 11,000 to 13,200 B.t.u. per

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coal. There is also some variation in material smelted lation by the different furnace crews. ry furnaces in different localities present their own l the lengths of furnaces will be found to be functions f fuel, draft, tonnage to be smelted, and investment. time the Garfield plant of the American Smelters . was being built near Salt Lake City, Utah, and for rnaces with 100-ft. hearths were adopted. Then the plant lt, Arizona, of the Consolidated Arizona Smelting Co. furnaces with 100-ft. hearths, using oil for fuel. The nt, at McGill, Nevada, followed with five furnaces with 0 ft. long, coal-fired, changed to oil-firing in 1911. The Cananea Copper Co. put in one furnace with hearth 100 ft. long for flue-dust in 1906, followed by a second in 1911. Experiments were tried with coal-dust firing, but cheap oil being available, the coal dust experiments were discontinued and oil substituted. The reader is referred to L. D. Rickett's article on experiments in reverberatory practice at Cananea (*Trans.* Institution of Mining and Metallurgy, Vol. XIX, p. 147).

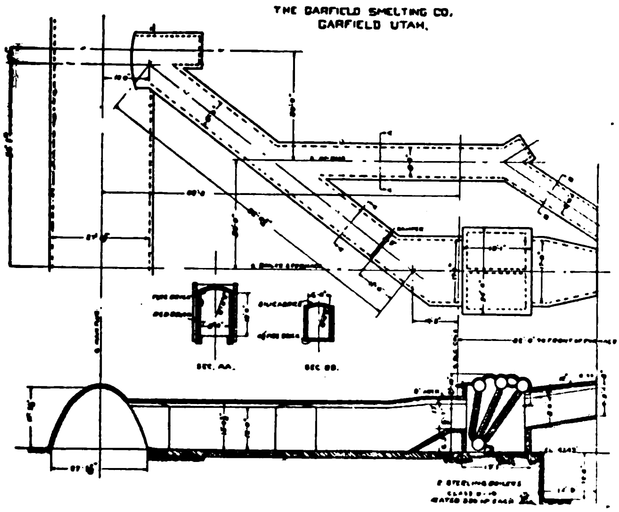

FIG. 20. ARRANGEMENT OF BOILERS AND FLUES AT GARFIELD.

The accompanying table A gives important data on the best modern installations of reverberatory furnaces, and the table of analyses, B, gives the details of the chemical composition of the charge, the slag, the matte, and the fuel. The features that distinguish the modern reverberatory are its length, solid bottom (monolith of fused quartz),

TABLE B. ANALYSES OF MATERIALS USED AND PRODUCED.

		Copper.	Silver.	Gold.	SiO_2.	Fe and Mn.	CaO.	S	Al_2O_3.
		Per Cent.	Oz. Per Ton.	Oz. Per Ton.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Charge	Anaconda	9.31	6.43	0.028	28.63	26.46	3.50	7.34	5.22
	Cananea	6.43	2.22	0.14	26.96	31.34	2.58	9.59	5.54
	Great Falls	9.47	3.77	0.0236	22.50	26.91	4.20	11.40	6.40
	Garfield	12.00	3.50	0.16	30.00	22.50	11.50	10.50	−4.50
	Steptoe (McGill)..								
	Coal-fired	13.60	0.401	0.078	26.30	25.90	5.80	9.50	5.30
	Oil-fired	13.10	0.345	0.078	27.60	24.10	5.40	9.30	6.90
	Tooele	2.90	3.80	0.11	30.50	35.00	3.80	6.50	......
Slag	Anaconda	0.39	0.19	0.0006	39.68	32.83	4.62	0.91	7.03
	Cananea	0.35	0.11	....	37.44	33.58	3.83	1.08	7.96
	Great Falls	0.35	0.12	Trace.	42.50	23.64	11.60	0.48	9.40
	Garfield	0.45[a]	0.08	Trace.	46.50	20.58	17.50	1.00 App'x	5.60
	Steptoe (McGill)..								
	Coal-fired	0.45[a]	0.06	0.005	40.20	28.90	10.50	0.40	7.60
	Oil-fired	0.50[a]	0.02	0.007	42.00	25.90	10.90	0.20	8.00
	Tooele	0.43[a]	0.45	0.008	40.50	36.93	4.00	....	....
Matte	Anaconda	41.68	28.40	0.130	0.27 (Insl.)	26.54	... ..	25.76	
	Cananea	31.91	11.34	0.07	0.54	37.18	....	26.87	
	Great Falls	31.96	12.00	0.085	No data	35.54	....	26.70	
	Garfield	42.00	10.00	0.50	.. ....	27.00	.. ..	26.00	....
	Steptoe (McGill)..								
	Coal-fired	41.76	1.25	0.25	0.70	29.00	. ...	25.10	....
	Oil-fired	42.99	0.91	0.22	0.50	29.10	....	24.80	.. ..
	Tooele.	23.00	27 50	0.85	0.50	43.00	....	26.50	... ...

[a] Converter-slag is poured into reverberatories at these plants, thus raising copper-assay, as only 75 per cent. of copper in converter-slag is recovered.

		Moisture.	V.C.M.	Fixed Carbon.	Ash.	B.t.u. (Dry.)	B.t.u. (Wet.)
		Per Cent.	Per Cent	Per Cent.	Per Cent.		
Fuel	Anaconda	6.13	36.28	45 42	12.17	12,390—	11,710—
	Cananea					No data.	18,600—
	Great Falls.	7.70	23.70	46.00	28.60	No data.	9,382—
	Garfield	8.00	39.40	46.40	6.20	No data.	12,000—
	Steptoe McGill: Coal-fired	2.00	39.10	51.50	7.40	13,000—	....
	Oil-fired					18,220—	......
	Tooele	6 00	38.00	45.00	11.00	....	11,500

	CO_2.	CO.	CH_4.	H.	N.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Gas—Great Falls	10.50	15.80	2.20	13.70	57.80

structural steel plate forming the bridge-wall, frequent charging, infrequent skimming of slag, the large body of molten matte retained in the furnace to assist in melting and distributing the charge, no leveling of charge by means of rabble; thick roof (15 to 20 in. of silica brick); fettling only once a month in many plants, practically continuous operation; recovery of waste heat in the form of steam; and, where coal is used for fuel, the recovery of the unburned coal from the ashes.

During the development of the reverberatory furnace to its present state of efficiency, a great many schemes were tried and abandoned; for example, the pre-heating of the air by passing it under the furnace bottom or around the walls of the flue or fire-box, and forced blast under the grate.

A great variety of fuel has been and is being used in reverberatory practice; for example, at Kyshtim, Perm Government, Russia, producer gas from wood is in use, with gratifying results. At this plant the hearth of the furnace is about 35 ft. between ports and 15 ft. wide; the gas is made from pine wood and the moisture and tar scrubbed from the gas before going to the furnace. The following data, furnished by A. H. Allen, regarding the furnaces will be of interest:

Surface of air checker, 869 sq. m.; useful area, 688 sq. m.

Sectional area of air checker, 49 sq. m.

Surface of gas checker, 698 sq. m.; useful area, 559 sq. m.

Sectional area of gas checker, 46 sq. m.

Average gas analysis, per cent: CO_2, 7.47; CO, 26.22; H, 8.3; CH_4, 5.86; C_2H_4, 0.79.

Analysis of products of combustion in stack, per cent: CO_2, 12.2; O, 6.3; N, 71.5.

Pressure at reversing valve, 3 mm. water. Suction at stack, 25 mm.

Temperature of gas entering regenerator, 68°C.; temperature entering stack, 385°C.

Average temperature of furnace, 1600°C. Maximum temperature at ports, 2000°C. Temperature entering checker, 1300°C.

The charge was mainly flue-dust, but there was some green ore; some silicious oxidized copper ore; some converter cleanings, etc., mixed with it. I cannot give the composition of the charge, but 90% of it was flue-dust and green fine. From January to September 1910 the furnace was in operation 166 days, with results as follows:

Amount smelted, tons	10,690
Wood consumed in producers, cords	3,417
Matte produced, tons	3,326
Contents of matte, tons of copper	283
Assay, % Cu	8.5
Contents charge, copper, tons	330.8
Average assay charge, % Cu	3.1

The work is chiefly interesting for the extremely low fuel cost, which has been verified by an experienced engineer now on the ground. Wood at Kyshtim cost $1.05 per cord delivered, so that the average fuel cost, over the 9-months period, was 33.6c. per short ton of charge. It results from the fact that the Russians in

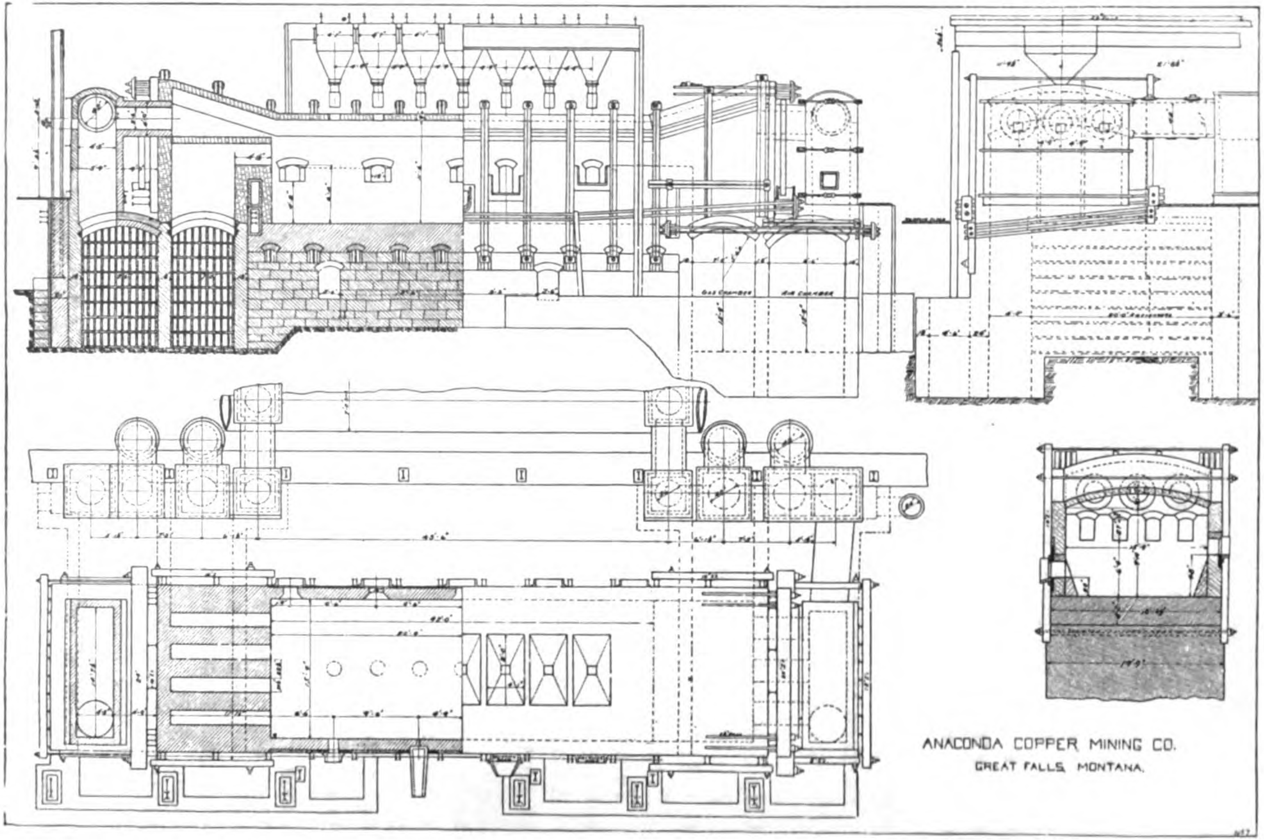

Fig. 21. Anaconda Plant; Plan, Elevation, and Boilers.

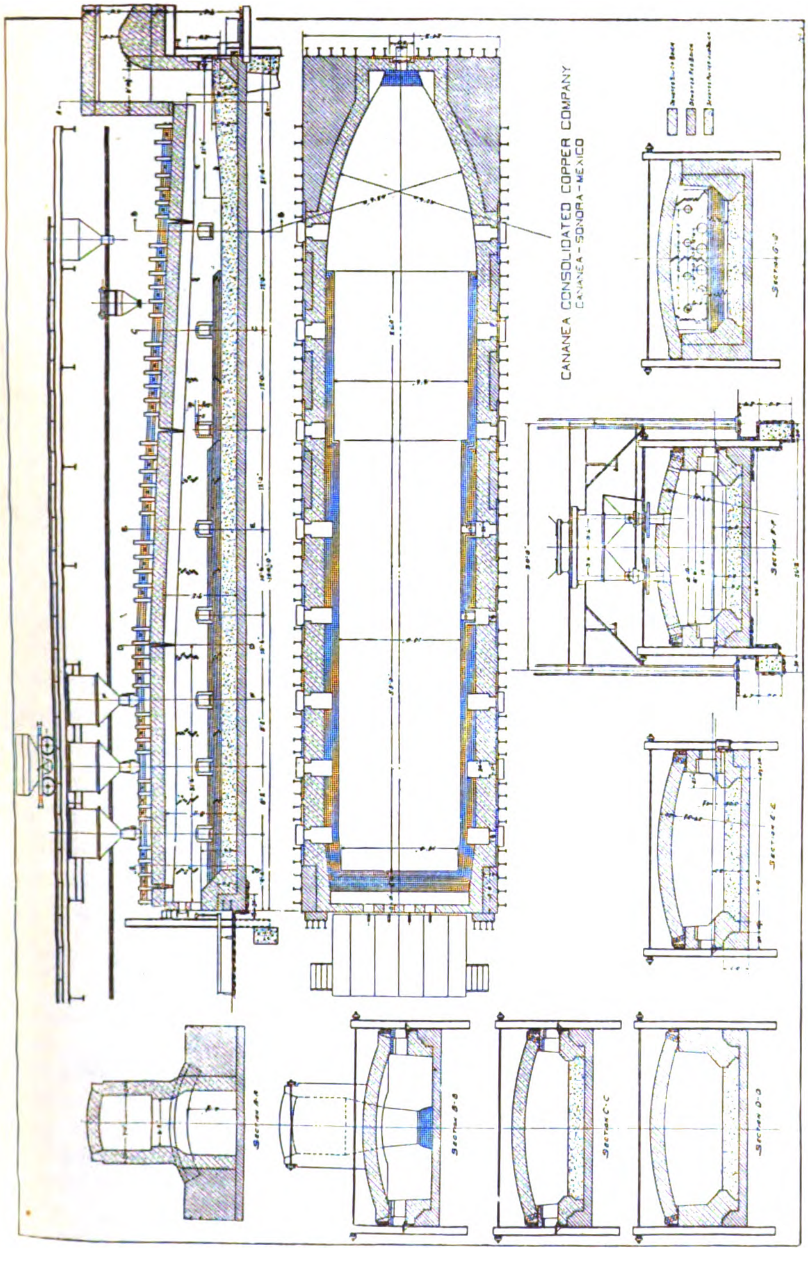

Fig. 22. Cananea Oil-Fired Furnace, Plan and Section.

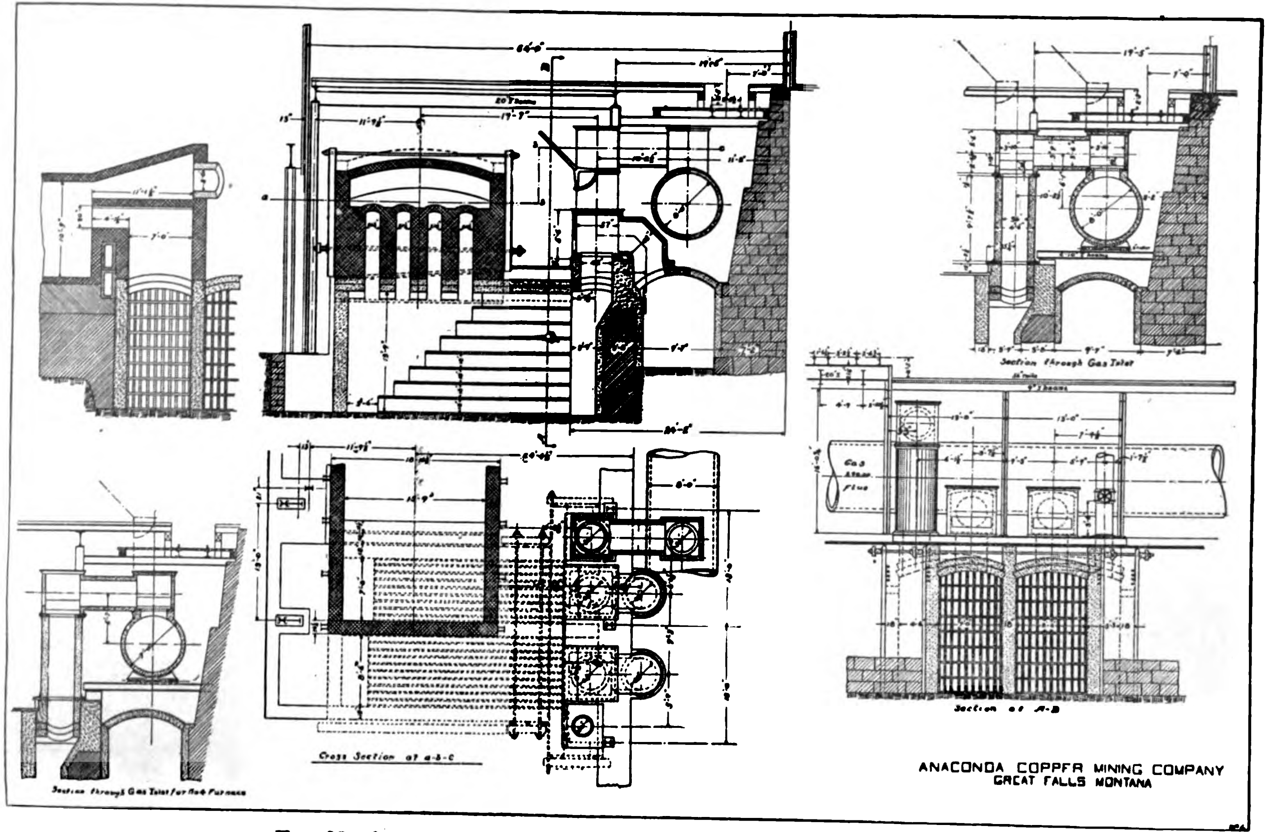

FIG. 23. GREAT FALLS, CHECKER WORK AND SECTION THROUGH FLUE.

charge of the furnaces have had experience with gas-producers in iron works and made a gas of high calorific power.

KYSHTIM FLUE-DUST AND ORE ANALYSES

Sample taken from one month's production of flue-dust:

	Per cent.		Per cent.
SiO_2	8.8	Cu	3.3
Fe	50.9	S	9.6

Sample from 6000-ton pile of flue-dust:

	Per cent.		Per cent.
SiO_2	8.9	Cu	3.3
Fe	48.9	S	10.0

	Cu, %.
Sample one week's production	3.38
Sample of large pile (weight not given)	3.2
Sample of large pile, made in 1907	1.8

Average sample of raw ore used with flue-dust:

	Per cent.		Per cent.
SiO_2	2.1	S	47.0
Fe	38.5	Cu	3.2
CaO	0.5		

At Cananea, Texas oil has been used; and at Humboldt, fuel oil from California. At the plant of the Steptoe company, at McGill, Nevada, fuel oil from California was introduced during 1911, with gratifying results. A great many different varieties of burners were tried, but a simple home-made burner, using air for atomizing the oil, has given the best results. A record performance at the Steptoe plant, McGill, Nevada, on December 17, 1911, is communicated by S. S. Sorensen, the superintendent, No. 1 furnace smelting 660 tons of total charge on an oil consumption of 5/8 bbl. of oil per ton of charge.

RECORD OF RUN OF NO. 1 REVERBERATORY FURNACE AND ANALYSES OF CHARGE.

Total charge per furnace day, tons	666
Oil fired per furnace day, bbl	421
Coal equivalent of oil fired, tons	124.00
Total charge per bbl. of oil, tons	1.58
Oil, bbl. per ton of total charge	0.63
Equivalent gross coal as percentage of total charge	18.60
Components of Charge as Percentage of Total Charge.	
Calcine	60.1
Seconds	16.8
Converter hot slag	9.0
Fettling	3.9
Limestone	9.6
Flue-dust	0.6

SLAG (Assay and Analysis)

Cu	0.40
SiO_2	44.00
Fe	34.30
CaO	8.60
Al_2O_3	7.40
Oxygen ratio	2.72
Grade of matte, % Cu	40.40
Draft:	Water, in.
Bridge	0.35
Throat	0.83
Stack	1.25
Temperature of verb, degrees F	1910
Infusibility factor of calcine	1.5
Burners, large Steptoe, high-pressure	7

At the Washoe plant, at Anaconda, coal is used for fuel, and a record run for one furnace was the smelting of 402½ tons of charge in 24 hours with a fuel ratio of 1 ton of fuel to 6.45 tons of charge. At Great Falls, Montana, the fuel is producer gas, but the producers are too far away from the furnaces to get the best results; new furnaces are to be built at this plant in which the producers will be placed in close proximity to the furnaces and a much greater efficiency will be obtained.

The Canadian Copper Co. is experimenting with pulverized coal as fuel, but the furnaces have not been in operation long enough to give data of any great value. David H. Browne, the superintendent, writes that he is greatly pleased with the performance of the furnaces up to date. The objections to the use of pulverized coal in reverberatory practice have been the settling on the top of the charge of ash and unburned fuel and the clogging of the flues with ashes. The Canadian Copper Co. has a peculiar condition, in that the charge is basic, and that the addition of silica, in the form of coal-ash, is rather welcome than otherwise. To avoid the other objection the coal is pulverized extremely fine, and the surplus ash goes out with the gaseous products of combustion.

At nearly all modern plants, waste-heat boilers are employed. Silica brick is used almost exclusively in the construction of the lining and roof of the modern reverberatory furnaces for copper smelting, and it is now possible to obtain silica brick, of excellent quality, all over the United States at reasonable cost.

COPPER SMELTING AT KYSHTIM

By V. P. Assaieff

(June 29, 1912.)

*Production of copper in Kyshtim began in 1907, but smelting in appreciable quantities began only in 1908. This was at the Soimonoffsky plant. It was subsequently determined to construct a new copper-smelting plant on Karabasheffsky hill with large water-jackets and converters, and also to build a large electrolytic refinery at Lower Kyshtim. It was proposed to produce 400,000 poods of copper per year. In September 1911 I saw at Karabash two new water-jacketed smelting furnaces, and a third similar furnace is now being built, smelting 25,000 to 30,000 poods of ore per 24 hours. The matte obtained is turned to blister copper in two 40-ton converters. At Nizhni Kyshtim the largest Russian electrolytic refinery has been built with a daily capacity of 1000 poods of copper, to be almost doubled, it is said, by the installation of new electric machinery. Just recently at Upper Kyshtim a Siemens' regenerative furnace, 145 ft. long, has been built for smelting fine ore and flue-dust.

Between the smelters and the mines a railway has been laid down, and a locomotive (steam and electric), electric cranes, and pneumatic lifts, are at work. The principal operations are as follows:

Karabash plant—

1. Pyrite smelting in large water-jacketed furnaces to matte with 25 to 35% Cu.
2. Converting the matte to blister copper in large converters.

Lower Kyshtim plant—

3. Refining blister copper and casting in anodes.
4. Electrolytic refining of copper and extraction of gold and silver.

Further important operations carried on are:

Upper Kyshtim plant—

5. Smelting fine ore and dust in large Siemens' regenerative furnaces.
6. Smelting of poor matte in small water-jacketed furnaces.
7. Converting concentrated matte to blister in small converters.

The board is sparing no expense in the search for the required ore reserves. The requirements will reach 90,000 to 100,000 poods per day. On this 400,000 rubles have already been spent, and 300,000.-

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*Translated by the *Mining Journal* from the *Gorny Journal.*

000 poods of ore of 3% copper content have already been proved. This year 200,000 rubles more will probably be spent, and five diamond-drills, two for 500 and three for 1500-ft. holes, are now at work. Three more 1500-ft. outfits may be bought this year. Most of the ore is now produced at the Konyuchovsky mine, then at the Smirnoff and Tisoff mines. Fortunately there is abundance of fluxing quartz and limestone at Kyshtim. The quartz assays 92 to 96% SiO_2, and the limestone 53 to 54% CaO. The pyrite smelting is done with very little coke, at most 1 to 1¼% of the mixture or 1.5 to 1.7% of the weight of the ore, so the annual consumption of coke does not exceed 300,000 poods. It comes from south Russia, England, and Germany, the cost being about the same. Wood is used for the boilers and furnaces, but recently lignite from the Cheliabinsk region is coming more into use.

Copper is now produced in the Karabash, Lower Kyshtim, and Upper Kyshtim plants. The first plant was begun in 1909, and it started full work in January 1911. The chief dimensions of the furnaces are 22 ft. long, 4½ ft. wide, and 19 ft. high. There are 52 tuyeres to a furnace, of which 47 to 48 are always in use; the blowers are of about 600 hp. each, and give 30,000 to 35,000 cu. ft. per minute, under a pressure of 6 to 8 in. mercury column. The blast is cold, and 40,000 to 50,000 cu. ft. is used to 1 ton of mixture, or 50,000 to 60,000 cu. ft. to 1 ton of ore. Gas and dust pass through two wide flues. About 5000 poods of dust is extracted per day with a content of 2.8 to 3.5% copper; thus about 50,000 poods of copper (value 500,000 rubles) is recovered per year. The third furnace (now building) will have the capacity of its dust-trap more than double that of the others.

The efficiency of the water-jacketed furnaces is as follows: The mixture in July 1911 for both furnaces was ore 75%, quartz 15%, limestone 5%, coke 1.25%. The quantity of mixture smelted was 2,040,276 poods, of which 1,512,397 poods was ore. The resulting matte was 135,448 poods, dust 159,530 poods, and slack 1,282,620 poods. The average analysis of the matte was copper 26.53%, iron 41.4%, and sulphur 25.1%. The total of sulphur in the mixture was 608,593 poods, of which 5.63% passed into the matte, 4.38% into the slag, and 1.76% into the dust. The result is considered to have been very satisfactory and to quite justify the expenditure involved in construction of the plant.

To convert the matte into blister copper two Pierce-Smith 40-ton converters are installed at Karabash. From the quantity treated in July 1911, 101,669 poods matte and 24,284 poods of copper was produced. The average copper content of the matte was 26.5%. The special laboratory at Karabash employs five chemists, who made 2500 assays last August.

The blister copper from the Karabash and Upper Kyshtim plants is treated at the Lower Kyshtim electrolytic works. The blister copper is smelted into anodes in regenerative furnaces with wood gas, and the output is 2400 poods of copper re-smelted per 24 hours. One cubic sagene of wood gives 425 poods of anodes averaging 99% Cu.

The electrolytic plant is a large one-story building with 396 baths of wood, lined inside with lead, 41.5 in. deep, 27 in. wide, and 68.5 in. long, in twelve series of three rows of eleven baths each. Each bath contains 9 anodes and 10 cathodes; the former are now being increased to 12. The current in the bath is 0.3 volts. The Upper Kyshtim plant provides the current by gas engines. The electrolyte consists of a solution of sulphate of copper and free sulphuric acid —solution 1.2 sp. gr.—which is led into the bed of the baths and rises up to be siphoned into subsequent series. Thanks to the purity of the anodes, the electrolyte has not been changed since starting the plant. The Upper Kyshtim plant is now an auxiliary one. At Kyshtim there are millions of poods of fine ore besides the millions of poods of dust from the furnaces, the annual production of which, at Karabash, is calculated at 2,000,000 poods. To smelt these and the fine ore a large Siemens-Martin furnace has been built.

WASTE HEAT BOILERS IN REVERBERATORY FURNACE FLUES

By S. Severin Sorensen

(October 11, 1913.)

As there are probably few of those who are interested who are also in a position to make comparisons of actual results between two leading types of boilers when used to recover heat from the flue gases of metallurgical smelting furnaces, the following notes may be useful.

The types here compared are the Stirling and the Babcock & Wilcox. The former are very widely used for this special purpose in copper smelters because of their large flue-gas area and small obstruction to draft. They are specified as class 'S' No. 19 and are 400 nominal boiler horsepower each. There are no baffles used in setting, the gases only taking one curved pass through the tubes before leaving.

The Babcock & Wilcox are a specially designed pattern for this particular installation, described by the makers as $\frac{14-26}{2-36}$ —10—D, each having 4054 sq. ft. of heating surface. No baffles are used in them either but their rows of tubes are staggered in respect to the flow of the gases, which is not the case with the Stirlings. The latter have straight open passages between rows of tubes. This is rendered necessary in order to provide means of getting their bent tubes in and out for repairs.

The illustrations, Fig. 24*a* and 24*b*, show the settings of the two types of boilers in elevation. Inserts A-A and B-B show the arrangement of their tubes by cross-sections on planes parallel to the flow of the bases. As far as constriction of the passage and cutting down of draft goes there is a little in favor of the Stirling type.

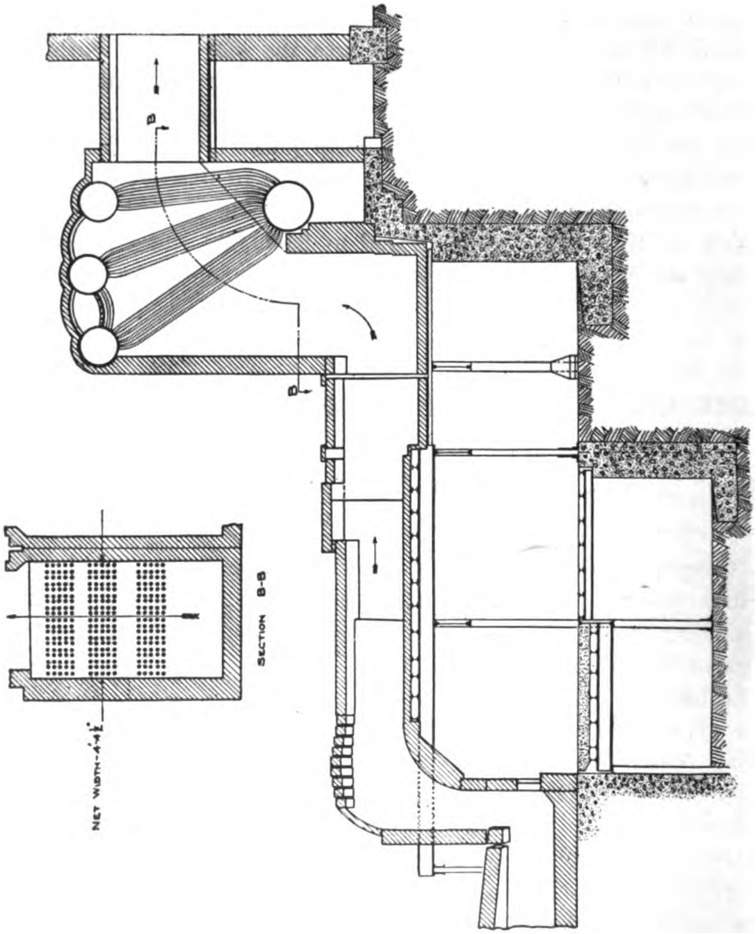

FIG. 24*a*. STIRLING BOILER SETTING.

When it is remembered that they were originally introduced in coal fired plants where draft is possibly both more essential and more sensitive, one reason for the favor they have enjoyed is explained. It will be noted that the Babcock & Wilcox tubes lie flatter than the average of the Stirling bent tubes. The latter are 3¼ in. diameter and the former 4 inch.

In the early days of this plant I was in favor of the Stirlings because they were so much more quickly accessible for cleaning, having only four manholes to remove against 732 hand-holes, each of which had a gasket to be removed and renewed. It thus took longer and cost more in direct charges for cleanings and repairs. In addition to the direct labor and material charges there was the extra time lost in the case of the Babcock & Wilcox boilers, which

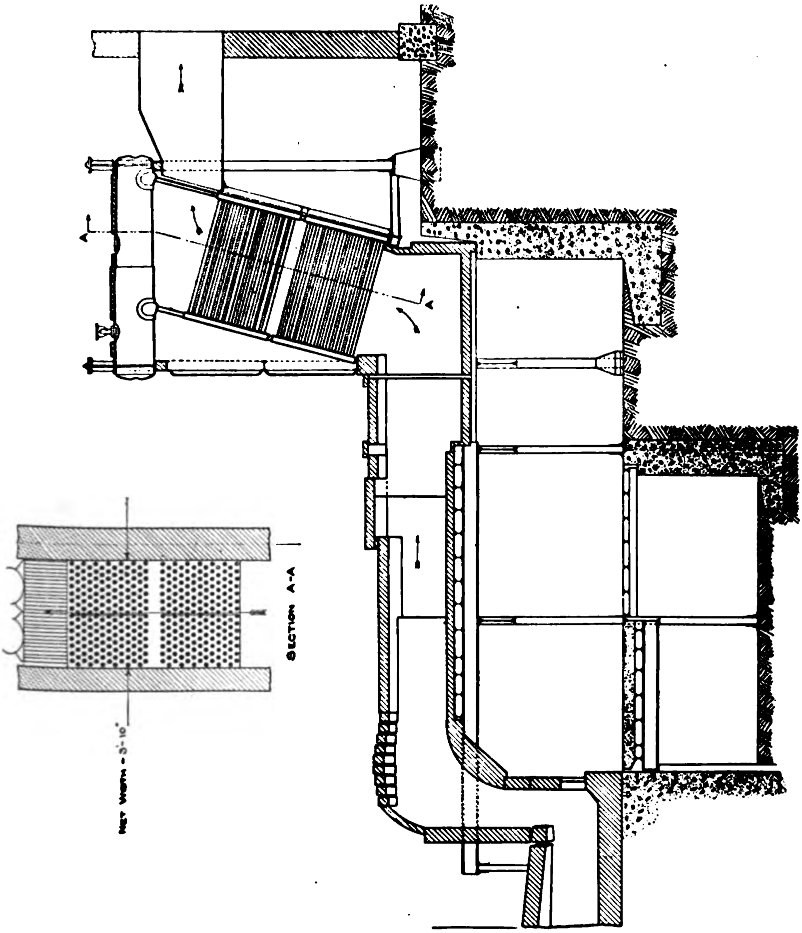

FIG. 24b. BABCOCK & WILCOX SETTING.

took longer than the Stirlings to clean. This extra time meant a loss when the cleaning was not done within the time required for repairs by its smelting furnace. So it looked like a walk-over victory for the Stirling type. For a long time this conclusion held undisputed sway, for in the absence of data it was based on the assumption of equal efficiencies of the two types. It was only after separate meters for measuring the feed water to each battery of boilers were obtained and comparisons of evaporative performances were made that it was seen that the Babcock & Wilcox were giving a higher evaporative duty. At first I suspected the figures. But having changed meters and checked results in various ways the truth was borne in on me that the money was being put on the wrong horse. For the higher evaporation obtained from the Babcock & Wilcox boilers far more than compensates for the extra cost and extra time lost in cleaning and repairing. In proof of this conclusion I append the figures on which it is based. They speak for themselves. I may add that the meters are calibrated every month at various rates by gauging in a measured tank, allowance being made for temperature differences. From the meter readings a deduction is made for blow-downs which are made and automatically recorded twice every 8-hour shift:

	Babcock & Wilcox.	Stirling.
Period covered in comparison, year	1912	1912
Number of boiler repairs	11	23
Average cost of repair and cleaning per repair, opening, cleaning, closing, testing	94.54	54.75
Material repairing	22.81	7.58
Labor repairing	30.19	29.07
Material cleaning	24.63	3.94
Total	$172.17	$95.34
Average number of days in service	83.50	88.40
Average number of days down, cleaning and repairing	7.54	6.56
	91.04	94.96
Time down as per cent total time	8.28	6.93
Number of boiler repairs per 100 days	1.10	1.056
Repair cost per 100 days	$189.39	$100.69
Water evaporated by battery of two boilers per day, lb.	861,111	621,472
Average boiler horse-power	520	375
Water evaporated in 100 days, less days lost cleaning and repairing, lb.	78,998,598	57,840,429
Evaporation lost during cleaning and repairs, lb.	7,112,802	4,306,803
Equivalent oil recovered at 14 lb. H_2O per 1 lb. oil, barrels	16,793	13,203
Recovered, dollars	27,708	21,785

Lost, barrels	1,512	917
Lost, dollars	2,494	1,512
Cost of repairing and cleaning per 100 days	189.39	100.69
Total loss and cost per 100 days	2,683.38	1,612.69
Net recovery in 100 days	25,024.61	20,172.31
Balance in favor of battery Babcock & Wilcox type, per annum	17,710.90	

Other figures point to the same conclusion. For, with the same assumptions and the same methods in testing, checking, and figuring, the pounds of water evaporated from and at 212°F. per pound of oil burned in the smelting furnaces was 3.315 for the Stirling boilers as against 7.91 for the Babcock & Wilcox type. To show that this was not due to deferred combustion favoring the boiler duty at the expense of the furnace smelting through faulty firing it is only necessary to state that for the same period the net oil per ton solid charge for the furnace with the Babcock & Wilcox boilers was only 77% that of the others with the Stirling. It is true the former burned 5% more oil gross per ton solid charge than the others. This may be due to inferiority of the furnace. But it is evident from the above figures that it more than compensated for that extravagance by its greater net efficiency.

From an examination of the two settings as shown in the sectional elevation in the illustrations it is evident that apart from the staggered as opposed to the straight line setting of the tubes the Stirlings lose by reason of the larger percentage of comparatively ineffective tube lengths. This might be improved both in front and at the back where structural conditions and levels permit. And on the other hand the Babcock & Wilcox records for cost of cleaning and repairing could be improved were the hand-hole joints of the metal to metal type instead of being made up with fabric gaskets.

STIRLING v. BABCOCK & WILCOX BOILERS

(Discussion, November 29, 1913.)

The Editor:

Sir—The article on 'Waste Heat Boilers in Reverberatory Furnace Flues,' which you published in the October 11 issue, is a very interesting example of partly calculated data. Mr. Sorensen has shown clearly that a battery of 520 hp. will save, or lose, more money than one of 375 hp. An analysis of his own figures will, however, disprove some of his conclusions.

The following 'per horse-power' figures are taken out with a 5-in. slide rule and so I do not claim great accuracy:

BOILER HORSE-POWER DATA

	Babcock.		Stirling.	
	Total.	Per hp.	Total.	Per hp.
Average boiler horse-power	520.00	....	375.00	....
Number of boiler repairs	11.00	0.02	23.00	0.06
Average cost of repairs, and cleaning and repair, per repair, total, dollars	172.17	0.33	95.34	0.25
Repair cost per 100 days, dollars	189.39	0.36	100.69	0.27
Water evaporated by battery of two boilers per day, pounds	861,111.00	1,660.00	621,472.00	1,660.00
Water evaporated in 100 days, less days lost cleaning and repairing, pounds	78,998,598.00	152,000.00	57,840,429.00	157,000.00
Evaporation lost during cleaning and repairs, pounds	7,112,802.00	13,700.00	4,306,803.00	11,500.00
Equivalent oil recovered, dollars	27,708.00	53.50	21,785.00	58.00
Equivalent oil lost, dollars	2,494.00	4.80	1,512.00	4.05
Cost of cleaning and repairs per 100 days, dollars	189.39	0.36	100.69	0.27
Total lost and cost per 100 days, dollars	2,683.39	5.18	1,612.69	4.30
Net recovery in 100 days, dollars	25,024.61	48.00	20,172.31	54.00
Balance in favor of the Stirling type per horse-power per 100 days	....	....	....	6.00
In favor of Stirling per horse-power year	....	....	....	22.00

The Stirling boilers needed many repairs compared with the others; it seems to be their only weak point, and, judging from my memory of the work at Anaconda, Montana, it seems abnormal. Unfortunately, the first cost of the two batteries of boilers, as installed, is not given, so no comparison can be made of the cost of the power obtained upon a correct basis. Mr. Sorensen, however, concludes with a balance of $17,710.90 in favor of the Babcock boiler. This divided by 520 gives a profit of $34.20 per horse-power year. My analysis indicates that the Stirling boiler is worth about $22 per horse-power year more than the Babcock. The difference is large and is worth attention.

The Stirling is apparently a much smaller boiler than the other. Probably the flue gases in the down-take behind the Stirling are much hotter than behind the Babcock, showing that a larger boiler could be used.

HERVEY GULICK.

Los Angeles, October 18.

STIRLING v. BABCOCK & WILCOX BOILERS

(Discussion, February 21, 1914.)

The Editor:

Sir—There are some calculations, the value of whose results would not be one whit affected whether a 5-in. or a 5-mile slide rule were used in working them out. If the assumptions and premises are wrong, accuracy in the figures counts for naught. This is the trouble with Mr. Gulick's letter in your November 29 issue on waste heat boilers in reverberatory furnace flues. It seems to me "a very interesting example of partly" digested reading, and of taking up the rôle of critic without making sure of one's facts.

His 'per horse-power' figures mean nothing, or rather they are quite misleading. The description of the boilers showed that they were all of the same nominal horse-power, namely, 400, so the evaporation duty of the two types under the given conditions of service were approximately as 1.37 to 1 per unit of heating surface for the B. & W. and Stirling types respectively. Moreover, the proper unit for a basis of comparison is not the horse-power, but the furnace. My notes were to show, and I still think do correctly show, which of two types of boiler attached to similar smelting furnaces saved most money per furnace under similar conditions, over similar periods.

S. SEVERIN SÖRENSEN.

Rancagua, Chile, January 5.

COPPER SMELTING PRACTICE IN THE SOUTHWEST

(October 4, 1913.)

By Thomas T. Read

A concept which has proved useful to geologists is that of the existence of petrographic provinces, or areas in which characteristic and often related types of rocks occur and in which characteristic minerals and mineral associations can be discerned. It therefore

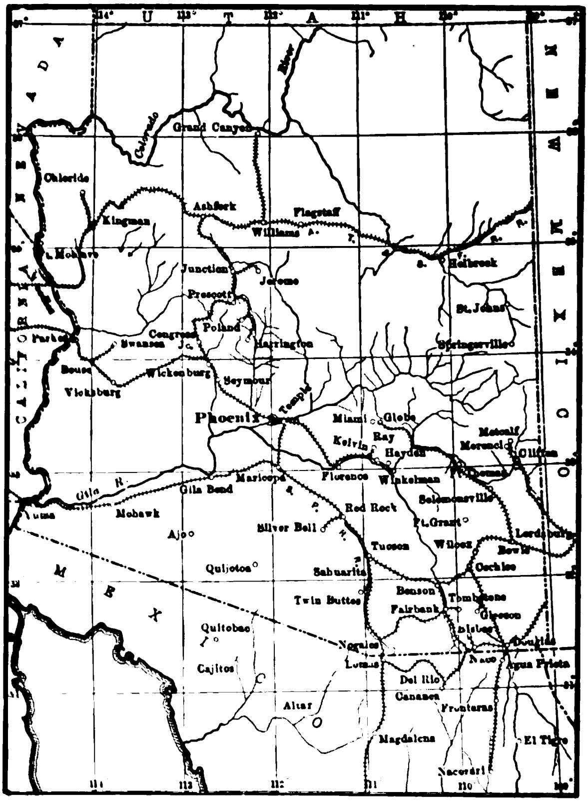

Fig. 25. Map of Arizona.

follows that metallurgical provinces can also be distinguished, since the choice of metallurgical methods employed in the treatment of [illegible] the ores is often decided by the characteristic minerals present in the ore. The states of Arizona and New Mexico, together with [illegible] parts of Mexico, may thus be considered as roughly [illegible] such a metallurgical province, especially if the inquiry [illegible] the ores of copper. In this area, in 1912, approximately one-third of the total copper output of the United States was produced, chiefly from ores that exhibit many characteristics in common. It is natural, therefore, that the metallurgical methods employed should also exhibit common characteristics. It is not my purpose to attempt to present here any profound research into the fundamental relations between the climatic, topographic, geologic, and economic conditions here prevailing, and their effects, as exhibited in past and current practice, but rather to indicate in a somewhat sketchy fashion only a few of their more salient features. It will be impossible because of the limits of space to enter into any discussion of the history of the area under discussion, and since this has recently been admirably done by James Douglas*, than whom no one could speak with more authority, there is indeed no necessity to make the attempt. These remarks are, therefore, based upon present practice as it has crystallized in the light of years of experience.

Bounded by the Rocky Mountains on the east, and the Sierra Nevada on the west, the region under discussion may be regarded as in part a southward extension of the Colorado plateau and in part the northern portion of the Mexican plateau. The Colorado river may indeed be considered to form its northern and western boundary as it threads through its profound cañons in its course from its headwaters in the Rocky Mountains to the Gulf of California. Though bordered by such a mighty river, the entire region is arid, and in some parts a true desert. It will be remembered that the prevailing westerly winds from the Pacific Ocean exhibit their maximum force, with its resulting damp climate, from Juneau south to San Francisco. South of San Francisco the coast sweeps away eastward, while the trade winds swirl away to the south and finally west. As a result, the climate of that part of the United States lying south of 40°N. latitude and between the Sierra Nevada and the eastern front of the Rocky Mountains is almost entirely arid. This peculiarity not only affects the character of the ore deposits, as altered by subaerial agencies, but by its effect in such important economic questions as food, water, and timber supply, and character of population serves to determine in a marked degree many of the salient features of the mineral industry. In its flora and fauna, including in the latter its human population, this region differs in many respects from every other part of the United States, with the exception of the adjoining parts of California.

The copper deposits found in this region are, as might be expected,

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*'The Copper Queen Mines and Works,' *Trans.* Inst. Min. and Met., March 1913.

characterized by outcrops of oxide and carbonate minerals, Arizona being famous for its specimens of malachite and azurite. At varying, but usually considerable depths, the oxidized minerals give way to secondary and even primary sudphides. The orebodies are of such multiformity and occur in such a variety of enclosing rocks that no attempts at generalization would be useful. These orebodies are commonly regarded as marked examples of the concept that copper orebodies are typically marked by a surface zone of oxidation, a succeeding zone of secondary enrichment, and a bottom zone of primary ore. This conception, at first extremely valuable, is beginning to outlive its usefulness and remains to hamper free thought. In a recent paper on the Bisbee deposits, A. C. Notman employs the term 'underground gossan'; surely a most confusing name for an obscure concept. As a matter of fact oxide minerals occur both above and below the sulphides. Thus, at the Chino mine, at Santa Rita, in New Mexico, the oxide ore in places extends to 1100 ft. in depth, while sulphides occur close to the surface and overlying the oxides.

Where the ores are rich enough to stand the cost, they are mined by methods appropriate to the size and character of the deposit and smelted directly. In the early work so small an amount of sulphide was available that the oxides were smelted to yield black copper. In practically every district enough sulphide ore is now available to allow smelting with the formation of matte, and in many cases a certain amount of sulphide has to be roasted, though in others the charge balance, for blast-furnace smelting, is somewhat short of sulphur. The tendency of present practice is toward concentration of the low-grade sulphides and leaching of the oxidized ore. More than a half dozen large concentrating plants are already at work, but, with one exception, the leaching work has as yet not progressed beyond the experimental stage. Most significant in its probable great importance is the experimental work with the flotation process which is being carried on at the Inspiration mine. If the 600-ton plant now in course of construction proves as successful as the experiments made with a 50-ton plant, it is unquestionable that profound changes in present practice in this district are to be expected.

There are now in operation in the district under review eleven important smelting plants, and preliminary plans for the construction of a twelfth are well advanced. The El Paso smelting plant of the American Smelting & Refining Co. is included in this list, since it draws its supply of copper ore chiefly from this area, though situated beyond its margin. As typical examples of the range of practice, the

Old Dominion smelter at Globe and the A. S. & R. smelter at Hayden may be cited. The ore supply of the former is lump ore, which is smelted in blast-furnaces. At the Hayden plant all the ore is subjected to preliminary wet concentration, the resulting concentrate being roasted and smelted in oil-fired reverberatory furnaces. Present conditions all tend toward an increase of reverberatory smelting. The amount of ore of high enough grade to permit direct smelting is limited, while, on the other hand, at the Chino 90,000,000 tons of concentrating ore has been developed, at the Ray 84,000,000 tons, at Inspiration 45,000,000, and at Miami 21,000,000. It is evident, therefore, that future tendencies will be toward an increase of reverberatory smelting. L. D. Ricketts has given the cost of reverberatory smelting at Cananea during the period from February to July 1911, as $2.22 per ton, from which must be deducted $0.82 as a credit for the value of the steam generated by the heat recovered in the waste-heat boilers, making the net smelting cost $1.40 per ton smelted. Of the total amount thus smelted, one-third was flue-dust, and the apparent cost of blast-furnace smelting must be increased by an amount equivalent to the cost of smelting in the reverberatory that excess of flue-dust produced in blast-furnace smelting as compared with reverberatory smelting. The average milling cost at the Chino last year was 58½c. per ton, so that it is evident that the field of milling and reverberatory smelting is likely to continue to make inroads into that of direct smelting. With so much as an introduction, I shall briefly describe each of the principal smelters in this district, without going into detail except as to such features as may be peculiar to that plant.

At El Paso the American Smelting & Refining Co. has recently added copper smelting equipment to a plant which has for years smelted lead ores. Two blast-furnaces, 40 by 144 in., smelt lump custom copper ore, derived from various parts of Arizona and New Mexico, yielding a matte of about 45% copper which is blown to blister in two Pierce-Smith converters; a newer 12-ft. Great Falls converter not yet having been put into service at the time of my visit. The main matte supply for the converter is derived from two 19 by 104-ft. reverberatory furnaces. These are oil-fired, but as this method of firing is general practice throughout the area it will not again be referred to except to give details of practice. Most of the oil used for firing is obtained from the Mexican fields, though Texas oil is also available, and has the advantage of not requiring as much preheating, as it is lighter in specific gravity. The reverberatory furnaces yield a 40% matte, the matte-fall amounting to about 30% of the charge, each furnace handling about 300 tons per day. The capacity is somewhat in excess of this, but was limited at the time of my visit by the capacity of the roasting plant. This consists of three 4-hearth Wedge furnaces, 22½ ft. outside diameter, and two more are in course of erection. These furnaces are designed to handle 100 tons each per day, but are being operated at a capacity of 130 to 150 tons each per day. It was originally expected that the average sulphur content of the Chino concentrate would not be so high as to require much sulphur elimination, and that roasting of only part of the total tonnage would be required. In actual experience it was found that the sulphur content ranged between 12 and 22%, according to the part of the ore deposit from which the mill feed is obtained; the necessity of directing the steam-shovels so as to carry on development work in the most efficient manner, making it impracticable to control the mill feed. Running a Wedge furnace at so much in excess of its intended capacity makes it impossible to completely dry the concentrate upon the top hearth. It therefore passes through the slot upon the third hearth in a somewhat sticky state, and in order to cause it to pass freely it was necessary to remove the distributer plate. This causes the concentrate to pile up on the third hearth, and from time to time coarse fragments roll between the central cylinder and the hearth ring, much to the disadvantage of the latter. Various small alterations have been made to minimize this and other minor troubles, but it is on the whole remarkable that a roasting furnace can be pushed so far beyond its designed capacity with comparatively little alteration beyond the use of an additional burner on the second hearth. This plant is under the direction of Kuno Doerr, general manager; J. J. Ormsbee, superintendent; H. F. Easter, assistant superintendent; and A. F. McCormack, assistant superintendent of the copper department.

The next smelting plant, proceeding to the west, is that of the Arizona Copper Co. at Clifton, Arizona. This is at the end of a rather long branch railway line, owned by the Company, which connects with the Union Pacific at Lordsburg, New Mexico. This is an old plant, shortly to be abandoned, which excites interest chiefly by the ingenuity which has been expended in compressing it into a narrow and inconvenient site at the junction of Chase creek and the San Francisco river, the boilers for steam supply being scattered about in niches between other equipment. It is also unique as to its chimney, which consists of a shaft, or winze, to the top of the nearby mountain. The experiment is one not likely to be repeated, since such a chimney defies attempts to repair or enlarge it. A new plant has been designed by Repath & McGregor, under the direction of

L. D. Ricketts, consulting engineer to the Company, and is already partly constructed, the site being farther down the San Francisco river, where ample dump room is available. The reverberatory furnaces will be 22 by 100 ft., space being provided for three such and for eight 21½-ft. Herreshoff roasters. This plant is similar in its design to the new plant of the Calumet & Arizona, which will be described more fully later.

An especially interesting feature of the work at the old plant is the leaching of oxidized copper ore with sulphuric acid. The acid, of 52°B. specific gravity, is made at the plant from pyrite obtained from one of the Company's mines. The jig tailing from milling operations, containing 2 to 2½% copper, 0.4% S, 60% SiO_2, and 16% Al_2O_3, is charged into vats and leached for 12 hours, the solutions being so managed that the first strong solution contains very little free acid, and, after decanting, its copper content is precipitated by passing it through a trommel filled with scraps of wrought iron and steel. The precipitated solution then goes to waste, and the wash water used in the vats is strengthened by the addition of acid to yield fresh leaching solution. This method can only be used on ore which is high as SiO_2, otherwise the loss of acid through the formation of Al_2SO_4 and $FeSO_4$ becomes prohibitive. The acid consumed in leaching is given as 2½ lb. per pound copper recovered. Allowing for the cost of the iron used for precipitation and labor there can be no great profit on the operation except as an adjunct to the wet concentration of ores of suitable composition. This Company is under the direction of Norman Carmichael.

Just across the river from the new Arizona Copper plant is the mill and smelter of the Shannon Copper Co. This consists of two blast-furnaces 42 in. by 15 ft. and one 44 in. by 30 ft., the latter handling about 700 tons of charge per day. These furnaces are provided with cast steel crucibles, which have given trouble by cracking. Pyrite is brought from Gleason, Arizona, to yield the sulphur required for the formation of matte. About 500 tons per day of the ore-supply contains no sulphides and is smelted directly; the remainder, 300 tons per day, is concentrated 10 into 1. The fine concentrate is then dewatered and mixed with flue-dust before charging into the blast-furnaces. Two small stands of acid-lined converters are sufficient to handle all the matte produced. This plant is under the management of J. O. Bennie, who deserves the greatest credit for the skill with which he has brought into shape a rather unsatisfactory plant and a mine which is on the border line of profit. Mr. Bennie has carried on an extensive series of leaching experiments; the general method of attack being roasting in specially constructed heaps, the progress of the operation being so controlled as to completely convert the copper into the sulphate, and to avoid the formation of ferric sulphate, as this consumes iron in the subsequent precipitation of the copper. No details of results obtained have been released for publication, but the method seems to offer much of promise. The experimental work is exceedingly tedious, as the time required to build and burn the heaps is great.

At Morenci, a few miles to the west of Clifton, but at a considerably higher elevation, is the smelting plant of the Detroit Copper Mining Co. This presents a number of features of interest; for example, the smelting of concentrate in blast-furnaces. All the mine ore is milled and the only lump ore smelted is that produced by lessees. Smelting is now done in a 42 in. by 22-ft. blast-furnace, which is driven slowly, smelting 350 tons per day. The concentrate contains 15 to 17% copper and 15% SiO_2; about 50 tons per day of limestone is used as flux. The slag contains 7.2% Al_2O_3 and 2% MgO, and the matte, of which 60 to 70 tons per day is produced, contains 42% copper. It is thus seen that a high degree of desulphurization must be maintained. The tonnage to be handled is too small to justify building reverberatory furnaces. It is planned to build a 33-ft. furnace which will smelt even more slowly, the idea being not to increase the desulphurization, but to cut down the dust losses. It would seem, on the face of the matter, that this end might be better obtained by first sintering the concentrate upon Dwight & Lloyd machines, but careful study of the local problems was made by the management, and it is safe to assume that the methods employed have only been adopted after due consideration. Converting is now done in acid-lined shells, but a Great Falls converter will perhaps be built, largely for the purpose of decreasing the amount of seconds to be rehandled. Another interesting feature of practice here is the use of Crossley and American Crossley gas engines for power supply. The gas is generated from bituminous coal, obtained from Dawson, New Mexico, in a Loomis generator. This is at the mine and the gas is brought to the smelter in pipes which sometimes give trouble in cold weather, due to freezing. The smelter is provided with round steel bins, which were built some years ago, and must be among the first bins of this type ever built. A. T. Thomson is manager of this plant, and E. W. Honeyman is superintendent.

The Old Dominion smelter, at Globe, is at the end of a 125-mile branch of the Southern Pacific, the Arizona Eastern, which connects with the main line at Bowie. At this smelting plant there are six blast-furnaces, of which three or four are usually in use. All, except one, are 48 in. by 16½ ft., the exception being 19¼ ft. long. An average daily tonnage of 325 per furnace is smelted. The smelting column is about 12 ft. high and 22 oz. blast pressure is used, with a coke percentage of 12.6. These features are due to the shortage of sulphur on the smelting charge, it being necessary to bring sulphide ore from Bisbee to make up the deficiency. The flue-dust made amounts to about 8½% of the tonnage smelted; this is drawn from the chambers and shipped to Douglas for smelting in the reverberatory furnaces. The matte averages 44% Cu and is handled by an 8-ft. Great Falls converter. This type of converter is in common use in the district, as its form, a vertical cylinder, offers obvious mechanical advantages over the horizontal cylinder of the Pierce-Smith converter. Since the basic lining is expected to maintain itself over a long period it should be of such a shape as to offer the maximum of strength with the minimum of exposed surface. Part of the tonnage smelted at this plant is lump ore, and part concentrate; about 500 tons of ore per day being concentrated 4 into 1. The relative proportion of concentrate to lump ore in the furnace is 1 to 4. P. G. Beckett is manager and L. O. Howard smelter superintendent.

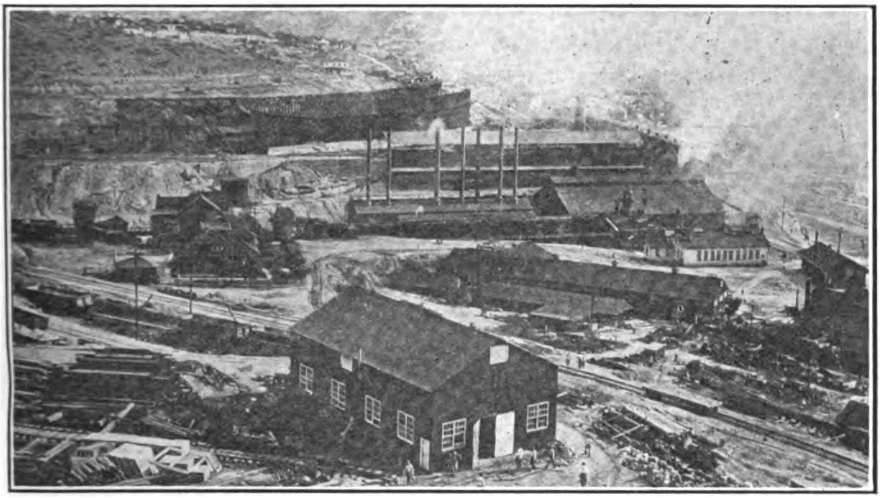


FIG. 26. OLD DOMINION SMELTER.

A short distance to the west the Miami is milling all its ore and shipping its concentrate over 200 miles, to Cananea. The Miami ore is also short of sulphur and the Old Dominion company, which has been involved in litigation for many years, apparently does not care to undertake the construction of a reverberatory plant for the handling of custom ore. As within a comparatively short time the Inspiration, which adjoins the Miami, expects to be producing concentrate from an 8000-ton mill, the International Smelting & Refining Co. will shortly undertake the construction of a new smelter at Burch, halfway between Globe and Miami, to consist of roasting and reverberatory smelting equipment.

About 20 miles directly south, but across a range of mountains, is the Hayden smelter of the A. S. & R. Co. This was recently built to handle the concentrate from the Ray Consolidated Copper Co., which it directly adjoins. The wet concentrate brought in railway cars, is allowed to dry as much as possible, then is sampled, dropped into bins, and when wanted for smelting is drawn out upon belts, which discharge into the McDougall roasters. Of these, five are 24-ft. and have 5 hearths, and three are 19-ft. and are 6-hearth furnaces. The concentrate contains 24% sulphur, and it is necessary to burn a little oil from time to time, on account of the high moisture. Each McDougall handles about 90 tons, the top of one 5-hearth roaster being used for drying. The reverberatory furnaces are 19 by 112 ft., and smelt about 350 tons per day each, including seconds, with a consumption of 0.8 to 0.9 bbl. oil per ton smelted.

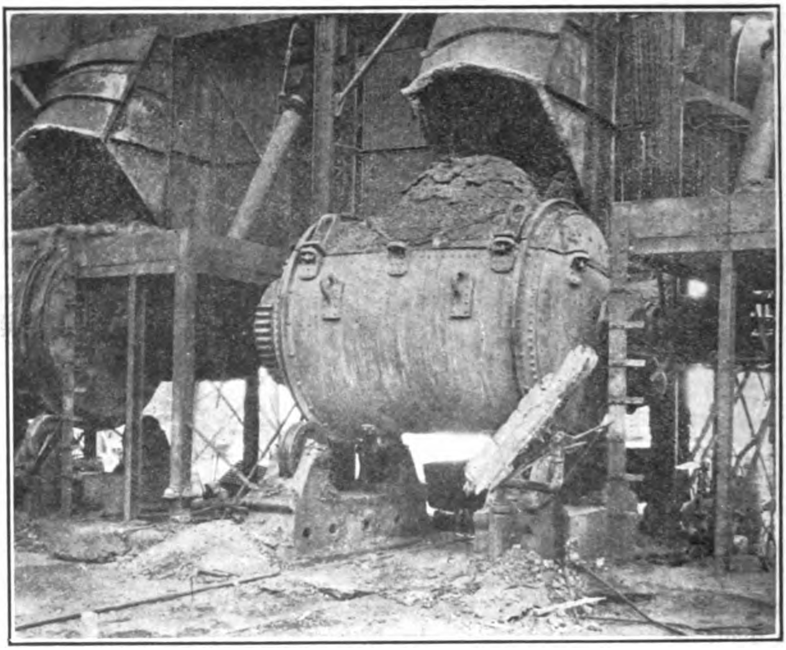


FIG. 27. CONVERTER STAND, CALUMET & ARIZONA SMELTER.

The steam recovered in the waste-heat boilers (Stirling) goes to the main power-plant, and suffices to meet all the power requirements of the smelter, with a substantial excess. The matte-fall amounts to about 25 to 40% matte. This is blown to blister in either one of two Pierce-Smith converters or in one 12-ft. Great Falls type converter. Slime concentrate is used for fettling the furnace. The blister is poured into an oil-fired receiving vessel in order to keep it hot while pouring the anodes. This smelter would naturally be the one for handling the Miami and Inspiration concentrate, but in order to get over the intervening short distance, several hundred miles of railway haul would be necessary. A short line connecting Winkelman with a point on the line from Bowie to Globe would form a short and convenient connection, but as the line would have to traverse the site reserved for a dam in a future government reclamation project there is but little likelihood that it will ever be built. J. B. Haggie is superintendent of this plant.

At Douglas, which is on the border line between the United States and Mexico, there are two smelting plants. The new plant of the Calumet & Arizona, which was designed by Repath & McGregor under the direction of J. C. Greenway and L. D. Ricketts, embodies all the important features of the New Arizona Copper Co. smelter and also that of the United Verde, at Jerome, therefore I shall describe it at some length. The blast-furnaces, of which there are two, are 48 in. by 40 ft. and are designed for the maximum convenience in charging and tapping. The ore from the beds is drawn out upon a series of belts, which in turn discharge upon a main belt which elevates the ore, fuel, and flux to the bins above the blast-furnaces. Each furnace is provided with four charging cars, each 20 ft. long, which rest upon track scales beneath charging hoppers, which are filled by belts provided with distributers, running between the pairs of hoppers. When the charge-car is filled, it only requires to be moved a short distance forward in order to discharge directly into the furnaces. The estimated capacity of these furnaces is 800 tons each per day. The settlers are oval in shape, the matte being tapped directly into pots set in the main crane aisle, the slag being drawn off into large pots and hauled by an electric locomotive to the slag dump. The reverberatory furnaces, of which there are four, with space for a fifth, are 19 by 100 ft., and present no special features beyond the convenience of their arrangement with respect to the rest of the plant. The calcine is brought directly over the charge hoppers in cars from the roaster. The converter slag is poured by the crane directly in at the feed end, and fettling and fluxing material, brought up by the same belt which elevates the lump ore, is drawn about as desired in small cars. Of converters, six stands of 12-ft. Great Falls type are provided.

The converter flue is hung directly upon the frame of the building and a hopper for the silicious addition is placed immediately above, delivering the silicious material directly through a chute into the converter, without its being necessary to move the shell from its normal blowing position. These bins are filled by a conveyor, thus relieving the crane of much work, so that one is able to serve the whole main aisle. Two casting machines of the Anaconda type will be provided, one at each end of the main aisle. In order to eliminate labor in breaking up skulls, a heavy steel plate with 8-in. holes is placed beneath a miniature pile-driver; as fast as the material is sufficiently reduced in size it falls through the plate into a hopper below. For roasting concentrate, twelve 21½-ft. Herreshoff air-cooled roasters are provided. These have six hearths and the roof is also used for preliminary drying. These as well as the furnaces are provided with dust flues hung in a steel frame, and with bottoms made from hollow tile. As this tile is manufactured at El Paso, steel plates will be substituted for it at Jerome, on account of the

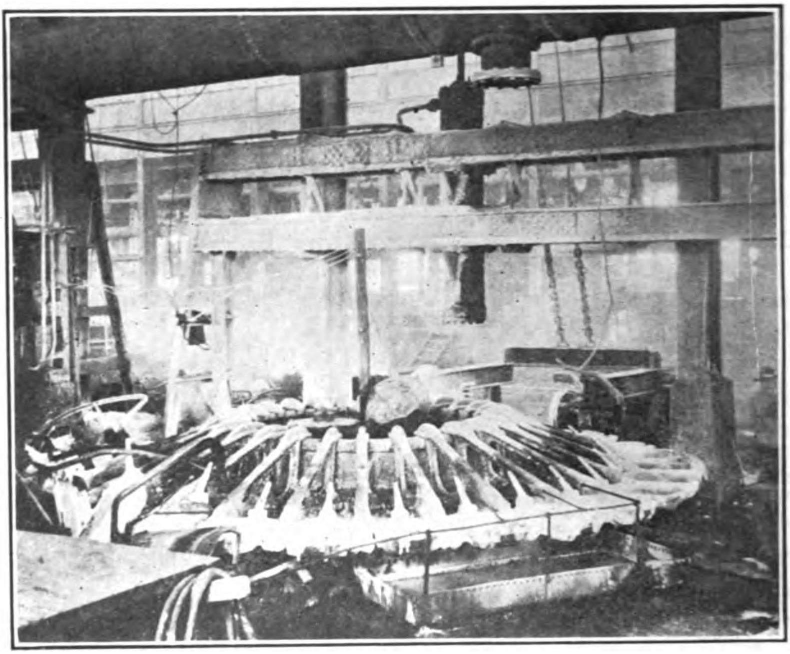


FIG. 28. AUTOMATIC COPPER INGOT MOLD, COPPER QUEEN SMELTER.

relative freight cost. The gases are distributed throughout the settling chamber by a row of flat bands, hung near the entrance; by sliding these the gases can be uniformly distributed over the whole cross-section. At the middle of the dust chamber a 20-ft. section is hung with No. 10 wires suspended from the roof. The plant has been designed to secure a maximum of capacity with a minimum of labor and equipment, thereby attaining low operating costs. The success of such an arrangement is largely dependent upon an accurate foreknowledge of the conditions to be met, and the whole forms an excellent example of advanced modern practice. J. C. Greenway is manager.

The other plant at Douglas is that of the Copper Queen company, built about ten years ago, and exhibiting characteristic differences from the plant just described. The most marked of these is in the bedding system employed. Long stalls, 21 car lengths long and 50 ft. wide, are provided with tracks upon the bottom for reclaiming, and tracks at a higher elevation on each side. Converter slag is placed on the bottom, lump ore is distributed over the top, and a small amount of concentrate is intermixed. About two days after a bed is completed, reclaiming of it is begun, using small steam-shovels for the purpose. The total storage capacity is 100,000 tons. The cost of reclaiming is given as 3c. per ton, 9 days being required to exhaust a bed. This ore is smelted in ten blast-furnaces, five of which are 20 ft. long and the others 18. The larger furnaces smelt about 360 tons per day each, using 30 oz. blast pressure, and making a 40% matte. The smelting charge is rather high in alumina, yielding a somewhat viscous slag.

The flue-dust and roasted concentrate from the Nacozari mines is smelted in two reverberatory furnaces, 19 by 95 ft., which handle about 300 tons per day each, using 0.86 bbl. of California crude oil per ton smelted. This plant is unusual in that Erie boilers, instead of the usual Stirling, are used to recover the waste heat of the reverberatories. As the result of extended experience the mechanical engineer of the plant reports that these boilers are of almost equal efficiency, lower first cost, and are easier to clean. The usual difficulty experienced with the Erie boiler is that it primes badly when it is forced, and a waste-heat boiler usually works under a heavy overload. The draft pressure on one reverberatory is 0.5 in. and on the other 0.3 in., the difference being due to a fuel economizer placed after the waste-heat boiler in the second. Better results are obtained with a higher draft pressure, but may be due in part at least to the fact that this furnace has been in operation for a longer period. It was necessary to raise the roof of these furnaces in order to give more combustion space, and it may be expected therefore that the 22-ft. reverberatory at the New Arizona Copper Co. plant will give somewhat better results than a narrower furnace. The necessity for ample space to ensure the proper combustion of a liquid fuel has been discussed at some length in recent issues of the *Mining and Scientific Press* and need not be further referred to here. This plant is under the general direction of Walter Douglas. S. W. French is manager, and Frank Rutherford superintendent.

The smelting plant at Cananea, a short distance south of the Mexican border, may be regarded as the type of the district, since it is here, under the direction of L. D. Ricketts, that the methods and equipment have been developed which are embodied in the newer plants now under construction or recently completed. The lump ore is smelted in eight blast-furnaces, 48 by 210 in., with a height of charge above the tuyeres of 7 ft. and using 26 oz. blast pressure. Each furnace smelts about 300 tons per day, using 9% of coke, and yield flue-dust amounting to about 10% of the charge. The material charged to these furnaces consists of lump ore, some custom ore, converter slag, and about two-thirds of the total amount of concentrate made in the Cananea mills, 600 tons per day in all. The charge is delivered into bins placed above the charge floor, and is run directly into the furnaces through chutes. The system of bedding for mixing the charge has been described in detail by M. J. Elsing, and will not be further referred to here.

The Cananea concentrate carries 30 to 35% sudphur and is roasted in 6-hearth 18-ft. McDougall furnaces, 10 being provided and 6 in use. The sulphur content is reduced to 6 to 7%, each roaster handling about 40 tons per day. The calcine and about 75 tons per day of raw Miami concentrate are smelted in two 19 by 100-ft. reverberatory furnaces which handle 260 to 275 tons per day each. The fuel used is California crude oil, less than 0.8 bbl. per ton smelted being required, having been reduced to that figure from 0.9 bbl. The oil is heated by a steam coil in a tank and is injected by air at 13 lb. pressure, the former burners which used both air and steam for injection having been discarded. Air for combustion is admitted through a checkerwork of brick above the burners, this arrangement producing a more even distribution of heat throughout the furnace. The Miami concentrate, still wet, is dropped against the walls to serve as fettling, and is better for this purpose than silicious ore, which is likely to produce 'floaters.' This concentrate only carries about 21% S in its raw state.

The remainder of the Miami concentrate, which amounts to 125 tons per day in all, is blown into the converters by use of an ingenious

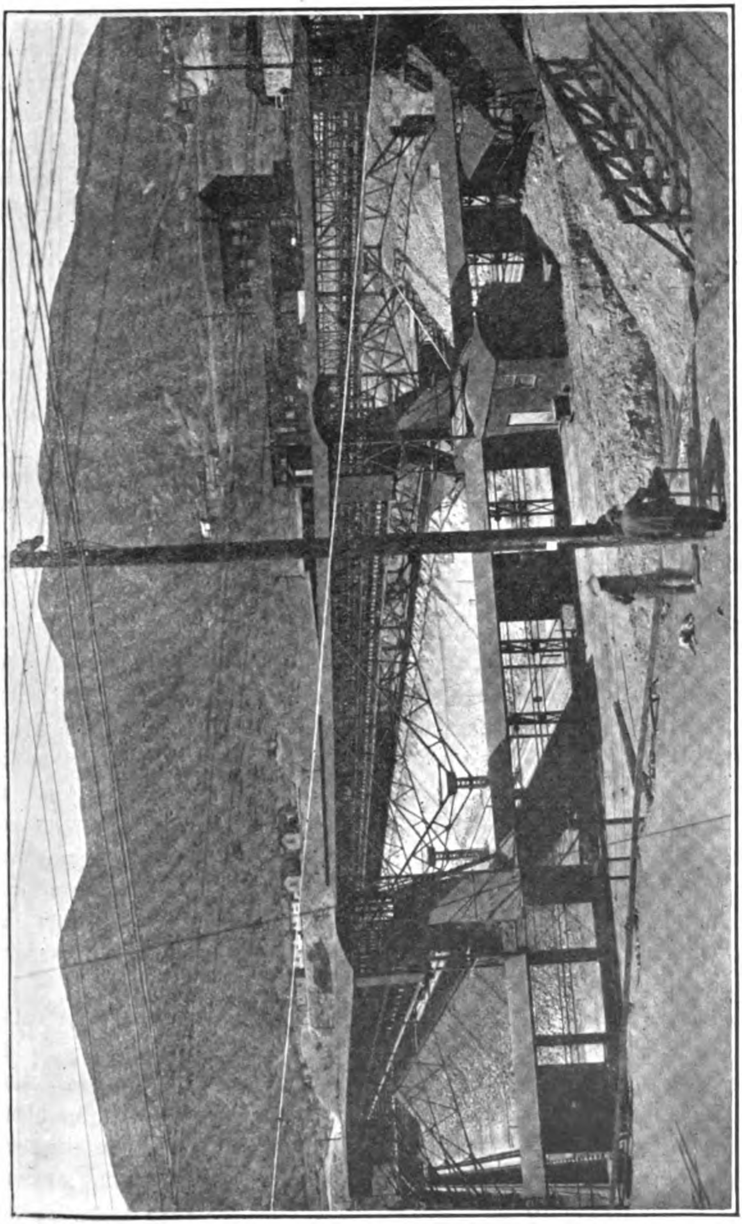
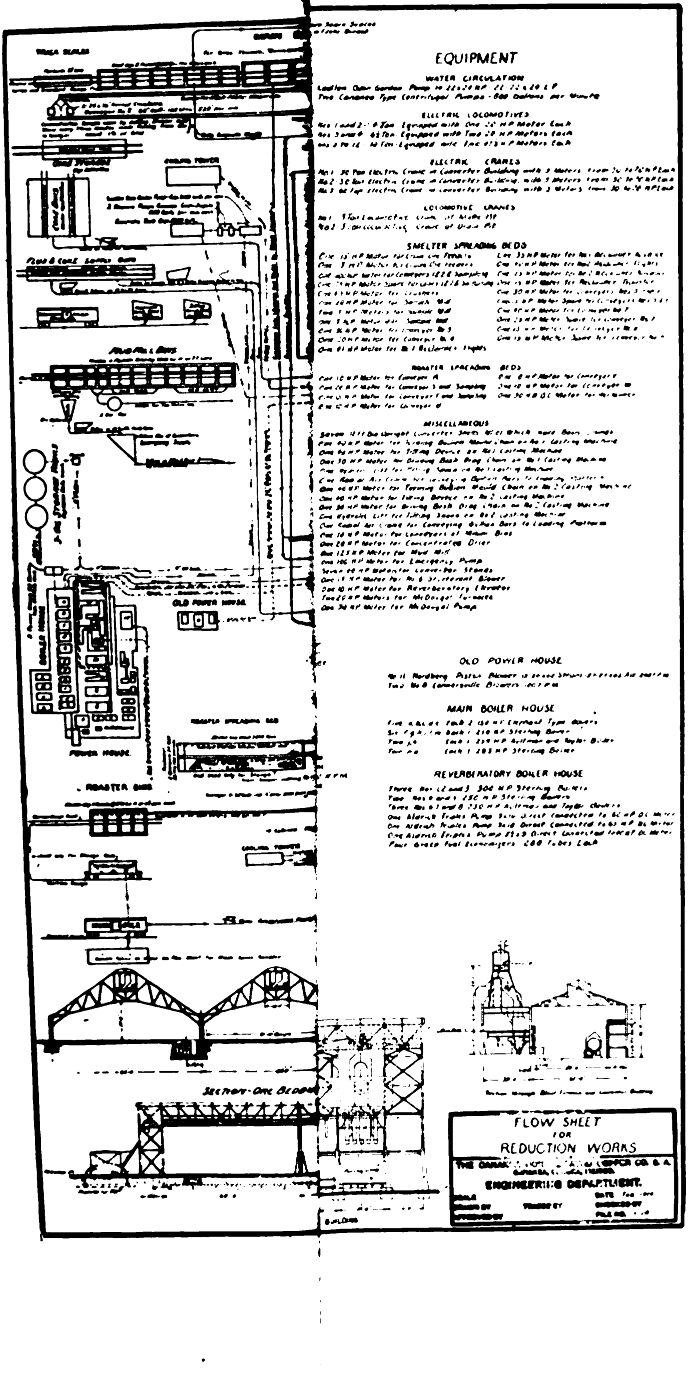


FIG. 29. BEDDING SYSTEM FOR COARSE ORE, CANANEA CONSOLIDATED COPPER CO.

blow-pipe. Three 12-ft. Great Falls converters are in use, and three more are under construction. The matte to be handled amounts to 400 to 500 tons per day, yielding about 150 tons of blister copper. The concentrate is first dried and in the course of the blow a certain amount of it is blown through the tuyeres, one at a time, in order to prevent local chilling. In this way 50 tons per day of concentrate is smelted, its silica content serving to displace a corresponding amount of silicious material which would otherwise require to be introduced through the mouth. The two reverberatories are provided with eight waste-heat boilers, three being 250 hp. each, two 225 hp., and three 200 hp. These yield about 1000 hp., the remainder of the power being furnished by a battery of oil-fired boilers. With admirable foresight the tailing from the milling plant has been impounded behind sand dams, looking forward to a time when it will be possible to recover a further part of its copper content.

At Jerome is the smelting plant of the United Verde. The old plant is just in process of being abandoned and a new plant is under construction. This will not differ materially from the new Calumet & Arizona plant, except that Wedge furnaces will be used for roasting. At Humboldt is the smelting plant of the Consolidated Arizona. This consists of blast and reverberatory furnaces, only part of the equipment now in use. I have not seen this plant, which is now practically shut down, pending changes in practice. An important project which has not yet reached the construction stage is the Ajo or New Cornelia property, which has been acquired by the Calumet & Arizona, after extensive drilling operations had disclosed the existence of large reserves of oxidized ore. Plans for working have not yet been fully matured, but it has been proposed to leach the ore with sulphuric acid. The resulting copper sulphate solution will be precipitated by metallic iron. The manufacture of sulphuric acid is rather expensive, however, and the consumption of iron so high that even with cheap iron the process is likely to be somewhat expensive. The process is being studied with care.

The range of ore to be handled in this district may be indicated on the one hand by the Cananea blast-furnace practice, with the elimination of 70% of the sulphur in the charge and the production of a 33% matte, and the Miami concentrate, which contains about 35% copper and 21% sulphur. The tendency toward increasing use of reverberatory smelting is not only due to the increasing amount of fine concentrate to be handled, but also because many of the lower-grade deposits are too low in sulphur for the formation of a satisfactory matte. The net cost of reverberatory smelting has been lowered to a point where, in many cases, it is better at once to smelt



EQUIPMENT

WATER CIRCULATION

[illegible] Pump [illegible]
Two Cameron Type Centrifugal Pumps - 600 Gallons per Minute

ELECTRIC LOCOMOTIVES

Nos 1 and 2 [illegible] Ton Equipped with One [illegible] HP Motor Each
Nos 3 and 4 6½ Ton Equipped with Two [illegible] HP Motors Each
Nos 5 to 12 [illegible] Ton Equipped with [illegible] HP Motors Each

ELECTRIC CRANES

No 1 50 Ton Electric Crane in Converter Building with 3 Motors from 30 to [illegible] HP Each
No 2 50 Ton Electric Crane in Converter Building with 3 Motors from 30 to [illegible] HP Each
No 3 [illegible] Ton Electric Crane in Converter Building with 3 Motors from 30 to [illegible] HP Each

LOCOMOTIVE CRANES

No 1 [illegible] Crane at [illegible]
No 2 [illegible] Crane at [illegible]

SMELTER SPREADING BEDS

One [illegible] HP Motor for [illegible]
One 3 HP Motor for [illegible] Ore Feeders
One [illegible] HP Motor for Conveyors 1, 2 & 6 Sampling
One [illegible] HP Motor Spare for [illegible] Sampling
One [illegible] HP Motor for Crushers
One 20 HP Motor for [illegible] Mill
Two [illegible] HP Motors for [illegible] Mill
One [illegible] HP Motor for [illegible] Mill
One [illegible] HP Motor for Conveyor No 5
One 20 HP Motor for Conveyor No [illegible]
One [illegible] HP Motor for No 1 Reclaimer [illegible]
One [illegible] HP Motor for No 1 Reclaimer [illegible]
One [illegible] HP Motor for No 2 Reclaimer [illegible]
One [illegible] HP Motor for No 2 Reclaimer [illegible]
One [illegible] HP Motor for Reclaimer [illegible]
One 30 HP Motor for Conveyors Nos 3 [illegible]
One [illegible] HP Motor Spare for Conveyors [illegible]
One [illegible] HP Motor for Conveyor No 7
One [illegible] HP Motor Spare for Conveyor No 7
One [illegible] HP Motor for Conveyor No [illegible]
One [illegible] HP Motor Spare for Conveyor No [illegible]

ROASTER SPREADING BEDS

One [illegible] HP Motor for Conveyor A
One [illegible] HP Motor for Conveyor [illegible] and Sampling
One [illegible] HP Motor for Conveyor [illegible] and Sampling
One [illegible] HP Motor for Conveyor [illegible]
One [illegible] HP Motor for Conveyor [illegible]
One [illegible] HP Motor for Conveyor [illegible]
One 30 HP D.C. Motor for Reclaimer

MISCELLANEOUS

Seven [illegible] Converter Shells [illegible]
One [illegible] HP Motor for [illegible] on No 1 Casting Machine
One [illegible] HP Motor for Tilting Device on No 1 Casting Machine
One 30 HP Motor for Driving Bosh Drag Chain on No 1 Casting Machine
One Hydraulic Lift for Tilting Spoon on No 1 Casting Machine
One [illegible] Air Crane for [illegible]
One [illegible] HP Motor for Turning [illegible] Mould Chain on No 2 Casting Machine
One [illegible] HP Motor for Tilting Device on No 2 Casting Machine
One [illegible] HP Motor for Driving Bosh Drag Chain on No 2 Casting Machine
One Hydraulic Lift for Tilting Spoon on No 2 Casting Machine
One [illegible] Crane for Conveying [illegible] Bars to Loading Platform
One 10 HP Motor for Conveyors of [illegible] Bins
One 20 HP Motor for Concentrates Drier
One 125 HP Motor for [illegible] Mill
One 100 HP Motor for Emergency Pump
Seven [illegible] HP Motors for Converter Stands
One [illegible] HP Motor for No 6 Sturtevant Blower
One 10 HP Motor for Reverberatory Elevator
Two 20 HP Motors for McDougal Furnaces
One [illegible] HP Motor for McDougal Pump

OLD POWER HOUSE

No 11 Nordberg Piston Blower [illegible]
Two No 8 Connersville Blowers [illegible]

MAIN BOILER HOUSE

[illegible] Each 2 [illegible] HP [illegible] Type Boilers
[illegible] Each 1 [illegible] HP Sterling Boiler
Two [illegible] Each 1 [illegible] HP Aultman and Taylor Boiler
Two [illegible] Each 1 [illegible] HP Sterling Boiler

REVERBERATORY BOILER HOUSE

Three Nos 1, 2 and 3 [illegible] HP Sterling Boilers
Two Nos [illegible] and [illegible] [illegible] HP Sterling Boilers
Three Nos [illegible] 7 and 8 [illegible] HP Aultman and Taylor Boilers
One Aldrich Triplex Pump [illegible] Direct Connected to [illegible] HP D.C. Motor
One Aldrich Triplex Pump [illegible] Direct Connected to [illegible] HP D.C. Motor
One Aldrich Triplex Pump [illegible] Direct Connected to [illegible] HP D.C. Motor
Four Green Fuel Economizers [illegible] Tubes Each

FLOW SHEET
FOR
REDUCTION WORKS

THE [illegible] COPPER CO. S. A.
[illegible]
ENGINEERING DEPARTMENT

SCALE
DATE [illegible]
DRAWN BY
TRACED BY
CHECKED BY
APPROVED BY
FILE NO. [illegible]

in reverberatory furnaces, rather than to attempt to handle much concentrate in the blast-furnaces, with the inevitable rehandling of a considerable part of it as flue dust. For roasting, both the water-cooled Wedge furnace and the air-cooled Herreshoff are coming into use, as more satisfactory from a mechanical standpoint than the ordinary McDougall. The Great Falls type of converter has come into common use, the longer life of the basic lining leading to lower costs, especially through the greatly decreased handling of seconds. Thus at Cananea it is hoped that much of the converter slag can be poured directly into the settlers of the blast-furnace, instead of requiring smelting. Sweeping changes in present practice are likely to result from the experimental work now being done on the use of flotation for the recovery of concentrate and of leaching processes for the recovery of copper. But through the similarity of general conditions which govern smelting operations in the territory which has been described, it is likely that smelting practice in the area will continue to exhibit certain characteristic common features.

THE SMELTING PLANT OF THE CANANEA CONSOLIDATED COPPER COMPANY

By Arthur C. Cole

The following is a general description of the equipment and essential points of practice of the Cananea Consolidated Copper Co.'s smelter at the beginning of 1914. The plant is at Cananea, in the state of Sonora, Mexico. It is illustrated by the general flow-sheet, Fig. 30.

Receiving Bins.—All ore, concentrate, flux, and secondaries for blast-furnace treatment are weighed on track scales and then dropped into the main receiving bins from self-dumping, steel, narrow-gauge railroad cars which contain 20 to 30 tons each, according to material. The bin system consists of 11 wooden bins and 8 steel bins, having a gross capacity of approximately 3000 tons. From the receiving bins the ore is discharged through a movable hopper, dropped upon a 30-in. conveyor belt and delivered to a 24 by 36-in. Farrel crusher, which crushes the material to about 4 in. maximum size. Then one-tenth of the entire stream is cut by a Vezin cutter, and passes through a 10 by 20-in. Cananea type crusher; then one-tenth of this is cut and passes through a Gates gyratory crusher; then one-fifth, which passes through a set of 17 by 27-in. rolls. The final sample is then cut, amounting to about 1/4000 part of the original material. The final sample is quartered down, dried, and prepared in the bucking room for the laboratory. All material, except concentrate, which is hand-sampled, is sampled in this manner. After the sample is cut, the reject is elevated back to the main belts, and joins the main stream of ore on its way to the spreading beds.

Here the ore is distributed over the entire length of the bed by means of an automatic traveling tripper on the delivering belt. The bed thus formed covers an area 450 ft. long by about 50 ft. wide and, when completed, reaches a height of about 18 ft. in the centre line. The bedding system is of steel construction and contains three ore-beds of 8500 to 9000 tons capacity each. Each class of material is distributed up and down the length of 450 ft. in continuous layers, one on top of the other, and is run out in lots of 100 to 200 tons at the rate of about 300 tons per hour. The weight and assay of each lot of material bedded is reported to the office, where the bed records are kept; the actual contents are calculated, so that upon the completion of a bed, there is available a self-fluxing mixture of known composition, ready for the blast-furnace.

Reclaiming.—The completed bed is reclaimed by means of a special machine, which advances into the face of the pile. A perfect cross section is removed by means of a harrow, which rests on the face of the pile at an angle of about 60°. This harrow has a reciprocating motion of about 18 in., thereby raking down the face of the pile into a scraper conveyor which removes the ore to a belt conveyor, and from there it passes on to the blast-furnace bins. The reclaiming machine has a capacity of about 175 tons per hour. There are two of these machines, one working at a time.

Blast-Furnaces.—The blast-furnace building is of steel construction, 33 by 312 ft., and contains eight furnaces. Charges are dropped by gravity, from the 75-ton steel bins above, into measuring hoppers of 2000 to 2200 lb. each. There are five hoppers to each furnace, and they are discharged direct by means of an arch gate. The coke bins are placed at the west end of the blast-furnace building, and have a capacity of about 3000 tons. Coke charges are wheeled in barrows to the furnace and shoveled in. The blast-furnaces are 48 by 210 in. at the tuyeres, at the top they measure 5 ft. 2 in. by 7 ft. at the inner edge of the jacket, and 7 by 17 ft. from the side of the hopper at the feed floor. They are 11 ft. 11 in. high from the top of the base plate of the inner edge of the jacket and 13 ft. 6 in. to the top of the hopper. Each furnace has six jackets on each side 2 ft. 11 in. by 12 ft. 6¾ in. and two right-hand and two left-hand jackets on the ends 9 ft. 7¼ in. high and 2 ft. 6 in. wide at the bottom by 3 ft. 1 in. wide at the top. The jackets are made of ⅝-in. steel plate on the inside and ⅜-in. on the outside, and have a 5-in. water space. There are three tuyeres of 5 in. diameter to each jacket. There are four settlers 10 ft. 5 in. by 19 ft., inside of the chrome brick lining, each serving two furnaces. The slag overflows continuously into self-dumping slag cars of 45 cu. ft. capacity. Trains, consisting of 6 to 8 cars, are hauled to the dump by an electric locomotive. The matte is tapped from the settlers into 66 cu. ft. cast steel ladles, which are conveyed to the converters by electric cranes. The gases from the furnace are discharged into a steel balloon flue 10 ft. in diameter and from there pass through two cross flues 12 ft. 6 in. diameter, and two goose necks 8 ft. 6 in. diameter, and then into the main dust chamber, which is 60 by 181 ft. by 32 ft. 6 in. From the main chamber, it then passes through a brick flue 20 by 242 ft. by 19 ft. 6 in., then through another chamber 84 by 34 ft. 8 in. by 23 ft. 6 in., to the brick-lined steel stack, which is 168 ft. high from the foundation. The foundation is 25 ft. high.

Following is a record of blast-furnace data for the last six months of the year 1912, and the months of October and November, 1913.

There have been a number of changes in metallurgical conditions and practice since 1912, and the months of October and November of this year are given for the reason that they are representative of conditions at present and as they will be for some time to come:

Composition of charge:	Last 6 months 1912.	October.	November.
	%	%	%
Ores and concentrates	74.66	85.51	85.57
Converter slag	12.95	5.40	6.19
Chips and cleanings	11.70	6.83	5.25
Flux	0.69	2.26	2.99
	100.00	100.00	100.00
Analysis of charge:	%	%	%
SiO_2	23.1	24.7	23.6
Al_2O_3	4.7	5.2	4.7
Fe	25.2	23.1	23.2
CaO	7.5	9.3	9.7
S	16.6	14.6	14.2
Cu	7.42	6.24	5.47
Au, oz. per ton	0.019	0.017	0.019
Ag, oz. per ton	2.95	3.18	3.53
Analysis of slag:			
SiO_2	37.6	38.2	37.3
Al_2O_3	6.9	7.9	7.6
FeO	34.7	32.6	33.4
CaO	13.0	14.7	15.0

Composition of charge:	Last 6 months 1912. %	October. %	November. %
MgO	1.20	1.65	1.58
MnO	1.23	0.80	1.04
Zno	3.05	2.30	2.10
Cu	0.34	0.41	0.32
Analysis of matte:			
SiO_2	0.6	0.6	0.5
Fe	34.2	38.0	38.0
S	24.4	24.5	24.0
Cu	35.66	35.5	33.5
Au	0.110	0.105	0.116
Ag	14.66	19.53	21.69
Analysis of flue dust:			
SiO_2	21.2	25.6	26.2
Al_2O_3	5.1	7.9	6.7
Fe	30.5	27.1	26.1
CaO	3.1	4.1	4.3
S	13.3	11.2	10.5
Cu	7.01	5.51	4.09
Au	0.019	0.020	0.020
Ag	2.91	3.38	3.26
Per cent wet coke to wet gross charge	8.87	9.5	11.48
Per cent dry coke to dry gross charge	9.24	10.88	11.92
Per cent dry coke to dry gross and concentrate	12.37	12.72	13.93
Furnace days	1174.849	123.833	119.322
Tons dry gross charge per furnace day	312.980	349.725	386.537
Tons dry gross charge per sq. ft. hearth area per furnace day	4.471	4.996	5.522
Per cent flue dust to dry gross charge	8.91	8.23	7.10
Blast data:			
Cubic ft. per furnace minute	16,708	19,900	20,235
Cubic ft. per ton dry gross charge	76,885	81,940	75,212
Cubic feet per sq. ft. H. A. per minute	239	283	289
Ounces blast pressure	27	28	28
S ratio in charge	2.240	2.567	2.599
S ratio in ore and concentrate	4.441	3.722	3.531
Per cent S volatilized	61.95	67.38	66.65
Per cent matte fall	24.98	19.53	19.36

Roasting Plant.—The roaster building is of steel construction and contains ten 18-ft. six-hearth McDougall calciners. The dust chamber is of brick, 115 by 25 by 15 ft. The bedding system consists of four bins of 500 tons capacity and two beds of 3000 tons each. The mixed roaster charge is reclaimed and delivered by conveyor belts into 60-ton bins over the roaster. The roaster product is hauled by electric motor in 5-ton cars to the reverberatories.

Analysis of charge:	Last 6 months 1912.	October.	November.
	%	%	%
SiO_2	19.6	10.9	13.0
Al_2O_3	4.5	2.8	3.0
Fe	26.5	34.7	34.1
CaO	1.3	0.9	0.8
S	29.0	38.9	38.0
Cu	9.09	7.76	7.29
Au	0.019	0.009	0.010
Ag	2.80	1.10	1.23
SiO_2	22.8	12.7	15.9
Al_2O_3	5.3	3.2	3.7
Fe	30.7	40.9	42.4
CaO	1.5	1.1	1.0
S	10.3	12.6	10.1
Cu	10.53	9.13	9.02
Au, oz. per ton	0.022	0.010	0.012
Ag, oz. per ton	3.24	1.28	1.52
Dry tons charge per furnace day	38.454	34.204	34.755
Tons calcine and dust per furnace day	32.456	28.918	27.904
Per cent calcine and dust to dry charge	84.401	84.545	80.29
Per cent flue dust to dry charge	5.52	5.49	5.51
Per cent S volatilized	69.26	72.64	78.60
Barrels of oil per furnace day	1.007	1.117	0.495
Barrels of oil per ton charge	0.026	0.019	0.014

Reverberatory Plant.—The reverberatory plant consists of two 19 ft. 6 in. by 100 ft. oil-fired furnaces. The waste gases fire three 300-hp. and five 250-hp. boilers, placed in parallel, and then pass through a fuel economizer system of 1152 tubes in four units, then through a brick flue 130 ft. long to stack 12 ft. 6 in. diameter, 152 ft. high. The slag is skimmed into 112-cu. ft. electrically tilted slag cars and hauled to the dump by electric motor. The matte is tapped into 66-cu. ft. ladles, and hauled to the converter building in ladle cars, from which it is transferred to the converter by electric cranes.

Composition of charge:	Last 6 months 1912.	October.	November.
	%	%	%
Calcine	35.21	39.96	40.72
Flue dust	38.01	27.85	27.56
Miscellaneous	24.08	31.86	31.58
Flux	2.70	0.33	0.14
	100.00	100.00	100.00
Analysis of charge:			
SiO_2	21.9	17.6	19.4
Al_2O_3	5.5	4.9	4.9

Composition of charge:	Last 6 months 1912. %	October. %	November %
Fe	25.7	28.6	28.2
CaO	3.4	1.9	1.8
S	13.2	15.3	13.4
Cu	13.61	16.83	17.22
Au, oz. per ton	0.018	0.014	0.019
Ag, oz. per ton	2.67	2.44	2.60
Analysis of slag:			
SiO_2	39.1	35.3	37.9
Al_2O_3	9.8	10.0	9.6
FeO	36.8	43.9	41.4
CaO	6.0	3.8	3.6
MgO	0.77	1.22	1.58
MnO	0.44	0.80	1.04
ZnO	2.26	2.30	2.10
Cu	0.44	0.45	0.48
Analysis of matte:			
SiO_2	0.6	0.5	0.6
Fe	29.7	30.0	29.4
S	26.7	24.0	23.0
Cu	40.35	42.19	41.80
Au, oz. per ton	0.055	0.036	0.046
Ag, oz. per ton	7.92	5.88	6.27
Per cent S volatilized	33.20	35.37	34.83
Ratio S to Cu in charge	0.968	0.909	0.780
Dry tons per furnace day	228.120	210.093	197.843
Fuel results:			
Barrels of oil per ton of charge	1.095	1.004	1.021
Steam, credit account boilers	0.478	0.468	0.429
Steam, credit account economizers	0.075	0.092	0.110
Total credit	0.553	0.560	0.539
Net account smelting	0.542	0.444	0.482

Converter Plant.—The converter building is of steel construction, 65 by 518 ft., and 56 ft. high to bottom chord of the truss. It is equipped with three electric cranes of 40, 50, and 60 tons capacity, respectively, six electrically operated stands, and seven basic lined, 12 ft. diameter, Great Falls type converter shells, with 1½-in. tuyeres. The converter slag is skimmed into 66-cu. ft. cast steel ladles. About two-thirds of the slag is poured direct into the balst-furnace settlers and the remaining one-third is hauled by electric motor and slag pots to pits and reclaimed by a 10-ton steam shovel. It is then hauled in narrow gauge cars to the bins at the spreading beds.

Copper Casting Machine.—There are two straight-line electrically operated casting machines equipped with two tilting ladles of 66 cu. ft. capacity. Each machine has a chain of 39 molds, casting bars of about 300 lb., dumping into a bosh from which they are removed by drag conveyor to the bullion platform. The blister bars are then stacked by an air-operated radial crane, picked up on trucks, weighed, and loaded into railroads cars.

Flues and Dust Chamber.—The converter balloon flues are of steel construction 10 ft. diameter and 201 ft. long, which discharge into a steel dust-chamber 104 by 26 by 20 ft. with a circular arch roof of 13-ft. radius. The gases then pass to a steel stack 12 ft. diameter by 125 ft. high.

Power Plant.—Following is a complete list of the units of the power plant. Power is generated for the reduction division, the concentrator, and a large part of the mining division, as well as for two small outside mining companies and the lighting of the town of Cananea. The power report for the month of November is appended and shows the distribution of power to the departments of the smelter:

ELECTRIC GENERATORS

General Electric 1500-kw. Curtis steam-turbine with 1500-kva. generator.

General Electric 1500-kw. Curtis steam-turbine with 1875-kva. generator.

General Electric 1500-kw. Curtis steam-turbine with 1875-kva. generator.

McIntosh-Seymore engine 16-32 by 30 with 300-kva. General Electric generator.

McIntosh-Seymore engine 16-32 by 30 with 300-kva. General Electric generator.

Nordberg engine 18-36 by 36 with 400-kw. DC General Electric generator.

Union Iron Works engine 13½-23 by 12 with 100-kw. generator (Reserve Exciter).

CONVERTER AIR

Allis blowing engine 20-36 by 42 steam 44-44 by 42 air, 148 fpr.
Rand blowing engine 22-40 by 48 steam, 46-46 by 48 air, 184 fpr.
Allis blowing engine 20-36 by 42 steam 44-44 by 42 air, 148 fpr.
Nordberg blowing engine 20-42 by 42 steam 44-44 by 42 air, 148 fpr.
Nordberg blowing engine 20-42 by 42 steam 44-44 by 42 air, 148 fpr.

BLAST-FURNACE AIR

Nordberg piston blower 13-24 by 42 steam 57-57 by 42 air, 70 r.p.m., 248 fpr.

Connersville blower No. 10, 300 fpr., direct connected to Murray engine 16-28 by 36.

Connersville blower No. 8, 100 fpr., direct connected to Murray engine 10-18 by 24.

Connersville blower No. 10, 300 fpr., direct-connected to Allis-Greene engine 16-30 by 36.

Connersville blower No. 10, 300 fpr., direct-connected to Murray engine 14-28 by 36.

Connersville blower No. 10, 300 fpr., direct-connected to Allis engine 14-28 by 36.

Two No. 8 Connersville blowers.

HIGH PRESSURE AIR (MINES AND INDUSTRIAL PURPOSES)

Ingersoll-Rand compressor, 14-24 by 6 steam 13-23 by 16 air, 1000 fpr.

Ingersoll-Rand compressor Type "C", 23-50 by 48 steam 44¼-26¼ by 48 air, 6000 fpr.

AUXILIARIES

Southwark Foundry & Machine Co., air pump 16½-30 by 24 belted to Bibins rotary pump.

Union Iron Works engine 13½-23 by 12, direct-connected to Root rotary pump.

Southwark Foundry & Machine Co., air pump 14½-34 by 30.

Electric circulating and feed pumps.

Main power house, two 13 by 10 Aldrich triplex pumps, direct-connected to 50-hp. induction motor.

Reverberatory boiler house—Aldrich triplex pump 9 by 10, direct-connected to 60-hp. DC motor.

Aldrich triplex pump 9 by 10 direct-connected to 65 hp. DC motor, reverberatory boiler house.

Reverberatory boiler house, Aldrich triplex pump, 5½ by 9, direct-connected to 40-hp. DC motor.

SMELTER ELECTRIC POWER

DC generator, direct-connected to 360-hp. induction motor.

300-kw rotary converter.

BOILERS (REVERBERATORY BOILER HOUSE)

Three, 300-hp. Stirling boilers.

Two, 250-hp. Stirling boilers.

Three, 350-hp. A and T boilers.

BOILERS (MAIN POWER-HOUSE)

Five each, 2-150-hp. Elephant type boilers.

Six each, 1-250 Stirling boilers.

Two each, 1-250 hp. A and T boilers.
Two each, 1-385-hp. Stirling boilers.

DISTRIBUTION OF POWER TO REDUCTION DIVISION, NOVEMBER 1913

Average indicated horse-power		5,647.50
Account power-house boilers	4,168.47	
Account reverb. boilers and economizers	1,479.03	
Total indicated horse-power-hours		4,066,030.00
Barrels oil, account reduction division		7,457.77
Barrels oil, account other departments		12,321.41

UNITS PER INDICATED HORSE-POWER

Total oil account steam:		Total water account steam:	
Barrels oil per i.hp	3.502	Gallons per minute	180.760
Gallons per i.hp.hr	0.204	Pounds per i.hp.hr	19.210

Water evaporated per pound of oil:	Net lb.
Power-house boilers	12.65
Reverberatory boilers	11.97

DIVISION OF POWER CONSUMED

Smelting:	Steam.	Air.	Electric.	Total i.hp.
Blast	730.8	....	....	730.8
Water supply	66.6	....	4.0	70.6
Slag expense	....	....	34.0	34.0
Matte pit	....	....	6.0	6.0
Total	797.4	....	44.0	841.4
Converting:				
Blast	899.5	....	....	899.5
Cranes	....	....	30.0	30.0
Clay mill	....	....	10.0	10.0
Operating	....	....	14.0	14.0
Casting machine	....	....	6.0	6.0
Total	899.5	....	60.0	959.5
Reverberatories:				
Atomizing oil, converters, air	40.5	....	6.0	46.5
Elevator	....	....	5.0	5.0
Cleaning boiler tubes	....	7.0	....	7.0
Slag expense	....	....	12.0	12.0
Total	40.5	7.0	23.0	70.5
Sampling:				
Sample mill	....	3.0	56.0	59.0
Miami samples	....	1.0	....	1.0
Total	....	4.0	56.0	60.0

Roasters:				
Receiving and spreading	....	3.0	16.0	19.0
Reclaiming	....	....	18.0	18.0
Water supply	....	....	15.0	15.0
Operating	....	....	12.0	12.0
Calcine haulage	....	....	5.0	5.0
Total	....	3.0	66.0	69.0
Spreading beds:				
Receiving	....	5.0	....	5.0
Crushing and conveying	....	....	66.0	66.0
Reclaiming	....	....	58.0	58.0
Total	....	5.0	124.0	129.0
Total reduction division	1,737.4	19.0	373.0	2,129.4
Total to other departments	0.7	719.4	2,798.0	3,518.1
Grand total	1,738.1	738.4	3,171.0	5,647.5

STEPTOE VALLEY SMELTING PRACTICE

(Discussion, October 7, 1911.)

The Editor:

Sir—At the Steptoe Valley smelter the average throat draught of the reverberatory furnaces with coal firing was 1.4 in. water and at the stack 1.8 in. With oil the throat draught averages only 0.5 in. and stack about 1.6 in. The best boiler to employ for the recovery of heat from the furnace gases is the vertical tubular type. This allows some self-cleaning of tubes from flue-dust by gravity. In the management of converters it is found that basic linings cannot stand excessively high temperatures so well as acid linings, neither can they stand the same range of temperatures without serious deterioration. This is a very important point in the successful use of basic lining.

C. B. LAKENAN.

Ely, Nevada, September 30.

PROGRESS AT THE STEPTOE VALLEY SMELTER

By S. S. SORENSEN

(January 4, 1913.)

Changes in the plant and equipment of the Steptoe valley smelter, McGill, Nevada, made during 1912 may be summarized as follows: At the smelter pressure-blast air cooling has been extended to all the roasting furnaces by the installation of belt-driven fan of 25,000 cu. ft. capacity at a pressure of 3½ in. of water, and the tonnage has been raised to about 90 tons per furnace day with a sulphur elimination of over 60%. No radical change has been made in the construction or working of the reverberatories. The three now running are smelting 440 tons of total charge each per furnace day. with an oil consumption of 0.86 bbl. per ton of charge. One has been rebuilt with an increase in size and a change in the design of binding. The latter has been concentrated into beams set at 5-ft. spaces, centre to centre, instead of the old method of close-set 8 to 12-in. I-beams. This furnace is ready to go into commission whenever required.

At the converters the greatest improvement has been the remodeling of the No. 1 Pierce-Smith basic converter. This converter, which is 30 ft. long inside, and has 46 tuyeres, is making 78 tons of blister per day from 38% matte. Charging is not done through the outlet as hitherto, but through a special charge hole from a fixed launder. A new copper casting machine has been built with oil-fired receiver which holds 50 tons of blister and is tilted by means of gearing from the old disused converters by an electric motor.

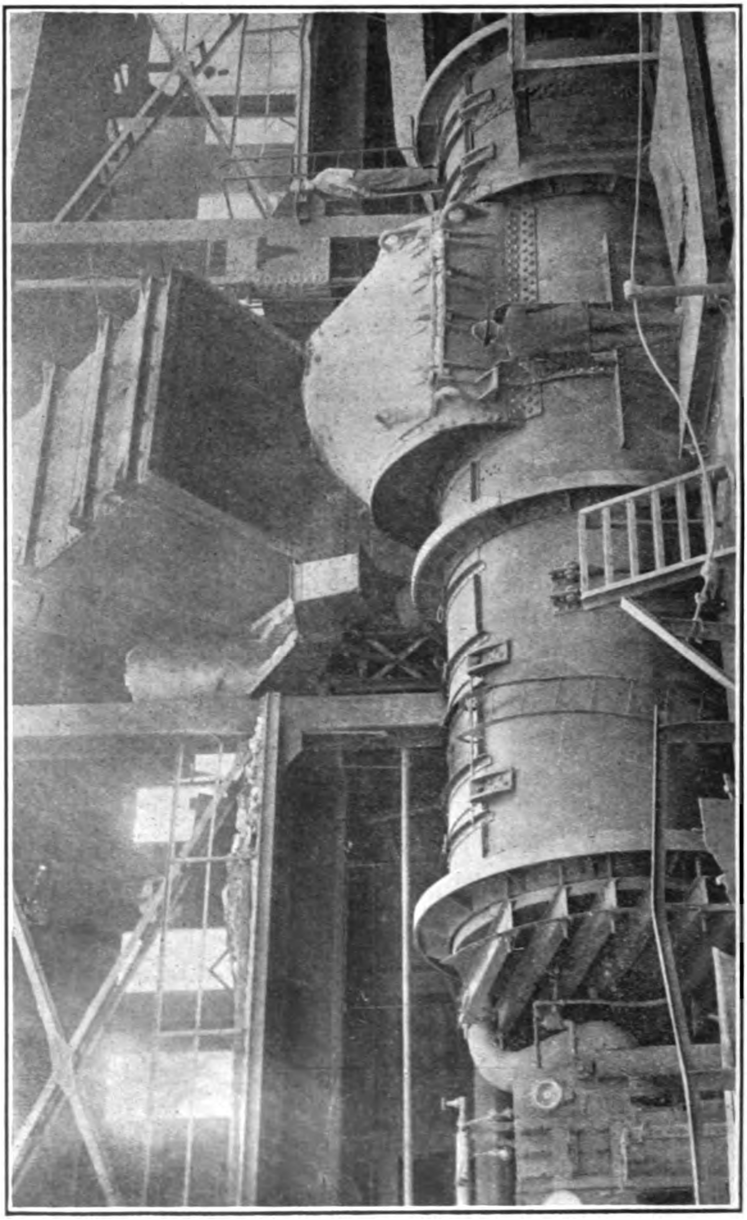


FIG. 31. PIERCE-SMITH CONVERTERS, SHOWING SPECIAL CHARGING DEVICE.

PROGRESS IN FUEL UTILIZATION

(Editorial, October 25, 1913.)

The community of interest in the field of metallurgy was well shown at the meeting of the Iron and Steel Section of the American Institute of Mining Engineers in New York last week. During the session a number of notable papers upon the use of powdered coal as fuel, and the recovery of waste heat from metallurgical furnaces were read and discussed at some length, and the results attained in the field of zinc and copper metallurgy were presented and listened to with interest by the steel metallurgists. Curiously enough, three of the papers had almost identical titles, though Messrs. R. K. Meade, H. R. Barnhurst, and W. S. Quigley treated the general subject of the use of powdered coal as fuel from slightly different aspects. These were preceded by two papers on the generation of steam by the waste heat of furnaces, read by Mr. G. C. Stone and by the secretary for Mr. F. Peter, professor at Leoben.

The present importance of the use of pulverized coal as fuel is illustrated by Mr. Barnhurst's estimate that 10,000 tons of coal in this form is burned daily in the cement industry alone. The development of direct-firing with coal dust has indeed advanced almost *pari passu* with the growth of the output of cement, the large quantity of coal used in the production of so cheap a commodity making necessary the use of fuel in a form which permits more efficient use of its heat content than is possible in any system of indirect firing. To a less extent, it has been applied to other fields of work, and its present use at the smelter of the Canadian Copper Company and the earlier experiments at the Highland Boy plant and at Cananea are well known to many of our readers.

The difficulties in the early work in all fields arose from the presence of moisture in the coal, insufficient grinding, and the attempt to use unsuitable types of coal. It is now possible to use any long-flame coal which contains over 25 per cent of volatile hydrocarbons, and coal containing as much as 25 per cent ash has been used in cement burning. In smelting, so high an ash content would not be permissible, especially in copper treatment, for if the ash is fusible it tends to clog the flues and waste-heat boilers, while if it is infusible it is an unwelcome addition to the already highly silicious charge on the hearth. Before pulverizing, the coal must be thoroughly dried. In addition to the bad effect of moisture on pulverizing and injecting the coal, each per cent of moisture in the fuel corresponds to a decrease of 52 degees Fahrenheit in the temperature attainable in the furnace; a decrease which cannot be overcome by burning additional fuel. This bad effect is accentuated by the fact that only part of the heat is utilized in the furnace; thus if the powdered coal contains five per cent moisture, the temperature in the zone of combustion is 1500, and the temperature of the outgoing gases is 600 degrees—the heat wasted in raising the water to the temperature of the escaping gases corresponds to nearly ten per cent of the total heat value of the fuel Standard practice requires the grinding of the pulverized coal so that ninety-five per cent will pass 100 mesh and eighty-five per cent through 200 mesh.

The aim in coal pulverizing is to make a maximum of what the millman used to call 'slime', namely, material too finely divided to permit measurement. Various types of mills have proved satisfactory; ball-mills, various tube-mills, Raymond, and Griffin mills. Certain types of these permit the burning of coal as fast as it is ground; in others it is caught in storage-bins. The ground coal is not explosive unless it is so stirred up as to form an explosive mixture with air.

In ordinary practice the coal is blown into the furnace by the use of twenty-five per cent of air, furnished either by a fan or by blowers which deliver it at higher pressures, the fan being preferred as cheaper. If a larger quantity of air is used, the flame becomes shorter, since the deflagration of the coal immediately in front of the nozzle is thus favored. The length of the flame attained depends, therefore, on the character of the coal, the amount of air used in its injection, and the draft. Regulation of the length of the flame is best done by manipulating the draft. It is, of course, necessary to admit an excess of air over that required for combustion, and the temperature attained can easily be controlled by regulating the amount of fuel admitted. The convenience and economy of this method of firing is leading to its increased use in open-hearth and puddling furnaces, and, within the limitations of local conditions, is likely to lead to its increased use in reverberatory copper smelting.

The high temperature of the gases escaping from such furnaces makes it especially desirable that some of their contained heat shall be recovered. This is commonly attained by the use of regenerators, but the gases escaping from these still contain recoverable heat. Unless their temperature exceeds 300 degrees Centigrade there is no economy in attempting to recover this heat, since the temperature difference between the gases and the steam generated is so small as to make the transfer of heat from the one to the other impracticably solw. The lowest practicable limit is perhaps attained at the

Palmerton, Pennsylvania, plant of the New Jersey Zinc Company, described by Mr. Stone, where gases (at 450°C.) escaping from the spelter furnaces are utilized in a battery of return-tubular boilers which furnish the steam required for the gas producers. It is there found that three horse-power can be recovered in the waste-heat boiler for each ton of coal gasified in the producers, without interfering with the results attained in the spelter furnaces. With reverberatory copper-smelting furnaces the use of waste-heat boilers is standard practice, and the important paper by Mr. S. S. Sorensen, published in our issue of October 11, 1913 [see p. 159], is of especial value, since it gives the comparative results attained by two different types of boilers, and thus contributes greatly to that more exact knowledge of this subject which is much to be desired.

METAL LOSSES IN COPPER SLAGS

(Discussion, November 13, 1909.)

The Editor:

Sir—The following communication to the secretary of the American Institute of Mining Engineers brings up a subject on which great interest centres, and it may stimulate further discussion: R. W. Raymond, Secretary American Institute of Mining Engineers.

In Lewis T. Wright's paper on 'Metal Losses in Copper-Slags,' read at the New Haven meeting, he starts in by saying: "It is commonly believed by the metallurgists that in copper smelting the copper in the slag, which is irreducible by continued smelting, is retained in the form of prills of matte." On a recent visit to Greenwood, British Columbia, I was discussing with J. E. McAllister, general manager for the British Columbia Copper Co., and the former metallurgist for the Tennessee Copper Co., this very question, and he strongly held to the idea that the copper in slag was in two forms, namely, one portion contained in occluded matte, and the other as an oxide. He believed that this rule would apply also to the silver in slag. From my experience I am strongly inclined to agree with Mr. McAllister and it appears quite reasonable that in furnaces treating oxidized copper ores, and to a lesser degree in furnaces treating sulphide ores, there is always bound to be present a crtain amount of copper oxide and silver oxide, which will behave like any base, and get into the slag as such. We all know that in Arizona, in the early days, when it was customary to produce black copper, because the ores were nearly all oxidized (or carbonates) and sulphur was scarce, the slags as a rule carried 2.5% copper and seldom less than 1.5%. The economic point was determined by the extra coke necessary to produce the reducing action. In connection with the extra saving in copper, Arthur L. Walker, now professor of metallurgy at Columbia University, and formerly general manager for the Old Dominion Copper Mining Co., at Globe, told me that with coke at $60 per ton it never paid him to make slags better than 2.5% copper.

In Tennessee, when treating well roasted and presumably well oxidized ore, we seldom, if ever, had our slags contain less than 0.5% copper, yet when treating this same ore pyritically, and producing the same grade of matte, the slag would not exceed 0.3% copper. The above instances are merely cited to show that the normal tendency of copper oxide is to act like iron oxide, lime, or any other base, and to go into the slag.

If we take Mr. Wright's first example and assume a 50% copper matte, corresponding with 0.3% copper slag, which, by the way, would be pretty low, we would have the following analysis of the matte and slag:

	Matte.	Slag.
Cu	50.000%	0.300%
Ag	31.400 oz.	0.147 oz.
Au	13.950 oz.	0.026 oz.

If we be permitted to assume that no gold is oxidized in the slag and that all there present is contained in the matte, we can assume the 0.026 oz. as a basis and from it calculate the amount of silver and copper corresponding to the grade of the matte produced. I have given below in the second column, the results of this calculation:

	Total loss.	Matte loss.	Oxide loss.
Cu	0.300%	0.093%	0.207%
Ag	0.147 oz.	0.058 oz.	0.089 oz.
Au	0.026 oz.	0.026 oz.	......

From the above it will be seen that of the total copper contained in the slag about one-third only is lost in the form of matte, and the other two-thirds is probably in the form of oxide. Nearly the same ratio exists for silver; and we all know that silver is easily oxidized in a furnace with a hot-top, such as is usual in copper smelting, and, while no doubt part of it is volatilized, a portion of it is ready to go into the slag. While neither Mr. McAllister nor myself have any means of actually verifying the above theory, it appears to us the simplest and most obvious solution of the problem, and much more in keeping with the facts than that the metals themselves are dissolved in the reject material from the furnaces, which is a mixture of slag with a small proportion of matte.

J. PARKE CHANNING.

New York, October 3.

METAL LOSSES IN COPPER SLAGS

(Discussion, November 27, 1909.)

The Editor:

Sir—I am glad to observe that J. Parke Channing has taken interest in points suggested by the data I recently published in an article on 'Metal Losses in Copper Slags,' and I sincerely hope that he and others will endeavor to collect and publish further information on these matters of high technical importance. The slags I submitted to examination came from both blast and reverberatory furnaces, making first copper matte of usual grade, namely, 25 to 50% copper. Both classes exhibited similar deficiencies in the ratios of the precious metals to the copper contained by the slag and matte respectively. That is to say, the slags contained less silver and still less gold than should have accompanied the copper of the slag if this copper had all been of the grade of the matte simultaneously occurring in the copper-smelting operation. I did not think it worth while to commence the enquiry by investigating converter, refinery, or black copper slags.

As stated in the paper, the facts observed leave room for belief that some of the copper of the slag may have been existent as silicate. M. Channing's suggestion that some of the silver might be in an oxidized form is worthy of investigation. I have no data that would throw much light upon this point. Such as I have, and some will be published later, show that in the ordinary migrations of copper through a cooling mass of slag, or in its precipitation by special methods of fusion, the silver follows the copper. It is well known, for instance, that a bar of lead bullion does not contain its silver equally distributed. The outside contains more than the inside. This is a case of what I have mentioned above as 'migration' in cooling.

In the same manner the shells of slag formed in slag-bowls contain more copper than the inner liquid portion. In cases I investigated, the copper in the shells was not uniformly distributed through the shell. It varied according to position but the variations of the silver follows the variations of the copper. It looks like a case of molecular segregation in which the three metals, copper, gold, and silver, behave similarly. Circumstances alter cases. I did not observe a falling off of 40% in copper of the slags when changing from oxidized to sulphide charges in blast-furnace work, as noted by Mr. Channing. I cannot help thinking that the temperature of the slag formation may affect the slag in the power of retaining copper.

LEWIS T. WRIGHT.

San Francisco, November 16.

METAL LOSSES IN COPPER SLAGS

(Discussion, February 12, 1910.)

The Editor:

Sir—In your issue of November 13 last, J. Parke Channing suggests a method of calculating the amount of matte in a slag, which seems to me to be correct. The method is to determine the amount of gold in the slag, and with the supposition that all of the gold in the slag is in the form of matte, to calculate the amount of matte in the slag, using the ratio of gold to copper as found from a matte sample. Mr. Channing also writes that, "the remaining copper in the slag, amounting to about two-thirds of the whole, is probably in the form of oxide." In regard to this, Lewis T. Wright in a more recent issue says, "As stated in the paper," meaning one read by Mr. Wright before the American Institute of Mining Engineers, "the facts observed leave room for belief that some of the copper of the slag may have been existent as silicate." In Peters' 'Principles of Copper Smelting,' page 113, we find the following: "When an ore is smelted, there is a considerable loss of values in the slag, resulting mainly from particles of matte or metal which have failed to separate properly from the slag, and, in a lesser degree, from the formation of oxides of the valuable metals—especially lead and copper—that have entered into combination with silica, and have thus become, chemically, a part of the slag." This view of the subject seems to be the one most favored at present by the metallurgist.

It is a recognized fact, that, if zinc is present in a slag, much of it is in the form of sulphide. Personally I have known a slag rather high in zinc to contain 1.7% of sulphur. Allowing 0.2% of this for any matte, etc., in the slag, it leaves 1.5% of sulphur to combine with the zinc, or sufficient to form 4.5% of zinc sulphide. This zinc sulphide is ('Principles of Copper Smelting,' page 393), in solution in the slag and evidently in solution in very much the same manner in the form of matte, is in the form of cuprous sulphide, seems reasonable, inasmuch as zinc has much the same power of replacement for copper that iron has, to suppose the following reactions to take place:

$$Cu_2O + ZnS = Cu_2S + ZnO$$
$$Cu_2SiO_3 + ZnS = Cu_2S + ZnSiO_3$$

Copper is in much the same condition in a slag, both as to quantity and physical characteristics, regardless of the presence of zinc. Therefore the assumption that the copper in a slag, aside from that in the form of matte, is in the form of cuprous sulphide, seems reasonable, and, furthermore, that it is dissolved in the slag in much the same manner as zinc sulphide.

In endeavoring to prove this, I have tried to find a slag, which contained no elements other than copper likely to be combined with sulphur. The only one at hand is that made at the reverberatory furnace of the Union Copper Mining Co., at Copperopolis, California, in the summer of 1907. An analysis of the roasted ore follows; also the copper and sulphur content of the slag. To flux the ore about 10% of quartz was added and the slag formed was a rather stiff one due to the large amount of alumina and magnesia in the ore, which probably caused more matte to be suspended in the slag than is usually the case.

ANALYSIS OF ORE

	Per cent.
SiO_2	21.80
Al_2O_3	13.00
MgO	6.30
CaO	1.00
Cu_2O	3.37
$CuFeS_2$	10.57
FeS_2	5.87
Fe_2O_3	20.20
$FeSO_4$	5.10
FeO	8.90
Combined H_2O	2.82
	98.93

	Cu, per cent.	S, per cent.
Matte	45.00	23.00
Slag	0.63	0.23

Assuming that all of the sulphur in the slag is either in the form of cuprous sulphide or matte, a determination of them separately is made as follows: In copper sulphide, from the atomic weights, the ratio of copper to sulphur is practically 4 to 1. In the matte, by means of the analysis, the ratio is 2 to 1.

Let

C = the copper in the form of Cu_2S
C' = the copper in the form of matte
S = the sulphur in the form of Cu_2S
S' = the sulphur in the form of matte

Then

$$C + C' = 0.63$$
$$S + S' = 0.23$$

and

$$C = 4S$$
$$C' = 2S'$$

Substituting

$$4S + 2S' = 0.63$$
$$4S + 4S' = 0.92$$
$$2S' = 0.29$$
$$S' = 0.145$$

$C = 0.34\%$ copper in the form of Cu_2S
$C' = 0.29\%$ copper in the form of matte

Some experiments were conducted, during the summer of 1907, at the above mentioned plant by the superintendent, H. W. Ross, as follows: The reverberatory furnace was operated as usual, with the exception that the charge before skimming consisted entirely of a concentrate of pure sulphide, containing about 8% copper. Assuming that this charge melted into a matte, containing about 10% copper and leached through the slag replacing the suspended matte containing 45% copper, with a 10% matte, but not affecting the cuprous sulphide dissolved in the slag, we can multiply 0.29 by 10/45 and obtain 0.06% copper in the form of matte under the new conditions. If we add this to the 0.34% as cuprous sulphide, we obtain 0.40% as the total copper in the slag. As actually found in practice this slag contained 0.45% copper. The slag and matte at Copperopolis contained no gold or silver, otherwise, it would be interesting to calculate the amount of free matte in this slag by the method suggested by Mr. Channing.

The conclusions therefore from these data would seem to be that the copper in the slag is in two forms; first, as cuprous sulphide dissolved in the slag in a manner similar to zinc sulphide; second, as matte particles suspended in the slag, which amount, in a good slag and one well settled, ought to be small. The data given are not by any means sufficient to prove these conclusions, and I shall be glad to see some more discussion upon this subject, either for or against the above statements.

N. M. ZOPH.

Kennett, California, January 22.

LOSS OF COPPER IN SLAG

(May 11, 1912.)

The slag losses of copper, other than mechanical, depend upon the composition of the slag, according to V. Vanyukov in *Metallurgie*. Both matte and slag are electrolytes, the concentration of the different ions varying with the different amounts of the elements present. It is evident that the concentration of the cations of a given base in the slag will be affected by the presence of sulphur compounds of this base. The higher the concentration of given base in the slag, the greater the tendency to dissolve sulphur compounds of this base, and along with it some associated Cu_2S. The degree of this tendency decreases with the affinity of the element for sulphur and the degree of dissociation of the element of the silicate and sulphur compound. The affinity for sulphur of various metals is in the following order: Cu, Ni, Co, Fe, Mn, Zn, Co, Mg, Al. With slags high in the metals near the end of the series the losses in copper are much lower than with slags high in the metals at the beginning of the series.

USING SULPHIDES AS FUEL IN SMELTING

By JOHN HOLLWAY

*When metals are extracted from their ores by fusion, the necessary heat is always obtained by the burning of coal, coke, or other form of carbon. But sulphides can be made to burn in air, and the metallic sulphides, consequently, are natural combustible minerals. My object is to prove that they can be utilized as sources of heat in certain metallurgical operations. The most important of the mineral sulphides is pyrite, with which are frequently associated sulphides of copper and arsenic; silver and gold are usually present in larger or smaller quantities. When pyrite is roasted in the open air, an increase of temperature takes place in its mass, so that the oxidation continues without the application of external heat. This operation is carried on in Spain and other countries, where vast quantities of cupriferous pyrite are exposed is heaps for several months to a slow process of combustion, which gradually converts the sulphide of iron into ferric oxide. A similar combustion is ef-

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*Extracts from a paper entitled 'A New Application of a Process of Rapid Oxidation by Which Sulphides Are Utilized as Fuel,' presented before the Society of Arts (London), February 14, 1879.

fected in the pyrite burners of the sulphuric acid manufacturers, the solid product of the operation being the so-called 'burned ore', an impure peroxide of iron. Other important sulphides are those of lead, zinc, and antimony. Sulphide ores of copper, lead, and zinc are usually roasted to render them reducible in the furnace and to make their non-valuable constituents capable of combining with the fluxes used, the requisite heat being always obtained by the combustion of coal or similar material of organic origin. This process of roasting extends over a considerable space of time, and the heat evolved by the oxidation of the sulphides is never very manifest at any period of the operation. The sulphur and metals frequently burn to waste, because the utilization of the heat resulting from their burning has not hitherto been considered a subject of much importance. If, however a rapid current of air is forced through molten sulphides, the maximum temperature of the combustion is attained, because all the oxygen of the air driven in is then utilized for oxidation, and the operation is concentrated into the space of a few minutes instead of occupying many weeks, or in the case of cupriferous pyrite, many months.

By calculation the comparative temperatures produced by the oxidation of the principal sulphides can be found. The data thus obtained indicated to me that the oxidation of sulphides could produce sufficient heat to render their smelting a self-supporting operation, and in consequence several experiments were made by forcing a current of oxygen from a gas cylinder into molten sulphide of iron contained in a fire-clay crucible. The tuyeres used were porcelain tubes and the stems of common clay pipes. When the current of gas was directed upon the surface of the mass, an intense incandescence with a violet light was observable, showing the high temperature resulting from the oxidation. Upon dipping the tuyere into the sulphide the temperature visibly increased, and the oxide of iron formed acted energetically upon the silica of the fire-clay; but the contents of the crucible soon became viscous in consequence of the absence of sufficient silica to form a fusible slag.

The experiments were continued on April 22, 1878, and larger crucibles used; the sulphide of iron was acted upon by a current of air, sand being added during the oxidation. Matte and slag were obtained. The sulphide of iron was made by fusing cupriferous pyrite in a steel melting-furnace, the fire of which was allowed to burn down as soon as the oxidation commenced. When the blast of air was turned on, the tuyere was dipped into the contents of the crucible, and the blowing was continued for over 30 minutes. Although the whole of the oxygen was probably consumed, the oxidation had not proceeded far enough to concentrate the matte to any extent. About half the iron was removed as ferrous silicate, in which was found 0.104% of copper, and the separation of the matte from the superincumbent slag was very distinct. It was evident that to enable the oxidation to take place with sufficient energy, it would be necessary to introduce the air very rapidly, and, after some further experiments, made also with crucibles, which showed that pyrite could be melted by the heat evolved by its own oxidation, a trial was projected with a bessemer converter, because a rapid transmission of air could be thus obtained, and several experiments were accordingly made during July 1878.

The piant employed consisted of an ordinary cupola, 4 ft. diameter at the tuyeres, and 5 ft. above, having eight tuyeres, four being 3 in. diameter and the remaining four 4 in. diameter. A ladle for conveying the molten metal to the bessemer converter; and a bessemer converter, capable of treating 6 tons of crude iron at a time, lined, as usual with gannister, and supplied with cold blast. The engines for supplying the blast were two cylinders of 42-in. diameter and 4-ft. stroke, working at about 45 r.p.m., at an average pressure of steam in the boilers of 73 lb. per square inch. The pyrite was put into the cupola with coke and treated like pig iron. When at each operation the cupola was tapped, the molten protosulphide was run into a ladle and thence into the converter.

In the first of these experiments, which took place July 10 and 11, 1878, about five tons of molten sulphide was blown for half an hour in the same manner as a charge of bessemer iron. After thirty minutes' blow the contents, consisting of silicate of iron and matte, were turned out of the vessel. No sand had been thrown into the converter, therefore the gannister lining was energetically attacked by the oxide of iron. The product was quite liquid when run from the vessel, the slag crystallizing on cooling in splendid crystals of ferrous orthosilicate. In the next experiment, sulphides containing 3.4% of copper yielded, after a quarter of an hour's blow, a matte containing 46% of copper, and the destructive action upon the gannister lining was greatly mitigated by throwing sand into the converter.

Six experiments were made on July 17 and 18. The pyrite used contained 2 to 3% of copper, 1.5 oz. of silver, and 3 gr. of gold per ton. Owing to the small quantity of sulphide employed in each of these experiments, the blowing had to be of very short duration, and great care was also necessary to avoid overblowing. To obviate this the blowing had to be arrested earlier than would be necessary to produce a matte containing the desired percentage of copper; but in operating with larger quantities of sulphide there would not be the same difficulty. The spectroscopic phenomena were observed through a six-prism spectroscope. The six experiments were all arranged and the duration of each blow in the bessemer previously settled; and the first, second, and third experiments were so carried out; the fourth was intentionally overblown, in order to observe the changes in the spectrum, and the result of these observations was that in the fifth and sixth experiments the blow was arrested at the moment indicated by the disappearance of certain lines in the spectrum.

The increments in the temperature of the gases at the mouth of the converter were measured by means of one of Siemens' differential pyrometers. The pyrometer was 8 ft. long, with platinum leading wires from the platinum spiral to the terminals. The end exposed to the gases was protected for 1 ft. 6 in. of its length by a porcelain tube 2 ft. long, and the next 2 ft. by loam placed around the iron tube of the pyrometer. The indicated temperature was always below that actually obtained, as the fire-clay coating protecting the exposed stem of the pyrometer retarded the transmission of heat.

The gases were sampled by means of a fire-clay bell, connected with ½-in. iron tubing, protected by clay from the hot sulphur vapor. The iron tube, at a distance of about 20 ft. from the converter, had inserted in it two pieces of brass pipe, to which were connected, by india-rubber tubing, two glass tubes for taking the gases. The free ends of the gas tubes were connected through a T-piece with a powerful aspirator holding 20 gal. of water. By turning a tap and shutting off the water-cock of the aspirator, both tubes could be filled with identical samples of gas evolved from the converter at any given moment. The sublimate appeared in the glass tubes about three seconds after turning the taps. To avoid any interruption in the experiments, two converters were fitted with similar appliances, both pyrometers being connected with the same instrument, but only one converter was used.

In the first experiment the blow occupied 17 minutes, during which period 14 cwt. of red sand was added. The amount of sulphide introduced into the vessel could not be exactly ascertained. The mean pressure of blast was 20 lb. per square inch. The phenomena at the mouth of the converter were remarkable. At the beginning dense white fume issued from the mouth of the converter, colored in varying tints by the volatile metals present in the incandescent state, affording a brilliant spectrum containing a great number of bright lines. When the temperature rose and the lead was in great part expelled, the flame showed less 'body' and was of a bluish-green color, possibly from the presence of minute quantities of zinc and copper. The blown product of the experiment was emptied into ingot molds and allowed to cool. When cold it was found to consist of three zones, the upper one being the slag proper, the central zone a mixed product of slag and matte, and the lowest a matte free from silica. The specific gravity of the matte was about 4.8 and the slag about 4.1. The products of the experiments gave, upon analysis, the following results: Slag, FeO 53.30%, Fe_2O_3 3.00%, copper combined with sulphur 0.16%, sulphur 3.39%, silica 29.90%. Matte, iron 57.10%, copper 15.85%, sulphur 21.96 per cent.

The second experiment was similar to the first, but about 30 cwt. of fused sulphide was used. The sand added was about 14 cwt., which was thrown in, as in the first experiment, and there was, as before, an intermediate matte. The slag was dense, black, and very crystalline; it showed a peculiar radiated structure at the edge near the iron molds, caused by the sudden chill. The zones were not horizontal, but were more or less conical, as owing to the sudden refrigeration of the mass the denser particles had subsided last in the central portion which remained fluid longest. The analyses of the slag and matte obtained are as follows: Slag, FeO 54.62%, Fe_2O_3 3.71%, copper combined with sulphur 0.22%, silica 30.05%. Matte, iron 56.05%, copper 16.59%, sulphur 23.47 per cent.

In the third experiment the charge was more fully blown. The sulphide used was about 25 cwt. and the sand added about 14 cwt. The blast pressure was 20 lb. per square inch, and the blow occupied 17 minutes, which was somewhat too long. The separation of the slag from the rich matte immediately beneath it was very distinct. The analyses of products in this experiment are as follows: Matte, sulphur 22.22%, iron 12.56%, copper 62.36%. Average of the slag, metallic iron 49.30%, metallic copper 1.55%, silica 29.55 per cent.

In the fourth experiment the charge was overblown by about six minutes. No sand was added in this blow; about 1 to 1¼ tons of fused sulphide was used. The product was apparently homogeneous in composition and gave on analysis: Silica 34.34%, FeO 25.10%, Fe_2O_3 33.83% (=43.27% metallic iron), copper oxide 2.39% (= 1.91% copper). This slag is less basic and far less crystalline than those ordinarily produced, and the copper exists principally as oxide. As soon as the sub-sulphide of copper began to burn, a splendid emerald-green flame suddenly appeared, lasting about a minute, and all the lines except those of copper and sodium left the spectrum. During the last few minutes of the blow the mouth of the converter was dull and without flame.

The sixth and last experiment of this series was made upon about 20 cwt. of the fused sulphide; about 11 cwt. of sand was thrown in with a shovel during the blew, which lasted 11 minutes. A central zone of mixed matte and slag was obtained, as in the first experiments of the series, and its composition was somewhat similar. It is of interest to observe that this contained neither silver nor gold, which were concentrated in the rich well separated matte. Slag, copper 0.42%, FeO 67.52%, silica 26.22%, sulphur 2.06%. Matte, copper combined with sulphur 59.71%, iron 13.16%, sulphur 21.94 per cent.

During the preliminary melting of the pyrite in the cupola, the flame seen at the charging door of the cupola exhibited brilliant colors, continually changing in tint. This flame brightly illuminated the visible spectrum. The greater part of the lead was volatilized during this preliminary melting. Indications were, besides, obtained of the volatilization of sodium, lithium, and thallium. The flame of sulphur and that of carbonic oxide give no distinctive lines, neither does arsenic volatilized as arsenic trioxide; hence no indication of the presence of these substances was to be expected, and zinc was probably not present in this flame in the form of vapor. The spectrum flame from the bessemer converter during the blow was a brilliant one. The lines of sodium, lithium, and thallium were recognizable, but the majority of the lines are of (as yet) unknown origin, though they are the most important, since the changes furnish indications of the progress of the chemical changes taking place in the vessel. Some of these unknown lines are those which were employed as test lines. No lines of lead were observed, nor of copper, except in the fourth experiment, in which all the lines except those of sodium disappeared about six minutes before the 'turn-down'. The charge took longer to melt in the cupola than a corresponding weight of pig iron under the same circumstances, but, when once molten, the sulphides formed a mobile limpid liquid.

Whenever during the experiments the converter was tipped forward the SO_2 poured out into the pit and made itself apparent by its strong odor. When the converter was in position, however, and the blast on, the smell was scarcely perceptible in the close vicinity of the flame in consequence, doubtless, of the rapid convection of the hot gas. The lining of the converter, after the six blows, was found to be not very materially acted upon, and therefore the reserve converter was not required. The products of the experiment which remained for examination consisted only of slag and matte and a few samples of the gases evolved. The sulphur and sublimates were principally lost in the cupola, which, as before mentioned, was only used for convenience. In practice, the volatilized substances would be collected in flues.

Further experiments were made on November 1 and 2, also with a bessemer converter. To start the oxidation, about two tons of fused sulphide was run into the vessel, which was then brought into position and the charge was blown for about five minutes without any addition. As the heat of mass increased, four tons of cold pyrite was thrown into the converter, gradually, in large lumps, together with 9 cwt. of sand containing over 16% of moisture. Although a large quantity of heat was expended in expelling the sulphur from the cold pyrite and in the expulsion of the water from the sand, yet the contents of the converter were maintained perfectly liquid by the heat of oxidation. After blowing some time, half the charge was tipped into the ladle, and, after the converter had been again placed in position, 18 cwt. more of pyrite was added, in lumps, and 3 cwt. more sand. After a short time the blowing was discontinued and the contents run out into the ladle.

During the experiment sulphur vapor burned at the mouth of the converter and phenomena were observed similar to those previously cited. It was proved to those present that the smelting of pyrite can be made a self-supporting operation, and that copper matte, silicate of iron slag, volatile sublimates, and SO_2 can be obtained without the employment of any extraneous fuel other than that employed in producing the blast. Another experiment of a similar nature had been previously attempted, but the addition of cold and wet raw material too early in the operation caused too great a reduction in the temperature, and the chilled contents of the converter had to be extracted by melting with coal. This refrigeration will be readily understood when the heat rendered latent by the fusion of the bisulphide of iron is taken into account, besides which large quantities of gas left the liquid mass at a red heat, and were not utilized for heating or driving off sulphur.

These experiments were carried out with an ordinary bessemer plant, but, for many reasons a bessemer converter is not suitable for the reduction of cold pyrite to matte, nor has it any arrangement for allowing the matte to accumulate out of the reach of the blast. Upon running the molten mass from the ladle into one of the ingot molds, the slag ran in a thick cylindrical stream of perfect continuity, falling without noise a distance of some feet into the liquid portion below. This lasted five minutes, and resembled the effect produced by illuminating a continuous column of liquid by a ray of light passed along its axis.

Further experiments with a bessemer plant were made on February 5, at which Prof. Roscoe and other friends were present. The first experiment was accidentally overblown, besides being chilled by the addition of more than the calculated quantity of sandstone. In attempting to continue the operation by the addition of lumps of pyrite, the strong current of air traversing the silicate of iron chilled it, and caused the mass to set in the vessel. It was partly extracted by the aid of coal, and partly by means of rods. About 2 cwt. of coal was now thrown into the empty vessel, and lumps of cold pyrite were added. The blowing being resumed, liquid sulphides accumulated on the tuyere hearth of the vessel, and pyrite and sand were continuously introduced during eight hours. While the operation was in progress a steam-pipe burst, and the blowing was discontinued for 20 minutes. The contents of the vessel, however, remained fused, and the operation was continued without any difficulty. During the 8 hours the blow lasted, a continuous jet of sulphur vapor burned at the mouth of the converter, where it came in contact with the external air. Round the edges, particularly at the base, the flame was of the well known blue color of burning sulphur; but the body of the flame appeared of a greenish tint. This was due to the yellowish brown color of the unburned incandescent vapor of sulphur and the blue color of the sulphur flame; viewed through a small direct-vision spectroscope, many absorption bands were seen occurring at apparently regular intervals, from the red to the violet. Dr. Watts believes the spectrum to be principally due to the sulphur. Over 8 tons of sulphur vapor was probably burned in the 8 hours, and about 18 tons of raw pyrite was treated.

The slag formed in the experiments principally consisted of a silicate of protoxide of iron, but contained some sulphur as sulphide of iron. If, after the matte has been separated from the slag, the latter is further bessemerized, and thus partly converted into silicate of peroxide of iron, the sulphide of iron will be entirely oxidized, and the slag consequently will contain no sulphur. The oxidation of FeO to Fe_2O_3 would produce the heat necessary to maintain the slag molten during this operation.

It is probable that the form of furnace eventually adopted will be a modification of the ordinary blast-furnace, fitted with a tuyere hearth. Such a furnace, built on pillars, with bosh and hearth of some substance not rapidly acted upon by the slag formed during the burning of the sulphides would, working continuously, treat a large quantity of material. Being built on pillars, the crucible hearth and tuyere bottom could be replaced when necessary without disturbing the remainder of the structure, and as these would be the only parts in contact with the fused materials, the furnace from the bosh upward should not experience much wear and tear. When a gannister lining similar to the ordinary bessemer lining is employed for the bosh and hearth, the corrosive action of the FeO would be neutralized and avoided by introducing with the pyrite sufficient silicious material to produce a slag containing at least as large a proportion of silica as compared with the bases, as the formula $2RO \cdot SiO_2$. If, however, a basic lining is employed, the slag should contain less silica, and in no case more than the proportion equivalent to the formula $2RO \cdot SiO_2$. Under such circumstances the blowing would be continuous, the hot charge coming down to a fusion zone, the height of which over the tuyeres would be determined by the amount of air blown in, and the frequency with which the blown products are withdrawn, varying likewise with the composition of the charge. The products could be withdrawn by tapping, as with a common blast-furnace, the matte being run off from a reservoir below the tuyeres, where it would collect. Being thus unacted upon and undisturbed by the blast, rich matte, or even metallic copper, could be produced. By continuing the oxidation and producing Cu_2S, and some metallic copper, the gold and silver will be found with the metallic copper. It is well known that small quantities of silver and gold are far more completely extracted from minerals which contain them by smelting, or treatment with fluid metal or metallic matte, than by any wet process. The fact that by such methods practically the whole of these metals present are collected and concentrated, is the fundamental principle of the analytical assay, and is proved by the accuracy of determinations made in this manner. A large side flue, at the top of the furnace, would carry off the gases and sublimates after their temperature had been reduced in heating the charge introduced above through a self-closing hopper. It is calculated that such a furnace, 30 or 40 ft. high, with a hearth capacity of 1 cubic metre, would be capable of treating annually 50,000 tons of pyrite, and a similar quantity of silicious flux, working 200 days in the year.

The theory of smelting sulphides with a blast-furnace is as follows: The operation is started by placing the tuyere hearth in its place, and throwing in hot coke at the top of the furnace. The blast is now turned on, and the coke develops a high temperature by its rapid combustion. The ordinary working charge of sulphide and flux is now introduced at the top hopper, and as the sulphides melt the coke burns away. As soon as a layer of molten sulphide lies over the tuyeres, the blast is increased and also the burden of the furnace. The charge above the fusion zone as it descends is gradually heated, losing much of its sulphur by volatilization before it becomes molten. On fusion, a considerable amount of lead sulphide will distill over, accompanied by the remainder of the arsenic as sulphide in the strong current of nitrogen and sulphurous acid. These gases, as they pass upward in the furnace, will be greatly reduced in temperature by the volătilization of the sulphur and moisture from the crude materials. There is some reason to believe that more than half of the sulphur in pyrite is volatilized in the free state by this operation. The sublimed oxides, sulphides, and sulphur would be collected in the wide chambers with which the side flue is connected. Below in the hearth the oxygen of the air forced in acts upon the sulphides of iron and zinc contained in the charge, and as long as a constant supply of these substances arrives at the hearth no other constituents present will be appreciably oxidized. A tap-hole near the top of the hearth allows the slag to be withdrawn. The blowing would be continuous day and night so long as the tuyere hearth lasted, and the heat from the gases after they leave such a furnace could be utilized so as to heat the blast or to produce steam-power for the blowing engines. The product of about six tons of material would be tapped every half-hour, so that in seven days work 1000 tons of sulphide would be treated. If desired, the products could be run direct into suitable reverberatory furnaces, when, after the matte had settled, the slag could be run off while yet in a molten state, and in which the oxidation of the matte could be completed. It is difficult to see how the charge could be overblown, but if it were, the product could be worked up again by adding it to a subsequent charge of sulphides introduced at the top.

The SO_2 evolved could be oxidized into sulphuric acid in chambers, or reduced to sulphur by H_2S. The latter decomposition might be accomplished by driving superheated steam into the furnace where the sulphides are oxidized. The SO_2 could also be utilized by Hargreaves' process, and there are other possible methods of treatment, such as dissolving it in water by spray jets in towers, or by condensing the gas to the liquid state. I am indebted to A. H. Allen for indicating to me a means by which large quantities of liquid anhydrous SO_2 can be produced, from which sulphuric acid free from arsenic could be made. This plan appears to me simple, and offers many advantages. "The gases, freed from impurities mechanically carried over with them, are first cooled and then led into towers, or other suitable vessels, filled with charcoal, which will absorb and retain the SO_2 and allow the nitrogen to escape.

The SO_2 is afterward obtained fom the absorbent by exhaustion or heat, and, being thus practically free from the nitrogen, can more readily be liquefied by compression than is possible in the presence of a large quantity of that gas. The SO_2 having been extracted, the charcoal will be ready for another operation, and may thus be used many times in succession."

I have thus endeavored either to prove by experimental data, or to logically demonstrate:*

1. That the whole of the oxygen of the air driven into a thin stratum of protosulphide of iron (FeS), is utilized for oxidation.

2. That by the heat evolved in the rapid oxidation of sulphides and without the use of extraneous fuel other than that employed in producing the blast:

   (a) About one-half of the sulphur contained in iron pyrite (FeS_2), is expelled in the free state.

   (b) The remainder of the sulphur, excepting that left with the matte, is principally evolved as SO_2.

   (c) Although only about 20% of sulphur is oxidized, the proportion of SO_2 to nitrogen by the new process is 14.9%, which is a larger proportion of SO_2 than is obtained by copper smelters who manufacture sulphuric acid. In the ordinary method of burning pyrite, where 45% of sulphur is oxidized, the ratio of SO_2 to nitrogen is only 16 per cent.

   (d) The volatile metallic sulphides, such as arsenic sulphide and lead sulphide, are distilled off with sulphur.

   (e) Iron being more oxidizable than copper, silver, gold, nickel, and certain other metals, these latter will be all concentrated in the matte, provided an excess of sulphide of iron is always present.

   (f) The FeO thus formed is converted into slag by the addition of the silica introduced with the pyrite.

   (g) The more perfect fusion of the slag thus obtained prevents loss of copper by entanglement with imperfectly fused material.

   (h) About 16 to 20 cwt. of incombustible material, having a specific heat of 0.15 to 0.25, can be added per ton of pyrite when a cold blast is employed, assuming that 1000°C. is the temperature necessary for the operation.

   (i) The quantity of similar incombustible material can be increased to 30 to 34 cwt. to each ton of pyrite operated on,

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*Much of the experimental data given in the original paper has, from considerations of space, been omitted.—EDITOR.

when a hot blast of 500°C. is employed, assuming that 1000° C is the temperature necessary for the operation.

(j) Such incombustible material may contain larger or smaller quantities of valuable metals as oxides which will pass into the matte or be volatilized as sulphides, after double decomposition with FeS present in the molten bath. Thus silicates of nickel or copper would be converted into sulphide of nickel or copper, and be concentrated in the matte.

3. That when employing a silicious lining for the furnace the corrosive action of the FeO formed is greatly mitigated, if not practically avoided, by the addition of sufficient silica with the charge of pyrite to produce a slag containing more silica than is required by the formula $2(MO)SiO_2$ (M representing an atom of divalent metal).

4. That the quantity of coal necessary to produce the blast, calculated on the oxygen requisite for the oxidation which takes place, is 1⅓ cwt. per ton of pyrite.

5. That to heat the blast to 500°C. an additional amount of less than 1 cwt. of coal per ton of pyrite is sufficient.

6. That the new process would materially avoid the destruction of vegetation now so bitterly complained of in Spain, and, at the same time, would greatly increase the profits of the pyrite companies that now employ the cementation process.

7. That the new process could be advantageously employed by the copper smelters for treating rich copper ores on account of the great economy there would be in labor and fuel.

8. That the cost of plant is small, compared not only with the quantity of material it would treat, but also on account of the additional profits derivable from the new process.

DEVELOPMENT OF CONVERTER PRACTICE

(Editorial, November 2, 1912.)

Less than a half century ago all the copper produced from sulphide ores was converted from sulphide into metal by a tedious and expensive series of repetitions of roasting and smelting processes. Now the use of the so-called 'bessemer' converter is universal and the successful development of an upright converter, 20 feet in diameter, at the Great Falls smelting plant of the Anaconda company, as described in earlier issues, affords an opportunity for a review of the development of converter practice. The primitive ancestor of the conversion of matte into copper through oxidizing the sulphur by means of a blast of air was the *mabuki* method, practised in Japan from early times. This method had no influence on progress, however, and it was not until after Japanese metallurgists had adopted the modern type of converter that the similarity of its principle of action to that of the *mabuki* method was pointed out. Development had its inception in the pointing out by Henry Bessemer that the removal of the carbon in pig iron could be accomplished by blowing air through the molten mass, as the heat of oxidation of the carbon and silicon would be sufficient to keep the mass molten until the completion of the operation. The analogy with the sulphur in copper-iron sulphides soon suggested itself to metallurgists, and experiments were made by Kupelweisse in 1868 and by Jossa and Lalitin in 1871, with the idea of producing copper from matte in the bessemer steel converter. But, as pointed out by Mr. James Douglas, in his review of converter development, these experiments were dropped when it was found that the copper chilled as it formed and choked the tuyeres, this unexpected result giving rise to the too hasty assumption that the method was impracticable. A few years later John Hollway made a series of experiments and later presented several good papers before the London Society of Arts, describing the results obtained. Elsewhere in this issue we reprint the more significant parts of the first paper, a fuller presentation of which the limitations of our space unfortunately serve to prevent. Hollway so clearly mapped out the general field that his work deserves to be better known by metallurgists, to whom the *Proceedings* of the Society of Arts is comparatively inaccessible, and we accordingly republish it for the benefit of our readers. Hollway points out that the heat generated by the oxidation of the iron and sulphur is sufficient to maintain the mass in a molten state during operation, and that the difficulties are lessened when the active mass is greater; the latter a factor of which the full force has only recently been realized. He further pointed out that the SiO_2 required to form a slag can either be derived from the silicious lining of the converter or from silicious material thrown into the converter during the progress of the operation, and suggests that in this way quantities of silicious gold and silver ores may be smelted. But Hollway was more impressed with the possibilities of his method as applied to the production of a rich matte from low-grade sulphide ores, having directed his experiments to that end, and after remarking that the ordinary form of bessemer converter is not altogether suitable for this work, proposes the use of pyrite smelting in a modified form of blast-furnace. Having blocked out the essential features both of the converting of matte and of pyrite smelting. Hollway was not equally successful in the development of the necessary apparatus. The many accidents of converter operation, now so familiar, served to effectually deter his financial backers, and this brilliant series of experiments came to an untimely end. Shortly afterward, M. Manhès attempted the making of copper from matte in the bessemer converter at his rolling mill at Védennes, and, having hit on the expedient of placing the tuyeres at such a height that the blast would not chill the metallic copper as it formed, he met with such success that he proceeded to erect a full-sized plant at Aiguilles, about 50 miles north of Marseilles. Here two 3-ton bessemer converters were used, in one of which the matte was concentrated to 60 per cent copper, then transferred and blown to blister copper in the second. It is truly remarkable that so important an advance in the art of copper smelting should be made in a country of so little importance as a source of copper that it does not ordinarily appear in the list of producers, and at a place so obscure. The honors may fairly be divided between Manhès and Hollway. Hollway made a scientific study of a general problem with commendable thoroughness, but unfortunately he slighted the most important possibility of his results, and his experimental work was inconclusive. Manhès, on the other hand, concentrated his attention upon the making of copper from matte, and being able to experiment at his own plant, quickly attained success.

Development of the method now transferred itself to this country. Franklin Farrel followed the lead of Manhès at the Parrot works at Butte, with the assistance of some of Manhès' pupils, so successfully that six converters were soon at work, the scheme of keeping the tuyeres free from copper by systematic punching, initiated at the Parrot, being a material aid to success in operation. By 1890 the advantages of the new method were generally recognized and converters were in use or under construction at Anaconda, Great Falls, Butte, and Ashio, Japan. All these plants used some modification of the upright type of converter, and it was not until later that the horizontal type of converter was first employed in America at the Vivian plant at Sudbury, Ontario. The management of the Copper Queen company had studied the operation of small converters of this type at an Italian smelter at Leghorn and 5-ton converters were constructed for experimental purposes. The purpose of the design of the horizontal or 'Bisbee' type, as it afterward came to be known, is to obviate trouble from the chilling of the copper at the tuyeres by placing them in a horizontal row on one side so that by tilting the converter the tuyeres may be kept above the level of the accumulated copper. For a variety of reasons this type of construction strongly appealed to metallurgists, and its use became so general that it was not long before only a few plants retained the upright construction. Among these was the smelting plant at Great Falls, where the staff not only adhered to the type, but developed it from 7 feet in diameter to 20 feet in diameter, and have earned the right to attach the name of Great Falls to this form of construction, originally known as the Parrot type. Meanwhile the changing flux of conditions had led, through a series of experiments which would require too much space for mention, to a general use of Hollway's method of throwing in silicious material through the mouth of the converter, and it was found useful to observe his suggestion that the difficulties of operation are less when the mass of material treated is greater. Messrs. Pierce and Smith accordingly devised a large horizontal converter with a basic lining which could be successfully used and was generally adopted at the plants of the American Smelting & Refining Company. The original form of this converter was mechanically imperfect, and exhibited in exaggerated form the drawbacks of the Bisbee type. The shape of the lining in the horizontal type is structurally imperfect to resist the strains resulting from changes of temperature and the effects of mechanical abrasion. The shape of the lining in the upright type, a vertical cylinder, is mechanically good, and as a result this type of construction is being adopted at the Cananea, Copper Queen, Calumet & Arizona, and Arizona Copper Company plants, and at the El Paso and Hayden plants of the American Smelting & Refining Company as well. The use of large upright converters with basic lining decreases the cost of converting nearly to the irreducible minimum of the power cost for air, and the minimum of labor for charging, pouring, and punching of tuyeres, and it seems improbable that any further great advance will be made, though metallurgists will continue, as in the past, to constantly improve their practice within the limits of the basal definition of metallurgy as the art of making money out of ore. Meanwhile Messrs. C. W. Goodale, H. A. Wheeler, and the staff of the Great Falls plant have the satisfaction of knowing that 'sticking to the job,' in the words of Mr. E. P. Mathewson, does bring gratifying results.

DEVELOPMENT OF CONVERTER PRACTICE

By HERBERT HAAS

(October 25, 1913.)

In the *Mining and Scientific Press* of November 2, 1912, there appeared an excerpt from the well known paper by John Hollway, 'A New Application of a Process of Rapid Oxidation by Which Sulphides Are Utilized as Fuel,' presented before the Society of Arts (London), February 14, 1879. In the accompanying editorial, entitled 'Development of Converter Practice,' as is usual in articles dealing with the history of copper converting, Pierre Manhès receives sole credit for having developed the first successful copper converter notwithstanding the fact that Paul David, who was then Ingenieur-Diréctеur of the Védennes works (Vaucluse), of which Manhès was the owner and furnished the means for the trials, did a great deal of original work, and as metallurgical engineer and coworker of Manhès should at least share equally with him in the honor of being the first to devise a successful copper converter. Nor can I agree with a possible inference from that editorial, namely, that John Hollway made his experiments to devise a process of bessemerizing copper matte. On reading his article critically it becomes apparent that what he advocated and was trying to do is pyrite smelting as practised today. Because he used the term 'bessemerizing' and gave to his experimental apparatus the name 'converter', a number of well known metallurgists ascribe to him the important initial work of bessemerizing. As a matter of fact, Hollway wanted to get away from the bessemer converter, and so states in his article, and describes a furnace as being more practical in carrying out his idea, namely, to produce matte from pyrite ore without the use of extraneous fuel.

Robert Sticht, in his chapter on pyrite smelting, in E. D. Peters' 'Modern Copper Smelting,' accurately remarks: "To John Hollway of England is due, therefore, the credit of being the founder of the smelting process advocated in these lines" 'A new application of

Bessemer's process of rapid oxidation by which sulphides are utilized for fuel.' "When metals are extracted from their ores by fusion the necessary heat is always obtained by burning coal, coke, or other form of carbon. I wish, however, to remind you that sulphides can be made to burn in air, and are thus combustible substances, while oxides are bodies that have already burned, which as you know is the conventional expression for entering into combination with oxygen. The metallic sulphides, consequently, are natural combustible minerals, and my object is to prove that they can be utilized as a source of heat in certain metallurgical operations. The most important of the mineral sulphides is pyrite, both on account of its occurrence and the extent of its deposits." But Hermann A. Keller, in his chapter on 'Bessemerizing Copper Mattes,' in the same work, as well as James Douglas,[1] again refers to Hollway as an experimenter trying to apply the converter to the conversion of copper sulphides. Hollway had in mind the treatment of sulphide ores, and their conversion into a molten state with a corresponding concentration of the copper sulphide into a regulus, without the use of extraneous carbonaceous fuel, but did not attempt to convert matte into copper.

A few quotations from his article will bear me out in my views. "These experiments were carried out with the ordinary bessemer plant, but for many reasons a bessemer converter is not suitable for the reduction of cold pyrite to matte, nor has it any arrangement for allowing the matte to accumulate out of the reach of the blast." "The blowing being resumed, liquid sulphides accumulated on the tuyere hearth of the vessel, and pyrite and sand were continuously introduced during the eight hours."

"It is probable that the form of furnace eventually adopted will be a modification of the ordinary blast-furnace, fitted with tuyere hearth * * *, the blowing would be continuous, the hot charge coming down to a fusion zone, the height of which over the tuyeres would be determined by the amount of air blown in, and the frequency with which the blown products are withdrawn, varying likewise with the composition of the charge. The products could be withdrawn by tapping, as with a common blast-furnace, the matte being run off from a reservoir below the tuyeres, where it would collect. Being thus unacted upon and undisturbed by the blast, rich matte, or even metallic copper, could be produced. By continuing the oxidation and producing Cu_2S, and some metallic copper, the gold and silver would be found with the metallic copper."

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[1] *Trans.* Inst. Min. & Met., Vol. VIII, 1899.

In this last sentence Hollway discloses that he wanted to make use of the well known Welsh process of bottom smelting; the concentrating of silver, gold, and impurities in a small amount of metallic copper which, if separated from the white metal, would leave the latter remarkably free from impurities, thus facilitating and cheapening the refining process of the bulk of the copper. This well known property of certain metals and metalloids to form with copper saturation products, alloys, or 'solutions,' in preference to remaining in the 'white metal' was made use of later by David in his ingenious *sélecteur* or best selecting process of making copper bottoms in the converter, in lieu of the Welsh practice of doing it in the reverberatory furnace. He provided his converter with a pocket, in which was collected the copper reduced during the first 10 minutes of blowing on white metal. This copper acts as a collector or *sélecteur* for the impurities, like arsenic and antimony, tellurium and selenium, gold, and, to a lesser extent, silver. This small amount of copper (aggregating about 10% of the total amount converted per charge) is tapped and leaves the rest of the copper reduced afterward (by continuing the blowing on white metal) remarkably pure and free from impurities. The importance of this practice for small works has been fully discussed by Paul David, Jannettaz, James Douglas, and others.

To continue Hollway's article: "If desired, the products could run directly into suitable reverberatory furnaces, when, after the matte had settled, the slag could be run off while yet in a molten state, and in which the oxidation of the matte could be completed." Evidently it was Hollway's intention to oxidize this matte or white metal by the then used methods of reaction smelting, by roasting a portion of the matte and having the dead roasted and partly oxidized calcine (sulphates and oxides) react with the molten sulphides, as was done in the well known Nicholls process.

These quotations should prove conclusively that Hollway had not in mind the conversion of copper matte into blister copper, but that he wanted to smelt raw pyrite in order to get the copper in it into a more concentrated form, that of matte, which he wanted to collect in a hearth "removed from the action of the blast." His experiments are indeed interesting as being the first authenticated attempt on record at pyrite smelting. His logic was sound as far as the well recognized facts were concerned, that sulphides can be made to burn in air and the metallic sulphides consequently are natural combustible minerals, but he made the mistake of trying to conduct the two distinct operations of smelting and of refining (which the bessemer process really is) at one and the same time and in the same apparatus. Starting with a pyrite ore, with 2 or 3% copper, and requiring the addition of ½ to ¾ ton of silicious ore, the concentration of this small copper content in the original furnace feed into blister copper would have required the handling of 60 to 100 tons of material for every ton of copper. This presents many practical difficulties, unless the two operations of smelting and converting are conducted separately. He approached this problem more as a laboratory worker than as a man who has practical knowledge of furnace operations. A parallel approaching Hollway's work is found in the Knudsen process, which attempts to smelt and convert simultaneously. The Knudsen 'converter' is, however, lined with magnesite. This process was at one time in operation at Sulitjelma, Norway, and was also tried at the United Verde smelter, Jerome, Arizona.

Reviewing briefly the essential features of the development of the copper converter, it will be found that the idea of applying the converter to the conversion of copper matte occurred to several soon after Sir Henry Bessemer's address at the Cheltenham meeting before the British Association on August 11, 1856, "on the manufacture of malleable iron and steel without fuel." Gossage, Baggs, and Keates secured patents to apply the process to copper matte in 1856. Four years later, Hunter secured a similar patent. In 1862 the Glass and Lecere patents followed.[2] In 1866 Peter von Rittinger, better known through his extensive work on ore dressing, advocated the adoption of the Bessemer process to copper sulphides, and made experiments in Hungary, in 1867, at the Schmollnitz works. In *Oesterreiche Zeitschrift für Berg- und Hüttenwesen*, 1886, pages 68-71, there is a description of his attempt to convert matte in the reverberatory furnace by passing a pipe through the matte and blowing air through the molten bath. This 'Zugutebringen der Kupferrohleche durch Verblasen derselben nach Bessemer Art' (Beneficiation of copper mattes by blowing according to Bessemer's process) met, however, with no more success than the previous attempts of others.

More successful was the work done at the Wotkinsk smelter in the Urals, in 1866, after Semenikow, a mining engineer, and director of the Bogoslovsk mining district, had proposed the application of the bessemer converter to the blowing of copper matte to black copper. The blowing was, however, only carried to the white metal stage, with 72 to 80% copper. The matte converted contained 31.5% Cu, 25.3% S, 39.4% Fe, and the concentrated product had

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[2] *Bulletin* de la Société Scient. Industrielle de Marseilles, 1901, pp. 78-79.

the composition. 77.5% Cu, 19.3% S, 1.76 Fe. The slag assayed 34.46% SiO_2, 4.73% Al_2O_3, 3.06% CaO, 55.26% FeO, 2.13% CuO, 0.11% S, and metallic copper 1.7%. About 1300 lb. was handled per charge. An English type of the standard steel converter was used. This blowing operation, taking about 20 minutes time, obviated at least one roasting and re-smelting of the matte by a slow and tedious process, the white metal thus obtained by converting being dead roasted and smelted to black copper in a reverberatory furnace.[3] The patents of Rath in America in 1866, of Dixon in England in 1869, of Gibb, Gelsthorp, and Tessie due Motay in 1870, and of Lavoissier in 1874 evince the interest men gave to this matter of devising a successful process for the bessemerizing of matte.[4] In 1878 and 1879 Hollway made his interesting experiments in pyrite smelting.

In 1880 Pierre Manhès and Paul David attacked the problem at the Védennes smelter, in Vaucluse, France, of which Pierre Manhès was the owner, and David the engineer in charge. Fortunately, they did not know of the many failures that had preceded their work, or it might have discouraged their efforts. They made their first experiments in a small steel converter, holding a charge of about 110 lb. It had a wind-box in the bottom, and the tuyeres were vertical, thus admitting the blast into the molten metal from the bottom. They began with a matte containing 25 to 30% copper, which was first melted in a reverberatory furnace. The lining was made of sand. The operation went along nicely until the matte had been concentrated into white metal. Then, after the tuyeres had been cleared, there was a violent boiling and foaming of the charge, and a portion of the charge was thrown out. It is quite possible that the slag had not been removed and that when blowing on white metal began, the well known phenomenon of 'foaming' was the result. This takes place when the white metal is blown with the slag still covering it. As the oxidation of the white metal proceeded, and the copper was thrown down, being immediately above the tuyere orifices, and having no heat producing impurities to give off, it chilled. The tuyeres clogged, and the blow came to a premature end. Variations of the copper and iron content of the matte had no appreciable influence on the result.

But Manhès and David were not discouraged by these first failures. They soon discovered that the chief difficulty appeared to be

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[3] Berg- und Hüttenmannische Zeitung, 1871, p. 7 *et seq.*

[4] P. David, *Bulletin* de la Société Scient. Industrielle de Marseilles, 1901, p. 78.

the clogging of the tuyeres by the chilling of copper, which cut off from the superimposed white metal the wind supply and thus stopped the blowing operation. They exchanged the vertical tuyeres for horizontal ones, placed a few inches above the bottom of the converter. The wind-box was changed into a hollow ring which supplied the blast to the different tuyeres. This arrangement is similar to the one used later on blast-furnaces instead of the bustle pipe and tuyere pipes, the main delivering the air into this wind-box, this type of furnace having been used at Bisbee at one time (1902-1903). With this construction they overcame the difficulties first encountered, the copper remained liquid until the entire charge was blown, even in this small apparatus.

In view of Hollway's work and that of Manhès and David, it is significant that the latter two started with matte, recognizing from the start the essential differences between a bessemerizing operation and a purely smelting operation, starting with raw ores. Another departure they made from steel bessemerizing is that they initially intended to have the sand lining furnish the silica required for slagging the iron, whereas Hollway painstakingly tried to avoid the attack on the gannister lining of the converter by attempting an impossible silicious slag. Since the extent to which iron and silica combine with each other in the acid copper converter is left to the free disposition of these two substances, the natural slag is a basic one, and for this reason a basic lining is the successful lining of the copper converter of today.

After these preliminary trials, conducted over a period of one year at Védennes, the process was introduced on a larger scale at Eguilles, near Sorgues, Vaucluse. The new works consisted of three blast-furnaces, two reverberatories, and three converters. In 1884 the plant was increased to five blast-furnaces, two reverberatories, six converters, and two refining furnaces.[5] As a variety of mattes had to be treated (mattes high as well as low in iron and correspondingly low and high in copper), with the variety in custom ores handled and treated, it frequently happened when blowing low-grade matte that the enriched matte or white metal would be *below* the tuyeres, so that the blowing was on slag, and the charge could not be blown to a finish, yielding copper. This difficulty was first overcome by removing the enriched matte or the white metal to another converter and finishing the blow in that manner. But it soon led to the adoption by Paul David, in 1885, of a cylindrical form of converter that could be turned around its central longitudinal axis. With this construction the relative position of the tuyeres

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[5] *Bulletin* de la Société de L'Industrie Mineral de St. Etienne, 1901, p. 614.

could be changed (raised or lowered), so that a variety of mattes could be blown in the same converter and in one operation. It also reduced the blast pressure needed. It is this cylindrical form of converter that was introduced at Livorno, Italy (known as the Leghorn converter), and adopted later by the Copper Queen Con. Mining Co. for its Bisbee works.

Soon after success had been achieved by Manhès and David at Védennes, the Société de Metallurgie du Cuivre (procédé Pierre Manhès) was formed to exploit the process. Vernis, who had formerly worked at the Eguilles works, built and operated for one year the first converters in America, those of the Parrot Silver & Copper Co., at Butte, Montana. Franklin Farrel, of course, was instrumental in introducing the process in the United States, and deserved the credit of 'pioneering.' This is all the more creditable, because the Anaconda was then, as it is now, the largest copper producer in Montana, its production then being nearly 50% of the total state production, whereas that of the Parrot works was materially less. The Parrot works was the only works which at that time shipped black copper, or blister copper, the other works shipping rich matte.

COLLAR PULLER FOR CONVERTERS

By L. O. Howard

(May 17, 1913.)

An important feature of converting practice is the keeping open of the mouth of the converter. The material thrown up by the violent agitation caused by the blast is thrown up through the open mouth, where, through cooling, it becomes attached and gradually builds up 'noses' or 'collars', which if allowed to grow unchecked will in time completely close the opening. This difficulty is especially marked in the operation of the basic lined converter, since the preservation of the lining requires that the temperature be kept moderate. The best method of dealing with these accretions is to remove them as fast as they form, the common practice being to pull away the crusts with a hook operated by the crane, the converter being tilted at an angle of 45°. Another device has been perfected at the converting plant of the Old Dominion C. M. & S. Co., at Globe, under my direction. This is shown in the accompanying illustration.

After giving hooks of different shapes and sizes a thorough trial at the Old Dominion plant, it was decided that they were impracticable, and experiments were made along other lines. Lee Young, one of the converter foremen, suggested using a heavy bar hung horizontally in such a manner that when dropped through the mouth of

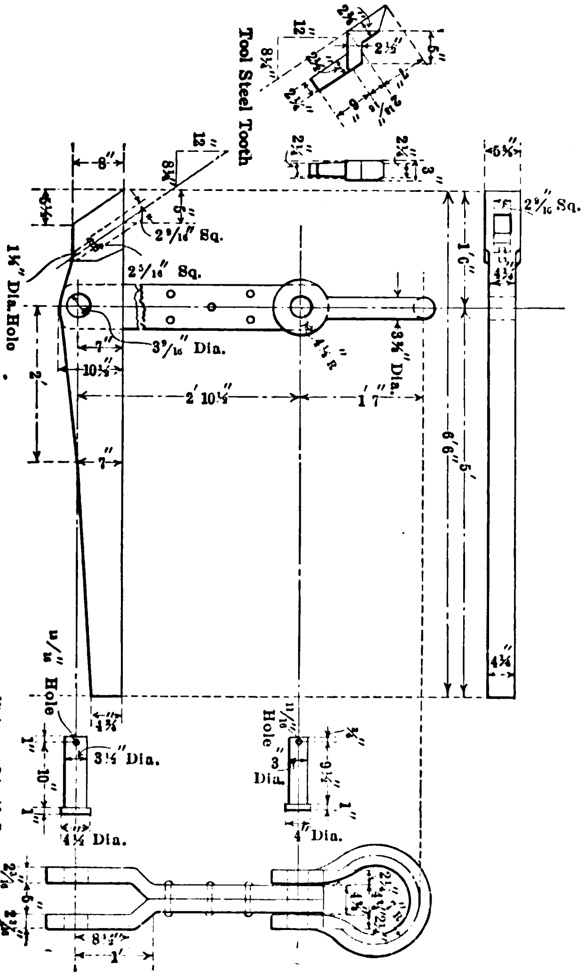

FIG. 32. DEVICE FOR CLEARING CONVERTER MOUTHS.

Mining and Scientific Press

the converter the ends would catch under the projecting crusts and remove them.

When a rim is to be pulled, the cutter is picked up by the crane and dropped tail first through the mouth of the converter. The cutting end is worked into position where the bite is to be made, and a pull taken with the crane, thus lifting the tail end until it comes into contact with the opposite side of the converter and forcing the cutting tool into the collar. A steady pull with the crane will draw the tool through the mass until near the top, when the collar breaks and comes away easily.

One strong point in its favor is that the steel cutter easily tears through the heaviest collar without lifting the shell from the rolls. A bite at any angle may be taken and a greater or less amount of material removed at will. Fifteen or twenty minutes work each shift is sufficient to keep the shell open to any size required.

OPERATING COPPER CONVERTERS BY ELECTRIC MOTORS

By G. B. Rosenblatt

(May 17, 1913.)

The use of electricity for operating copper converters is coming more and more into vogue in progressive smelters throughout the West, supplanting to a great extent hydraulic operation which used to be the rule in the old days. Electric driving has proved so satisfactory that many installations have been made with converters of all sizes. The largest converter at present electrically operated is the one of 250 tons capacity recently constructed at the Great Falls plant of the Anaconda Copper Mining Co. The converter proper, not including bearings and gears, weighs about 65 tons, and will take approximately 50 tons of charge. This converter is to be operated by a 100-hp. Westinghouse, type 'MC', direct-current motor.

The first converters to be electrically driven were equipped with direct-current motors, but an entirely satisfactory alternating-current motor has now been developed for this service, and a number of installations of this type have recently been made which have proved entirely satisfactory. With either direct or alternating-current motors, the most approved practice is to gear the motor to a jack-shaft which in turn drives the large gear on the converter proper through a worm-gear. This prevents any slippage and eliminates the possibility of the converter pulling the motor past any position desired. Motors for this work are usually totally enclosed so as to make them dust-proof, but in some of the more modern installations it has been the practice to install the motor alongside the converter in a sheet iron housing of its own, in which case, while there is no disadvantage in having a totally enclosed motor, absolute dust-proofing is not altogether necessary. The accompanying illustration, Fig. 33, shows a barrel-type converter with basic lining, at the plant of the International Smelting & Refining Co. at

Tooele, Utah. A totally enclosed Westinghouse motor is used, but installed in the housing shown at the right of the converter. The type of motor used is the same as that shown in the illustration of the converter crane.

The chief advantage of motor operation over hydraulic operation is the smoothness of the motion. The converter starts gently, without halt or jar, and this is bound to increase materially the life of the lining. Smoothness of operation is of particular importance with quartz-lined converters, but is not to be disregarded in the operation of basic-lined machines. The smoothness and ease of motor control also obviates much of the danger from the splashing of molten charge within the converter. An electric brake is usually attached to the motor, and is applied automatically whenever the current is shut off from the motor. It holds the converter dead

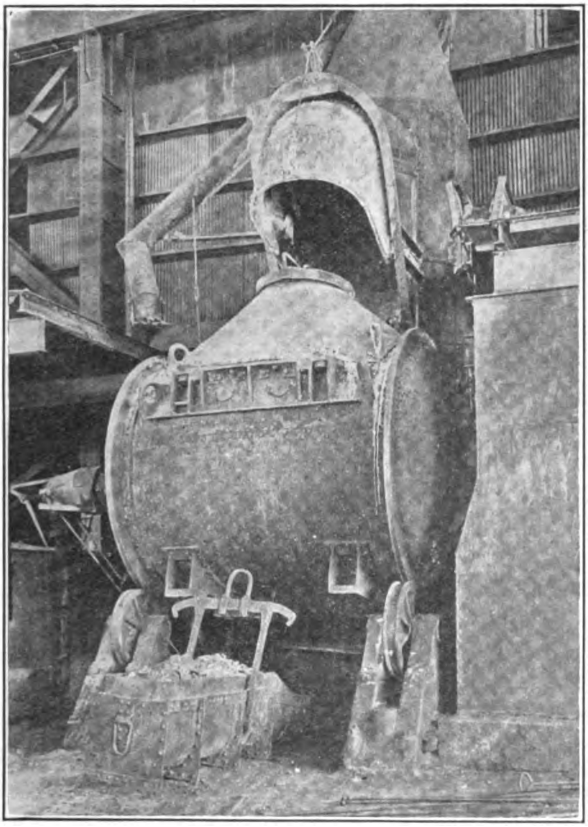

FIG. 33. CONVERTER AT I. S. & R. PLANT.

even in case of failure of power, and makes an entirely satisfactory accessory.

Motors for converter service are usually built with some special characteristics. Absolute reliability is of prime importance, as the converter operation must not be interrupted. To secure this, a strong and rugged construction is employed similar in many respects to that employed for motors used in steel mills. A high overload capacity is provided to take care of emergencies, and the installation is designed to withstand high temperature, so that even a splash of molten metal from the converter mouth on the motor will not burn it out. Totally enclosed dust-proof construction is recommended, but is not an absolute requirement when a separate housing is used over the motor. Ease of inspection or repair in case of some accident should be given great consideration, and these motors for converter service are therefore usually built so that any part can be quickly and efficaciously removed and substituted.

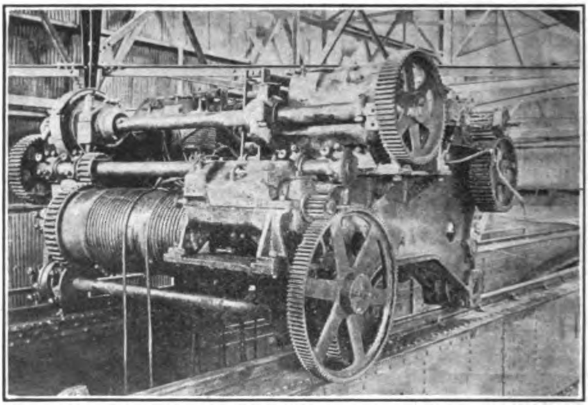

FIG. 34. MOTOR-DRIVEN CRANE.

The best practice is to equip the cranes for charging and handling the converter with the same type of motor equipment that is used to operate the converter proper. These motors on the cranes are subjected to much dust, fumes, and high temperatures, so that the rugged construction of the converter motor is not at all out of place. Further, the use of identical motors on the crane and on the converter simplifies the matter of spare parts, an advantage by no means to be overlooked when the absolute necessity of keeping things moving is considered.

MINERAL LOSSES IN GASES AND FUME

By F. G. Cottrell

(March 30, 1912.)

*In considering the mineral wastes passing out of the stacks of our smelters and metal refineries, distinction must be clearly made between the gases themselves and the clouds of suspended solid and liquid particles which they mechanically carry along with them. In which class a material belongs often depends, to be sure, on temperature. Many metallic compounds are actually vaporized in the furnaces and gradually condense from gases into clouds of solid particles with the progressive cooling of the gas streaming through the flues. Arsenic in the form of the trioxide is usually the last of these materials to condense, since even down to 150°C. its vapor tension is sufficient to permit the loss, in the gaseous state, of several tons per day for the largest plants. But below, say 125°C., for ordinary technical purposes it may be considered that the only important element in the gaseous state is sulphur, in the form of dioxide.

For many centuries the material nature of gases and the fact that they actually possess weight escaped the chemist. Even today a distinct effort of mind is required to sense the vast tonnage of the clouds to be seen floating away so lightly from smelter and power-house stacks. There are many single stacks in this country, from each of which over five tons of gas issue per minute, while in isolated instances this is exceeded several-fold.

Aside from carbon, the element which is lost in the greatest tonnage is unquestionably sulphur. Many of our Western ores will run from 25 to nearly 40% sulphur, and a plant smelting 1000 tons of ore per day is at present considered to have a very moderate capacity. The largest plants will easily touch the 1000 mark in tons of sulphur passing up their stacks when at full capacity. On stopping to think that this represents three thousand tons of concentrated sulphuric acid, were it made into such, and that one such plant could have supplied all the acid used in the whole phosphate industry of the United States a year ago, it is easier to appreciate the skepticism of the smelter companies regarding acid manufacture as a general solution of the sulphur fume problem. The disposal of this sulphur presents unquestionably the gravest problem confronting the metallurgical industry today.

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*Abstract from the *Journal of Industrial Engineering & Chemistry.*

Suggestions have not been wanting for methods of direct absorption of this gaseous sulphur dioxide by water, limestone, lime, or charcoal, either as a means of final disposal, or as a step in its concentration and final liquefaction under cold or pressure. One of the most ingenious and attractive is that it be used for decomposing fine granulated and moistened slag,[1] thus fixing the sulphur in harmless compounds and, at the same time, leaching out any metal still held in the slag. While many of these are perfectly practicable on a laboratory scale, the tonnage to be handled and the cost, either in first installation or operation, have prevented them from producing any practical results up to the present.

From a chemical standpoint, the other obvious alternatives consist either in oxidation to sulphuric acid or reduction to elementary sulphur. Of these two, the first has the advantage of requiring no additional substance save the oxygen of the air, and of running itself, when properly catalyzed, without need of external energy, but unless an immediate market is at hand for the sulphuric acid, its disposal in turn becomes difficult and often impossible.

The greatest consumption of sulphuric acid in this country today is in the manufacture of superphosphate fertilizer, but, until very recently, the chief demand for fertilizer has been in the Southern and Atlantic states; here, too, were the chief known deposits of phosphate rock, while, on the other hand, the great smelting industry is for the most part in the far West, and cut off by high transportation costs. The recent discovery and description in Idaho, Wyoming, and Montana, of what promise to be the largest phosphate deposits in the world, will, it is hoped, materially change this condition.

Much of the land in the West is still practically virgin soil; where crops are falling off, the reason for this and their relations to different kinds of fertilizers to tillage and crop rotation, are still so much in dispute that in trying to build a rational and permanently helpful phosphate industry, the facts must first be thoroughly established from the standpoint of agriculture, manufacture, and transportation. At present, these are not available to any extent justifying an immediate general development of the phosphate industry in the West; but coöperative work, aimed at a broad and practical study of these conditions, is already well under way between the U. S. Department of Agriculture, the Agricultural Experiment Stations of various Western states, the U. S. Bureau of Mines, and the large metallurgical interests of the West, and it is hoped that

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[1] "The Westby-Sorensen Process,' E. P. Jennings, *Eng. and Min. Jour.*, 86, 418-19, August 1908.

the results may soon justify and lead to a healthy development of the industry.

An encouraging example is before us in the work already accomplished by the Tennessee Copper Co. at its plant at Copper Hill, Tennessee. This company is successfully treating the weak gases from copper blast-furnaces on a scale for which many predicted failure, and which certainly did require time, courage, ability, and indefatigable effort to perfect. The very difficulties and delays which the work had to encounter have in one sense been a help to the industry, as the present annual capacity of 250,000 tons of chamber acid has been gradually reached over a period of some years, and while it has materially reduced the price of acid in the South, the market has had a chance to adapt itself with the minimum of hardship to those already in the field, and with great gain to both the metallurgical and agricultural communities. Although much may undoubtedly be done, not only to increase the consumption of sulphuric acid in superphosphate manufacture, but also in developing other and newer uses, still the chemist should not rest content with this.

I have considered so far only the results of oxidation of the sulphur to dioxide. On the other hand, it may also be reduced to elementary sulphur, for which purpose a number of methods have been proposed, some depending on the action of solid carbon or of carbonaceous reducing gases, while others use hydrogen sulphide generated by the action of steam or acids on alkaline or metallic sulphides. An interesting general discussion both of the possibilities and the limitations of these methods together with some very suggestive new facts[2] has recently been attracting considerable attention and may not improbably lead to practical results, at least in certain special cases.

This naturally leads to the inquiry: "Why should all the sulphur of the ores be oxidized in the first place?" This question comes home still more forcibly on looking through the charge doors of a pyritic copper blast-furnace shortly after a fresh charge of ore has been dropped on the column and seeing the great clouds of unburned sulphur subliming out from the charge itself and burning above it, while all the heat thus produced is wasted. If copper blast-furnaces could be run with gas-tight tops, as is the case with iron blast-furnaces, and full utilization of the air blown in the lower part of the charge column, it certainly seems as though a far better

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[2] 'The Thiogen Process for Reduction of SO_2 in Smelter Fume,' S. W. Young, *Mining and Scientific Press*, 103, 386,387, Sept. 23, 1911. (See p. 269.)

thermal efficiency should be attainable, besides the possibility of delivering a very considerable proportion of the sulphur of the ore to the flue in unburned form, either as a gas or cloud of flowers of sulphur, depending on how cool the top of the ore column could be maintained.

That such a mode of operation is not entirely impossible is curiously evidenced by the fact that the first pyrite copper blast-furnace ever run[3] was operated under these very conditions, the run finally terminating, after a few hours, by so much free sulphur accumulating in the flues as to stop the furnace. In this classic article of John Hollway's, particular stress was laid upon the recovery of the sulphur, but the development of smelting practice soon began to follow other aims. It was naturally in the hands of the copper men, and sulphur did not so directly interest them. Then, too, in the far West, where pyrite smelting was first carried on successfully on the large scale and where even today probably 90% of it is still practised, the limited market for sulphur and the high cost of transportation and low value per ton, as compared with copper, have naturally influenced metallurgical development to strain for the greatest degree of economy in copper production, with absolute disregard for sulphur as a by-product.

In working the vast tonnage of low-grade sulphide ores of the West, effort is concentrated on burning out and slagging off the maximum amount of iron in the blast-furnace in order to produce as high a percentage of copper as possible in the resulting matte, as it is much more expensive to burn out a pound of iron in the converter than it is in the blast-furnace. This greater cost is due chiefly to two items; the cost of the refractory linings, and the greater power-consumption due to the higher pressure of air in the converter. Hollway looked on his blast-furnace simply as a means of melting the ore into low-grade matte, the remaining iron of which might then be burned off in some of the then well known processes or in a converter, which he was also the first to use in copper metallurgy.

In copper smelting, it has been the cost of converter practice which has led to crowding as much of the work of oxidation upon the blast-furnace as possible, and this insures the burning up of all the sulphur instead of recovering a good portion of it. Within the last few years, one of the two items of converter cost, that of linings,

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[3] John Hollway, 'A New Application of a Process of Rapid Oxidation by Which Sulphides Are Used for Fuel.' *Jour. Soc. Arts, London*, 27, 248-270, 292-295, 488-495, 606-607, 728-730. (See p. 201.)

has been greatly reduced by the practical development of basic linings, so that today this factor is rapidly becoming almost negligible, and all that stands in the way of shifting most of the work of oxidation from the blast-furnace to the converter is the cost of the high-pressure air (10 to 15 lb. per square inch as against 3 lb. for blast-furnace) required to force its way in against the hydrostatic pressure of the deep liquid bath in the converter. If a mechanically more efficient way of mixing this air with the molten matte in the converter could be secured, it would mean the opening of a new chapter in metallurgy.

Even with the greater cost for air, the relation between blast-furnace and converter operations is already steadily changing in the sense of taking a lower-grade matte to the converters, and this in turn may be expected to react in a study of the blast-furnace from a somewhat different point of view than has hitherto obtained.

The copper blast-furnace is today confessedly one of the crudest appliances of engineering practice, and we know with less certainty the mechanics and chemistry of its inner workings than of almost any other apparatus of equal industrial importance. The slow development of a really fundamental scientific basis for this field of metallurgical engineering has undoubtedly been due in great part to the exceptional difficulty, if not impossibility, in this particular subject, of interpreting small-scale experiments in terms of full-scale operation, and the great expense of full-scale experiments.

Another reason for metallurgy having lagged behind other engineering branches is perhaps to be found in the fact that commercially it has usually been pursued as an adjunct to the private ownership of some natural resource. If the mine were a rich one, the ultimate economies of smelting practice were apt to be overshadowed by the more immediately attractive study of increasing tonnage; if, on the contrary, the low grade of the ore made careful smelting necessary, the company felt it could seldom afford to go into expensive experiments for radical developments of the art, but must stay close to standard practice and make its improvements on minor details.

Even if elementary sulphur were produced, its present consumption outside of sulphuric acid manufacture is too small to be at all significant, but it at least has the advantage over acid of permitting safe accumulation and storage to an unlimited extent, as illustrated by the practice in Louisiana, where it is cast into blocks some 200 ft. square and 50 ft. high. What is really most needed for the solution of the sulphur fume problem is the diccovery of new uses and a wider extension of existing uses for both elementary sulphur and sulphuric acid. Chemists have become accustomed to looking at sulphur almost exclusively from its chemical aspect, but if it is ever to be utilized in anything like the proportions in which it is now being thrown away, it would be well to consider possible mechanical uses, as, for example, a binder for sand or wood fibre. Experiments on the manufacture and use of drain tile made of such material are at present being carried out in the West, where the recent rapid development of irrigation is constantly bringing new and special demands for material of construction.

If metallic sulphides are dissolved in molten sulphur, its toughness and other mechanical properties are greatly improved, as first pointed out by J. B. Spence.[4] A detailed study of this product from the standpoint of structural materials has long awaited the investigators and seems much to be desired. If a sufficiently large and permanent outlet for sulphuric acid, on the one hand, and elementary sulphur on the other, can be assured reasonably near the sources of supply, the metallurgical industry itself can be safely relied upon to meet the demands and find the way to supply them.

DUST EXTRACTION FROM SMELTER SMOKE

(July 23, 1910.)

The problem of precipitation of dust from smelter smoke has engaged the attention of metallurgists for many decades, and it still offers opportunity to trained investigators. The staff of the Copper Queen Consolidated M. Co. at Douglas, Arizona, under the direction of G. B. Lee, the smelter superintendent, has been conducting a series of experiments for the purpose of determining the conditions under which the solid matter in the smoke may be precipitated and recovered. The details of this study will soon be available in a paper to be published in the Bulletin of the American Institute of Mining Engineers. The effect of temperature, expansion of volume, velocity of the gases, and the like, have been carefully investigated, and an enormous amount of data accumulated. The experimental plant in which these observations were conducted consists of a 6-in. pipe running up the side of the stack, and turned by elbows so as to take the gases from the centre, an exhaust being so applied to this pipe as to secure an accurate sample, leading the fume away at the same

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[4] 'A New Metallic Compound' (Spence's Metal), Granville Cole. *Jour. Soc. Arts, London*, 28, 225-229 and 279, Feb. 13, 1880. See also Thurston's 'Material of Engineering', Vol. III, p. 205.

velocity as that at which it rises in the stack. Chambers provided with means for observation of temperature, pressure, etc., receive the gases below. These lead to an experimental bag-house. The immediate result of the work done at this plant has been a decision to enlarge the existing dust-chamber. At the Copper Queen smelter the dust-chamber depends upon volume rather than length for efficiency, and is not provided with baffle-walls, nor with means to increase friction by extension of the surface-area in contact with the gases. The flue opens directly into a wide high chamber, which becomes narrower by successive steps toward a short flue leading to the stack. Thus the advantage accruing from the chamber is that due to expansion. The chambers are merely being enlarged in their horizontal dimensions.

The amount of dust mechanically carried into the atmosphere through the stack of the Copper Queen smelter is large, and its recovery is entirely an economic measure. The precipitation of the sulphuric anhydride does not enter into the problem, as there is no contest with the farmers at this point. The agriculturists at Douglas seem to be as content to have the smelters in their midst as are the farmers of New Jersey, who abide in peace with their metallurgic neighbors. In fact the situation at Douglas is unique. The diurnal variation in temperature is normally great, and at certain seasons the change is so severe as to prove fatal to agriculture. The smoke from the smelters, however, spreads a gaseous mantle over the valley which opposes the radiation of the heat absorbed by the earth during the day, thus rendering possible the utilization of a considerable area for raising a large variety of crops which formerly could not be grown at all.

The Calumet & Arizona smelter, situated near the Copper Queen, on the outskirts of Douglas, has also taken in hand the problem of recovery of the fine dust which fails to settle in the dust-chambers. It must be said that at neither smelter has the provision for arrest of dust been as elaborate as that at the majority of large smelters, and the temperature of the gases in the stack is relatively high, owing to the short travel from the furnaces and to the small dilution with air. The Calumet & Arizona is just installing an experimental apparatus for dust recovery, the elements of which are shown in the accompanying sketch (Fig. 35) in vertical section. The gases are led to the apparatus through the flue D, which connects eccentrically with the settling cylinder R. Within this cylinder, which is 12 ft. diam., is another cylinder set concentrically, expanding from a diameter of 6 ft. at the top to 8 at the bottom. This produces an annular chamber J J. When the gases enter

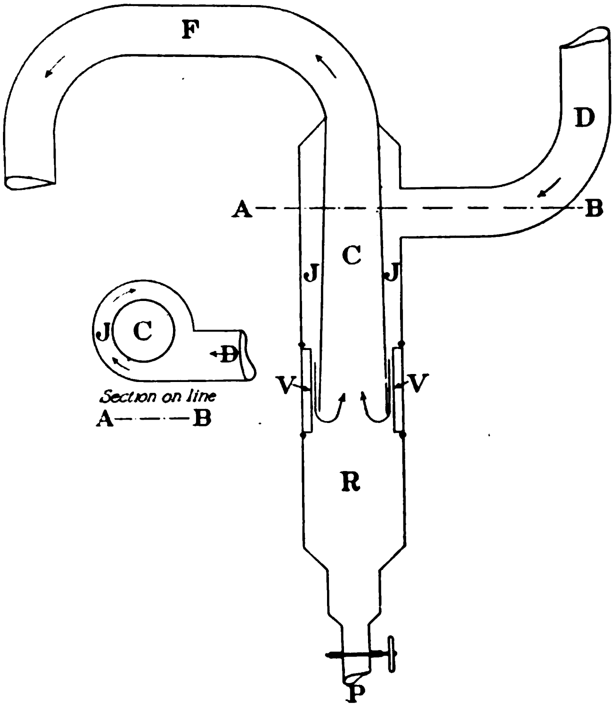

FIG. 35. DUST EXTRACTION AT CALUMET & ARIZONA SMELTER.

this on one side they are given a rotary motion as they sweep around the inner cylinder R. On passing below the open mouth of the inner cylinder C they come into an expanded area, and thus their velocity will be checked. The reduction of velocity is aided by the opposition of a series of vanes, V V, projecting 1 ft. inward from the shell of the outer cylinder R. The dust that falls will be drawn into bins through the valve P, while the gases will issue through C to the flue F. A similar dust extractor was applied to the gases issuing from an iron blast-furnace in the East, and is said to have given good results. The new appliance will soon be in operation at Douglas, under the direction of James Wood, the smelter superintendent. The plans were drawn by A. J. McDermott, assistant superintendent, and James A. Potter, civil and mechanical engineer to the company.

SHASTA COUNTY SMELTER-FUME PROBLEMS

(March 8, 1913.)

By J. NELSON NEVIUS

*The object of this trip was to obtain at first hand for the Chamber of Mines and Oil information on both sides of the controversy between the smelting companies and the agricultural interests in Shasta county, California, and the following report embodies the information given me by the respective parties. I was most corteously received, both at the smelter and by T. W. H. Shanahan, attorney for the Shasta County Farmers' Protective Association, and my questions were answered freely by both parties.

When operating under normal conditions, the Mammoth Copper Mining Co. operates three blast-furnaces, each having a capacity of approximately 400 tons of charge, or a total of between 1100 and 1200 tons of charge. The ore is massive iron sulphide, containing some copper sulphide, and ranging from 3 to 5% in copper, with some gold and silver. The object of the bag-house system is to eliminate all dust or solid matter from the fume, and also to eliminate entirely the sulphuric acid gas (SO_3) and to reduce the volume of sulphurous acid gas (SO_2) to less than three-quarters of one per cent, in order to comply with the conditions imposed by the court.

By way of explanation it should be said that SO_3 unites readily with atmospheric moisture and forms sulphuric acid. The SO_2, or sulphurous acid, is a pungent odor familiar to all in the burning of a sulphur match, and is less active than the SO_3, though capable of damaging vegetation. The solid matter, or dust, is a whitish powder, weighing about 13 lb. to the cubic foot, and has approximately the following chemical composition:

	Per cent.		Per cent.
Copper	1.04	Arsenic	4.3
Insoluble	7.8	Zinc oxide	4.8
Iron	6.2	Zinc sulphate	47.2
Calcium oxide	1.8	Gold, oz	0.03
Lead	7.0	Silver, oz	4.08

The total amount of metallic zinc ranges from 20% upward. In this dust the arsenic is the chief objectionable element, but it is also claimed that, by settling on foliage, the dust mechanically retards vegetable growth.

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*From a report made to the Board of Directors of the Los Angeles Chamber of Mines and Oil.

Formerly the fume containing the acids and this dust escaped into the atmosphere and caused damage to vegetation. Under the bag-house system, all the fume from both the blast-furnaces and converters is taken at the base of the former smokestack in four round pipes, 8 ft. in diameter and about 100 ft. long. These convey the fume to a square flue 15 ft. in diameter and about 120 ft. long. The fume is thence conducted to two fans, built by the American Blower Co., each being 11½ ft. in diameter and driven at 300 r.p.m., requiring 400 hp. each. The form of these pipes is merely a matter of convenience and economy, and would be subject to variations at any other plant, as the topography might require. The fans form the suction to draw the fume from the smelter and force it into the 'fan-discharge chamber,' whence it is led into a series of round cooling pipes, 4 ft. in diameter, of which there are 45, arranged in 9 series of 5 pipes each, contained on a rack 180 ft. long. The gas thus travels through 900 ft. of cooling pipes at a rate of 2700 ft. per minute. These pipes discharge the gas into a 'distributing chamber' 15 ft. square and 220 ft. long. The gas is still too hot to go to the bags, so cold air is blown into the top of the 'distributing chamber' through twelve 24-in. pipes leading from a larger supply pipe above, and supplied by a special fan. The temperature of the gas on reaching the distributing chamber is about 130°C., and sufficient cold air is blown in to reduce the temperature to 93°C. The cold air has the further effect of diluting the gas. At the fans the temperature of the gas is 280°C., consequently the loss of temperature in the cooling pipes is 150° Centigrade.

From the distributing chamber the gas is led to the bag-house through 5 apertures, each of which is under independent control, and leads to one of the 5 bays, or compartments, of the bag-house. The bag-house is 220 ft. long and is divided into 5 bays; there are 4 divisions in each bay, each with a hopper for the removal of the dust. The bags are specially woven of pure wool, 18 inches in diameter and about 34½ ft. long. These are set 21 in. apart, centre to centre, and there are 5 rows of bags to each hopper, making a total of 2960 bags. These bags are suspended vertically from a series of racks, the lower end being fastened in the floor, through which the fume enters, and the closed top tied to a short arm which is a part of an apparatus designed by J. E. Egleston, engineer for the smelter, and operated by electric motors, by which a vertical shaking motion can be given the bags. At fixed intervals the fume is cut off from the bays, the motor turned on, and the bags shaken vertically, causing the dust to fall into the hoppers below. Originally the dust was removed from the hoppers dry, but the impalpable powder was so distressing to the workmen that a wooden trough was built below the hopper, as shown in the accompanying sketch, so that the dust now falls in water and is drawn off in tanks and settles. It is interesting to note that the dust already accumulated contains sufficient lead, zinc, gold, and silver to more than pay for the cost of the bag-house plant, but that as yet no method has been found by which these metals can be economically recovered.

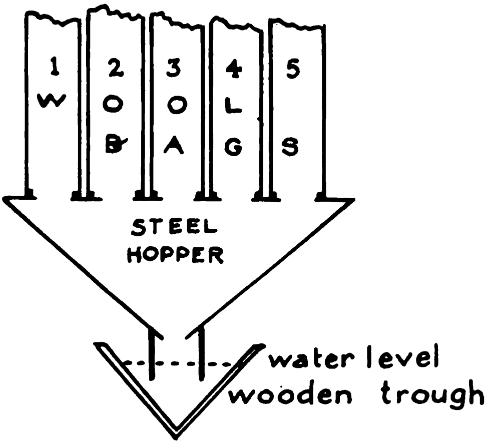

FIG. 36. HOPPER UNDER BAGS.

The fume enters the bags through their lower ends, under pressure from the fans, and filtering through their sides in the closed bag-house, escapes at the roof through 5 large square towers, which distribute the gas to the atmosphere in a much less concentrated form than if it escaped from one flue. The bag-house is open along the sides of the floor and roof to admit free air to dilute the SO_2. The action of the apparatus is as follows: The ore from this mine contains considerable zinc sulphide, which, in the blast-furnace, is changed to zinc oxide. The zinc oxide has a strong affinity for SO_3, and unites with it to form zinc sulphate. In the case of this particular smelter, there is sufficient zinc to completely neutralize the SO_3, but to be certain that none of this gas escapes, hydrated lime is introduced at the fans, at the rate of about 30 lb. per hour, which neutralizes any SO_3 escaping the action of the zinc oxide.

At the distributing chamber a minature bag about two feet long is operated, from which samples are taken every two hours for purposes of testing for the presence of SO_3. The SO_2 is neutralized simply by the addition of air to the fume, and reduced below the requirements of the court, which, as already stated, is 0.75%. I

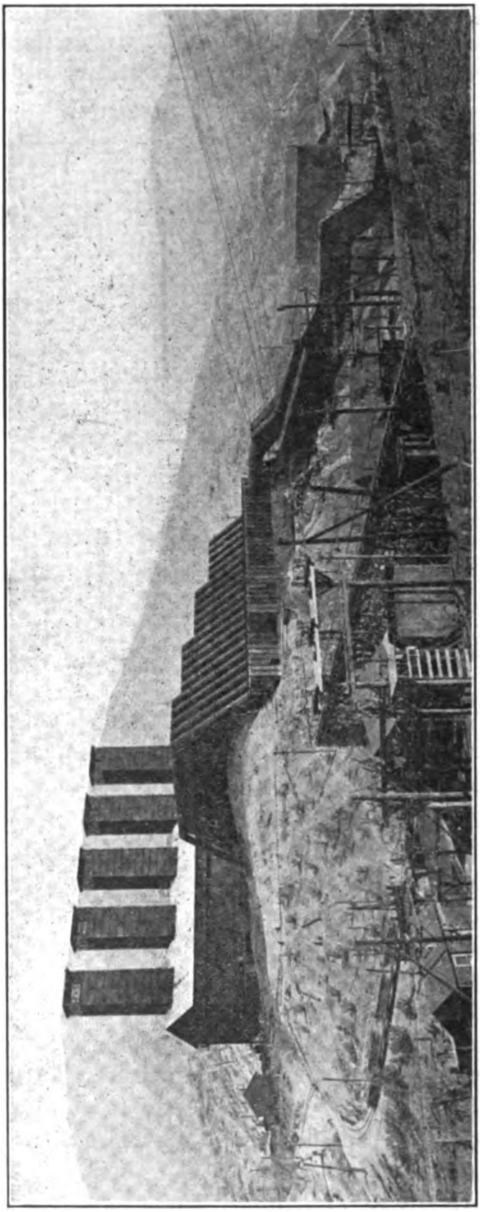

Fig. 37. Bag-house and Cooling Pipes at Kennett.

was informed that the escaping gas varies from 0.4 to 0.6% by volume. The solid matter, or dust, is filtered out by the bags and entirely removed. It is the custom of the company to keep very full and complete records of the results obtained by the entire apparatus, and it claims that the result is as follows: (1) That the escaping gas contains from 0.4 to 0.6% SO_2 by volume; (2) that it contains no SO_3; (3) that it contains no dust.

It is interesting to note that one day's operation showed that the volume of gas passing the fans in 24 hours operation was 333,000 cu. ft. The volume of cold air introduced was 23,000 cu. ft. This was in winter, when the outside air was cold. In summer the volume of outside air introduced may run up as high as 125,000 cu. ft., the reduction in the volume of gas being due to cooling. The volume of gas filtered per minute per square foot of bag surface was 0.631 cu. ft. The amount of dust accumulated in 24 hours varies from 12 to 15 tons, and averages about 13 tons. The gas escaping into the atmosphere carries a little carbonic acid, and an excess of nitrogen, in addition to the sulphurous acid. Tests for the presence of sulphuric acid gas (SO_3) are made every two hours at the miniature bag attached to the distributing chamber. So long as these tests show a deficiency of acidity, it is impossible that SO_3 is escaping. Furthermore, tests have shown that the texture of the bags will show depreciation within a few minutes if any SO_3 comes in contact with them; but the above tests are made and the record kept for the company's protection in case claims may be made of anyone having sampled the outside air and detected traces of SO_3. The bags cost about $6 each, and their life is from 18 months to 2 years.

On December 14 I interviewed T. W. H. Shanahan, attorney for the Shasta County Protective Association, who pointed out that great damage can be done by smelter fumes, without its being apparent at any particular place or time; that is, it may retard the growth of orchards without killing the trees; also, it may retard the development of the size of the fruit, thereby reducing its value; also that damage done is cumulative—a small amount of damage done day after day finally assumes serious proportions; also that the small, almost intangible damage, is very difficult to demonstrate legally. He explained that on being retained by the farmers and looking into the subject in general, he found that the United States Smelting Co., operating at Midvale, Utah, had been closed down by the courts on account of damage done by smelter fumes, and that the company, after making certain agricultural experiments, had petitioned the court to be allowed to operate, provided it eliminated all SO_3 and all dust from the fume, and reduced the SO_2 to less than 0.75% by volume, at the same time so diffusing the gas that it did not damage vegetation. The petition was granted by the court. The United States Smelting, Refining & Mining Co. is the interest controlling the Mammoth smelter, and Mr. Shanahan secured an order from the court that Shasta county smelters must either equip themselves to fulfill the following conditions, or shut down, being given a certain length of time in which to obey its mandate, which provided as follows:

1. That all free sulphuric acid shall be removed from the fume, by the application of zinc oxide or some other neutralizing agent, before reaching the bag-house.
2. That all solid matter shall be removed by the bags in the bag-house.
3. That after the neutralization of the free sulphuric acid, and the removal of all solid matter, that not to exceed 0.75% of sulphur dioxide (invisible gas) in volume, shall be permitted to escape into the atmosphere.
4. That only the amount of sulphur dioxide be permitted to escape into the atmosphere, as not to damage complainants; or complainants' crops, trees, or vegetation; or do any injury to their live-stock; and so as not to cause any discomforts or inconveniences to the complainants or any members of the families of any of the complainants.

Thus it is apparent that unless the Shasta county smelters operate in conformity with the court's decree, they are immediately in contempt of court, and it is not necessary for the farmers to prove any actual damage done, it being only necessary for them to show that the court's order is not being obeyed in order to close any smelter. The Mammoth Copper Mining Co. protected itself by constructing the bag-house at a cost of about $250,000, and the company states that it is now operating within the law.

The Balaklala Consolidated Copper Co., owning the smelter at Coram, Shasta county, installed the Cottrell process at a cost of about $126,000 for equipment, and some $25,000 additional for experimenting. After it had been in operation for a while, the farmers still complained, though the company maintained that it was complying substantially with the conditions imposed by the court. Thereupon, Mr. Shanahan engaged the services of Gould & Burt, consulting chemists, to test the fume. They made 44 tests, and found that in 8 out of the 44 the percentage of SO_2 was in excess of 0.75% allowed by the court; that free SO_3 was escaping, as was shown in several of the tests, and also that dust was escaping. The tests showed that the maximum amount of SO_2 found in any one test was 0.921%, and their summary showed that the efficiency of the Cottrell process was as follows: Maximum, 96% plus; average, 74%; minimum, 20%; and they concluded that the process was worthless for the purpose intended. Mr. Shanahan stated that the company did not dispute the accuracy of these tests; consequently that the company was disobeying the injunction on three counts, and was, therefore, in contempt of court. The result was that the company compromised by paying the Farmers' Protective Association $4000, which was about the costs of chemists' and legal fees, and closed the smelter. The Balaklala was smelting 700 tons of ore per day.

The Bully Hill smelter is situated 17 miles east of the other two, and is surrounded chiefly by government land and forest reserve. The capacity of the plant is in the neighborhood of 400 tons per day. Mr. Shanahan stated that when the Government saw the success of his efforts, it enjoined the Bully Hill smelter and gave that company a short time in which to equip a fume destroyer, or close down. This was in 1910, at which time the price of copper was low, and apparently the company closed down rather than go to the expense of questionable experiments. At the present time the Bully Hill smelter is closed, and no one seems to know the plans of the company.

The Balaklala smelter has been closed and partly dismantled, and the Mammoth smelter is the only one operating. Mr. Shanahan stated that a marked improvement in conditions is apparent since the Mammoth smelter installed the bag-house, and since the other smelters closed. He was not prepared to state whether, in his opinion, the bag-house was entirely successful, or whether there might not be further disputes with the farmers; but he expressed the belief that it was by far the best scheme yet devised, and that he sincerely hoped it would be effective. He stated also that, while most of the farmers appeared to be satisfied with present conditions, some of them were still complaining, but that during the past season the French prune crop in the vicinity of Anderson was the best it has been in years. At the annual meeting of the Shasta County Farmers' Protective Association, held in Anderson December 13, the new officers elected were from the dissatisfied element, and that at least some of the farmers are still dissatisfied appears to be shown in the newspaper accounts of that meeting as appended to this report. Mr. Shanahan protested at the smelter being permitted to experiment with new processes, because of the fact that damage was being done meanwhile, but he was not prepared to state how a successful process could be developed other than by actual experiment. He freely acknowledged that this was the only way the question could be determined, and his only suggestion was that such experiments be conducted on the desert.

There appears to be two fundamental reasons why the smelter situation in Shasta county imposes peculiar difficulties: (1) the ore is a massive sulphide, in other words the sulphur content is unusually high, which produces an unusual amount of sulphur gases; (2) the two principal smelters were within three miles of each other, and in a narrow valley in which the winds are almost invariably either north or south, consequently the fume was more concentrated than if the region were subjected to varying winds. The natural conclusion seems to be that the smelters should not be congregated near an agricultural region for the reason that while one smelter may operate within the law and without injury to the agricultural interests, it does not follow that more than one smelter, even though they operate within the law, will not have a damaging effect. As a test of present conditions at Kennett, it is interesting to note the lawns, shrubs, and flower gardens surrounding the houses within half a mile of the smelter. I was informed that it has been possible to maintain those gardens only since the bag-house was built.

SMELTER FUME IN JAPAN

(Editorial, March 9, 1912.)

When facing an intricate problem it is always interesting, and often profitable, to learn how your neighbors have met the same difficulties, and in the treatment of smelter-fume, help from every source is desirable. In densely settled countries with well developed industries, such as the Eastern part of the United States, and Europe, it is possible, by finding a market for sulphuric acid, to dispose of the sulphur in the fume economically and satisfactorily. In Utah, Montana, and California this is only feasible on a limited scale. There is no present or imminent market for the sulphuric acid that might be made at Anaconda alone, and, if it were economically possible to ship acid from Montana to the Eastern states, this would merely result in disorganization of established industries. It is unnecessary to review here the various attempts made in the Western states to meet the difficulties of an admittedly serious situation. These have been chronicled from time to time in our news pages, and while there is distinct promise of final relief, it is fair to say that at present no entirely satisfactory means of handling smelter fume in the states mentioned has been devised. The final solution must be a process which prevents escape of injurious material from the stack, and which is applicable, if not with profit, at least without serious loss to the smelting company.

Pending perfection of existing processes or invention of better ones, and owing to our imperfect methods of legal procedure, it has been customary to close the offending works by injunction. In determining whether such injunction shall be made permanent, the relative measure of damages is not, unfortunately, the principal factor. It should be clear that closing a smelter cuts off the revenue from the only interest having sufficient at stake to warrant the heavy expenditures necessary to perfect a satisfactory technological process. This is aside from the incidental damage to the community in destroying what is often the chief source of income of the particular community. This bears certain analogies to the universally condemned 'cutting off one's nose to spite his face,' and also the killing of the 'goose that lays the golden eggs.' If the smelters are not to perfect the technology in this case, who will? And how are they to do it if their revenue is impaired? In fairness, it must be stated that in several cases the courts have refused to issue or enforce injunctions for a period during which experimental work was being conducted. This, we believe, is a sound policy and may well be extended.

In Japan they do even better. Conditions in that country are not unlike those in our Western states in main essentials. In particular there is the same condition of undeveloped industry that prevents the direct utilization, in the form of sulphuric acid, of the sulphur in smelter fume. On the other hand, agriculture is highly developed and every acre of cultivable land is used. Damage to crops is a serious matter where nearly fifty million people must be sustained in an area about equal to that of California and with but 12 per cent of the land suitable for cultivation. Establishment of smelters in such a country, necessary as it is, brings up smelter-fume problems in their most acute form. The smelting industry of Japan is still in an early stage so far as industrial organization is concerned. There is no great 'smelting trust' and custom smelting is not general. Practically each large mine has its own smelter, or, rather, each large mining partnership has at least one smelter. The three leading copper mines of the country each supply a separate plant. The Kosaka mine of the Fujita company is situated in the mountainous country of the far northern province where little damage is possible. The Osaruzawa mine of the Mitsu-Bishi company is in the same situation. The Ashio mine of the Furukawa company is in a region where the valleys are devoted to rice farming, and great care is therefore necessary to neutralize both the smelter fume and the waste water from concentrating mills. For the first the Asano process of treatment with lime-water has been adopted, and while not entirely successful, it accomplishes such a degree of neutralization as permits the smelter to continue in operation. In the treatment of the waste water from the mills neutralization with lime-water has been combined with sand filtration so effectively as to produce an actual profit from copper otherwise lost.

At the Besshi mine of the Sumitomo company, perhaps, the experience has been most informing of all. This mine is now producing about 600 tons per day of ore containing 30 to 33 per cent of sulphur. The ore is smelted in blast-furnaces, with pot-roasting of the fine material, in an excellent modern plant built in 1904 on Shisaka island, to which the works were removed from the mainland to prevent further damage to agriculture from fume. The island is in the Inland sea ten miles from the nearest important land. Despite this distance, complaints of damage continued after the erection of the new works. Experiments have accordingly been undertaken to determine methods of preventing damage. These experiments have been made by the engineers of the company under the supervision of a commission of technical men, appointed by the Government. This same commission, it may be noted, has full charge of all matters relating to fume damage, and complaints are made to, and adjusted by, it rather than the courts. In other words, technical matters are judged by technical men, rather than left to decision by judge and jury. The commission includes professors of mining, forestry, and agriculture in the Imperial Universities, and the particular committee handling the experimental work includes Mr. Wataru Watanabe, the director of the Engineering College of Tokyo Imperial University, chairman; Mr. Jisaburo Yokobori, professor of metallurgy at the Kyoto Imperial University; and Mr. Yoshitaro Watanabe, professor of metallurgy in the Kyushu Imperial University. Agricultural experiment stations are maintained on the main islands on three sides of the plant and careful observations are being made of the effect of each change as made. One interesting test was as to the effect of dilution resulting from the height of the chimney, which rises 212 feet above the plant. It was found that the percentage of SO_2 in the air on the mainland was not different when the smoke was released at the base of the stack from what it was when released from the top. In other words, any reasonable height of stack was insignificant in comparison with the horizontal distance within which damage was claimed. Attempts were made to neutralize the SO_2 by mixing coal gas with the furnace gases, and the amount was brought down to one per cent. Interesting experiments are now under way, in the direction of clearing the fume by sucking through a 2-inch layer of filtering media of various sorts. Many of the lines of research undertaken proved unprofitable, others are promising, none have as yet proved entirely successful and at the same time economical. Under conditions obtaining in the United States probably an injunction would be issued and the works closed. Instead, they are running, and good crops are being regularly harvested. In the agricultural experiment stations conducted as a part of the general investigation it was found that damage was done by the fume in only a certain limited portion of the growing season. In the case of rice, the main crop, the injury occurs almost altogether within 20 days of the year, though danger extends through 40 days. Accordingly the crops are watched by experts, and when the danger season arrives, notice is sent to the smelter. The output then is reduced by one-third, and the plant continues to run at the lower capacity for 40 days, during the 20-day period a further reduction of one-third is made, or until notice from the experts indicates that the rice is past danger of damage. The remainder of the year the plant runs at full capacity. This appeals to us as a simple and effective method of meeting present difficulties pending perfection of process such as will permit smelters to run profitably at full capacity and without question of damage to agriculture. We commend it to the attention of state and national authorities in the United States.

THE ELECTRICAL PRECIPITATION OF SUSPENDED PARTICLES

(August 26 and September 2, 1911.)

By F. G. COTTRELL

[*]The removal of suspended particles from gases by the aid of electric discharges is by no means a new idea. As early as 1824 it was suggested by Hohlfeld[1] as a means of suppressing ordinary smoke, and again a quarter of a century later by Guitard.[2] These suggestions, which do not seem to have stimulated any practical study of the question, were soon entirely forgotten and only brought to light again by Sir Oliver Lodge[3] many years after he himself

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[*] Abstract from *Jour. Ind. Eng. Chem.*, August 1911 and revised in part March 1914.

[1] 'Das Niederschlagen des Rauchs durch Electricität,' Hohlfeld, *Kastner Archiv. Natural.*, 2, 205-6 (1824).

[2] C. F. Guitard, *Mechanics Magazine*, Nov. 1850.

[3] Historical note on 'Dust Electrification and Heat,' O. J. Lodge, *Nature*, 71, 582 (1905).

had independently re-discovered the same phenomena and brought them to public attention[4] in a lecture before the Liverpool Section of the Society of Chemical Industry, November 3, 1886. The first recorded attempt to apply these principles commercially appears to have been made at the Dee Bank lead works. The general principle of electrical precipitation of suspended matter was at this time patented by Alfred O. Walker, of the above firm, in several countries,[5] but these patents have long since expired. The apparatus was erected in 1885 by the works manager, W. M. Hutchings, with the coöperation of Mr. Lodge, and briefly described by the former just before its completion as consisting of a system of metallic points situated in the flue from the lead furnaces and excited from two Wimshurst influence machines with glass plates 5 ft. in diameter. each machine being driven by a 1-hp. steam-engine. Almost simultaneously with Walker and apparently without knowledge of his and Lodge's work, Karl Moeller, of the firm of K. and Th. Moeller of Brackwede, Germany, secured a patent (Ger. Pat. 31911, Kl. 12, Oct. 2, 1884) on the electrical precipitation, or at least agglomeration of particles in connection with the filtration of air through cloth screens. The patent specification itself appears, however, to be the only published record of this work. The idea was, it is understood, suggested by an article by Robert Narwald (*Wied. Ann.* 5, 460-499, 1878) dealing with the disturbing influence on electrometer measurements due to dust in the air.

The apparatus undoubtedly did not in practice fulfill expectations, as nothing further is found of it in the literature. The most apparent weakness of the project lay, perhaps, in the reliance on the Wimshurst machine, which had then just been brought out and from which a great deal more was anticipated[6] than has been justified by experience, at least as far as commercial applications are concerned.

Since that time an occasional patent[7] or article[8] has served to

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[4] 'The Electrical Deposition of Dust and Smoke with Special Reference to the Collection of Metallic Fumes and to a Possible Purification of the Atmosphere,' *Jour. Soc. Chem. Ind.*, 5, 572-6 (1886), with appended bibliography.

[5] Great Britain, 11,120, Aug. 9, 1884; Belgium, 68,927, May 19, 1885; Spain, 7211, July 10, 1885; Germany, 32,861, Feb. 27, 1885; Italy, 18,007, Mar. 31, 1885; United States, 342,548, May 25, 1886.

[6] A. O. Walker, *Engineering* (Lond.), 39, 627-8 (1885). G. Tissandier, *London Electrician*, 17, 33 (1886).

[7] Lorrain, British Pat., 6495 and 5667 (1886); Thwait, U. S. Pat., 617,618, Jan. 10 (1899); Hardie, U. S. Pat., 768,450, Aug. 23 (1904).

[8] J. Wright, *Elect. Rev. Lond.*, 47, 811, Nov. 23 (1900); see also *Jour. Roy. Sanitary Institute*, 27, 42.

keep the subject in the public eye, and in 1903 Lodge himself took out a patent[9] covering the use of the then new mercury arc for rectifying high potential alternating currents for this purpose. Up to the present time none of these patents seem to have been carried into successful commercial operation on a large scale in the chemical or metallurgical industries. Some four years ago while studying various methods for the removal of acid mists in the contact sulphuric acid process, I had occasion to repeat the early experiments of Lodge, and became convinced of the possibility of developing them commercially. The work here described may fairly be considered as simply the reduction to engineering practice as regards equipment and construction, of the fundamental processes long since laid open to us by the splendid pioneer work of Lodge, a feat vastly easier today than at the time of the original attempt.

The precipitation of suspended matter, whether in gases or liquids, may be accelerated by electricity in the form of either direct or alternating current, but the mode of action and the type of problem to which it is best applicable differ in certain important respects. Where an alternating electromotive force is applied to a suspension, the action consists for the most part in an agglomeration of the suspended particles into larger aggregates out in the body of the suspending medium and a consequently more rapid settling of these aggregates under the influence of gravity. Thus, if powerful Herzian waves are sent out into foggy air, the alternating fields set up in space cause an agglomeration of the particles of liquid into larger drops which then settle more rapidly. Considerable work aimed at the application of this phenomen to the dispelling of fog on land and sea has recently been done in France and England, but little as to definite results seems as yet to have been published. The field appears, however, one of considerable promise. Another application of alternating current along these lines is found in a process now in use in California oilfields for separating emulsified water from crude oil. This process grew out of the work here described and was developed some two years ago by Buckner Speed and myself to meet certain peculiar conditions existing in these fields.[10] Alternating current may thus be used to advantage where the masses of gas or liquid to be treated are fairly quiescent and a simple agglomeration of the suspended particles into larger aggregates is sufficient to effect separation by gravity or otherwise.

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[9] Brit. Pat., 24,305 (1903); U. S. Pat., 803,180, Oct. 31 (1905).

[10] 'Dehydration of Crude Petroleum, a New Electrical Process,' Arthur T. Beazley, *The Oil Age*, 3, 2-4, Apr. 21, 1911.

In the case of the large volumes of rapidly moving gases in smelter flues, the agglomerating and settling process is however, too slow even when the flues are expanded into as large chambers as are commercially feasible. It is in such cases that direct-current methods have been particularly important. If a needle point connected to one side of a high-potential direct-current line to be brought opposite a flat plate connected to the other side of the line, the air space between becomes highly charged with electricity of the same sign as the needle point; irrespective of whether this is positive or negative. Any insulated body brought into this space instantly receives a charge of the same sign. If this body is free to move, as in the case of a floating particle, it will be attracted to the plate opposite charge and will move at a rate proportional to its charge and the potential gradient between the point and plate. Even if there are no visible suspended particles, the gas molecules themselves undergo this same process, as is evidenced by a strong wind from the point to the plate even in perfectly transparent gases. The familiar experiment of blowing out a candle flame by presenting it to such a changed point, is simply another illustration of the same phenomenon.

As above indicated, the first step toward practicability was of necessity a commercially feasible source of high-tension direct current. The obstacles to building ordinary direct-current generators for high voltages lie chiefly in difficulties of insulation, and if this is avoided as to individual machines by working a large number of series, the multiplication of adjustments and moving parts intrudes itself. On the other hand, high-potential alternating-current technique has in late years been worked out most thoroughly, and commercial apparatus up to 100,000 volts is available in the market. The mercury arc rectifier has been made practical for series arc lighting service up to some 5000 volts direct current, but although higher voltage units were at one time attempted by the electrical companies, they were later withdrawn from the market as unsuited to practical operating conditions. For high voltages they seem, like the static machines, to work well under the careful management and light duty of the laboratory, but fail in practice. This is particularly so in the application to metallurgical and chemical work where the electrodes in the flues have to be placed close together and worked near the potential of disruptive discharge, the occasional occurrence of which latter places much more severe requirements upon generating apparatus than in the case of the series arc lighting, to which the rectifier seems particularly well adapted. This restriction does not necessarily apply to the conditions under which fog and smoke would be treated in the open air. It is to these latter cases that Lodge's efforts in late years seem to have been the more particularly directed, and there the mercury rectifier in connection with high-potential transformers may find a useful field.

The procedure actually followed in the work described below consisted in transforming the alternating current from an ordinary lighting or power circuit up to some 20,000 or 30,000 volts and then commutating this high-potential current into an intermittent direct current by means of a special rotary contact-maker driven by a synchronous motor. This direct current is applied to a system of electrodes in the flue carrying the gases to be treated. The electrodes are of two types corresponding to the plate and point in the experiment above cited. The construction of electrodes corresponding to the plate presents no special problem, as any smooth conducting surface will answer the purpose. With the pointed or discharge electrodes it is quite otherwise, and the working out of practical forms for these has proved the key to the first commercially successful installations. In laboratory experiments, when the discharge from a single point or a few such is being studied, fine sewing needles or even wire bristles answer very well, but when it is attempted to greatly multiply such discharge points in order to uniformly treat a large mass of rapidly moving gas at moderate temperatures, great difficulty is met in obtaining a powerful and, at the same time, effective distribution of current.

It may be of interest to note that the clue to the solution of this difficulty came from an almost accidental observation. Working one evening in the twilight when the efficiency points could be roughly judged by the pale luminous discharge from them, it was noticed that under the particular conditions employed at the time, this glow only became appreciable when the points had approached the plates almost to within the distance for disruptive discharge, while at the same time a piece of cotton-covered magnet wire which carried the current from the transformer and commutator to the discharge electrodes, although widely separated from any conductor of opposite polarity, showed a beautiful uniform purple glow along its whole length. The explanation lay in the fact that every loose fiber of the cotton insulation, although a relatively poor conductor compared to a metallic point, was still sufficiently conductive from its natural hygroscopic moisture to act as a discharge point for this high-potential current, and its fineness and sharpness, of course, far exceeded that of the sharpest needle or thinnest metallic wire. Acting on this suggestion, it was found that a piece of this cotton-covered wire when used as a discharge electrode at ordinary temperature proved far more effective in precipitating the sulphuric acid mists, which were then the object of study, than any system of metallic points which it had been possible to construct. Perhaps the greatest advantage thus gained lay in the less accurate spacing demanded between the electrodes of opposite polarity in order to secure a reasonably uniform discharge. Much of the importance of this discovery at the time lay in the limited potentials of a few thousand volts then available in laboratory work.

In practice a more durable material than cotton was demanded for the hot acid gases to be treated, and this has been found in asbestos or mica, the fine filaments of the one and the scales of the other supplying the discharge points or edges of the excessive fineness required. These materials are twisted up with wires or otherwise fastened to suitable metallic supports to form the discharge electrodes in such wise that the current has to pass only a short distance by surface leakage over them, the slight deposit of moisture or acid fume naturally settling on them serving to effect the conduction. With the further development of the electrical technique to provide the far higher voltages now being used in commercial operation, the choice and design of electrodes has become much more flexible, including simple metallic wires, sharp metallic strips, and the like. In fact, the very phenomena of so-called corona loss or direct leakage from the wire into the air on high-tension transmission lines which is today the chief stumbling-block of the power companies in going very far from 100,000 volts on these lines, becomes exactly what is desired in the processes of precipitation, and with the voltages now used a bare metallic wire of moderate size or the relatively thin edge of a metallic sheet may be made to furnish an excellent discharge. The construction and arrangement of the electrodes, as also of the chamber containing them, naturally vary widely with the conditions to be fulfilled under the varied applications to which the process may be put. Some of the more general features of this work are described in the patents[11] already issued in this and foreign countries, while further details and modifications are covered in other patents not yet issued.

Following development of satisfactory laboratory apparatus the attempt was made to duplicate the experiments on a scale some two-hundred-fold larger. This was carried out during the summer of 1906 at the Hercules work of the E. I. du Pont de Nemours Powder Co. at Pinole on San Francisco Bay, where the contact gases from one of their Mannheim contact sulphuric-acid units were available.

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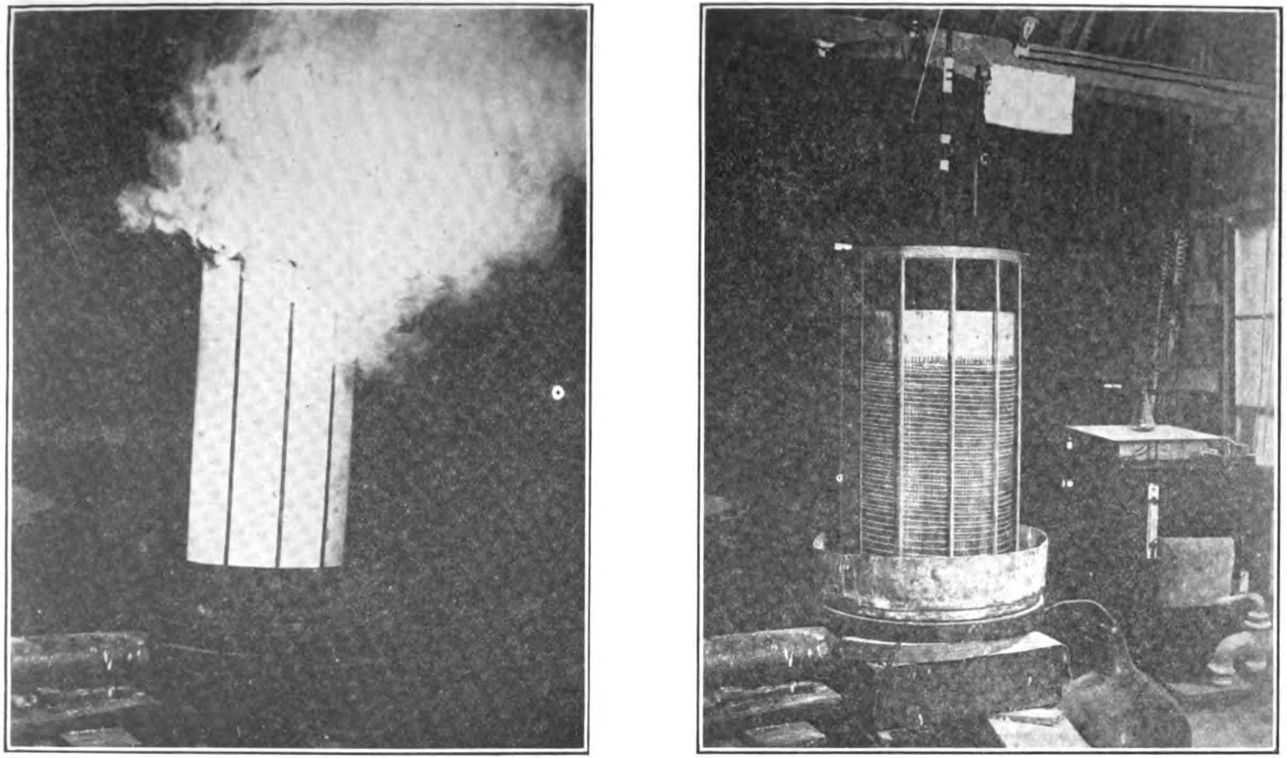
[11] U. S. Pat., 866,843, 895,729, 945,917, 1,016,476, 1,035,422, and 1,067,974.

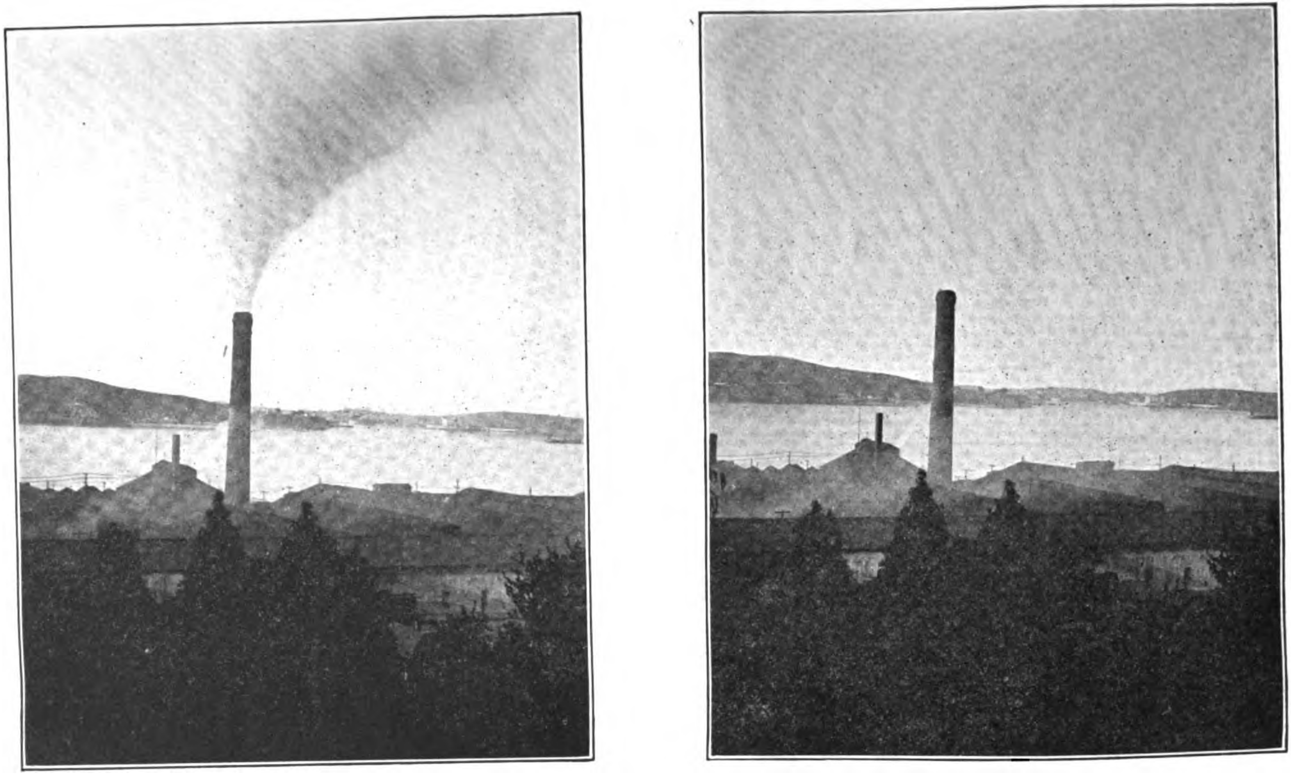


a, Acid fumes at powder works without current on apparatus. b, Acid fumes at powder works with current on apparatus, one minute after.

FIG. 38. FUME PRECIPITATION AT POWDER WORKS.

These gases at the point selected contained about 4% by volume of dry gaseous sulphur trioxide, and in order to convert this into sulphuric-acid mist they were brought into contact with water. Under these conditions very little of the sulphur trioxide is absorbed by the liquid water, but the latter evaporating into the gas combines with the sulphur trioxide to form the far less volatile sulphuric-acid which immediately separates as a dense white cloud of suspended particles so fine as to represent one of the most difficult of all materials to remove by filtration. Fig. 38*a* and Fig. 38*b* are from photographs taken a minute apart with the same current of fume-laden gases passing into the precipitation-chamber, but with the electric current respectively off and on. The apparatus consisted of two concentric cylinders which formed a discharge electrode, while intermediate in diameter between these two and resting on the lead pan-bottom of the apparatus was a third cylinder, also of iron-wire screen, but without asbestos winding, which together with the outer leaded glass cylinder constituted the collecting electrodes. The precipitated acid drained off from this precipitation chamber into the carboy on the right. The space between successive cylinders of opposite polarity was as nearly as possible an inch and a quarter. Current was supplied from three 1-kw., 110 to 2200-v. transformers connected in series on their 2200-v. side to give 6600 v. The fume was delivered to the bottom of the precipitation-chamber at its centre and had to pass through the three-wire cylinders and up between the outer of these and the glass container, thus being subjected to three electric fields in succession. In this apparatus the power-consumption was about one-fifth of a kilowatt, and between 100 and 200 cu. ft. of gas per minute could readily be treated. This apparatus was merely for experimental and demonstration purposes, but a still larger unit, built for permanent operation and to handle all of the gas from a Mannheim unit, has since been erected at this plant and put into steady commercial operation.

These experiments at Pinole attracted the attention of the Selby Smelting & Lead Co., whose smelter situated at Vallejo Junction, a few miles farther up the bay, was at that time the object of injunction proceedings brought by the farmers of the surrounding country. At the time the suits originated, three separate stacks at the smelter contributed to the alleged nuisance. The first, and admittedly the most serious offender, handled the gases from the lead blast-furnaces and discharged several tons of lead fume daily into the air. Shortly before the commencement of experimental work at the plant, this had been obviated by the erection of the bag-house. After correcting this evil, there still remained, however, a



a. Gases from blast-furnaces and roasters, without current on apparatus.

b. Gases from blast-furnaces and roasters, with current on apparatus.

FIG. 39. FUME PRECIPITATION AT SELBY WORKS.

stack discharging the gases from the roasters, which, besides the invisible sulphur dioxide, furnished dense white clouds consisting chiefly of sulphuric acid, arsenic, and lead salts, and to which the bag-house was inapplicable on account of the corrosive action of these gases upon the bags. Lastly, there was the stack of the refinery carrying the mists escaping from the pots of boiling sulphuric acid used to dissolve the silver out of the gold and silver alloy coming from the cupels.

The blast-furnace and the roaster stacks each carried something over 50,000 cu. ft. of gas per minute, while the refinery stack represented scarcely a tenth of this volume. As a first step, operations were begun on this latter, and after several months' experimenting as to the best form of construction, a system of vertical lead plates 4 in. wide by 4 ft. in length and spaced about 4 in. apart, was developed. Several rows of such plates were assembled in a 4 by 4-ft. lead flue. Between each pair of plates hung a lead-covered iron rod carrying the asbestos or mica discharge material, the latter finally proving the more serviceable in this highly acid atmosphere. These rods or discharge electrodes were supported on a gridwork of buss bars extending over the heads of the plates and through apertures in the sides of the flue to insulators on the outside.

The electric current is taken from the power circuit of the plant at 460 volts, 60 cycles, and transformed up to 17,000 volts, thence through the synchronous contact-maker or rectifier to the electrode system. At first a glass-plate condenser was connected across the high-potential line in parallel with the electrode system in order to assist in maintaining the potential of the electrodes between the intervals of contact, but was found troublesome and unnecessary in practice, and in this and other installations is now omitted. The power-consumption is about 2 kw., including the driving current for the synchronous motor. The switchboard transformer and rectifier are in the engine-room of the plant and require no more attention than a feed-pump or a blower. The plant appears to have been the first commercially successful installation of electrical precipitation ever made, and had in 1911 been in successful daily operation for over three years at a cost for labor, attendance, and repairs of less than $20 per month. In fact, while the plant was making enough bluestone to utilize all the weak acid recovered, the saving on purchase of the latter paid for the entire cost of operating five times over.

In the seven years which have elapsed since this plant was equipped, the processes have undergone steady development, and incidentally have passed through the many vicissitudes common to innovations in the industrial world. One of the first extensions of the work was, naturally, a series of experimental precipitation chambers in the main roaster flue of the same works. The former was of lead construction similar to that of the first installation, but of some tenfold larger capacity, while the latter was of brick and iron construction and intended for the collection of dryer and less highly acid-bearing material. Fundamental changes in the furnace equipment and metallurgical practice at this plant obviated the need for electrical precipitation, and the permanent installation originally contemplated for those particular flues has never been carried out.

The next plant in order of size that was undertaken was at the Balaklala smelter at Coram, Shasta county, California. The vast body of low-grade copper ore reaching for many miles across this county and commonly known as the copper crescent has recently been described in detail by L. C. Graton[12] and characterized as the second largest copper deposit which can be considered as a single geological unit in the United States. The first commercially successful smelter in this region was erected at Keswick in 1896 by the

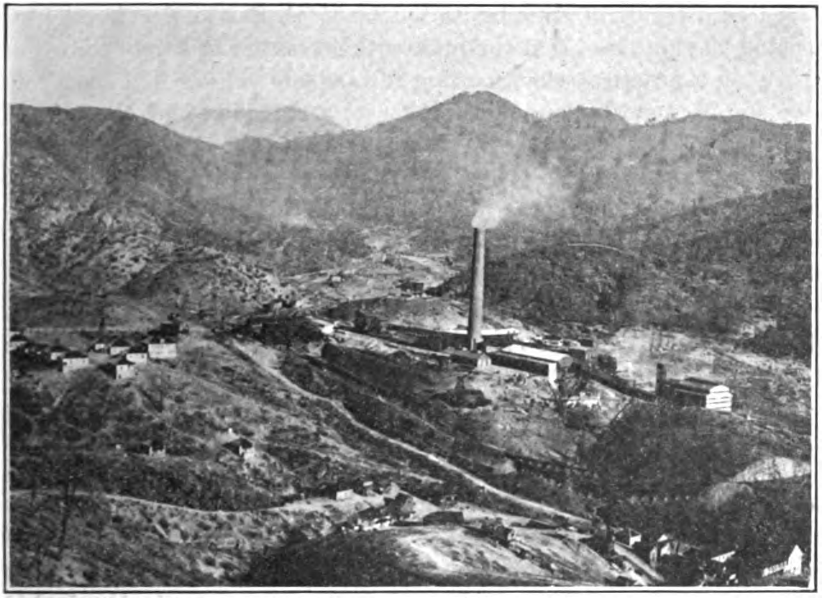


FIG. 40. THE BALAKLALA PLANT AT CORAM IN 1911.

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[12] U. S. Geol. Surv., Bull. 430b.

Mountain Copper Co., Ltd., under the management of Lewis T. Wright, and was of especial interest as one of the pioneers in pyrite smelting. Extensive heap roasting was also carried on at this plant, with the result of widespread deforestation of the surrounding country and the final closing down of the plant in 1905 through injunction proceedings instituted by the U. S. Forest Service. The company has since built a small smelter and acid phosphate works at Martinez on San Francisco bay and now ships its ore to this point, nearly 250 miles distant, for treatment, but even this latter plant has at various times come in for its share of fume litigation.

In 1901 the Bully Hill smelter at Delamar, with a capacity of 250 tons per day, was started. This was later purchased and is now owned by the General Electric Co., but since last July it has been closed as a result of complaints by the U. S. Forest Service, which insisted, at that time, that the plant either close or at least commence efforts on a practical scale looking toward controlling its fumes. In 1905 the Mammoth Copper Mining Co., a subsidiary of the United States S. R. & M. Co., blew in its present smelter at Kennett. This plant, when running full, has a capacity of some 1200 tons of ore per day. The Balaklala, which belongs to the First National Copper Co., is the most recent of the Shasta county smelters, having blown in its first furnace in 1908.

These smelters are all situated in the narrow, precipitous cañon of the upper Sacramento river and its tributary, the Pit. The region itself is too steep and rocky for agriculture, but was once heavily wooded, although now swept bare of vegetation for miles. As far as the cañon itself is concerned, probably all the damage possible has already been done, unless reforestation were undertaken. This latter even would probably be slow and difficult work, as, since the loss of vegetation, the steep hillsides have been washed bare of soil for miles around. At Redding, however, some 13 miles below Coram and 17 miles below Kennett, the cañon widens out into the fertile Sacramento valley, and from this point southward, for some 12 miles farther, lies the region from which for the past two years have come increasingly insistent complaints against the smelters. These culminated, in the spring of 1910, in agreements between the farmers and the smelters under which friendly suits were brought in the Federal courts and injunctions issued by stipulation requiring the smelters to remove the suspended matter from their exit gases and dilute the latter to such an extent that their sulphur dioxide content should not exceed 0.75% by volume as discharged from the stacks, with the further general and sweeping provision that they should do no damage.

To accomplish this the Mammoth smelter built a bag-house which has been in successful operation since July 1910. The gases discharged from the five stacks (each 21 ft. square) are, to all intents and purposes, free from suspended matter and consequently invisible. This represents a notable achievement, being the first time that the bag-house, so efficient in lead smelters, has been successfully applied to copper blast-furnace gases on the large scale. It is made possible in this instance through neutralization of the sulphuric acid in the gases by the zinc oxide carried over in the fume from the zinc content of the ore smelted. The company is also the owner of patents[13] on the introduction of finely divided metallic oxides into the gases for this purpose. In addition it was necessary to provide an extensive system of cooling pipes in front of the bag-house. There are 40 of these pipes, each 4 ft. in diameter and averaging about 200 ft. in length. They represent a large part of the cost of the plant. It was expected that they would be sufficient to cool all the gases of the plant to a safe temperature to protect the $30,000 worth of woolen bags with which the house is filled, but upon starting it was found that only a little over half the full capacity of the plant could thus be treated with safety, and operations were accordingly restricted to this. In the cold weather of winter a considerably larger tonnage can be handled with safety than during the hot summer. Subsequently, by rearranging the pipe connections so as to have eight groups in parallel, each consisting of five pipes in series, the company succeeded in greatly increasing their efficiency as coolers, and thus materially increased the plant capacity over that first obtained. Provision was also made to supplement the pipe cooling by blowing in of outdoor air when necessary. The fan power necessary to move all this vast weight of air and furnace gases through the bags and pipe system is, of course, considerable, reaching at times well up toward the 1000 hp. mark.

Notwithstanding this, however, the bag-house is to be considered a decided success; at least for the particular conditions met with at this plant, and the management deserves great credit for the courage and skill with which it has carried through this new, and after all largely experimental, undertaking, representing as it does the expenditure of over a quarter of a million dollars.

In the case of the Balaklala smelter, the use of a bag-house was also considered, and in fact a small experimental unit containing a few bags was run for some months in comparison with tests both

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[13] Clarence B. Sprague, U. S. Pat. 931,515, Aug. 17, 1909, and 984,498, Feb. 14, 1911.

by the electrical process here described, and also a centrifugal apparatus in which the gases passed through a rapidly rotating cylindrical shell equipped with radial baffles to insure the gas being raised to full velocity. As a result of these tests the electrical process was adopted for the full-sized installation.

The smelter treats from 700 to 1000 tons of 2.5 to 3% copper ore containing over 30% sulphur with considerable but varying amounts of zinc. The greater proportion of this is handled in blast-furnaces, but the fine material, including everything under an inch and amounting to less than 10% of the whole, goes through McDougall roasters and an oil-fired reverberatory. The plant has also two converter stands. The gases from all these departments pass into a common flue, 18 by 20 ft. in cross-section. The volume of gases passing through this flue varies with operating conditions from a quarter to half a million cubic feet per minute, which means, in round numbers, a mean linear velocity in the flue of 10 to 20 ft. per second.

Before attempting to design the full-sized equipment for treating these gases a small precipitation-chamber capable of treating about 1% of the total gases, was erected and an extended set of experiments made with it. With this miniature unit as a guide, the equipment of the whole plant with similar apparatus was undertaken in March 1910. This was completed and first put into operation at the end of the following September, since which time it remained in continuous operation until July 24, 1911, when the whole plant shut down until such time as a practical method could be found for removing from the gases not only the suspended solids, but the sulphur dioxide gas as well. Experiments to this end with the Hall process already referred to, upon at least a semi-commercial scale, are now being pursued by the Balaklala company at the smelter, but no decision seems yet to have been reached regarding their commercial practicability on the full-sized operating scale of the smelter. These experiments on sulphur dioxide have, however, nothing fundamentally to do with the electrical precipitation, and returning to this it may be said that the nine months of operation which the plant had, amply demonstrated the entire practicability of extension of the process to operation of this on even a larger scale. As was naturally to be expected, many difficulties were encountered, some of which have already been entirely overcome, while others are giving way more gradually before the systematic study of operating conditions. Fig. 41 is a plan of the nine electrical precipitation-units or chambers in their relation to the flue system and stack. It should here be noted that the two large fans indicated in the drawing are not required for the operation of the precipitating system nor to overcome any added resistance due to its introduction, as this latter is slight. The fans were made necessary by that section of the court's decree requiring dilution of the sulphur dioxide to 0.75% or less. When the furnaces are running on a high sulphur charge this feature of the decree necessitates a considerable dilution of the gases with fresh air and corresponding diminution of stack draft. At these times the

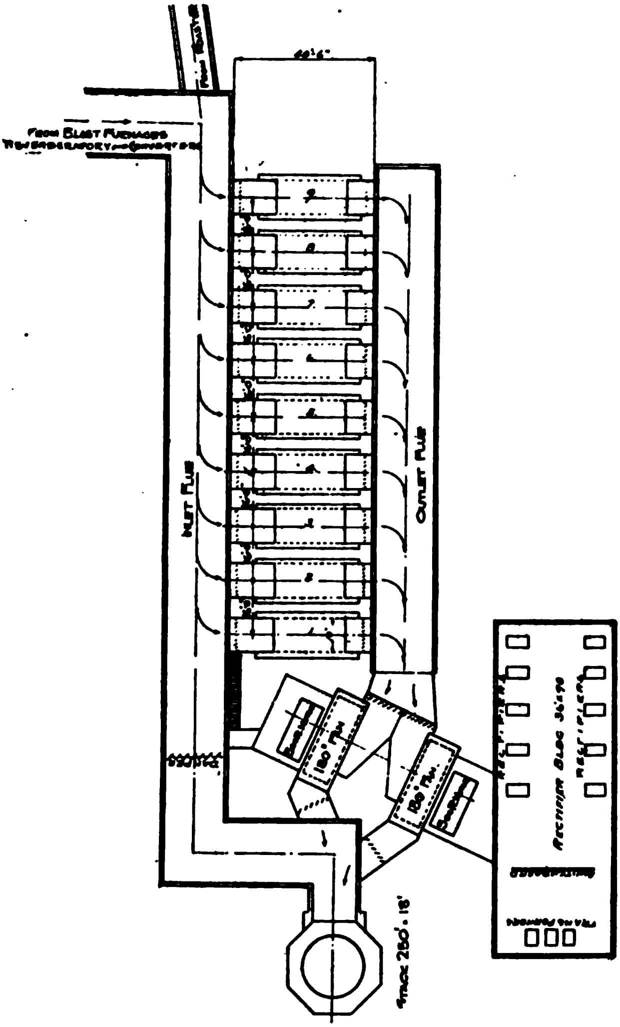

FIG. 41a. GENERAL ARRANGEMENT OF PRECIPITATING PLANT, BALAKLALA CON. COPPER CO., CORAM, CALIFORNIA.

fans are operated, but during a considerable portion of the time the sulphur dioxide in the gases can be brought low enough without interference with the draft, and during these periods the fans are stopped entirely, although the gases still pass through them.

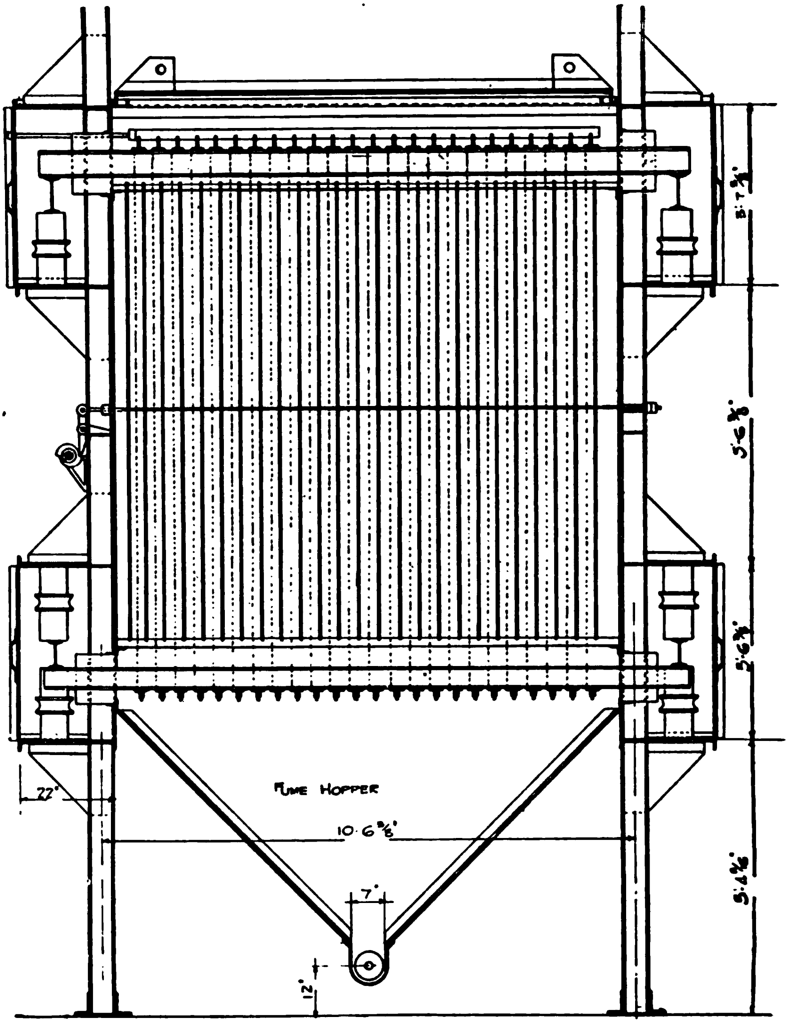

FIG. 41b. SECTION THROUGH PRECIPITATING PLANT, BALAKLALA CON. COPPER CO.

At the rectifier building the current is received from the company's three-phase power-circuit at 2300 volts, 60 cycles, and after being transformed up to from 25,000 to 30,000 volts under the control of the operator through the variable resistance and induction regulators, is rectified into an intermittent direct current, as already explained, and distributed to the individual precipitating units. Fig. 43 shows a cross-section through one of these units or precipitating flues as first installed. The double vertical lines represent the collecting or grounded electrodes, each 6 in. wide by 10 ft. high, made of No. 10 sheet iron. The dotted lines represent the discharge electrodes, consisting of two iron-wire strands, between which is twisted the discharge material, for which both asbestos and mica preparations have been used in this plant. Each unit contains 24 rows of 24 electrodes of each type. The collecting electrodes are carried by bars connected directly to the frame of the chambers themselves, while the discharge electrodes are spanned by springs between a system of buss bars carried on externally placed insulators as shown in the figure. To the auxiliary chambers surrounding these insulators a small regulated amount of air is admitted to prevent conductive dust or fume from working back and settling on the insulators. The cam and shaker rod extending across the middle of the unit was originally designed for the purpose of vigorously shaking the electrodes, as it was greatly feared that the removal of precipitate from the electrodes in units of this size might be one of our most serious problems. In actual operation it has been found, however, that the electrodes can easily be shaken entirely free from dust by hand from the top; the whole operation, including cutting the unit in and out of the system and the removal

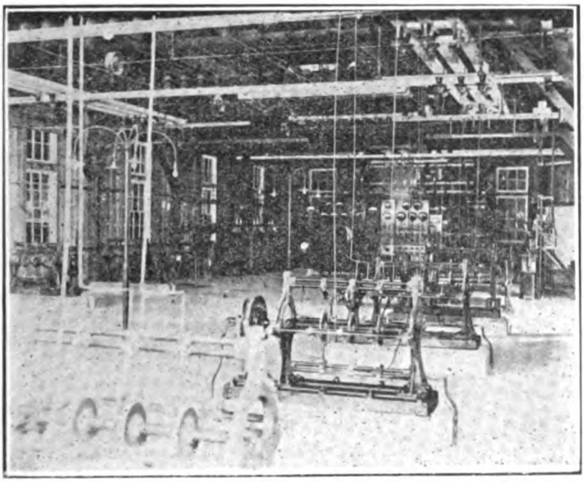

FIG. 42. RECTIFIERS AND WIRING OF DYNAMO ROOM.

and replacement of its covers, requiring only about ten minutes, and this having to be repeated every six or eight hours, depending on the dust content of the gases. The precipitated dust and fume as it falls from the electrodes is carried by the conveyor in each unit to a common longitudinal conveyor which in turn discharges into cars carrying it away for treatment.

Filtration tests show that this plant, under favorable working conditions, precipitates between 80 and 90% of the suspended matter in the gas, the average over the whole period of operation to date being somewhat less, while after introducing improvements in detail of construction on one of the units shortly before the final shut-down of the plant, the efficiency of this unit has carried well up into the nineties under average operating conditions at the smelter; this represents some 6 to 8 tons of precipitate per 24 hours. The gas-treating plant as a whole, including flues, fans, motors, and electrical apparatus, cost, up to the time it was put in operation, a little less than $110,000. Although many minor changes have since been made, none of the large or more expensive elements of construction have been greatly altered.

The total average power-consumption for the precipitation-plant when running is in the neighobrhood of 120 kw. One man can readily control the whole operation in the rectifier house, although, as a matter of precaution for a new plant under the high tension here used, two have usually been on duty. Two laborers and a foreman are employed on the precipitating units and dust-handling system, although this can probably be reduced somewhat by automatic shaking devices, since as yet the main efforts at improvement have been directed elsewhere. The volume of gases to be treated varies considerably with the condition at the furnaces, but at present may fairly be taken as averaging between 200,000 and 300,000 cu. ft. per minute, and entering the units at from 100° to 150° Centigrade. Fig. 42 is the interior of the rectifier house or control station, showing the general arrangement of apparatus and wiring.

This plant, while not able to save the Balaklala smelter from an eventual shut-down, formed a very important link in the development of the processes to their present status. The interest of the smelter companies up to this point depended almost entirely on the hope of eliminating smoke as a nuisance and its attendant litigation with their neighbors, and had it not been for the life or death struggle that this meant to them, it is probable that the success of the electrical processes might have been delayed many years longer. Once brought, however, under this powerful stimulus to the state of development above described, importance of these processes for the saving of metals lost through waste gases began to claim general interest and at present a large number of precipitating plants are under construction and several are already operating in copper, lead, and iron smelters, metal refineries, cement mills, and a varied set of chemical industries.

It is unnecessary here to describe any of these plants in great detail as they represent essentially the same principles as those already discussed. Fig. 43*a* and 43*b* show current off and on in an experimental unit erected about the end of 1911 at the Garfield copper smelter in Utah to treat the gases from basic lined converters handling a leady matte. In this case, the five-inch steel pipes carrying the gases act also as grounded or collecting electrodes, the discharge electrodes being iron wires stretched axially within them. The deposited material was shaken down from time to time by striking the pipes with a system of hammers attached to the rocking shaft seen in the picture passing behind the front row of pipes. A pile

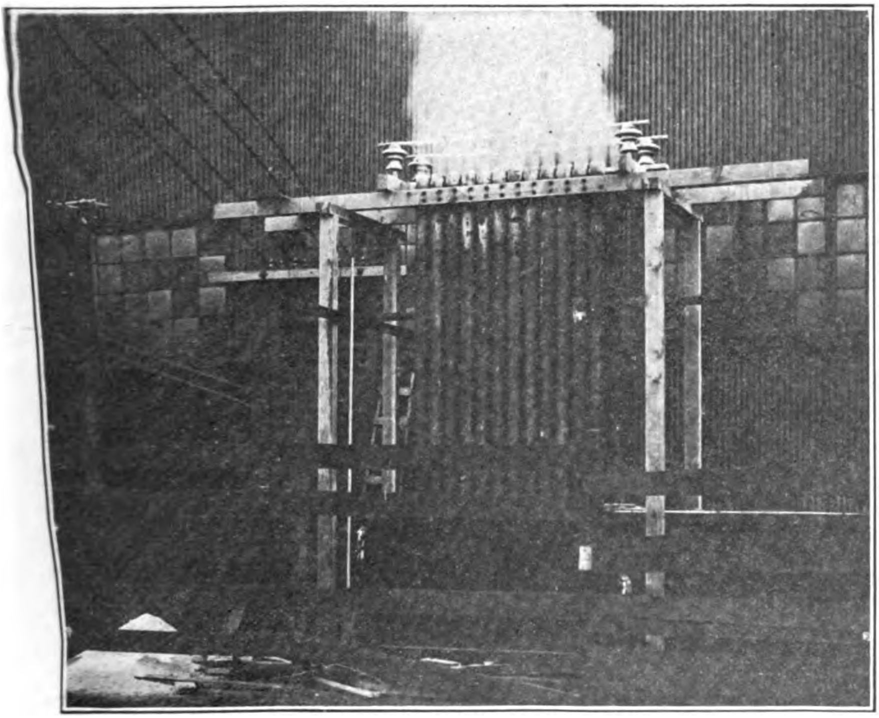

FIG. 43*a*. EXPERIMENTAL PRECIPITATING UNIT AT GARFIELD WITHOUT CURRENT. AT THE LEFT IS SHOWN A SMALL PILE OF PRECIPITATED MATERIAL.

of the deposit consisting chiefly of lead sulphate but also carrying gold, silver, and other metals is seen on the ground to the right. This unit consisting of 24 pipes was followed by another one of 600 pipes which was successfully operated for about a month on steady tests, taking the whole of the gas from one large basic converter and part of the time a portion of that from a second converter as well. A part of this equipment was then moved to a point where gas from the other departments (i. e., blast-furnaces, reverberatories, and roasters) could also be secured and experiments continued on each of these and on mixtures of them. They were all found to be easily handled, the others in fact were more easily handled than the converter gases above illustrated, but these latter contained the material of highest value. Having determined to construct a new stack and flue system to take the gases from the blast-furnaces and converters of the plant it was decided to incorporate an electrical precipitation plant for the treatment of all the converter fume. This complete plant is now (1914) under construction and is expected to be in

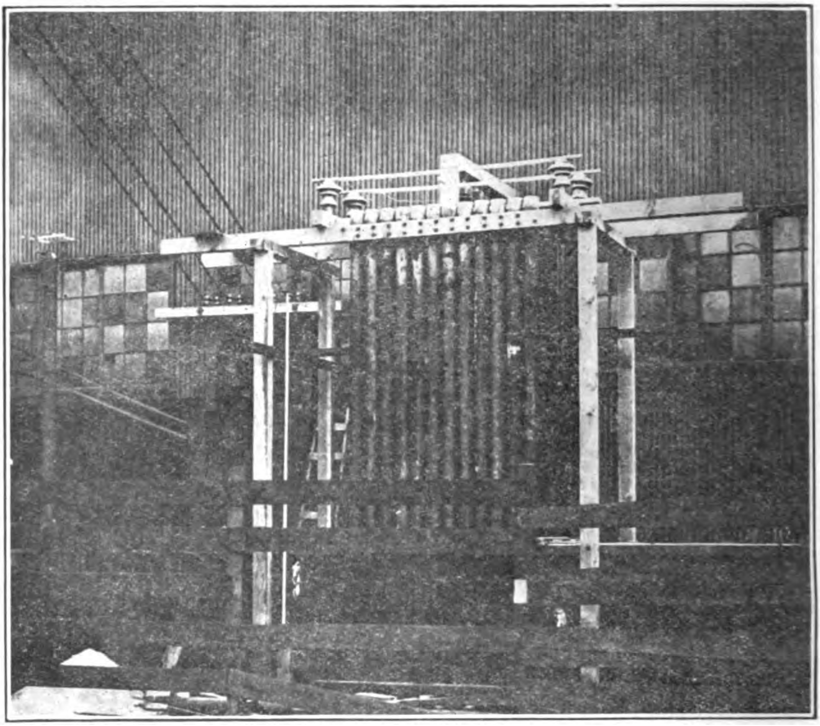

FIG. 43*b*. EXPERIMENTAL PRECIPITATING UNIT AT GARFIELD WITH CURRENT ON.

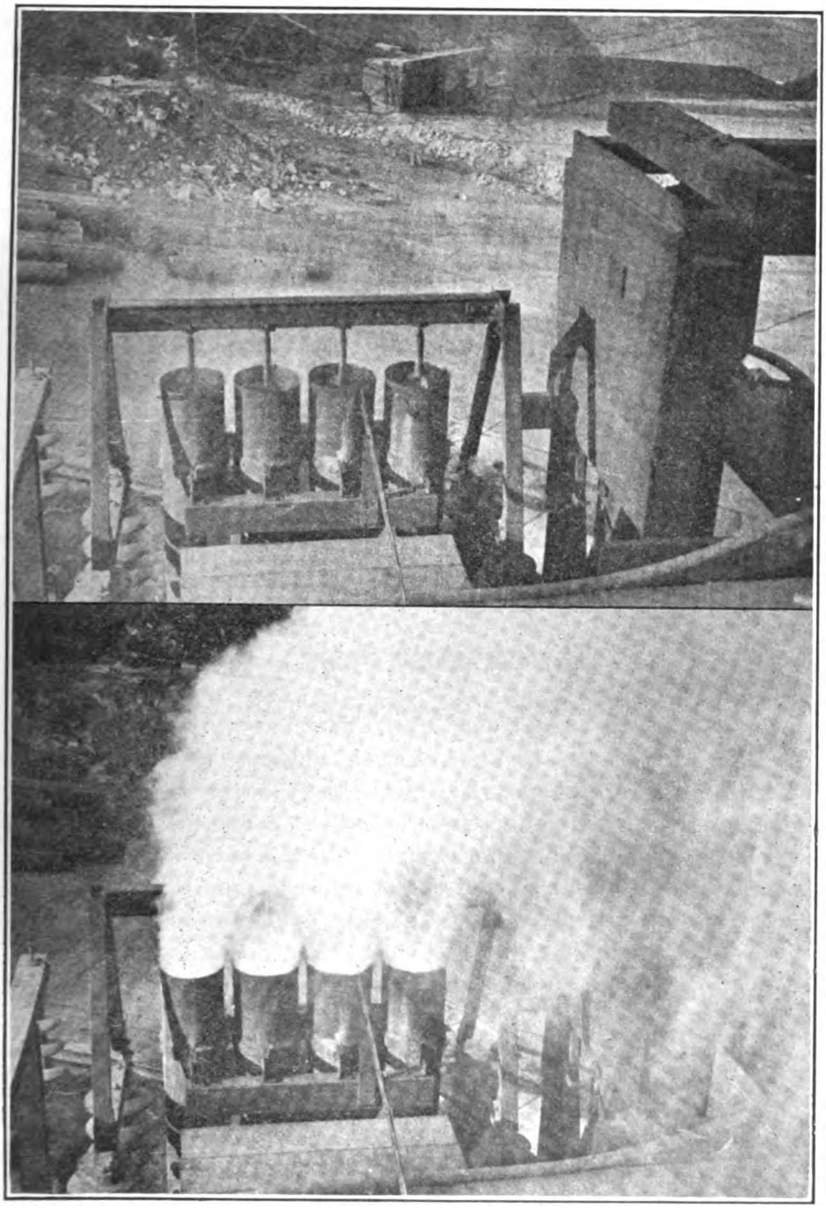

FIG. 44. COTTRELL PRECIPITATING PLANT AT ANACONDA; ABOVE WITH CURRENT ON, BELOW WITH IT OFF.

operation by the coming summer. Provision has also been made for the adding of electrical precipitation plant for the blast-furnace gases, should it later be decided to do so.

Another plant to treat about 100,000 cu. ft. of gas per minute and consisting of 384 pipes each 13 inches in diameter is also under construction at the lead smelter of the Consolidated Mining & Smelting Co. at Trail, British Columbia.

Another unit of tubular type in this case with pipes of lead 12 inches in diameter and 12 ft. long, was constructed at Anaconda to collect the copper and acids volatilized from an 80-ton experimental McDougall furnace, roasting low-grade sulphide tailing with salt preparatory to leaching. In this plant an electric current of some 60,000 volts is used. Where the material collected is largely liquid or saturated with acid as in this case both wood stave and vitrified terra cotta pipes up to 2 ft. in diameter have been used with success, and with dry material iron-pipe construction has been successfully employed up to 36 in. diameter and 20 ft. in length. In these larger pipes, several discharge electrodes are generally employed carried on a central support. This type of construction where applicable has done much to simplify and reduce cost of installation.

SMELTER FUME

(Editorial, September 23, 1911.)

Smelter fume has been for some years past a critical problem in the metallurgy of the metals which chiefly occur as sulphides, of which copper, lead, and zinc are the most important. Smelters, like many other industrial plants, have always been considered undesirable neighbors, because of the odors emanating from them, though when compared with tanneries, glue-factories, stock yards, fertilizer works, and many other useful and necessary forms of industry, their offense seems almost negligible. The solid particles in the fume, gradually settling in the neighborhood of the stack, have also been sources of complaint. Dust of any sort is objectionable, and it has also been urged that the chemical effect of the dust settling from smelter fume is deleterious to the vegetation on which it may settle and to livestock that may browse upon this vegetation. It is unquestionable that injured vegetation and dead livestock have been found in the vicinity of smelters but that the damage was the direct result of smelter fume can rarely be definitely proved.

The solid particles commonly present in smelter fume are of two kinds, fine particles of ore and flux which are carried off by the force of the draught, obviously no more objectionable than the ordinary dust of a country road, and fine solid particles commonly present which have been produced in the smelting operation itself. The most important of these are condensed sublimates of some of the volatile metals, notably oxide of arsenic, and H_2SO_4. The latter is produced in roasting and smelting both by the catalytic action which certain oxides, principally of iron and silica, exert in SO_2, and by the decomposition of various sulphates by heat. The SO_3 thus formed. takes up moisture as soon as it has cooled sufficiently, forming a fog of sulphuric acid. Except when smelting a charge high in sulphur, the amount thus formed is undoubtedly small, and it is unquestionable that 'the damage produced has often been greatly exaggerated. Where any considerable amount of arsenic is present in the ore it will largely escape as fume and may poison livestock grazing near the smelter. There has been much controversy over the subject of smelter fume and only a few points seem to be clearly established as yet. One is that the damage complained of has been greatly exaggerated and in many cases can be proved to be due to causes other than smelter fume. The second is that damage has been done in some instances. The third is that it is commercially feasible to remove solid particles from fume before allowing it to escape. The Cottrell electrical process of precipitating solid matter had given good results at the Balaklala when that smelter was closed for a variety of reasons, and it has also been found possible to neutralize the sulphuric acid formed and then to remove all the solid by filtering through bags, in much the same method as that employed in the production of sublimed pigments of zinc and lead. Part of the cost of this work is defrayed by the increased recovery from the solids; the remainder may be regarded as an excise tax upon the privilege of carrying on smelting operations in agricultural districts. That a smelting industry should be compelled to carry this burden when the smelting operations antedate the agriculture would seem unreasonable, but legal privilege does not always coincide with justice and equity. The third cause of complaint is the most difficult to deal with. In roasting and smelting operations large quantities of SO_2 are discharged into the air; practically all of the sulphur driven off in roasting and all the sulphur of the smelting charge which is not drawn off in the form of matte escaping in this way. That SO_2 will injure vegetation is known to everyone who has had occasion to disinfect a sick room containing growing plants; that the gas must be of a certain degree of concentration to produce injury is capable of equally simple demonstration. Theoretically, the gases escaping from the smelter stack quickly mix with the outer air and are so diluted as to become harmless. Practically, it appears difficult to secure complete mixing of escaping gases with the great mass of the atmosphere and they frequently travel to a considerable distance in nearly their original state, as the muddy water of a small tributary is often distinguishable in the clearer water of a large river for long distances below the point of inflow. Where the percentage of sulphur on the charge is low, this is no great matter, as the undiluted gases are not concentrated enough to produce injury, and smelting works have, for example, long operated undisturbed among the fertile farms of New Jersey. But where the sulphur content is high, as in pyrite smelting, the degree of concentration becomes dangerous, and the injurious effects of the gas may be increased by a moist climate, which tends to heighten the effect of SO_2, or by the topography, which may produce atmospheric currents that carry the escaping fume, undiluted, into contact with nearby vegetation.

Blowing air into the base of the stack by means of fans has been tried, but that the power cost is a disadvantage of this method is obvious. Converting the SO_2 into SO_3, which may be made into sulphuric acid, is done at the plant of the Tennessee Copper Company, and the objectionable gas is thus made a source of profit. But this cannot form a universal solution of the problem; the initial outlay required is too great for small companies and if all smelting plants engaged in the manufacture of acid the market would be inundated. It is also necessary that the fume shall have a high concentration in SO_2 to allow its economical conversion into acid The problem is accentuated by the legal requirements of some states, which provide that escaping fume must have a sulphur content so low as to be procurable only at a prohibitive cost. As described on a following page, Mr. Stewart W. Young has attacked the problem boldly by proposing the reduction of the SO_2 to sulphur by hydrocarbons using calcium sulphide as the intermediary. The suggestion is a novel one and the results of trial upon a working scale will be of much interest. Evidently one problem will be to bring the fume thoroughly into contact with the calcium sulphide without interfering seriously with the draft. In this connection it may be noted that in the early stages of the Gayley dry-blast process it was suggested that the blast be passed over calcium chloride, to which it would give up its moisture, the chloride to be regenerated by passing the gases of combustion over it. The mechanical difficulties of this method were found insuperable, though recent reports from Germany are to the effect that the method is being successfully practised there. In many other cases where a reversible reaction has been made the basis of a continuous process it has been found that after a number of reversals the reaction becomes slow and incomplete. The question of cost is the vital one, and the sulphur produced can scarcely be counted on as an asset. The cost data of experiments upon a working scale will be the criteria of success, and meanwhile we assume the prerogative of extending the thanks of the profession to Mr. Young for his efforts toward the solution of an urgent metallurgical problem.

THE THIOGEN PROCESS FOR REDUCTION OF SO_2 IN SMELTER FUME

By Stewart W. Young

(September 23, 1911.)

The difficult situation of the smelting industries of the United States at the present time is too generally understood to need any extended discussion here. The fumes which escape from the various furnaces, especially from roasters, carry in many cases quantities of substances that are destructive to vegetation and in some cases poisonous to stock. The result is that in some cases it becomes impossible to carry on the two industries of agriculture and smelting in the same district, and a number of important smelters have been enjoined from operating by court orders issued on complaint of farmers.

The most important of the deleterious substances contained in smelter fumes are arsenic compounds, sulphur trioxide, and sulphur dioxide. Various devices are at present in successful use for handling the first two of these substances. Both the bag-house systems and the Cottrell apparatus are able to remove the arsenic and the sulphur trioxide from the fumes, but neither is able to remove sulphur dioxide. The reason for this is that the sulphur dioxide is present in the form of a true gas, while the above mentioned methods are able to handle only substances in the form of solid particles, which is the form of occurrence of the arsenic and the sulphur trioxide.

The only device for mitigating the sulphur dioxide evil that seems to be in use is that of diluting the stack gases with large quantities of fresh air so that the concentration of the sulphur dioxide is at a minimum. The benefits of this method are probably considerable where comparatively small amounts of sulphur dioxide are released, but in case very large quantities are released its beneficial action is at least questionable. It might be imagined that since sulphur dioxide is very readily soluble in water, the stack gases might be sprayed, and the sulphur dioxide eliminated in this way. But the solution of sulphur dioxide is equally as poisonous as the gas itself, and there is no practicable way of getting rid of it. The problem, then, seems to narrow itself down to some process for chemically converting the sulphur dioxide into some other substance that may be utilized, or that at least is innocuous. To the chemist there are two, and only two, means which seem at all feasible for the accomplishment of this purpose. They are:

(1) The conversion of the sulphur dioxide into sulphuric acid.

(2) The conversion of the sulphur dioxide into free sulphur or brimstone.

There are various methods whereby the first of these reactions may be brought about, and these are used in some places, as, for example, at the Ducktown smelter. The chief obstacle to its extensive use is the comparatively limited market for the sulphuric acid. It is practically impossible to store the enormous quantities of this substance that would be produced, and still more is it impossible to throw it away in considerable quantities. Like sulphur dioxide, it will kill vegetation and fish. The possibility of converting the sulphur dioxide into free sulphur seems to have received very little attention up to the present time. Sulphur is an innocuous substance and, provided a reasonably economical process is devised, its elimination from smelter gases in this form would seem to offer an ideal solution of the problem. Supplies of sulphur for many years to come would be established, and the present serious check to the smelting industry would be overcome. It is the purpose of this paper to give a brief sketch of a process whereby this result may be accomplished.

The elementary chemistry involved in the reduction of sulphur dioxide is extremely simple. The reduction may be accomplished by means of carbon or by means of hydrogen, or, still better, by means of compounds of carbon and hydrogen, such as crude petroleum and the like. Assuming, for simplicity's sake, a hydrocarbon of the formula CH_2, the reduction should proceed as follows:

$$SO_2 + CH_2 = S + CO + H_2O$$

That is, the carbon is converted into carbon monoxide, the hydrogen into water, while the sulphur is left in the free state. It may be stated at the outset, that since the conversion of sulphur dioxide into free sulphur and free oxygen is a reaction that involves an absorption of energy, such reaction cannot be accomplished without supplying such energy. Thus there will always, of necessity, be a fixed cost for materials used in any process operating in this way, and no purely catalytic process can accomplish the result. In the above reaction the energy used is supplied by the hydrocarbon.

While from the above statement of the case, the problem seems to be readily solved, this is, as a matter of fact, not the case. Very serious difficulties stand in the way and these are partly chemical and partly mechanical. The more important of these difficulties are the following:

(1) The presence of free oxygen in the gases to be treated, bringing about a serious waste of hydrocarbon.

(2) The slowness and incompleteness of the reactions even when oxygen is not present.

(3) The fact that all the available energy of the hydrocarbon is not utilized, since carbon burns to monoxide instead of dioxide.

All of these difficulties have been met and solved in the laboratory, and although it is not possible at present to discuss in detail the devices whereby these things have been accomplished, there is no objection to making public a general sketch of the main points of the process.

The Thiogen process has been put into a practical form by carrying out the reaction discussed above in an indirect way. The sulphur dioxide is first absorbed by means of a basic sulphide, either in the wet or dry state, the one chosen for most work being calcium sulphide. The reaction which takes place here results in the formation of calcium sulphide and free sulphur. The presence of oxygen in the gases does not interfere in this reaction to any appreciable extent. That is, the reaction with the sulphur dioxide is very rapid, while that with oxygen is very slow.

The second step in the process consists in removing the free sulphur from the calcium sulphite, which may be accomplished by distillation or other suitable means. The third step consists in the reduction of the calcium sulphite by means of hydrocarbons. This reaction results in the regeneration of calcium sulphide which again enters into the reaction in its first stage. That is, it is used for the further absorption of sulphur dioxide. Under ordinary conditions the reductions of the calcium sulphite to sulphide is a matter of considerable difficulty, the reaction being rather slow, even at high temperatures and resulting in the conversion of the carbon into monoxide instead of dioxide as is desirable. A method has, however, been devised whereby the reaction takes place readily even at relatively low temperatures, and whereby the complete combustion of the hydrocarbon is assured. An adaptation of the process has also been devised whereby gases which are free, or nearly free, from oxygen may be handled with a much increased mechanical efficiency. In fact, the fundamental principles upon which the process depends offer very considerable possibilities of adaptation to suit special conditions. Incidentally, considerable progress has been made toward adapting the process to the recovery of fine dust. The reactions involved in the indirect method, expressed in chemical symbols, are as follows:

(1) $2CaS + 3SO_2 = 2CaSO_3 + 3S$

(2) $2CaSO_3 + 2CH_2 = 2CaS + 2CO_2 + 2H_2O$

To anyone versed in chemistry it will appear at once that here, in an indirect manner, is accomplished precisely what is done by the more or less theoretical equation given in the earlier part of this article, except that here the combustion of the hydrocarbon is more complete. In any event, the fixed cost of the process is a quantity of a reducing agent (here hydrocarbons) sufficient to reduce the sulphur dioxide.

Another reaction which is made use of to some extent depends upon the fact that when sulphur dioxide is passed over a heated sulphide ore, the sulphide ore is roasted, and that almost as completely as if air were used. Expressed chemically, this reaction is a follows:

$3FeS_2 + 2SO_2 = Fe_3O_4 + 8S.$

In this case the ore itself becomes the reducing agent and the fuel oil charge is avoided. The reaction is, however, slow and requires external heating to give any great capacity. On the other hand, it is found to be a valuable adjunct to the other processes above described.

The ultimate efficiency in the application of these principles will naturally be greatest in a plant in which the adjustment is so made that the greatest amount of sulphur dioxide may be handled with the least expenditure of hydrocarbon, and with the least expenditure of installation and operating expenses. Whatever may be the most economical plant for accomplishing this purpose can be told only from actual operation, but this much at least seems certain, that, from the chemical point of view, the maximum economy of these processes must probably be the maximum economy attainable in any process for the production of free sulphur from sulphur dioxide. The reasons for this are briefly as follows:

(1) A certain perfectly definite expenditure of energy in the form of a reducer is absolutely necessary to produce sulphur from sulphur dioxide.

(2) The processes above discussed operate by applying this energy in the cheapest known form, namely hydrocarbons, and further take advantage of a certain possible saving through a reduction by means of the ore itself.

Thus in order to devise a process whose theoretical chemical efficiency is greater than that of the Thiogen process, it will be necessary to discover a reducing agent cheaper than the hydrocarbon oils, which seems not likely to happen. It would not be fair to close this article without expressing my deepest thanks and appreciation to Messrs. John T. Overbury, H. G. McMahon, and Hamilton Sanborn, of San Francisco, whose assistance, financial as well as personal, has made possible the development of the Thiogen process to its present state.

EXPERIMENTS WITH THE THIOGEN PROCESS

By FRANK L. WILSON

(April 6, 1912.)

At a lecture recently delivered before the Mining Association of the University of California, S. W. Young, professor of physical chemistry at Leland Stanford Jr. University, gave a very interesting and detailed account of his Thiogen process and the results so far obtained. His remarks, in slightly condensed form, follow.

The gases and waste products issuing from the flues of roasters, blast-furnaces, reverberatory furnaces, and converters, are made up for the most part of flue-dust, SO_2, SO_3, CO_2, and nitrogen in varying proportions. The main sources of these products are the roasters and blast-furnaces. The flue-dust is largely caught before leaving the stack, while the gases pass out into the atmosphere. The most troublesome of these gases are the SO_2 and SO_3. The SO_3, being but a small part of the gas, has not proved of importance, but the SO_2 issues from the stacks in large quantities and travels great distances. One method of present-day practice is to dilute the gases by mixing with sufficient air to meet the requirements of the law before discharging into the atmosphere, and heretofore no attempt has been made to recover the sulphur as such.

The present sulphur supply is obtained from the sulphur wells of Louisiana, and the natural deposits of Japan, Sicily, and Mexico. At the present time sulphur is being discharged into the atmosphere from the stacks of different metallurgical plants (1000 tons of pyrite ore discharging approximately 800 tons of SO_2). To recover this sulphur it is necessary to have some reducing agent to combine with the oxygen of the SO_2. Carbon has been tried, the SO_2 gas being passed over red-hot coke. This process was found to be expensive and only partly successful. In the Thiogen process an oil spray is used to furnish the necessary carbon, and the reaction is shown by the following equation:

$$SO_2 + CH_2 = S + CO + H_2O$$

Ordinarily the CO would reduce more SO_2 as is shown by

$$4CO + 2SO_2 = 2S + 4CO_2$$

but this reaction takes place very slowly. In the presence of a catalyzer, however, the reaction is accelerated. It has been shown by the Mannheim process that in the manufacture of sulphuric acid the reactions are accelerated by means of a catalyzer in the form of Fe_2O_3. The SO_2 gases are passed over hot Fe_2O_3

$$3Fe_2O_3 + SO_2 = 2Fe_3O_4 + SO_3$$

and the Fe_3O_4 again reacts with the free oxygen in the gas

$$2Fe_3O_4 + O = 3Fe_2O_3$$

A suitable accelerating reagent was found in calcium sulphide. Potassium and other alkaline sulphides might be used, but calcium is the cheapest and the most infusible of them all, standing quite high temperatures. As a catalyzer the calcium sulphide acts as follows:

$$2CaS + 3SO_2 = 2CaSO_3 + 3S$$

and in the presence of a reducing agent, as vaporized oil, the $CaSO_3$ is reduced to CaS

$$CaSO_3 + CH_2 = CO_2 + H_2O + CaS$$

In laboratory experiments these reactions are rapid, especially so when iron salts are present in the catalyzer. No CO was formed, and all of the carbon combined with the oxygen to form CO_2 direct. The mixture of CH_2 and SO_2 may be passed over almost any calcium compound, as $CaCO_3$ or $CaSO_4$, and the resultant compound is CaS. In case the gases contain free oxygen, this must be taken care of before the CH_2 will replace the oxygen in the SO_2, and this difficulty presents the worst drawback to the process from an engineering standpoint. A suitable temperature must be maintained for the gas passing over the catalytic agent.

It is preferable to clean the gas as thoroughly as possible and concentrate it before allowing it to pass to the combustion chamber. An 8% SO_2 content has been obtained by passing the resulting gas products from one roaster through a large drum (*a* Fig. 45) to another similar roaster, the draft being controlled by means of dampers (*b*). The gas finally passes through the connecting flue (*c*) to the apparatus.

The roasters at Campo Seco, where this process is being tried experimentally, are of an old type, not constructed to control the amount of oxygen passing in so that all the admitted air may be utilized, and 8 or 9% gases are the best that can be obtained.

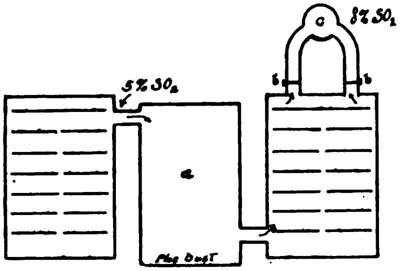

FIG. 45.

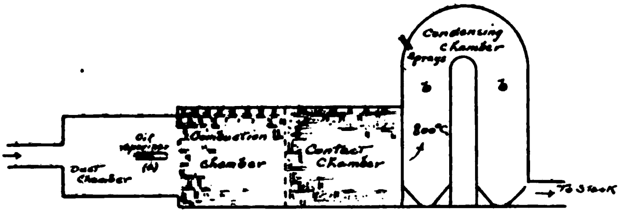

FIG. 46.

The vaporization of the oil is accomplished by the use of steam injected burners, as shown at *a*, Fig. 46. This spray of oil is directed into the combustion chamber, which consists of a checkerwork of brick, and a temperature of about 800°C. is obtained. Beyond this is a reaction chamber, similarly filled with a checkerwork of brick. This chamber is 4½ ft. square and 30 ft. long, and is more than able to handle the gases from the two roasters. The spaces in this checkerwork of brick are nearly filled with a mixture of equal volumes of plaster of paris and sawdust wetted with water, to which a small amount of iron salts is added. This mixture has the advantage that it decreases the amount of plaster of paris used and gives a porous mass. The sawdust burns out and the $CaSO_4$ reduces to CaS quickly, the mass shrinking so as to fill about half of the space between the bricks. Condensing chambers are necessary to collect the sulphur liberated. Cast-iron condensers were found to be too leaky, and at present an inverted V-shaped pipe is used (see *b*, Fig. 46), each condenser having a sloping floor with a trapdoor. Difficulties were encountered in this chamber. Any selenium, arsenic, or antimony compounds that the ore contains distill over and condense with the sulphur. To overcome this a low, broad, slightly inclined, condensing chamber can be used, in which water-sprays cool the gas below the boiling point of selenium and arsenic (about 750°C.), or about 500°C., the selenium, arsenic, and antimony being deposited on projecting tiles placed in the walls of the chamber for that purpose, and finally drawn off through traps in the lower corners. The sulphur, having a lower boiling point, passes on and is caught in the regular condensing chamber described above.

Tight construction is required throughout, and the brick walls so far used have been too porous to withstand the inward pressure of the air due to the vacuum caused by the draft up the stack. Leakage of air has been guarded against by means of sand sealing. Trouble also occurred through ends of the chambers being forced out, due to expansion of the brick, and it is hoped this will be overcome in a great measure by building the separate chambers independently and allowing for telescoping, using sand sealing to guard against air leakage inward.

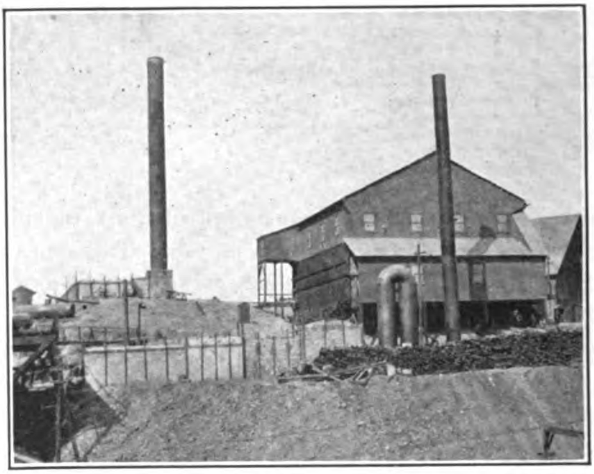

FIG. 47. THIOGEN PROCESS PLANT, CAMPO SECO.

Three trials have been run so far at Campo Seco. The first was largely to show the supervisors that the process was feasible. The apparatus used consisted of a 4-in. pipe connecting with the flue and discharging into an 8-in. pipe, 5 to 6 ft. long, in which the contact material was placed. A gasoline burner was placed at the junction of these two pipes and the necessary CH_2 and heat obtained. Leading from the 8-in. pipe, a 2-in. pipe discharged into a large iron drum. At a point in the drum above the 2-in. pipe a spray of water was allowed to play on the gases, the resultant liquid being tapped from the bottom of the drum. This resultant liquid showed free sulphur and was milky white in appearance. Connected with the drum by means of another pipe was a blower that drew the gases through the apparatus with considerable velocity and discharged the soot and the gas on a piece of corrugated iron. Analysis of this soot showed free sulphur. The escaping gas was nearly free from SO_2.

The second trial was made about a month ago, and on a larger scale. There was considerable leakage of air and consequently considerable oxygen came in contact with the free sulphur and combined with it. Heat was generated by this combustion, and the temperature at the condenser was raised to slightly over 950°C. Analysis of the gas showed 5.6% SO_2 at the inlet and 0.2% at the condensing chamber, a decrease of 5.4%, or above 97% of the sulphur dioxide present.

On March 11 the present condensing chamber was ready and the apparatus tried (see Fig. 46). Trouble was encountered shortly after the burner was started. The temperature ran up beyond 1100°, and on shutting down it was found that someone had shut off the spray-water, and as a result the sprays were burned out and rendered useless. A tremendous draft was developed, due to the excessive heat. The present sprayers are an improvement over the former ones, in that they cause the spray to be thrown in all directions, thus cooling the gases more uniformly.

The cost of repairs cannot be estimated at the present time. Gases from the roasters have been passed for some time through the apparatus without settling out the flue-dust, but if the dust is settled out, the apparatus can run indefinitely without becoming clogged. Gypsum is not very expensive and the old contact material is not wasted, as it can be used in the smelting operations. Indicators are now being devised to register the percentage of SO_2 in the gas automatically, the supply of gas in turn being regulated by dampers placed at convenient points. Any excess oil burned will be taken care of by the $CaSO_3$, and any excess SO_2 by the CaS.

SMELTING AT CAMPO SECO, CALIFORNIA

By M. W. von Bernewitz

(December 6, 1913.)

In a hilly and somewhat rough district, although only at an altitude of a trifle over 300 ft. on the Calaveras county side of the Mokelumne river, is the property of the Penn Mining Company. The mine has been opened to a depth of 1400 ft. by the main incline shaft, while below this a winze has been sunk another 300 ft. The country rock may be described generally as a greenstone, and the ore contains about 5% copper, a large amount of baryta, and a gold and silver content of $1 per ton. This, it will be admitted, looks like a fairly complex smelting problem, but as is shown by the following description of the process, the ore is not troublesome.

From the main and No. 2 shafts the daily tonnage hoisted is 100 and 40 tons, respectively, that from the latter being higher in silica than from the main part of the mine. All over the surface of the property are tracks laid for the Vulcan steam locomotive, which hauls the ore in Weiner side-dumping 5-ton capacity cars to a 90-ton storage bin. From this the coarse ore is fed to a Gates No. 3 D-type crusher, and the fine passes through a grizzly. All of the crushed material and that from the grizzly is taken by an elevator to a trommel having ⅝-in. holes. The undersize from this goes direct to a conveyor-belt, while the oversize passes through a McFarland 16 by 36-in. set of rolls, then on to the belt with the undersize. An improvement is to be effected here before long in that the roll product will go through a second trommel with ⅝-in. holes, the oversize to be fed to a pair of 12 by 24-in. rolls. Roll shells last three months. All the ore is next conveyed to bedding bins having a total storage of 2100 tons, of which 700 tons constitutes one lot, which is independently sampled.

Cars holding 1.3 tons of the bedded ore take it to the 8 McDougall roasting furnaces of the Pacific Foundry type. These are 11½ ft. diameter and have 7 hearths and a drying floor on top. Originally they had only 6 hearths, but the addition of another, and the drying floor, increased each furnace's capacity by about 20%. Rabbles are air cooled by natural draft, but if necessary, air may be forced through by fan. The speed is one revolution in 65 seconds, with worm drive, no fuel is used, and the sulphur is reduced from 26 to 8%. Dust losses are 1½% of the crude ore, but a fair quantity of this is saved in the 10 by 10-ft. flue, leading to the 6 by 85-ft. steel stack, which also serves the smelting furnace, being 125 ft. above it. A 15-hp. motor drives the roasting furnaces.

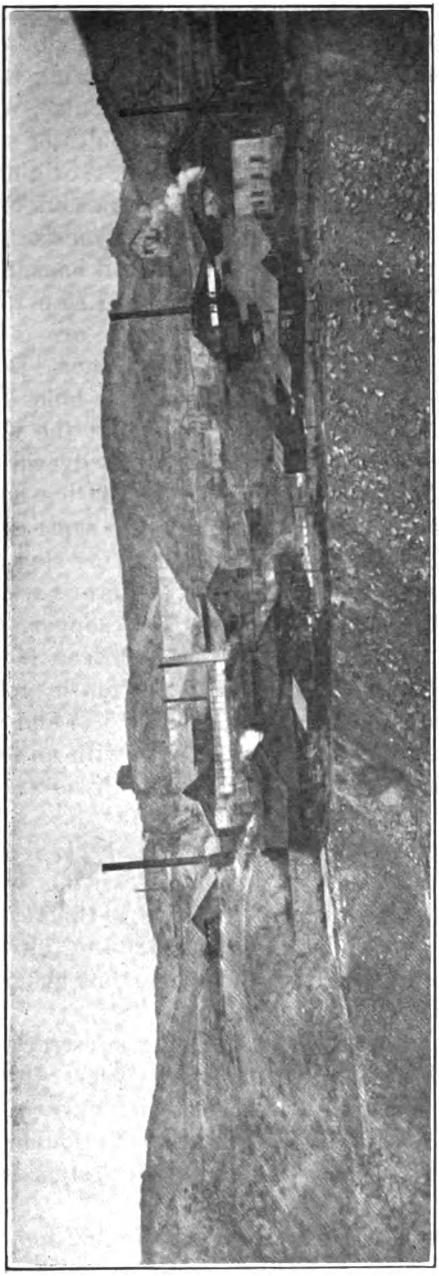

FIG. 48. THE PENN MINING CO. SMELTER AT CAMPO SECO.

The roasted ore is trammed to the charging hoppers over the reverberatory furnace after being weighed.

The reverberatory is of Anaconda type, 20 by 56 ft., of 788 sq. ft. effective hearth area, the roof and sides tapering toward the firing end. The roof has a rise of 20 in. at the centre, while the bottom is of fused silica of local quartz, and the sides of silica brick, then magnesite brick on the slag line, the thickness originally being 27 in. The furnace is oil fired, there being three burners of local make, oil being delivered at 80 lb. and steam for atomizing at 100 lb. pressure. Over a period of nine months the oil consumption averaged 43 gal. per ton of ore smelted, this costing $1.15 per barrel delivered at the works. Charging the furnace with ore is done about six times per 8-hour shift, each charge being 6 tons. During this work, the oil blast is shut off. Ordinary charge holes in the top of a reverberatory burn out in 6 to 8 weeks, but the water-cooled cast iron holes have been in 10 months. Slag is drawn off about every five hours into 25 cars holding 2000 lb. of this material, and the matte is drawn off once or twice daily, the same cars holding 2700 lb. of this product. The cars are taken to the slag or matte dumps as desired, by an electric locomotive. There is a concentration of 10 into 1, and the matte averages 55% copper, and is crushed, sampled, sacked, and shipped to the American Metal Co. of New York for refining. There is also $20 per ton in gold and silver in the matte. Copper content of the slag is 0.5%, and with 45% matte it decreases to 0.4%. It will be seen that with no fluxes and at one operation this complex ore is reduced with no trouble, giving an unusually high-grade matte and low slag.

In the flue from this furnace, which has been in successful operation since February last, is a 250-hp. Stirling waste-heat boiler which generates enough steam to drive a Buckeye 150-hp. cross-compound engine, which is belted to a 120-kw., alternating-current, 440-volt generator, also a small direct-current generator, these supplying the power for the whole plant.

A new reverberatory furnace is being constructed near the one described and will be blown in early next year. It will have a similar roof, but the walls will be of 9-in. brick only, of silica and magnesite as usual, 19 by 60-ft. hearth area, and double ended, the fires to be reversed after every charge. This system is hoped to give good results.

During 1912 the Penn mine produced 51,162 tons of ore yielding 6,058,449 lb. of copper, 112,020 oz. silver, and 2867 oz. gold. A. P. Bussey is superintendent of this interesting property, with D. C. Smith as metallurgist.

THE HALL SULPHUR PROCESS

(Editorial, July 5, 1913.)

Copper metallurgists everywhere, but especially in California, will be keenly interested in the Hall process for dealing with the sulphur in smelter fume, with which the First National Copper Company is about to undertake experiments upon a working scale at its smelter at Coram, California. Mr. William A. Hall, who has devised the process, is a graduate of the Massachusetts Institute of Technology, and a chemist of distinction who has made notable successes in the field of industrial chemistry, being a cousin of the chemist of the same name who devised the process for the production of aluminum by electrolysis of bauxite in a fused bath of cryolite. The Hall sulphur process is essentially a controlled oxidation; the sulphide ore being roasted in a specially-constructed furnace in an atmosphere of reducing gases and steam at a temperature between 700 and 900°C. As a result, the metallic bases are oxidized, but the sulphur, owing to the dissociation of the steam, escapes without becoming oxidized and, passing off in the fume in the form of 'flowers of sulphur,' is easily collected. Careful tests made under the direction of Messrs. C. F. Chandler and A. L. Walker, indicate that the chemistry of the new process is sound, Mr. Walker finding that ore containing nearly 40 per cent of sulphur was roasted to a sulphur content of 3 to 5 per cent at a rate which indicates that approximately the same tonnage per square foot of hearth area can be handled in this way as is done in current practice with the ordinary type of McDougall furnace. No data have been made public as to the fuel consumption to maintain this roasting speed, exact information as to this critical point being one of the objects of the large-scale experiments about to be started at Coram. The collection of the sulphur will be done by the aid of the Feld washer, though the possibilities of collection by means of the Cottrell electric precipitation process are also to be tested. The process will at first be applied to the McDougall roasters.

The large question involved is, of course, the disposal of the sulphur to be produced. Application of the process to all the ore and operation at the rate maintained at Balaklala in 1910 will involve the production of about 250 tons of sulphur per day, or somewhat in excess of the present visible local market, assuming that the sulphur can be laid down at points of consumption at a price which would enable it to supplant the pyrite now in use, as well as the rather limited amount of Japanese and Louisiana sulphur now being used in California. The estimated cost of sulphur production is placed at $5 per ton at a maximum, at which rate the sulphur would easily be able to dominate the market and perhaps increase consumption, as a lower selling price commonly does. In any case, any financial loss, not to exceed ½ cent per pound of copper produced, for example, incurred in the production of sulphur may properly be charged to the cost of smelting under the conditions obtained in Shasta county. Under previous conditions of operating at Balaklala, a production cost of 10 cents per pound of copper produced was estimated, though operating difficulties caused it to be somewhat exceeded in practice. The new process, if it proves feasible, will meet fully judicial restrictions, and if the cost of operation proves sufficiently low, will solve the smelter-fume situation as far as the First National is concerned. It is proposed to spend considerable sums on the experimental work, under the direction of Mr. H. F. Wierum, who has had extensive experience with the Tennessee Copper Company, and the prospect of success seems decidedly favorable. But, like the manufacture of sulphuric acid from smelter fume, the process is not one which is of universal application, as its general adoption would at once swamp the market on which dependence is placed for meeting the cost of operation. The foreign rights to the Hall patents have been acquired by the Sulphur Company, Ltd., and the American rights are controlled by the Federal Sulphur Company, Ltd. We wish both companies the fullest measure of success in their efforts to meet a situation trying in the extreme to copper metallurgists throughout the world.

RELATION OF CHEMISTRY TO METALLURGY

By Arthur C. Claudet

(March 4, 1911.)

*Among the large number of processes that might be quoted as illustrating the debt of metallurgy to chemistry, I venture to select the following:

The 'cementation', or precipitation process as applied to cupriferous pyrite at the mines in the south of Spain, and in adjoining districts of Portugal. This group of mines is traversed by a belt of cupriferous pyrite, containing 2% copper, with 46 to 47% sulphur. The processes used on the spot for the extraction of the copper belong both to chemistry and metallurgy. This is a case where chemicals are formed naturally, and in place, for the purpose of rendering the copper soluble. Briefly, the process consists in building up enormous heaps of mineral, so arranged that air can penetrate through them; water is pumped on them, and, aided by the heat of the sun, chemical reactions are set up; the copper sulphides are more easily oxidized than the iron pyrite, and are dissolved by the water. The resulting solution finds its way through the mass of the ore, and, after passing through a filter-bed of small pieces of pyrite, is led by suitable channels to precipitation-vats, where the copper is precipitated by means of pig iron, the precipitate being subsequently collected and sent to the smelting works. The pyrite heaps, after their copper content has been extracted, are sold to sulphuric acid works, whence, after burning out the sulphur, the oxidized residue is sold to iron works to be smelted. The copper precipitate is sold to the copper smelter.

This appears to be a simple process, but it has its complications. The chemist must thoroughly understand the reactions in order to regulate the cost, and to be able to produce a copper precipitate as free as possible from impurities, so that the best prices may be obtained. This requires metallurgical as well as chemical knowledge, that is, information concerning the requirements to which such products must conform at the hands of the smelter and in the market. In this process the main reactions are as follows: Oxidation is promoted by the action of the air, which is naturally warm owing to the climate, penetrating through the heaps, helped by the water percolating through them. When once the oxidation commences,

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*Abstract from *Journal of the Society of Chemical Industry*, December 31, 1910.

considerable heat is generated; this must be carefully controlled. The copper sulphide becomes copper sulphate, and the iron sulphide becomes converted into ferrous and ferric sulphates. The ferric salt exerts a powerful action on the oxidation of further quantities of copper sulphide, being thereby reduced to the ferrous state, then re-oxidized to ferric sulphate, and so on. The liquor issuing from the heaps contains a large proportion of iron as ferric sulphate, and in order to lessen the consumption of pig iron, this liquor is led through filter-beds of fine ore, so that the ferric sulphate becomes reduced mainly to ferrous. It may here be remarked that much more iron is consumed in precipitating the copper than is required by theory, and this excess consumption is reduced considerably, according to the proportion of the ferric salt present in the liquor, before it enters the precipitation-vats. The number of these vats, which are filled with cast-iron pigs, and the length of the series is determined by the work done by the iron; that is, when the iron consumed costs more than the worth of the copper precipitated, the liquor is run to waste. In the iron precipitation process, the copper precipitate gets contaminated by the impurities present in the iron, and this affects the quality of the precipitate produced during the final stages. The pig iron used varies from 92 to 94% metallic iron, the remainder being carbon and phosphorus. In most descriptions of this leaching process the phosphorus is rarely mentioned. At certain stages of the process the precipitate will therefore contain considerable quantities of these two elements, carbon and phosphorus, and in addition arsenic from the ore itself. The arsenic also is chiefly thrown down in the final stages of the process. It is therefore desirable to collect the copper precipitate in grades of quality, as by doing so the precipitate will command a higher average price. The heaps, after having been denuded of their copper, are sold to sulphuric acid makers all over the world. On the average, the washed pyrite contains 45 to 50% sulphur, and the United States takes a considerably larger tonnage than any other nation.

This is a brief description of an important industry. It requires considerable skill to keep costs down to the lowest limit, and to effect this object it requires well-trained technical men who possess sound knowledge of mineral chemistry and metallurgical requirements. Some years ago the ore was roasted in heaps, but such enormous quantities of sulphurous acid fume were evolved that the agricultural country for miles around the mines was devastated, and the Spanish Government insisted that the mining companies should gradually abandon roasting and adopt the natural weathering process that is now universal—an enlightened policy having regard to the magnitude of the industry locally.

The treatment of Spanish cupriferous pyrite exported direct to Great Britain and other European countries for sulphur and copper extraction. The composition of this ore is much the same as given under No. 1, except that it is richer in copper. This class of ore is first burned by the sulphuric acid manufacturer to extract its sulphur. This reversal of the order of operations as practised in Spain is an example of the influence of economics. In both cases the ore is essentially the same, but one or two per cent more copper permits of its direct shipment to centres where metallurgical facilities are greater, and where consequently a closer recovery can be made, the additional profit outweighing the freight. The burned ore is calcined with salt, and lixiviated with waters containing hydrochloric acid recovered from the furnace-gases by ordinary condensation. The residue, representing the bulk of the ore, is thereafter sold to the iron smelter. The liquors pass to vats where the silver and gold (present in very small proportions) are separated by means of soluble iodide, and then pass to vats where the copper is precipitated by wrought-iron scrap. The silver and copper precipitates are sold to smelters.

Perhaps I may be allowed to touch upon the Claudet iodide process used in conjunction with the ordinary Longmaid & Henderson wet copper process as practised on this class of ore, more especially as this process is now suffering a lingering death, and I have no interest in it, the patent having expired some years ago. I think I am correct in saying that no other chemical process has ever been able to make a profit out of an ore containing only 1 to 1½ oz. silver per ton and 3 gr. gold, as the Claudet process did some years ago. This was only possible on account of the conditions existing at the time. First, the price of silver was 5s. per oz. when the process was invented by my late father, Frederic Claudet; now the price is under 2s. 6d. per ounce. Second, it was only applicable to copper ores when treated by the Longmaid-Henderson process; these were practically limited to Spanish cupriferous pyrite. The use of this silver process does not interfere with the subsequent precipitation of the copper. In the Longmaid-Henderson process the roasted pyrite, called 'burnt ore', obtained from the sulphuric-acid makers and containing 3 to 4½% copper, is calcined with 10 to 15% rock salt, and the chloridized ore lixiviated with acid water; the resulting solution is then run into vats and the copper precipitated by iron. The strong chloride liquor held the silver chloride in solution, while silver iodide was found to be practically insoluble. The method adopted was to place a series of vats in front of the copper precipitation-vats, and separate the silver therein as insoluble argentic iodide by means of a soluble iodide; the liquor was allowed to settle, decanted, and passed to the copper precipitation-vats. The silver iodide was collected and reduced to spongy metal by means of zinc, the zinc iodide being used for a further precipitation of silver. About 1 oz. of silver and 2 or 3 gr. of gold per ton of original ore were saved, and large profits were made. Under normal conditions, about 10 to 15% of the iodide was lost. The only condition required in the iodide process is that the calcination of the burnt ore with salt needs careful regulation in order to prevent the formation of subchloride of copper, which, if formed in any considerable quantity, militates against the success of the process, cuprous iodide being produced with a loss in iodide. The silver residue, containing 500 to 15,000 oz. silver per ton, was smelted in the usual way.

Pyrite smelting.—In this case it is the metallurgist who comes first. The direct smelting of low-grade pyrite copper ore, perhaps containing silver and gold, which by any other process would be of no value, is now successfully conducted in various parts of the world. The object to be attained is to perform the smelting operation in such a way as by a first smelting to obtain a matte containing 20 to 25% copper, then by a second smelting to obtain a matte containing 45 to 50% copper, the slag from this operation being rich enough to add to the first smelting. This rich matte is then run into converters of the bessemer type, and by blowing air through the mass the combustion of the sulphur is effected, a blister copper containing over 90% of the copper originally present in the ore is obtained, and 95% of the silver and gold is collected. Pyrite smelting has the advantage of avoiding the first roasting operation, as it deals with the metallic sulphides direct, in place of their oxides, and by utilizing the calorific value of a large proportion of both the sulphur and the ferrous iron in the blast-furnace, permits of a great reduction in the consumption of coke, an expensive item in cupola smelting. That portion of the iron which is oxidized forms a base for the silica and thus reduces the consumption of extraneous flux. The portion of ferrous sulphide that remains unoxidized falls as matte, carrying the copper with it.

Pyrite smelting is also practised with pyrite gold and silver ores containing but little copper. Some proportion of the latter is, however, necessary to act as a collector of the precious metals, which the ferrous sulphide matte alone is incapable of doing completely. As small a quantity as ½ to ¼% copper may, in some cases, be sufficient to effect the collection of gold and silver into the matte. Pyrite smelting, in spite of its apparent simplicity, demands constant watchfulness on the part of the metallurgist to secure its maximum advantages. The final product is sold to the copper refiner and treated by electro-chemical means, separating copper of 99.9% and over, practically free from impurities, the silver and gold with the impurities remaining in the form of a mud, which is again sold to smelters for re-treatment.

The electrolytic refining of the blister copper is a task belonging to the chemist, but he plays an even more important part in the recovery of sulphur from the furnace-gases evolved in huge quantities during the smelting processes that yield the copper matte. This work has been forced on the chemist on account of the devastating effect of the sulphurous acid gases on vegetation. Agricultural land has suffered so much from their action that unless some method for their elimination from the air were found possible, pyrite smelting in most cases would have to cease, and large smelting works would be compelled to stop. The best way to suppress a nuisance is to turn the cause into a source of profit. During the last few years these deleterious gases have been turned to account, and large quantities of sulphuric acid are now made from them by the 'contact' process, and the intensive chamber process. In dealing with immense quantities of ore, as in pyrite smelting, every endeavor in the way of reducing working costs has to be made, and as the plant must be made and designed for as nearly automatic working as possible, the mechanical engineer takes a prominent part. I may mention here that the sulphuric acid is mainly used for the manufacture of superphosphates, the raw phosphates being shipped to the works from a long distance. Thus, instead of pyrite smelting being a curse to the agriculturist, it becomes a blessing.

The manufacture of copper sulphate.—At first sight this would seem to be a chemical process, pure and simple, but on looking into it, various metallurgical operations are found to play a part in the process. Copper sulphate is produced on a large scale, either by the direct treatment of smelted copper or as a by-product from some other process, such as in the refining of silver and gold bullion, where copper is used as a reagent, and has to be recovered.

As a direct process, the smelter in its operations produces copper bars, which may contain silver and gold; these are granulated and treated with sulphuric acid, and, with the aid of air, the copper is brought into solution, from which the copper sulphate is crystallized out, leaving an insoluble residue which is smelted for its silver and gold. An important factor in this process is that iron sulphate, the enemy of copper sulphate, is nearly absent from the copper sulphate crystals. As a by-product, this salt is obtained in the process of refining silver and gold bullion, a suitable alloy being made up by melting the bullion with silver 2¼ times the weight of gold and copper ¼ the weight. This mixture is granulated and boiled with sulphuric acid in suitable vessels; the gold remains as a mud, which is collected and melted into bars; the silver goes into solution and is precipitated by metallic copper, to be melted into silver bars. The liquor is then subjected to crystallization and yields copper sulphate, thus recovering in a marketable form the original copper added in the refining process to precipitate the silver.

The Pearce process of treating auriferous copper was the product of both chemist and metallurgist. The main feature in this process, which has now been replaced by later methods, was the discovery by Richard Pearce that a granulated alloy of copper and gold, when melted with iron pyrite, yielded some of its copper to form a matte, but the gold remained in the unattacked copper; by repeating the process a number of times the whole of the copper is separated from the gold. This method was worked in conjunction with the well known Swansea smelting and the Ziervogel processes, the latter of which is of a purely chemical nature.

WET METHODS OF COPPER EXTRACTION

(Editorial, September 21, 1912.)

Perhaps the most confidently expected advance in the art of metallurgy is the devising of a wet method for the extraction of copper from its ores upon a large scale and at a good profit. Legions of methods have already been devised, but none which meet the governing conditions of applicability upon a large scale, and yield of a satisfactory profit. How great a demand exists for such a method can be readily realized from the fact that in Utah 4,500,000 tons of material carrying 10 pounds of copper per ton is yearly escaping through the tailing launders of the copper concentrators, in Montana an equal amount of even richer material is similarly escaping, while in Nevada, Arizona, and other states the losses probably serve to bring the total up to 200,000,000 pounds of copper annually run to waste in the tailing from ore-dressing plants treating sulphide ores. Ore dressing is a highly developed art, but it is one that is not well adapted to extract small quantites of a valuable mineral from a large quantity of gangue unless the valuable mineral is free from the gangue and of much greater specific gravity, as in the case of gold-bearing gravel. Metallic ores are commonly intimately associated with the gangue, the difference in specific gravity is often not great, and, most unfortunately, ores are frequently so friable as to yield, during crushing, considerable amounts of material which is in so finely divided a state as to bid defiance to all ordinary ore-dressing methods. While it may be reasonable to hope that the losses in tailing may be decreased, for example, from 10 pounds per ton to 6 pounds per ton by improvement in ore-dressing methods, there can be little doubt that the ultimate 5 or 6 pounds of copper in the tailing from the concentration of sulphide ores will forever continue to resist all efforts for its recovery by mechanical concentration.

The ease with which a part of the copper in ores can be extracted by wet methods has stimulated efforts to discover a commercial method, and in a few cases success has been attained. But, as a general statement, it is safe to say that wet methods for the extraction of copper from its ores have not yet been successfully applied on a large scale. Some of the difficulties are those common to all wet processes, others are peculiar to the treatment of copper minerals. Before the cyanide process had reached its present development the hope of ultimate success was much more slender than now. The cyanide operator has largely solved the many mechanical problems incident to the handling of crushed ore and solutions, and their separation, and the metallurgist who wishes to treat copper ores by wet methods has, therefore, an accumulated fund of experience and achievement which forms an invaluable asset. Cyanidation experience cannot be applied *en bloc*, however, for the richest silver and gold-bearing material which has been treated by cyanidation carries only 85 ounces of precious metal per ton. Treating material worth $1.50 per ton carrying 160 ounces of metal offers a decidedly different problem from the treatment of material worth $500 per ton and carrying 85 ounces of metal.

The chemical problems involved in solution methods for the extraction of copper are the most baffling. The tailing previously mentioned consists largely or entirely of sulphides, and no feasible method has yet been devised for the treatment of raw sulphides. The material must first be roasted, and even where this can be accomplished for less than 50 cents per ton, it forms a heavy addition to the total cost of treatment of material worth only about $1.50 per ton. After roasting, the ore may be treated by solutions of alkaline cyanides, sulphites, sulphates, or chlorides. The alkaline cyanides have the advantage of extracting the precious metals present along with the copper, but the consumption and mechanical loss of cyanide, and the cost of the heat required in boiling out the ammoniacal copper solution has so far proved a bar to its successful use. Similarly, in the case of sulphurous acid and sulphites, the dilute solutions obtained and the difficulty in securing complete precipitation of the copper are serious disadvantages. The use of sulphuric acid or sulphates is perhaps the most promising of the chemical methods. It is not practicable here to go into the difficulties involved in the various special cases in which this method has been tried. But, speaking generally, the action of sulphuric acid on the bases other than copper present in the ore, the difficulties due to the formation of -ous and -ic compounds, the difficulty of securing complete solution and precipitation, and the whole group of problems centring around mass action, have so far prevented the profitable operation of the many processes which have been devised for the use of sulphuric acid or sulphates as a solvent for copper. The use of chlorides offers many of the same problems in accentuated form, since the other bases present are more easily attacked by chlorides than by sulphates. A mechanical difficulty of all such chemical processes not shared by the cyanide process is the attack by solutions upon the apparatus used. Iron and steel are the common materials of construction, and chemicals which will attack copper will more vigorously attack iron. Possibly some chemical will be found that, like potassium cyanide, is more effective in dilute solutions, but so far it has been found necessary to use strong solutions which are chemically active toward the vats and pumps necessary for treatment. None of these difficulties are insuperable, but each adds to the cost of treatment and the margin for profit is perilously small.

After the copper has been taken into solution, its precipitation and subsequent conversion into marketable form is a necessary operation that cannot be performed without cost. Elsewhere in this issue reference is made to the precipitation of copper from mine water at Butte by the use of scrap iron, and its precipitation at the Besshi mine with iron and coke. Pure pig iron costs 1¼ to 1½ cents per pound at Birmingham and Pittsburg, and its cost at a copper mine is enormously increased by freight charges. In the average case the cost of pig iron for precipitation of the copper would possibly be 4 to 6 cents per pound of copper produced, a heavy item in the cost of operation. Electrolytic precipitation offers advantages in remote places, and it is stated that it will be adopted in the leaching plant at the Braden mine, Chile. But there are many difficulties in the electrolytic percipitation of copper from dilute and often impure solutions, and a universal solution has not yet been found. The task of finding a profitable wet method is like that which faced the small boy sent to dig out a woodchuck, it must be accomplished. Able investigators are at work upon the problem in many laboratories and plants, and it is not too much to hope that a satisfactory solution will before long be found.

LEACHING COPPER ORES IN CHILE

(Editorial, June 21, 1913.)

The technological problems involved in the exploitation of the copper ores in the extensive deposits of the Chile Copper Company at Chuquicamata are of unusual interest. Some details concerning this property have already appeared in our columns. The new company has taken over a group of properties formerly worked on a modest scale by a number of European and Chilean companies, following exploratory work under the direction of Mr. Pope Yeatman, which has demonstrated the existence of 100,000,000 tons of ore averaging 2.48 per cent copper, and nearly double that amount of possible ore. Previous mining had been confined to shallow workings, in which the ore is of lower grade than in depth. The methods employed were exceedingly crude, consisting of mining the surface material, called *llampera,* in open pits and screenings. The copper mineral present has commonly been designated atacamite, $Cu(OH)Cl{\cdot}Cu(OH)_2$, but the careful study incident to the forming of a plan for working the deposit on a large scale has disclosed the interesting fact that the atacamite is present in only minor amount. the principal copper mineral present being brochantite, $CuSO_4{\cdot}3Cu(OH)_2$. Since a small amount of chalcanthite, the natural 'blue vitriol,' is also present, it is not astonishing that brochantite has been confused with atacamite, both being crystalline and emerald or dark green in color. The copper mineral being most friable, the fine screenings were comparatively rich and were shipped to smelters, while the coarse fragments were thrown upon the waste dump. Exploratory work upon a large scale disclosed an orebody over 8000 feet long and of variable width, up to 700 feet. The orebody is somewhat irregular and its exact shape and extent have not yet been thoroughly defined by drilling, but the ore has been proved to persist to depths as great as 1000 feet in places, changing in depths from brochantite to chalcocite, bornite, and chalcopyrite; about three-fourths of the ore developed so far consisting of brochantite, and one-fourth of sulphides. As might be expected, the brochantite ore tends to become richer near the zone of transition into sulphides. Small veins also occur at intervals throughout the main mass, giving rise to local enrichment, so that assays as high as 15 per cent copper are in places found, though the deposit as a whole averages slightly under 2½ per cent. The deposit is usually well adapted by nature to steam-shovel mining, since the ore outcrops in the crest of a low ridge some distance from the main range, so that the stripping is practically confined to the minimum of side slopes. A short branch from the main railway line from Antofagasta now reaches within one mile of the deposit, so that the physical conditions of exploitation are unusually favorable.

Careful study of the metallurgical problems involved has been made by Mr. E. A. Cappelen Smith. Solution of the copper content of the brochantite ore presents no difficulties, as that mineral is easily soluble in dilute sulphuric acid even when crushed to between ½ and ¼ inch size. The ore yields little or no slime in crushing, so that leaching is easy. An 8 or 9 per cent solution of sulphuric acid will be used. Since the copper is already present in the form of sulphate, its subsequent precipitation yields a larger amount of sulphuric acid than was used in its solution, so that there is no consumption of acid. Precipitation of the copper offers more difficulty. Dissociation of cuprous sulphate by the electric current yields free sulphuric acid at the anode, which liberates the chlorine present. This attacks the copper being deposited on the anode, leading to a considerable loss in electrical efficiency. Little trouble is experienced from ferrous sulphate, a source of difficulty in ordinary leaching, as the iron minerals, perhaps because of the coarseness of ore, seem not to be attacked by the solution used, and no increase in iron content, beyond 2 or 3 grains per litre, follows the repeated use of the solution. An additional difficulty at Chuquicamata is the nitric acid produced by the action of the sulphuric acid upon the nitrates which are present in small amounts on the surface deposits of this part of Chile. By the use of proper means, chlorine is eliminated from the solution, but this leads to an increased attack upon the anode. Details of the methods pursued to obviate these difficulties are not available, but plans for the construction of an 8000-ton plant are already well advanced, so that the experimental work, based upon several hundred tons of ore treated, must have proved correspondingly satisfactory. There is practically no gold and silver present in the ore, and arsenic, antimony, and bismuth are fortunately also absent. As a result, the solutions from leaching do not become foul through repeated use. This is indeed a fortunate combination of circumstances which greatly facilitates the work of leaching in a country of limited water supply. All stages of the treatment, as far as the electric precipitation, can thus be done at unusually low cost, so that the comparatively expensive precipitation is not expected to bring the total cost of production above 6 cents per pound of copper. Electric power for the precipitation of copper will be furnished by an oil-fired power-plant at the sea coast, electric power costing about $60 per horse-power per year.

*Further details are now available concerning the huge metallurgical plant to be erected to treat copper ore by leaching at Chuquicamata, as discussed at some length in our editorial columns of June 21. The ore, which is mined by steam-shovels, will be subjected to a preliminary crushing in gyratories, crushers, and rolls, and then distributed, by belts, into concrete vats, lined with acid-proof materials and holding 9000 tons each. It is estimated that one day will suffice for filling a vat, two to three days for acid treatment and washing, and one day for discharging, each vat holding one day s ore supply. The extraction process might perhaps be more accurately termed soaking, rather than leaching, since it is not necessary to cause the solvent to percolate through the ore to the same degree as is required in cyaniding gold ores. The pregnant solution will be drawn off into tanks, preliminary to electrical precipitation. The leached residue will be excavated from the vats by clam-shell buckets operated from a traveling bridge, somewhat resembling the system used in reloading coal from stockpiles, and the residue will be piled upon the waste dump by a series of belt conveyors. The plant is to be built upon a side-hill having a 6° slope and of sufficient extent to provide ample storage room for the great quantity of tailing which will in time be produced. The contract for the construction of the electrical power-plant and transmission line has already been let to the lowest bidder, a German firm, for approximately $3,000,000; the equipment including a 40,000-kilowatt, oil-fired power-plant on the seacoast, a 100-mile transmission line to the mines, and the necessary auxiliary sub-stations and transformers. It is hoped to have the plant in full operation within three years, but construction work on so large a scale at so great a distance from headquarters may easily involve unexpected delays.

†Leaching work at the Braden mine has been held somewhat in abeyance pending the solution of the problems connected with concentration of the ore; the mine, concentrator, and flotation plant

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*July 12, 1913. See also article by E. A. C. Smith, May 2, 1914.

†June 21, 1913.

having, up to the present, taxed the capacity of the hydro-electric plant upon the Cachapoal river. Recently another unit was added to the power plant, and it is stated that a test run will soon be made with the leaching plant, a brief description of this plant by Mr. Pope Yeatman, appeared in the *Mining and Scientific Press* of December 16, 1911. The process to be employed consists of roasting the concentrate from milling operations in Wedge furnaces and utilizing the escaping SO_2 gas to manufacture in lead chambers, the H_2SO_4 to be employed in leaching the calcine. The copper sulphate solution resulting from the leaching will then be electrolyzed. It is hoped in this way to produce cathode copper at a very low cost, since the cost of the hydro-electric power available is low. Only a part of the concentrate will be thus treated at first, though there is a possibility that similar treatment may be applied to the ore. Later in the year we hope to be able to present more data concerning the interesting work in progress at this and the Chuquicamata.

SULPHURIC ACID LEACHING

(Editorial, August 16, 1913.)

By one of the frequent vagaries of circumstance it is with sulphuric acid leaching, which has been tried times without number with at best but limited success and most commonly with no success at all, that much of the most promising experimental work on the extraction of copper from its ores by wet methods is being done. On the face of it, this method seems the simple, easy, and obvious means to employ. It is one of the classic theorems of economic geology that iron and copper sulphides oxidize an exposure to the air, yielding free sulphuric acid which takes the copper into solution as sulphate and carries it away to take part in other reactions, possibly the forming of a rich copper orebody. At Rio Tinto this natural reaction is utilized in a metallurgical process for the recovery of copper, and from time immemorial it has been customary to recover copper from underground waters by causing them to flow over scrap iron. Except under special circumstances the use of sulphuric acid to dissolve copper from its ores, subsequently precipitating the copper from the sulphate solution, has miserably failed to meet that inflexible definition of metallurgy as the art of making money out of ores. In attempting to make sulphuric acid the basis of a general process, such as the use of cyanide solution for the recovery of gold, a host of technical difficulties are met, many of which are apparently insuperable, that have so far, except under special circumstances, combined to prevent the successful use of this method. So much buncombe regarding leaching processes for copper is to be heard or seen that it will be well worth while to review the facts.

The first difficulty with sulphuric acid as a solvent for copper ores is that it will not attack metallic copper, fresh unaltered sulphides, or chrysocolla. The last is somewhat in doubt, as apparently the acid will attack some varieties of chrysocolla, though it is maintained, with some show of reason, that these are not true chrysocolla, but an intimate mixture of a large quantity of malachite with a small quantity of chrysocolla. Cuprite also is not soluble in sulphuric acid, though the oxide of copper formed in roasting sulphides is soluble. Now as a matter of practical fact, in dealing with a copper orebody of profitable size it is usually necessary to deal with the whole range of copper minerals, oxides, carbonates, silicate, metallic copper, and sulphides ranging from chalcocite to lean chalcopyrite. To attempt to use sulphuric acid on such a melange as this is like trying to put salt on the collective tail of Noah's ark. Where nature has been especially kind the use of sulphuric acid is feasible. Thus at Chuquicamata the copper occurs as a basic sulphate which is easily soluble in acid; at other places, north of Butte, for example, considerable bodies of carbonate ore occur and the amount of insoluble copper mineral present may be small enough to yield a working profit. At times an ore may be concentrated in the ordinary way, and leaching applied to the tailing, which is largely composed of oxidized minerals, as at the Arizona Copper mill at Clifton. But as a general thing the presence of metallic copper or unaltered sulphides will make ordinary leaching with sulphuric acid unprofitable.

Assuming that it is practicable to dissolve with H_2SO_4 the copper present in the ore, a fresh crop of difficulties is encountered. The acid not only dissolves the copper, but a variety of other substances present in the ore as well. Thus clayey minerals will be attacked yielding aluminum sulphate, and iron oxides will be in part converted into ferrous sulphate. These and other substances remain in solution when the copper is subsequently precipitated and accumulate to such an extent that the solution becomes so foul that it must, in part at least, be thrown away. Sulphuric acid is now worth about $7 per ton at the copper and zinc smelters where it is being produced; at more remote places its cost may considerably exceed this figure. It is easily seen that the cost per pound of copper for the sulphuric acid thus consumed is an important item of working cost. Various schemes have been tried to prevent this loss of acid, but no general remedy has yet been devised. The iron thus dissolved is a source of further trouble in case it is desired to precipitate the copper from the solution by means of electrolysis. When the current is passed through the solution, copper is precipitated and SO_3 ions are released which oxidize the ferrous sulphate present to the ferric state, and the ferric sulphate promptly redissolves the copper, reducing itself to the ferrous state again. The ultimate result is much like that of a puppy chasing its tail, and converts the electric energy into useless heat. Some promising work is being done on the use of porous diaphragms to prevent the ferric sulphate reaching the cathode; but while effective these increase the resistance of the cell and merely alter the difficulty without solving it, though it is possible that a satisfactory solution may be found. Another troublesome difficulty in electric precipitation is the securing of a satisfactory substance for an anode. The best that has been found is lead, but even when this is used it is slowly converted into the peroxide of lead by the oxidizing reaction mentioned above and is gradually destroyed. The peroxide can be collected and reduced to metallic lead, but this operation forms another item of working cost.

It is possible to reduce the copper from the solution by causing it to pass over metallic iron. Theoretically, 56 pounds of iron will thus reduce 63 pounds of copper, but in practice this is greatly exceeded, for a certain amount of free sulphuric acid is present and dissolves iron, thus at once consuming iron and sulphuric acid. Pig iron is not very satisfactory for precipitation, since it contains phosphorus, sulphur, silicon, and carbon, and is, in addition, in the form of pigs which present only a small active surface in proportion to their mass. Scrap iron and steel are used for this purpose in small way, but a large plant would quickly exhaust the scrap heaps of the world. It is usual to reckon that three pounds of iron is required to thus precipitate one pound of copper, though much better records are made in many cases, and on this basis the cost of precipitation alone would be 3 cents per pound of copper. Some interesting work has been done on the roasting of pyrite to yield H_2SO_4 and the reduction of the resulting porous iron oxide to metallic iron in a reverberatory furnace, but even by this method the cost is still heavy.

Collateral with these fundamental difficulties are a variety of mechanical and chemical problems which vary according to local conditions. If the ore is ground fine the action of the acid on the gangue minerals is greatly increased and in addition colloidal material is produced which renders difficult the separation of the dissolved copper from the residue. It is reported that in the leaching work at Butte only 80% of the copper is obtained in the solution, the rest, though apparently dissolved, remaining with the ore. A muddy solution is obtained which leads to the inclusion of floating particles in the precipitated copper. On the other hand, if the ore is coarsely crushed these difficulties are avoided, but the copper may not be dissolved by the acid. But enough has perhaps been said to indicate clearly that sulphuric acid leaching, while applicable where conditions favor its use, has not yet been sufficiently developed to play any such part in the metallurgy of copper as cyanide does in that of gold.

THE LEACHING OF COPPER ORES

(August 23, 1913.)

By John Rooke-Cowell

The bodies of low-grade and, for the most part, oxidized copper ores which occur in the southwestern part of the United States, in Canada, Mexico, Central and South America, Australia, Africa, and in fact throughout the world, which, owing either to their inaccessibility, metallurgical condition, or their low copper content, cannot be profitably treated by smelting methods, and the huge accumulations of tailing from the great concentrating plants of the West, in which millions of pounds of copper is being lost every year, have turned the attention of metallurgists to the investigation of hydro-metallurgical or electro-chemical processes by which this copper may perhaps be profitably recovered at a small cost and added to the world's supply.

At the mines at Rio Tinto the leaching of copper ores has for many years been successfully accomplished by a somewhat complicated and very lengthy process, by which the copper in the ore is converted into soluble copper sulphate by natural oxidizing agencies of the air and moisture and the copper precipitated from the solutions on iron. This process is too lengthy and entails the tying up of too much capital to find much favor under present commercial conditions. At Clifton, Arizona, the low-grade oxidized ores of copper have been for some years past leached with sulphuric acid manufactured on the ground at a small cost from a nearby deposit of pyrite, and the copper precipitated on scrap wrought iron and old tin cans.

Although the copper produced by this method is of inferior quality, this is the simplest and, where possible, the ideal method of leaching copper ores, but it is only applicable to oxidized ores and cannot be used to advantage in localities where sulphuric acid and iron are not obtainable at a moderate cost. The problem, therefore, which now confronts the investigator, and which during the past decade has given rise to half a hundred processes, most of which are now happily defunct, is to find a method suitable to both sulphide and oxidized ores of copper, by which the solvent may be regenerated at a small cost and the heavy expense of precipitation by means of iron eliminated.

With the thorough knowledge now possessed regarding the chemical reactions which occur among the compounds of copper, among themselves and also with relation to the known copper solvents and the associated minerals in the different ores, and the amount of accumulated data regarding the electro-deposition of copper from leaching solutions, and with the many improved mechanical devices at our disposal, the invention of such a process, with a fairly universal application, should not present any great difficulty. Yet the costly failures of many processes within the past few years have created a certain atmosphere of skepticism, and the inventor of a new process is likely to be regarded in some quarters as a harmless crank, or as having sinister designs on the pocket of the investor.

Ammonia has been used as a solvent for certain oxidized copper ores, the copper being subsequently recovered as cupric oxide by simply heating the solution, but mechanical difficulties, the loss of ammonia during its operation, and the high cost of this solvent, have limited the usefulness of the process. Sulphurous acid, made by dissolving sulphur dioxide in water, has also been used as a solvent, the copper being recovered as cuprous oxide by heating the solution. In this, as in the ammonia process, the solvent is simultaneously regenerated with the precipitation of the copper. Sulphurous acid is an excellent solvent of copper ores, especially as it does not attack the other metals present with the same avidity as many other solvents. Only mechanical difficulties and the necessity for using extremely dilute solutions have prevented the successful application of the process. Sulphuric acid has been generally found to be the best and most convenient solvent for copper ores, while ferric sulphate and chloride, which have been found to readily attack certain chalcocite ores and act with less energy on chalcopyrite, have also been successfully used. In certain processes the copper in the ore is converted into a soluble sulphate by a careful roasting at a low temperature, and then leached with water; in others, a soluble chloride of copper is obtained by roasting the ore with common salt or passing free chlorine, obtained by the electrolysis, of a solution of common salt, over the ore while it is being roasted. Sulphide ores for the most part require roasting before they can be successfully leached, but modern large-capacity mechanical roasting furnaces enable this to be accomplished at a low cost.

There appears to be no great difficulty is dissolving the copper out of the ore, though cuprous oxide is not soluble unless heat is applied, and chrysocolla (the silicate) is practically insoluble. The ore does not require fine crushing, as is the case in the ores of gold and silver; about 10-mesh will generally be found sufficiently fine, while in some cases a reduction of the ore to pieces about one inch in size, and even larger, will be all that is necessary to obtain a good extraction of the copper. This naturally reduces the cost of crushing and does away with the difficulties of percolation encountered in other processes, as practically no slime is formed.

The action of the solvents on copper ores is increased by the application of heat to the ore charge, is extremely rapid, and the extraction of the copper from properly prepared copper ores is usually high. The numerous mechanical devices which are now available, such as stoneware and hard-rubber pumps for handling acid solutions, lead-lined pipes, and the several excellent non-corrosive compounds which are now on the market for protecting the tanks and thus avoiding the necessity for using an expensive lead lining for the tanks, have done a great deal toward permitting the successful leaching of copper from its ores. In fact, it may be considered that most of the difficulties encountered in the actual leaching of copper ores have now been surmounted, and the others relegated to the precipitation of the copper from the solutions.

With the exception of the ammonia and sulphurous acid processes in which the copper is recovered as a high-grade oxide of copper, or those processes where the production of copper sulphate is the ultimate object, and neglecting the precipitation of copper by means of iron as generally too expensive, the only practicable methods in use for the recovery of the copper from its solutions are either chemically by the use of hydrogen sulphide, which is passed into the solution and precipitates the copper as copper sulphide, or electrolytically, by which the copper is obtained as cathodes of pure copper. In the former method the precipitation of the copper is rapid and complete, but at the same time the other metals of the acid group, such as arsenic, antimony, and bismuth, are precipitated with the copper, while other metals which are not precipitated in an acid solution, such as iron, continue to accumulate in the solution until it becomes so saturated and foul that it has to be thrown away and the whole of the contained acid lost. In any case, only the acid in combination with the precipitated metals will be regenerated, and so a constant addition of part of the acid at least will always be necessary. In some places the manufacture of hydrogen sulphide would be extremely costly.

In the electro-deposition of copper from leaching solutions many difficulties have been encountered. They are partly due to the fouling of the electrolyte, owing to the accumulation of the sulphates of iron and other metals which make it necessary to periodically send some of the solution to waste, thereby causing a loss of all the acid; to the difficulty of finding a suitable anode for sulphate solutions other than lead, which is at present used but is rapidly oxidized; and to the reactions which occur among the foreign metals present in the electrolyte. Although a complete regeneration of all the acid is effected, and in fact when sulphur dioxide is injected into the electrolyte as a depolarizer, an excess of acid is obtained by its combination with the nascent oxygen liberated at the anode, so many reactions and counter-reactions occur among the foreign metals present in the electrolyte, and so much difficulty has been encountered in properly regulating the current density, that a great deal of electical energy is wasted and the consumption of current is greatly in excess of that required for the actual electro-deposition of the copper. Theoretically, the electrical efficiency of the deposition of copper from sulphate solutions is 2.14 lb. per kilowatt-hour. In practice, from leaching solutions, only about one pound per kilowatt-hour, or about 50% of the theoretical efficiency, has hitherto been obtained. With a pure copper sulphate electrolyte, free from other metals, and the proper precautions taken to prevent the impoverishment of the copper ions around the cathode, there is no reason why at least 90% of the electrical efficiency, or about two pounds of copper per kilowatt-hour, should not be obtained. The difference between this and the theoretical is due to the resistance of the conductors. This would mean a reduction in the cost of the electrical installation and of the electro-deposition of the copper of about 50%, which would be of considerable importance in a large plant.

Such a condition could be obtained if the electrolyte were freed from the presence of foreign and interfering metals, by chemical means, before being introduced into the electrolysis, instead, as is the practice at present, of precipitating the foreign metals when the electrolyte has become thoroughly saturated and foul and a great loss of electrical energy has already occurred. The great advantage of recovering the copper by electrolytic means is the complete regeneration of the solvent and the production of pure copper which can be readily marketed. The first cost of such an installation, especially if a high degree of electrical efficiency can be obtained, is not prohibitively high, as many people imagine, and the cost of precipitating the copper, especially where water-power is available, will in very few cases exceed one cent per pound. There seems every reason to believe that electrolysis will in time take the place of all other methods for the precipitation of copper from leaching solutions. Roasting, by rendering insoluble in acids many foreign substances which in the crude unroasted ore would be soluble, and by converting insoluble into soluble copper compounds, has been found beneficial before leaching. Yet in most cases where both the sulphides and oxides of copper occur in the same ore a preliminary concentration of the sulphides and the leaching of the oxidized tailing and roasted concentrate will prove most economical. Ferric salts, which may be generated during the operation of the process, will dissolve part at least of the small amount of sulphides which escape concentration. Ores containing much lime are, for the most part, unsuitable for an acid-leaching process on account of the excessive consumption of the solvent, but in certain cases this has been overcome by roasting the ore at a low temperature and converting the lime into an insoluble sulphate.

So thoroughly have the chemical reactions of the various copper compounds been investigated during the past few years that the process which will have the most general application in the treatment of low-grade copper ores is not likely to depend on the discovery of some new solvent or be a panacea for the treatment of all low-grade copper ores. It will comprise a simple, scientific, and economical arrangement of certain well known and thoroughly understood chemical and electrolytic reactions, and form a process capable of modification so as to meet individual metallurgical and economical conditions. By taking advantage of modern labor-saving devices, and, os far as possible, eliminating the cost of chemicals, it will enable copper to be produced at a price which will prove profitable under almost all possible market conditions. Such a process would not only add considerably to the world's supply of copper by enabling the low-grade deposits and old tailing dumps to be profitably treated, but would prove a great boon to the small mine-owner, who is at present between the Scylla and Charybdis of courting almost certain disaster by erecting his own smelter, or of sacrificing a large part of his legitimate profits by selling his ore to customs smelters. There is no reason that such a process should not be used in the treatment of high-grade as well as low-grade copper ores, especially if freight rates to the smelter are high, and its introduction would lead to increased activity in all classes of copper-mining.

THE IRVING LEACHING PROCESS

By L. S. Austin

(January 3, 1914.)

Joseph Irving, of Salt Lake City, has patented a process* for the leaching of oxidized and sulphide ores of copper containing gold and silver. Essentially it consists in crushing mixed carbonates, oxidized, or sulphide ores of copper to 12 or finer mesh. The crushed ore is subjected in leaching vats to the action of the mother liquor containing ferric sulphate from a prior leaching, to which has been added some sulphuric acid and common salt. The ore and solution together are mechanically agitated and heated by injecting steam. Upon conclusion of the agitation the mixture is allowed to settle. It is then drained and washed, yielding a copper-bearing solution, which is passed through a filter consisting of sand and fresh iron pyrite, and then through the precipitation vats. Recovery of the metals may be accomplished by electrolysis or by passing solution over scrap iron. The remaining solution can be regenerated for further use; it is oxidized by a steam jet which agitates it violently, bringing it in contact with air. The solution is then reinforced with sulphuric acid and salt, and is ready to be again used for a fresh charge.

It would appear that the free sulphuric acid reacting on copper carbonate would give cuprous sulphate which would decompose the salt as follows:

$$(1) \qquad CuSO_4 + 2NaCl = Na_2SO_4 + CuCl_2$$

and the copper chloride, acting upon silver sulphide, would decompose it:

$$(2) \qquad Ag_2S + CuCl_2 = CuS + 2AgCl$$

In presence of salt in the solution the silver chloride dissolves. Gold would be acted on in the same way.

Ferric sulphate, acting on chalcocite, changes it to copper sulphate thus:

$$(3) \qquad Fe_2(SO_4)_3 + Cu_2S = CuSO_4 + 2FeSO_4 + CuS$$

and the resultant CuS, as well as that in the same condition in the ore, would be changed as follows:

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*U. S. patent 1048541, December 31, 1912, *Salt Lake Mining Review*, XV, 17; *Met. & Chem. Eng.*, XI, 160.

(4) $Fe_2(SO_4)_3 + CuS + 3O + H_2O = CuSO_4 + 2FeSO_4 + H_2SO_4$

When the ore has been leached, the resultant solution, before going to the scrap-iron boxes for precipitation, must have the ferric sulphate changed to the ferrous form, otherwise the consumption of scrap iron will be largely increased. This is done, as already specified, by running the solution through a filter containing fresh pyrite.

(5) $7Fe_2(SO_4)_3 + FeS_2 + 8H_2O = 15FeSO_4 + 8H_2SO_4$

This ferric is changed to ferrous sulphate and sulphuric acid is regenerated.

After the copper, gold, and silver have been precipitated the barren liquor, being agitated by a steam jet, its contained ferrous sulphate is changed to ferric sulphate.

Fig. 49 is a plan of a 50-ton mill, designed for the effective carrying out of the process along lines suggested by the long experience of the inventor in the hydrometallurgy of copper.

The ore, after coarse crushing, goes to the 100-ton storage bin *B*, whence it is fed regularly to a Hardinge mill, being crushed in the iron sulphate solution flowing from the iron sulphate storage tank. The leaching vats *E*, *E′* are alternately filled with the pulp, being at the same time agitated by a four-arm stirrer. When two-thirds full the proper quantity of sulphuric acid is run in from the acid storage tank or acid-bearing liquor from the dilute copper sulphate storage tank *D*. When a vat has been filled with pulp, the stirrer is stopped and the contents allowed to settle. The supernatant liquor is now decanted, going to the strong liquor storage tank *J*. The pulp left in the leaching vat is drawn off to the cone-classifiers *F* and *G*. In these classification is effected by aid of a rising current of wash-water, while the underflow from both cones goes to the concentrating tables in case it contains heavy sand of value, otherwise it is wasted. From the concentrating tables the tailing, if worth while, can be pumped back to the Hardinge mill for further treatment. The overflow from the second cone *G* goes to the pulp storage tank *H*. From this, as much of the clear liquor as possible is decanted before the remaining slime is withdrawn at the bottom to one or other of the filter-tanks *I* and *I′*. The filter-tanks have false bottoms of porous tiles which will be unaffected by the acid liquid. The filtrate from either tank is withdrawn to either of the liquor storage tanks. One of the tanks is for the stronger, the other is for the weaker solution. From these the solution is delivered in regulated flow to the electrolytic-deposition boxes *K*, *K*, *K*, *K*, where most of the copper is precipitated electrolytically. The partly impoverished liquor then goes to a sump-

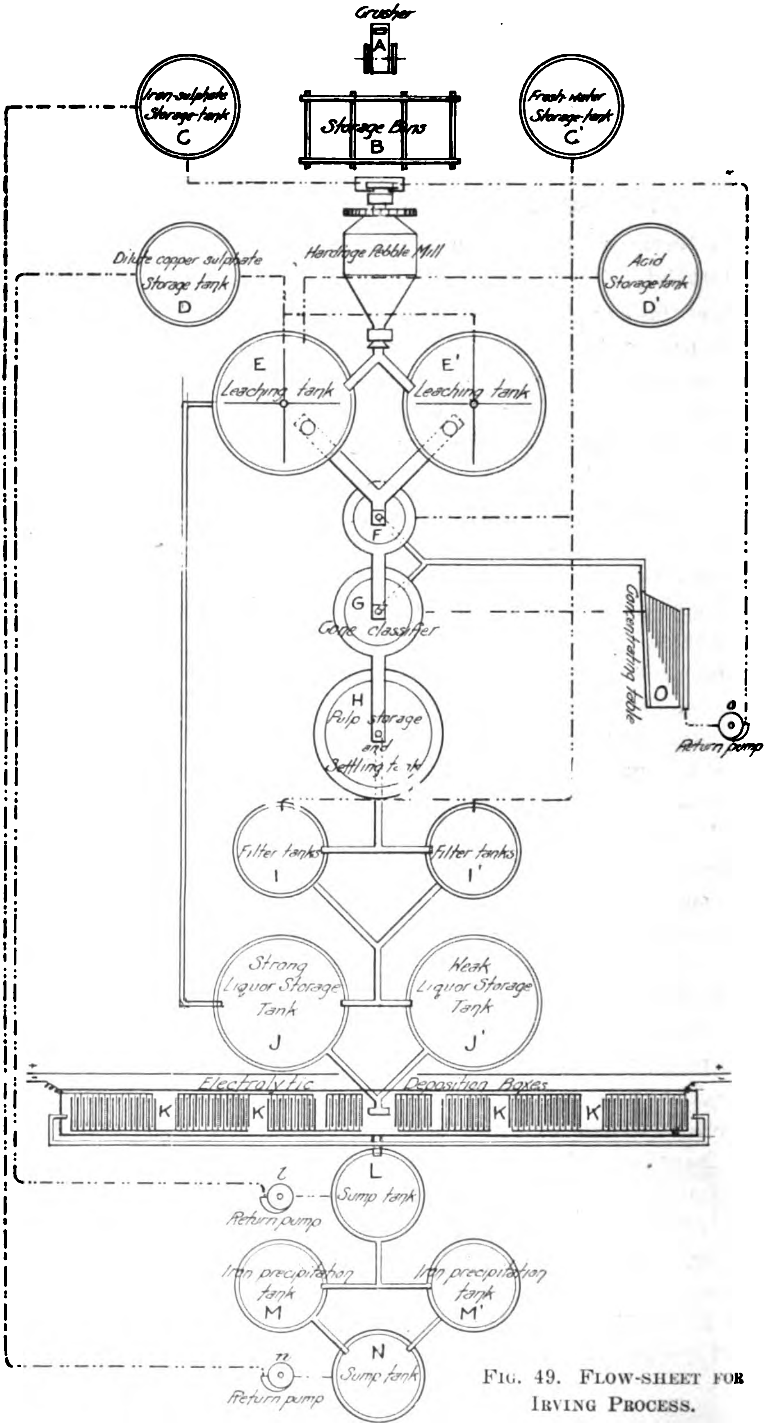

FIG. 49. FLOW-SHEET FOR IRVING PROCESS.

tank *L* and to the iron precipitation tanks *M*, *M'*, where the remaining copper is recovered by scrap iron; or the solution at *L* containing a little copper, may be sent back by a centrifugal pump to the copper sulphate storage tank *D* to be again used. The barren solution in the second sump tank *N* is also sent back by a centrifugal pump to the high level iron sulphate tank *C* and is there agitated by blowing in steam, whereby the ferrous sulphate is changed into ferric sulphate, and is then ready for use on a fresh charge. The cycle of operations for crushing, filling tank, leaching, and washing contents, and finally emptying tank, will average 48 hours.

A number of laboratory tests in 1 to 5-lb. lots have shown an extraction of copper varying from 77 to 99% and with an average of 91.4%; while several ores which had appreciable amounts of gold and silver with the copper were also found amenable; for example:

		Gold.	Silver.	Copper.
Utah ore:	Original	0.06	0.56	1.60
	Tailing	0.01	0.09	0.13

This amounts to an extraction of 75% of the gold, 83% of the silver, and 92% of the copper.

On a larger scale, using 700 lb. of copper ore containing 1.32%, an extraction of 79% was made; on another lot of 1500 lb. containing 1% copper there was a recovery of 74%. In precipitating the copper there was consumed 1.28 lb. iron per pound of copper recovered.

On these 50-lb. lots, some Utah copper ore, containing Cu 0.81%, together with 0.145 oz. Au, and 5.06 oz. Ag per ton, gave, as the result of three tests, an average extraction of 56% of the copper, 41% of the gold, and 60% of the silver with a copper precipitate containing 68.52% copper with 1 oz. of gold and 189 oz. of silver per ton.

On an operating scale, March 1913, on some Nevada ore a charge of 23.19 tons, containing 4.65% copper, gave the following results using decantation and leaching, the ore having been crushed in the returned barren solution; there was actually extracted by the solution 93% of the copper. Precipitation was slow because all the iron was new. There was used 1603 lb. of acid; the milling time was 9½ hours.

On another charge, 30 tons dry weight, March 1913, the ore contained 2.86% copper. The ore was crushed in the barren solution, and after settling 12 hours the supernatant solution was decanted. Acid was then added and the whole mass was stirred and allowed to settle. The decanted solution contained 1.57% copper. The total copper dissolved was 1610.66 lb., equal to a 94% extraction and 3200 lb. of acid was used. This charge contained a very large portion of copper glance (chalcocite). It is interesting to note that ores containing copper glance leach slowly.

On copper carbonates and oxides, and with acid quoted at an average price of $25 per ton, the acid consumption would be 2c. and average treatment costs 4 to 5c. per pound of copper. On heavy sulphide ores acid consumption would be 3c. and average treatment costs 5 to 6c. per pound of copper. It is estimated that a crushing and leaching plant to treat 100 tons per day would cost $35,000. This includes building and equipment complete. Scrap iron delivered would cost $15 per ton. The consumption of iron is placed at not to exceed 1.5 lb. per pound of copper recovered and of acid not to exceed 2 lb. per pound of copper, and it may be considerably less than that.

THE SLATER LEACHING PROCESS FOR COPPER ORES

By H. W. Morse

(January 24, 1914.)

Among the new processes for the extraction of copper from its ores in the wet way, there is one of much interest, chemically as well as technically. This is the cyclic hypochlorous acid process of H. B. Slater, of Riverside, California. The chemical reactions involved are simple enough fundamentally, but they are sufficiently unusual to be worth careful consideration and some study. Inasmuch as the process is a cyclic one, it will perhaps be best to describe the operations and reactions in a series of steps. The completeness of the cycle can then be checked from point to point of the process.

The leaching liquor comes from the anode compartment of an electrolytic cell of ordinary type. A 15% solution of common salt is used here, and in it there is suspended ferric hydroxide (produced at the third step), the quantity used depending on the copper content of the ore to be treated. Electrolyzing, the reaction is:

$$Fe(OH)_3 + 6Cl = FeCl_3 + 3HClO$$

half of the chlorine going to form ferric chloride and half to form hypochlorous acid. Electrolysis is continued until only a small amount of suspended hydroxide is left. This means that there is but very little free chlorine in the solution; and it means, in practice, that very little chlorine escapes into the air. For an ore containing 2 to 2.5% copper, the lixiviant will contain 5 to 7 gm. per litre of iron as ferric chloride, and a corresponding amount of hypochlorous acid. In the cathode compartment there is formed during electrolysis sodium hydroxide, in amount equivalent to the quantity of chlorine produced. This is used at a later stage of the process.

Ore ground to 40 mesh is easily treated. If chalcocite be assumed to be the chief copper mineral present, the leaching reaction, as far as the hypochlorous acid is concerned, is:

$$Cu_2S + 4HClO = 2CuCl_2 + 2H_2O + SO_2 \text{ (or, } S + O_2\text{)}$$

as a part of the sulphur is freed in elementary form. The SO_2 produced in this reaction reduces part of the cupric chloride to cuprous chloride:

$$2CuCl_2 + 2H_2O + SO_2 = 2CuCl + 2HCl + H_2SO_4$$

so that even at this point in the process a considerable portion of the extracted copper is present as cuprous chloride, held in solution in the sodium chloride solution.

At this point the leaching liquor contains ferric chloride, ferrous chloride, cupric chloride, and cuprous chloride, all in salt solution. To it there is added sufficient of the sodium hydroxide solution, from the cathode compartment of the cell, to completely precipitate all of the iron as hydroxide, according to the reactions:

$$FeCl_3 + 3NaOH = Fe(OH)_3 + 3NaCl \quad \text{and}$$

$$FeCl_2 + CuCl_2 + 3NaOH = Fe(OH)_3 + CuCl + 3NaCl$$

This last reaction is an interesting one chemically, especially because cupric chloride acts here as an oxidizing agent, raising ferrous to ferric iron. This reversal of the usual order of things takes place because of the complete and immediate removal of the ferric iron as hydroxide as fast as oxidation takes place. A slight excess of the sodium hydroxide causes no precipitation of copper as hydroxide, for the copper is all locked up in the complex (perfectly colorless) form with the sodium chloride, and the concentration of ferrous iron is so slight that a considerable excess of hydroxide is required to produce a precipitate.

The combined reduction effect of the ferrous chloride at this stage and the sulphur dioxide in the previous one results in the complete reduction of all the copper to the cuprous form. This is of practical importance, since it is only necessary to supply a single equivalent of chlorine for each copper molecule instead of two equivalents, which would be needed if the copper were to be extracted as cupric chloride. As a matter of fact, it is in practice only necessary to supply a very slight excess of chlorine in the leach over that calculated for the cuprous chloride equivalent.

The copper can, of course, be recovered by the usual methods, or it can be precipitated with the aid of the sodium hydroxide as cuprous hydroxide. If this precipitation is made with hot sodium hydroxide solution, air oxidation is rapid and the ordinary black oxide of copper is formed. Probably the reaction: $2CuOH + O = 2CuO + H_2O$ will express this step. The black oxide so formed might well be cast into blocks and used as cathodes in the electrolytic cell giving metallic copper, and resulting in a noticeable decrease in the working voltage of the cell.

It is evident that the process is completely cyclic chemically. No chemicals are brought in from outside except the salt, which may be lost in the final wash waters, and this can be reduced to as low a point as may be desirable in practical operation.

The following points are worthy of special consideration: (1) Ores of low total sulphur content can be treated. (2) The ore need not be roasted. The only reason for roasting any part of the ore treated would be to bring up the iron content of the leach in order to produce sufficient ferric hydroxide to restore the original concentration for the next leach. This might be necessary in the case of an ore containing much lime. (3) If the ore is not roasted, practically none of its iron is leached out with the copper. (4) The materials used are produced in a single operation in a single electrolytic cell, except the salt lost in the final wash waters. (6) The sodium hydroxide produced in the cathode compartment is completely utilized in separating the iron from the copper and in the subsequent precipitation of the copper. (7) The precipitated ferric hydroxide separated in (6) is used in the preparation of the succeeding lixiviant. (8) Extraction is very rapid and complete. With agitation, using a 40-mesh ore, 95% extraction can be had in four to six hours. (9) Copper produced by this process is extremely pure. None of the other metals can follow it into its complex salt with sodium chloride, and they are almost completely removed with the iron by precipitation as hydroxides. For the 'porphyry ores', so called, and for non-roasting ores in general, the Slater process appears to combine nearly the maximum number of advantageous features with but very few objectionable ones.

THE WEIDLEIN LEACHING PROCESS

(April 4, 1914.)

For some time past E. R. Weidlein has been studying the application of leaching processes to the treatment of copper ores, working in the Mellon Research Laboratories of the University of Pittsburgh, under the auspices of the Metals Research Co., which is a subsidiary of the W. B. Thompson interests. Mr. Weidlein has devised a process, for which U. S. patent No. 1,089,096 was issued on March 3, and is now engaged in the construction of a small plant at Wabuska, Nevada, to test the process on a working scale under actual operating conditions. In the patent specifications the process is described as below.

The present invention relates to the recovery of copper in the wet way, from its ores, and is based generally upon the reversible reaction

$$CuSO_4 + SO_2 + 2H_2O = Cu + 2H_2SO_4$$

More specifically, it contemplates a mode of operation which makes the utilization of this reaction of high commercial value, in that in actual practice the copper can be precipitated substantially quantitatively in the metallic form. This substantially complete precipitation is likewise accompanied with the regeneration of twice the amount of sulphuric acid necessary to dissolve out the same quantity of copper precipitated, so that the solution, while it is still hot, and after the copper has been precipitated out, may be used for re-leaching the original body of ore, or for leaching a new batch of ore, as the case may be.

In the preferred practice of the invention, the ore is leached, by percolation, or by agitation with or without the admission of air, with a 3.6% sulphuric acid solution. The free acid content of the copper sulphate solution thus obtained is lowered during the leaching operation to less than 1% of free sulphuric acid. The free acid is thereupon neutralized by adding the calculated amount of calcium carbonate or ordinary limestone, or, in fact, any alkaline substance. If the alkaline substance added is one which forms an insoluble sulphate, as is the case when calcium carbonate is employed, the precipitated insoluble sulphate is separated by filtration or otherwise removed in any known way.

The copper sulphate solution will be neutral and will contain 1.5% copper, in which condition it is supplied to the absorption tower, of any suitable or convenient type. Sulphur dioxide (for instance, sulphur dioxide contained in smelter fume) will thereupon be pumped into the solution until the desired concentration of (1.5%) sulphur dioxide for the amount of copper in solution is obtained. The solution is then pumped to the precipitation tank or tanks, which may be made of iron, lead-lined throughout, and which are adapted to withstand, with safety, a relatively high pressure. Heat is then applied to the solution, in any suitable way, until a temperature of 150°C. is attained, giving a pressure of 100 lb. per square inch. As soon as these conditions of temperature and pressure are obtained, the pressure is released, the tanks emptied upon a filter, and the hot solution is run back on the ore for re-leaching purposes. The solution is very easily filtered, and the copper remains on the filter in a finely divided form. It may then be conveniently handled by melting and poling, or by compressing it to form anodes.

The typical example given embodies the conditions necessary for completing the reaction for the attainment of the best results. Thus, it has been found that the best results are obtained with 6% solution of $CuSO_4 \cdot 5H_2O$ (containing 1.5% copper) and under a pressure of about 100 lb. per square inch, at a temperature of about 150°C. It is commercially feasible, however, to increase the strength of the solution up to 3% metallic copper and to obtain a precipitation of approximately three-fourths of the copper.

The treatment of the solution, as it comes from the leaching tank, with an amount of calcium carbonate or other alkaline substance sufficient to neutralize the free acid, together with the conditions of temperature and pressure and of concentration of the sulphur dioxide in solution relatively to the concentration of copper in solution in the sulphate liquor, determine the nature of the precipitate which, under these conditions, closely approximates 100% finely divided metallic copper. The quantity of sulphur dioxide passed into the solution is determined by the degree of concentration of the copper, that is to say, the concentration of the sulphur dioxide solution expressed in per cent by weight should not materially exceed the concentration of the copper in solution expressed in the same terms. In this connection, it is to be carefully noted that solutions of copper sulphate stronger than 6% $CuSO_4 \cdot 5H_2O$ may be, in part, precipitated by sulphur dioxide as metallic copper when the solution is first neutralized by the calcium carbonate or other neutralizing agent employed, but, in such case, the precipitation as metallic copper will not be complete. So also, at higher temperatures and pressures than 150°C. and 100 lb., complete precipitation of the copper in the solution may be obtained, but this precipitate will consist only partly of metallic copper and will contain, in large quantities, oxides, sulphites, and sulphides of copper; although sulphuric acid will nevertheless be regenerated under these conditions in sufficient quantity for re-leaching purposes.

The process is particularly applicable to the extraction of copper from low-grade copper ores, principally the carbonate, oxide, and basic sulphate ores. The sulphur dioxide may conveniently be obtained from smelter fumes, inasmuch as the percentage of sulphur dioxide in such fumes is normally sufficient to saturate a 6% $CuSO_4 \cdot 5H_2O$ solution. Furthermore, the process finds a special application in connection with sulphide ores, inasmuch as the conversion of the sulphides usually occurring in such ores into the oxide by roasting will yield, in most cases, even more sulphur dioxide than is required for subsequent treatment of the solution. It may, therefore, be said that the process is applicable to all ores or other products containing copper, as, for instance, in addition to those above noted, to roasted matte from which the copper may be efficiently brought into solution by sulphuric acid. It is proposed to leach with the solution as it comes hot from the precipitation tanks, thereby utilizing its maximum leaching efficiency, and making the process continuous in the sense that the leaching solution is utilized over and over again for re-leaching the ore, after the copper has been precipitated each time out of the solution.

In order to maintain a constant volume of solution, the leaching is effected, as hereinbefore indicated, with a 3.6% sulphuric acid solution, assuming a chemical and mechanical loss in the leaching operation of about 25%. The precipitated copper is not contaminated with other metals more electro-positive, such as iron, as they will not be precipitated under the same conditions. The ferric iron present in the solution will use up some of the sulphur dioxide and convert it into sulphuric acid which will be an added advantage to the process, and, at the same time, the reduced ferrous sulphate in the solution, when agitated with air, is an excellent leaching agent for copper.

It will be noted that the amount of copper precipitated, expressed in percentage by weight, corresponds, within certain limits, to the ratio of the amount of SO_2 employed to the amount of copper contained in the solution. For instance, if the solution contains 1.5% copper and the amount of SO_2 employed is 1.5%, then approximately 100% of the copper present will be precipitated as metallic copper. Or, if the same 1.5% copper solution contains 1% SO_2, then approximately 66% of the copper present will be precipitated as metallic copper. If the percentage of SO_2 exceeds 1.5%, a complete precipitation of the 1.5% copper solution will be obtained, but will consist more or less of oxides, sulphites, and the like.

It will, of course, be understood that I do not claim broadly the use of sulphur dioxide as a precipitating agent in the treatment of copper sulphate solutions, inasmuch as the general reaction, as hereinbefore indicated, is well known, and its utilization, for commercial purposes, has been heretofore attempted. In every such attempt, however, in so far as I am aware, the precipitation of the copper in the metallic state substantially quantitatively has not even been approximated, and even its precipitation in the form of a mixture consisting largely of oxides. and sulphides accompanied sometimes with a relatively small proportion of metallic copper has been incomplete. By the practice of the present invention, on the contrary, practically complete precipitation of all the copper in the solution may be obtained in the metallic form, by observing the conditions of temperature and pressure and of concentration of the sulphur dioxide solution relatively to the concentration of the copper in solution in the sulphate liquor, as hereinbefore specified; and, even with higher concentrations of copper in solution, and at higher temperatures and pressures, the preliminary neutralization of the free acid permits the complete precipitation of the copper partly in the metallic state. So also, with the employment of the preferred conditions of temperature and pressure hereinbefore specified, even though the concentration of the copper in solution may exceed the optimum, a much larger proportion of copper in the metallic form will be precipitated than is obtainable, in so far as I am aware, in the commercial practice of any other process based upon the employment of sulphur dioxide in the reversible reaction quoted.

By the term "substantially quantitatively" as used in the specification and claims is to be understood such a precipitation as will result in a residual leach liquor, which when used as a final washwater, will leave associated with the spent ore an amount of copper so small that it shall not represent a material loss; that is, a loss whose maximum may be said to be one-fourth of the amount of copper originally present in the ore.

HYDRO-METALLURGY OF COPPER IN 1913

By Thomas T. Read

(January 3, 1914.)

It is somewhat illogical to place the discussion of the hydro-metallurgy of copper before that of ordinary smelting methods. But this reversal is perhaps justified by the present keen interest in wet methods for the extraction of copper from its ores, arising from the necessity for devising some means to recover the copper present in the form of oxidized minerals in the enormous tonnages of low-grade ores now being mined and milled in many different places. The oxides, carbonates, and silicate of copper are all rebellious both to ordinary wet concentration and to flotation, and the use of hydro-metallurgical processes seems the only possible means of effecting the recovery of their copper content. The general criteria of the hydro-metallurgy of copper has been discussed at some length on several occasions,[1] and needs no further references. All the most promising of recent work has centred around leaching in sulphuric acid solution. The simplest method, to be applied on the largest scale, is that proposed for the brochantite ore of the Chile Copper Co., Chuquicamata. This has already been described at some length,[2] but may be summarized by saying that careful large-scale experimental work done by E. A. C. Smith has demonstrated that the copper content of this hydrous sulphate of copper can be extracted by leaching the ore, after crushing to 3-mesh, with 8 or 9% sulphuric-acid solution. This is to be done in concrete vats holding 9000 tons each, and it is estimated that one day will suffice for filling, two or three days for leaching and washing, and one day for discharging, which is to be done with clam-shell buckets. The copper in the pregnant solution will be precipitated electrolytically, a 40,000-kw. generating station on the seacoast and a 100-mile transmission line being already under construction. This ore contains no gold, silver, bismuth, arsenic, or antimony and, being a sulphate, the solution builds up in sulphuric acid, so that acid consumption is *nil*. Small amounts of chlorides and nitrates are present and cause difficulties which have been successfully overcome, but the means by which this is done have not yet been disclosed.

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[1] 'Wet Methods of Copper Extraction,' Editorial, Sept. 21, 1912; 'Sulphuric Acid Leaching,' Editorial, Aug. 16, 1913; 'Leaching of Copper Ores,' John Rooke-Cowell, Aug. 23, 1913.

[2] 'Leaching of Copper Ores in Chile,' Editorial, *Mining and Scientific Press*, June 21, 1913.

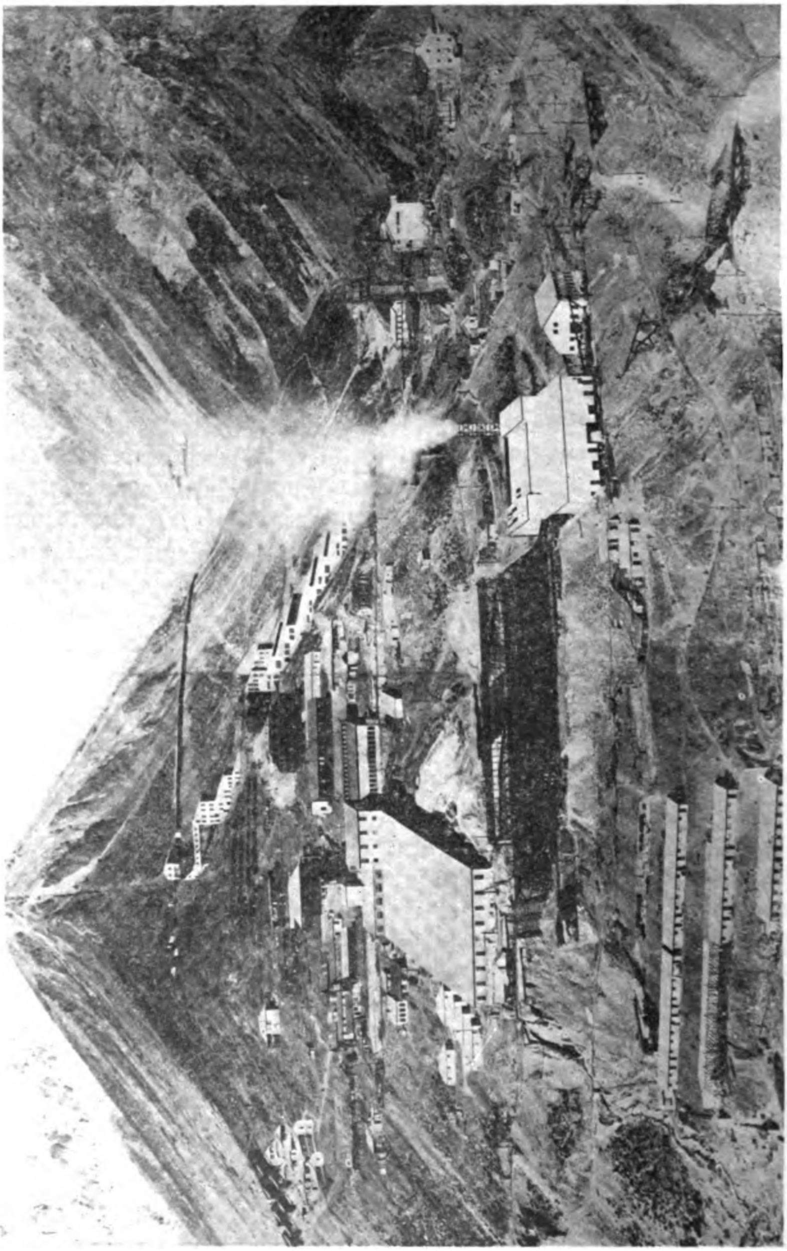

FIG. 50 BRADEN MINE, MILLS, AND SMELTER, RANCAGUA, CHILE.

Leaching work at the Braden has been delayed by more urgent problems, and the necessity of using the available electric energy for other purposes. E. A. Cappelen Smith has recently described the proposed process as follows: "The process decided on was roasting of the concentrate in a Wedge furnace in such a manner as to eliminate practically all of the sulphur and at the same time oxidize all the iron to the ferric state. The concentrate, after roasting, contained 2 to 3% sulphur, practically all in the form of sulphate, 15 to 18% copper, and about the same amount of iron of which only a very small amount was present in the ferrous state. The roasted concentrate was leached with dilute sulphuric acid, the leaching operation being divided into two stages: first, neutralizing leaching; second, acid leaching. The solution from the neutralizing leaching was electrolyzed direct, whereas the solution from the acid leaching was neutralized by the following batch of calcines. Several leachings made in the plant indicate an extraction of between 90 and 95% of the copper present in the concentrate."

The most interesting announcement of the year[3] was that roasting and leaching with sulphuric acid, following a method devised by Frederick Laist, was to be tested on a large scale for treating the tailing from wet-concentration processes at Anaconda. An 80-ton experimental plant was constructed, and the results of the first work were published[4] during the summer. As this paper is readily accessible and contains many important details, it should be read by all. Briefly, the process consists in drying the tailing from wet concentration on the upper hearths of a McDougall roaster, adding salt, and roasting at a low temperature to keep down the volatization of copper chloride and the production of ferrites. The calcine is then leached with sulphuric acid solution. Experimental work later in the year has been highly successful. It was at first proposed to precipitate the copper from this solution by H_2S, thus regenerating H_2SO_4, but more recent work indicates the desirability of using an iron sponge produced by reducing 20-mesh calcine with coal dust in a McDougall furnace. The sulphuric acid required for leaching is to be made in lead chambers, from the SO_2 gas obtained in roasting rich coarse concentrate, at an estimated cost of $4 per ton. Tentative plans for a 2000-ton plant are well advanced. As a result of the success of this work experiments along similar lines are being made at several places.

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[3] Editorial, *Mining and Scientific Press*, May 3, 1913.

[4] 'Roasting and Leaching Tailings at Anaconda,' Frederick Laist, *Bull.* Amer. Inst. Min. Eng., July 1913.

In preceding years I have usually closed the discussion of this topic with an expression of hope that by the following year details of the practical operation of leaching plants would be available. That hope has at last been rewarded, P. E. Peterson[5] having published descriptions of the Butte-Duluth and Bullwhacker plants, which were erected at Butte last year. It is to be deplored that these descriptions do not include any account of the numerous difficulties encountered in the early operation of these plants, as being more helpful to operators elsewhere than the methods found successful. These plants operate under special conditions, in that they treat silicious ore comparatively free from the iron oxides present in ordinary oxidized ores. The Butte-Duluth plant is 100-ton and the Bullwhacker 125-ton. At the Butte-Duluth the ore is crushed, by gyratory and Symons disc crushers, only through a ½-in. screen. The ½-in. ore is dumped into V-shaped leaching vats, provided with a filter bottom of boards bored with ⅝-in. holes. After leaching with acid, the vats are discharged through doors in the bottom. The solution is drawn from the vats, allowed to settle, elevated to lead-lined tanks, where it is heated to 60°C., and thence to the electrolytic cells. Each of these is 30 by 39 in. and 8 ft. long, lined with hard lead, and provided with 20 anodes of hard lead, weighing 20 lb. per square foot. The cathodes are ordinary starting sheets of copper. The current density used is 12 to 13 amperes per square foot, and the cathodes assay 99.96% copper, according to Mr. Peterson. The present cost is estimated by him as 14c. per pound of copper, but he thinks this can be considerably reduced, as the sulphuric acid required now costs $27 per ton or 4¾c. per pound of copper produced. If locally manufactured on a large scale, the sulphuric acid should not cost over 1½c. per pound of copper.

The percentage of extraction on the ½-in. ore was so low by the method described by Mr. Peterson that it was necessary to abandon it and the following modified process has now been adopted. The Company is building a 1000-ton dry-crushing plant to reduce the ore to 10 mesh. This consists of a swing-jaw crusher, intermediate gyratory crusher, Symons disc crusher, rolls, and impact screens. The dry ore is fed to a mixer with approximately four times its weight of 8 to 10% sulphuric acid which discharges to a Dorr classifier, which is the first of a series of 6 arranged in such a manner that the sand will discharge from one into the next. Various times of contact with the acid in the first four Dorr classifiers were tried out and it was found that approximately 40 minutes

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[5] *Min. Eng. World*, Sept. 6 (p. 423) and Oct. 4 (p. 585), 1913.

gave an extraction between 85 and 90%. The last two classifiers, No. 5 and 6, in the series, are used for washing. The overflow from No. 5 is used to make up for the loss of acid in the strong acid solution.

The overflow from No. 1 classifier carries off approximately about 13% of the total weight of ore as slime. This, as well as the overflows from classifiers 2, 3, and 4 are run to the acid leaching-tanks in the bottom of which is a layer of crushed ore to act as a filter. The underflow from these filters passes to the electrolytic cells and on to the strong acid storage-tanks where the solution is brought up. This method of running the slime to the acid leaching-tanks is only temporary as later on it is intended to recover the dissolved copper content by passing this pulp through a series of Dorr continuous thickeners operated so as to give counter-current washing.

FIG. 51. CANAL GATE SPILLWAY, AND AUTOMATIC SPILLWAY, BRADEN.

The second water wash, which is applied to the last classifier, goes to scrap iron precipitation. The sand discharged from the classifiers approximates 18 to 20% moisture. No. 1 classifier in the series is a machine 30 ft. long by 4 ft. 6 in. wide. The rest of the classifiers are of standard duplex size, 15 ft. long by 4 ft. 6 in. wide.

FIG. 52. PENSTOCKS AND POWER-HOUSE, BRADEN MINE.

These machines are built with wooden boxes and all moving parts that come in contact with the acid, are also built of wood (hard maple). The wear of the wooden teeth is surprisingly small, but the experimentation has shown that plate glass teeth are the proper ones to use here, and will be installed in any future classifiers built for this purpose. This series of 6 classifiers is operating at the present time and has a capacity of approximately 200 to 220 tons per 24 hours.

The Dorr Cyanide Machinery Co. is building another series of 5 machines each 30 ft. long by 8 ft. wide, which should have a capacity of 400 to 500 tons of ore per day. At the present time strong acid solution is being added with the ore to No. 1 classifier, but later on this practice will be altered and the overflow from No. 5 Dorr classifier will be mixed with the dry ore; the advantage of this being that the strong solutions can be kept from coming in contact with the slime, the weak acid solution not taking iron and aluminum into solution so readily as the strong. All solutions in the plant are kept at approximately 55°C. The cost of this in the northern winter climate is likely to form a considerable item of the total working cost.

Since the above was written, the leaching of the Butte-Duluth ores in Dorr classifiers has given such favorable results that another machine has been added to the first series, making seven in all, and a second line of large classifiers, each 30 by 8 ft. wide is now being installed. The whole of the slime and solution from these machines will be run to acid proof Dorr thickeners, the clear overflow passing to the electrolytic cells and the thick under-flow being washed with barren solution and water.

Walter H. Weed has made[6] the following statement regarding the operations at this plant: "The mill feed for the past six months has averaged 2.01% copper. * * * * The ore passes through the leaching machine in 32½ minutes and 88% of the copper content is extracted by the solution. The consumption of acid is 67½ lb. of 60% sulphuric acid per ton of ore. No trouble is experienced from fouling of the solution, the constant accession of first-wash water and of fresh acid diluting the amount of iron and alumina. The solution is returned to the machine after settling in tanks until it contains 3 to 4% copper and 9 to 19% acid. It is then pumped to the electrolytic cell room, where the copper content is recovered on cathode sheets.

"The waste solutions and wash waters are sent to the concentration plant and copper content recovered as cement copper. The output of the plant in January, 1914, was 55,000 lb. of electrolytic and about 40,000 lb. of fine copper in the form of cement copper.

"The estimated costs of mining and treatment given herein are based on figures taken from the company's records. The plant has been so much bettered in the past few weeks and the present treatment, with lowered costs, has been in operation so short a time that the figures given must be accepted as a reliable estimate, and not as a record of actual experience for the past year. The

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[6] *Min. & Eng. World*, Feb. 21, 1914, p. 377.

present cost, using both leaching tanks and the leaching machine. is $3 per ton of ore. The present plant is treating a little more than 100 tons per day, the total tonnage in January amounting to 3200 tons of ore. I firmly believe very material improvement made by Mr. Sherwood in the cell room and the taking out of the rectangular leaching tanks will reduce the costs, even with the present plant, to $2.70 per ton of ore treated."

At the Bullwhacker plant, which is near by, the ore, crushed to 16 mesh, is agitated with 10% H_2SO_4 solution, 2 tons of solution to 1 of ore, in a Hendryx agitator. The effluent solution, containing as much as 4 to 5% copper, is decanted from settling tanks, and sent to electrolytic cells which differ from those previously described in being circular, 9 ft. in diameter and 5 ft. deep, holding 24 cathodes. About half of the copper content of the solution is deposited electrolytically, and it then goes back to be used for further leaching. The cost of production of copper is here also about 14c. per pound.

The difficulties of the method employed in these two mills are, perhaps not unnaturally, not discussed at length. They may be summarized as follows: fine crushing is necessary in order that the copper minerals shall be completely soluble, but leads to the production of colloids ('slime') which make the solution cloudy and difficult to settle. It is, of course, impossible to deposit a pure cathode from a solution containing impurities. The leached ore is difficult to wash for the same reason, and also because the use of washwater involves the discharge of solution, which carries away with it not only the expensive sulphuric acid but copper as well. This copper can be precipitated on iron, but the consumption of iron by the acid cuts down the net return. The difficulties and excessive consumption of energy in precipitating copper electrolytically when using an insoluble anode are too well known to need repetition. Here the resistance of the solution has been decreased by preheating it, and its conductivity is kept as high as possible by only precipitating half of the copper present. At several other mines, notably the Nevada-Douglas, preparations are being made for the construction of similar plants, but it is not to be expected that these can be put into operation without first solving local problems.

At nearly every important copper mine in the West some member of the chemical or metallurgical staff is studying hydro-metallurgical problems in the light of local conditions. The work at the Shannon, Calumet & Arizona (Ajo), and Arizona Copper mines was described in my review for the last year, and no reports of progress have since been given out. At the Keystone and Inspiration, both at

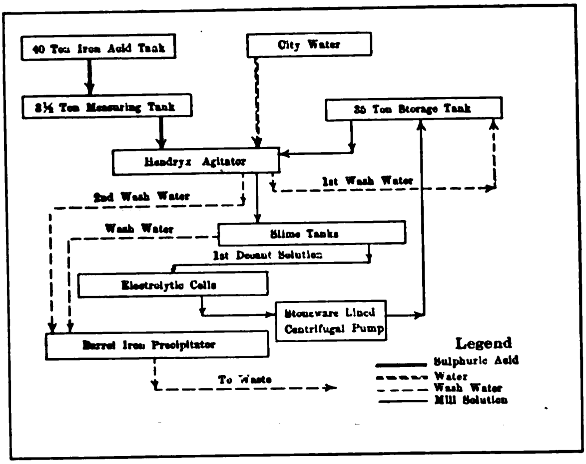

FIG. 53. BULLWHACKER FLOW-SHEET.

Miami, interesting working is being done. J. Parke Channing has described this work in the *Bulletin* of the Mining and Metallurgical Society of America. R. S. Canby, at the Keystone, suggests the possibility of using cast iron hearths as at the Steptoe Valley smelting plant, in the McDougall roasting furnaces, so that the fuel used on one hearth would better heat those above. He proposes to use a special design of burner which will permit the use of oil as a fuel along with the maintaining of a low temperature on the hearth. The research for the Inspiration is being done at the University of Pittsburgh, under the direction of R. K. Duncan. At the Copper Queen the leaching of low-grade ores is also being studied.

W. Y. Westervelt describes in some detail, in the November *Bulletin* of the Mining and Metallurgical Society, the following record of a 33-day test on the average ore at the Ray mine in Arizona.

"The ore in the vats at the beginning of the tests was carefully sampled and assayed by the vat. That added during the test was weighed, sampled, and assayed by the vat charges of 200 lb. each. Tailing rejected during the test was sampled, and assayed by the vat discharged. The ore remaining in the vats at the end of the test was sampled and assayed by the vat.

"At the beginning of the tests the solutions in the vats were carefully drawn off, measured, and sampled. The same was done independently with that in the electrolytic tank, and again independently with that in the storage tanks. All acid added (commercial

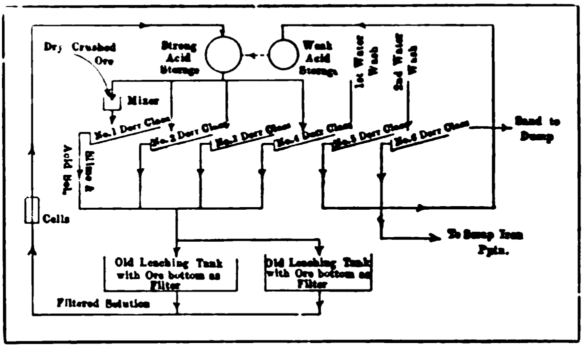

FIG. 54. NEW FLOW-SHEET OF BUTTE-DULUTH PLANT.

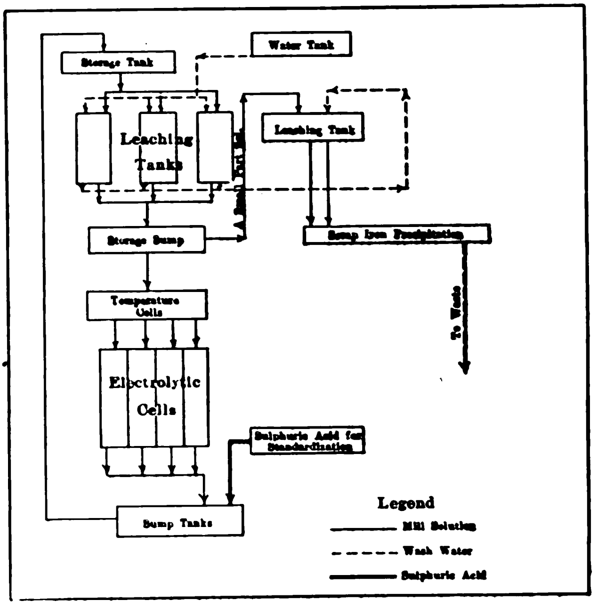

FIG. 55. ORIGINAL FLOW-SHEET, BUTTE-DULUTH PLANT.

66°B. sulphuric acid) was carefully weighed as added. Additions of water were measured, the condition of the solution was daily determined by assay, and on completion of the test, complete measurement and assay was again resorted to.

"The electrolytic copper deposited on copper cathodes previously made from the Ray ore by stripping the deposition on rolled copper sheets. These cathodes were removed and weighed daily. The voltage maintained at the tank was determined hourly by a Weston voltmeter capable of being read to 0.01 volt. The amperes were determined both by hourly readings of a Weston ammeter and by checking against the daily weighings of a standard copper voltmeter.

"The heating was done solely in the leaching vats by means of closed lead-pipe coils placed in the bottoms. Steam was supplied to these coils from a main line running over the five vats and the condensed water was all secured by connecting the ends of the coils with the main drain pipe, the latter itself discharging into a measuring can. The steam supply was regulated by a valve to each coil, as was also the discharge into the drain pipe, by another valve. The drain pipe itself, in which the the condensed water accumulated, was kept closed by a valve so that no uncondensed steam could escape.

"The ratio of volume to surface of the vats employed was 3.03, while that of 26-ft., 100-ton vats is 0.487, or less than 1/6. In other words, the heat radiating and conveying surface of the experimental plant was over six times as great proportionately as would be that of a 100-ton vat plant. Again, the evaporative surface of an operating plant would not be ¼ as great proportionately as that of the experimental plant, while a considerable part of the steam condensing surface in the experimental plant was outside the vats, thus doing no work, while registering as condensed steam. Due allowance was accordingly made for these which may be summarized as below:

Copper deposited, total run	40.28 lb.
Copper deposited, per diem	1.22 "
Amount ore in vats	1000.00 "
Copper deposited per diem, per ton in vats (0.122%)	2.44 "
Days required to extract 40 lb. per ton	16.4 days
Acid consumed	91.4 lb.
Acid consumed per pound copper deposited	2.27 lb.
Ampere-hours consumed	22.382
Ampere-voltage	1.7
Kilowatt-hours	38.0494
Kilowatt-hour per pound copper	0.945
Ampere-hours consumed per pound copper	556.0
Steam condensed per pound copper deposited	33.0 lb.
Extraction	80%
Average temperature of vats	72°C.
Assay of tailing	0.3%

At the Bwana M'Kubwa mine it was proposed to use the Siemens-Halske process; presumably in much the same form described above, but this process has now been dropped, and the Mackay process will be tried. The Bradley leaching plant at Anaconda has closed down, having proved a failure. The various other patented processes seem not to have made much headway during the year. The McKay process, which is now being developed by Archibald Carmichael, was under investigation by important interests, but negotiations have been dropped. The owners of the Alexander process are carrying on experiments, but have made no report of progress. The same may be said of the Robertson process. The Irving process is elsewhere described by L. S. Austin. The leaching of the copper contained in the burned cinder from sulphuric acid making is being done is a number of places, but is quite a different operation from the leaching of raw or roasted ore. In the precipitation of copper from mine-water, J. W. Richards has suggested that if the scrap iron used for this purpose were laid upon sheets of copper, a galvanic couple would be established and the surface of the iron kept cleaner and consequently more efficient. It is proposed to use sponge iron for this purpose in place of the usual scrap iron, but the experiment has not yet been made. In conclusion, it may be said that the experimental work at Anaconda has demonstrated the possibility of roasting sulphide tailing tailing at a low cost, and the leaching plants at the Bullwhacker and Butte-Duluth have demonstrated the practicability of sulphuric acid leaching and electrolytic precipitation for oxidized silicious ores, with prospects of attaining a low cost. It must not be forgotten, however, that local conditions create fresh problems at each mine.

PRECIPITATION OF COPPER FROM MINE WATERS

By Floyd Bushnell

(November 18, 1911.)

Although copper has been hovering around the 13c. mark for the past four years, the precipitation of copper in the Butte district has not decreased as much as might have been expected. Several of the smaller and less prosperous plants on Silverbow creek on the south side have been compelled to discontinue operations, but the larger are running at full capacity.

Fred Miller was the first to undertake the precipitation of the copper contained in the water pumped from the mines at Butte, and while his methods were crude, compared with those of the present day, he was fairly successful. Miller dammed back the water in small ponds, into which he threw tin cans and scrap iron; after the metal disappeared he drained the ponds and scraped up the precipitate. Others became interested and encroached upon his rights, whereupon he became discouraged and quit the business.

William Ledford, in 1901, was the first to put copper precipitation on a business basis. He secured a three-year lease on the water from the St. Lawrence and Anaconda mines at a 25% royalty. Ledford was subjected to much criticism because of his so-called foolish venture, but during the life of the lease he cleared $100,000. During this lease the copper water became so strong that in a short time it corroded the pipes through which it was pumped to the surface, whereupon the companies weakened the water by passing it over iron underground. Ledford contested this on the ground that he

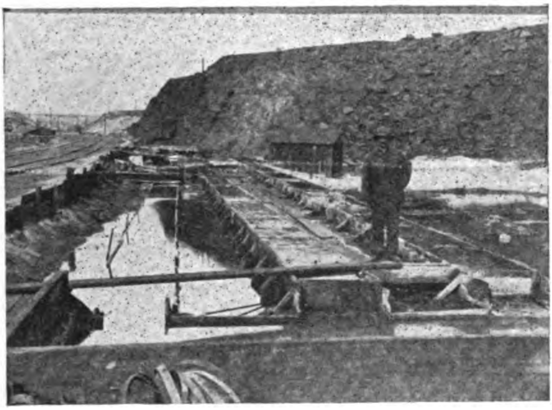

FIG. 56. MANY SMALL PLANTS EXIST ON OLD SMELTER SITES, WHERE WATER SEEPS THROUGH TAILING AND SLAG PILES.

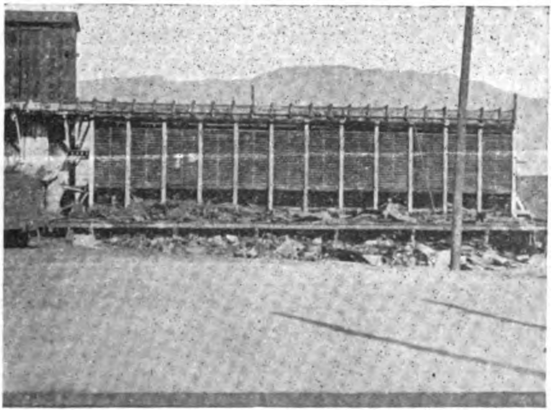

FIG. 57. ONE OF THE TOWERS BELONGING TO THE B. & M. COMPANY.

was to have the sole right of precipitation. He sued the Anaconda company and was awarded several thousand dollars damages. After this successful venture the companies refused to give further leases, and the larger companies have since been doing their own precipitation; the Amalgamated, the Boston & Montana, Gagnon, Original, and other mines have plants at their properties. However, there are a few smaller companies that lease their copper water.

The plant at the Original mine has about the most economical system in Butte. Three men take care of 1400 linear feet of boxes. This company is the first to use the hose-washing system, which is decidedly effective. City water, under good pressure, is piped to all parts of the boxes, and at regular intervals the precipitate is washed from the half-eaten metal and from the bottom of the boxes. The precipitate is protected by means of a shelf, 2 in. from the bottom, and once it is separated from the metal, is unimpeded in its course to the tanks. This system is so arranged that the copper water may be shut off from any part of the plant. The high-grade precipitate is flushed into tanks 2 by 7 by 9 ft. in a steam-drying room. There are two tanks, each holding 21 tons, and these are filled several times per month.

The plants which secure their water direct from the mines and those operated on the sites of abandoned smelters have water with by far the highest copper content. The water as it comes from the mines averages from 0.04 to 0.06% copper. With the 0.06% water precipitate as high as 90% is secured, and from this it runs as low as 30% copper. Some of the operators are so fortunate in the situation of their plants that they can cause the water to flow through the tailing heaps of abandoned plants, thereby materially increasing the copper content of the water.

There are at present two methods of precipitation, the box system and the tower system, the latter being generally conceded to be the more successful, as the towers occupy less space and the water is brought more thoroughly into contact with the iron. The box system consists of a series of troughs from 1 to 3 ft. wide and 1 ft. deep, covering a great deal of surface. The boxes are so arranged that there is a gentle incline and the water flows naturally through them. The scrap iron and tin cans are thrown into these troughs. The copper precipitate, after being washed through the many compartments, sinks to the bottom of the receiving-tank, which is cleaned out about once a month.

The tower system was introduced about eleven years ago by Huddleston & Turner. The usual tower is about 3 ft. wide, 20 ft. high, and 30 ft. long, and is partitioned off by means of floors 18 in. apart. These floors are made of 2 by 4-in. scantlings, placed 3 in. apart, running lengthwise. Tin cans and scrap iron are placed on the several floors. By means of a small pump the copper water is raised to the top of the tower, where it is distributed equally over

FIG. 58. BOX SYSTEM AT THE ORIGINAL MINE.

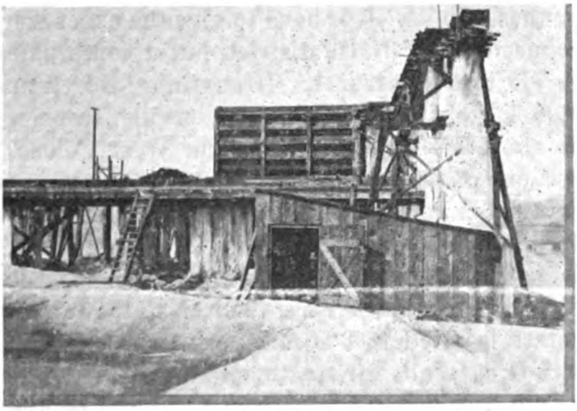

FIG. 59. COMBINATION BOX AND TOWER PLANT IN THE WINTER.

the scrap iron. The tin cans and smaller pieces of iron are eaten away in a comparatively short time, but the larger pieces must be scraped regularly to remove the precipitate. The latter is washed to the bottom and then into tanks alongside the towers.

The operators sell their output to the smelters at 4c. less than the market quotation for copper; the shipper must pay a treatment charge of $5 per ton. The cost of making a pound of copper is estimated at between 7 and 8c., so that with the present price of copper there is not a very large profit. Theoretically a pound of tin or iron will precipitate its chemical equivalent in copper. Tin cans are used where the water is weaker, and iron and steel are sought after by those who have stronger water. A test made with the same water shows that tin cans produced a 33% copper precipitate, and iron 50%. At many of the plants one or two men do nothing but remove the paper from and punch holes in cans in order that the free passage of water may not be interfered with.

It might be thought that the plants would be compelled to close in winter, but such is not the case. The water coming in from the mines is quite warm, and the chemical action of precipitation liberates heat. The water has a greater percentage of copper in it during the winter months because there is no surface water to dilute it. As a matter of fact, almost as much copper can be made in winter as in summer. One peculiar feature of this work is that the streets and alleys of Butte are kept free of tin cans and scrap iron. Because of the increased demand for scrap iron for precipitation purposes during the past five years the price of old iron has increased from $7 to $12 and $14 per ton; $10 is the average price for a ton of tin cans. Men and boys make a living in the larger towns of the state securing material for these plants. It is hard to give the exact amount of precipitate produced in the Butte district, but a conservative operator places it at 700 tons per month. If anything, this estimate is low.

The Boston & Montana plant, at Meaderville, securing its copper water from the two Leonard mines, the West Colusa, and other properties, is one of the largest and most up to date in the district. Two flumes. 1000 ft. long and 4½ ft. wide, 800 linear feet of boxes, and four large towers are operated by this company. The towers are 100 ft. long, 30 ft. high, and 6 to 8 ft. wide. The lower towers have two electric elevators to raise the iron to the top of the towers. The precipitate is dried to 8% moisture, barreled, and shipped in 30-ton lots to Great Falls for treatment; three shipments per month is the average output. The average strength of the Boston & Montana copper water is 0.05% copper. But during the fire that raged several years ago in the underground workings of the West Colusa and other mines, the percentage of the water ran as high as 0.32%. While the water was at this great strength 90% copper precipitate was produced and the average monthly shipment of precipitate was 9 carloads. Under normal conditions the precipitate averaged 60% copper.

The charge was made in a recent issue of a mining journal that the methods employed in Butte are crude and wasteful as compared with the plants at the Rio Tinto mines, where the water runs as high as 0.2 and 0.3% copper. A comparison between the methods employed at these two places is absurd; the water from the Rio Tinto mines, after having run through the precipitating plants, is as strong as the water coming from the Butte mines. The charge was also made that strong copper water flowed down Silverbow creek, past all the precipitating plants. Such a statement is absolutely false. An assay of the water after it passes the last precipitating plant shows the exceedingly small percentage of 0.0002 copper.

PRECIPITATION OF COPPER FROM MINE WATERS

(Discussion, February 3, 1912.)

The Editor:

Sir—The article, 'Precipitation of Copper from Mine Waters,' contained in your issue of November 18, invites comment and correction—certainly here in Spain, the 'home' of the ancient process. It is well known in Europe that this process of the precipitation—in Spanish *cementacion*—of copper contained in vitriolous waters was patented and practiced at Rio Tinto as early as the sixteenth century.

[1]According to the Official Record, the following operations took place at Rio Tinto on the 9th of May, 1752, and following days (translated):

"Note of iron placed in the *canales* of cementation tanks on that day at 12 o'clock noon and on the following days:

	Arrobras.	*Libras.*
'*Planchuela de Viscaya.*' Sheet iron	24	1
'*Hierro Viejo.*' Scrap iron	6	21
Dia 10, á las 11 del dia	2	7
" 15, á las 7 de la tarde (p.m.). Sheet iron	49	2
" 16, á las 7 de la mañana (a.m.). *idem*	3	14
" 19, á las 7 de la mañana (a.m.), *idem*	23	24

The well recorded history of this process may be recapitulated thus: Basil Valentine, who wrote about 1500, in his instructive *Currus Triumphalis Antimonii*, indicates this method of the extraction of copper.

It was in practice in the Lower Hartz until the middle of the

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[1]Figuier, '*Lalchimie et les alchimistes.*'

sixteenth century, when it was substituted by another process, as took place at the mines of Rammelsberg on the banks of the Oker.[2]

Alonzo Barba[3] describes this process as it was in operation in Peru before the year 1637.

In 1661 a *licencia* was granted by the Crown[4], at Madrid, to Alvaro Alonso de Garfias; while in the year 1695 a similar privilege was conceded by the same authority to Roguc Salas y Ulloa: to extract the copper from the vitriolous waters of Rio Tinto.

The same process, or the precipitation of the copper contents of 'mine-water' by placing iron in it, has also been treated of and described by Hermann, Figueroa, Heron de Villapose, Guenyvean, and others.

But what is equally surprising is the allegation made also in the last paragraph of the article under examination, that "the water from the Rio Tinto mines, *after* having run through the precipitating plants, is a strong as the water coming from the Butte mines." Presumably he means us to infer, as to these latter, in their untreated state. This is quite in harmony with the other allegations. But if that actually is so, then the shareholders of the Rio Tinto company can with good reason charge the local management with a wicked waste of copper: for the inference is that a certain quantity of copper is daily being lost because of unskillful treatment and of wasteful supervision! But fortunately for their pockets, and, too, for the credit of those engaged in the huge work of *cementacion* or precipitation as at Rio Tinto is performed, *that is not so.* The final passage: "that the water (at Butte) after it passes the last precipitating plant, shows the exceedingly small percentage of 0.0002 of copper," should be very satisfactory to the Butte interests—*if that actually is the case.*

W. G. Nash.

Minas de Rio Tinto, Spain.

[Mr. Nash's unfamiliarity with the conditions existing in Butte leads him somewhat astray in his criticism, and for that reason part of his letter is here omitted. The work at Butte and Rio Tinto is not strictly comparable; at the former the mine water, free from harmful impurities except small amounts of arsenic, is the solution used, at the latter a comparatively strong solution of iron and copper sulphates is employed. Under the conditions obtaining at Butte it is found that but little of this arsenic is precipitated, and

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[2] Schlutter, '*De la fonte des Mines.*' Vol. 2.

[3] Alonzo Barba, '*Arts de Met,*' I, 121.

[4] Nash, 'Hist. of the Rio Tinto Mines,' p. 140.

though present in the water in the proportion of 1 of arsenic to 17 of copper, only 1 to 500 occurs in the high-grade cement copper and 1 to 20 in the low grade. It is thus possible to carry the precipitation of the copper to a very low content.—EDITOR.]

PRECIPITATION OF COPPER FROM MINE WATER AT BUTTE

(September 28, 1912.)

The large amount of copper annually recovered from copper-bearing water at Butte is not generally realized. In 1910, a total of 4,558,850 lb. of copper was recovered by precipitation on scrap iron. In 1911 the water pumped from the Leonard shaft yielded 600,000 lb., while the water from the High Ore shaft yielded about the same quantity. Not all of this large quantity is produced from the mine water, however, as the leaching of old tailing, slag, and waste heaps by lessees adds very materially to the output. The recovery of copper from the mine water is of the greatest interest, however, since the quantity thus recovered is sufficient to pay the cost of pumping and yield a small profit. Handling the underground water, instead of being a heavy burden on the cost of working, is at Butte a small source of profit.

It is difficult to give exact data regarding the precipitation of the copper, since the character of the mine water varies a good deal, both according to the season of the year and according to the part of the workings from which it is derived. At one time much water was used in the Leonard mine in an attempt to extinguish an underground fire. The copper-content of the water obtained from this part of the workings was considerably above the average, but for some unknown reason this water formed a curious precipitate upon the iron, resembling fungus growths, which greatly interfered with its efficiency as a precipitant. Similar growths are also found within the mine, often on the mine timbers, and have given rise to much discussion as to whether they are vegetable or mineral growths. The following is an analysis of the water from the Leonard mine:

	Per cent.
Copper	0.0122
Iron, suspended	0.0127
Iron, solution	0.0067
Arsenic	0.00026
Antimony	0.000088
Free H_2SO_4	0.0139

The water is handled by pumps with bronze working-parts. and pumped through a column-pipe lined with wood or lead. On the surface the water is passed through a series of boxes and trays. At first the water flows at a rapid rate over large pieces of scrap iron and steel, and the precipitate collected at this point is high-grade, containing as much as 80% copper. Farther down smaller pieces of iron are used and a slower rate of flow. The final precipitation is accomplished on small scraps, chiefly old tin cans spread in layers on trays, the water dripping slowly from one layer to another. In this way practically complete precipitation is secured, analysis showing that with mine water which carries 0.03893% copper, the final effluent carries only 0.00036% copper, corresponding to an extraction of 99.1% of the copper in the water. The low copper content of the escaping water has been compared with that of the effluent solution at Rio Tinto, but it must be remembered that at Rio Tinto the copper-bearing solution carries 0.4% copper and the 0.002% copper content of the effluent solution corresponds to an extraction of 99.5%. Another analysis of Butte mine water shows an original content of 0.0099% copper, the water entering at 73°F. and escaping at 65°F. with a copper content of 0.0008%, corresponding to a 92% extraction. The average copper content of the precipitate is 45%. A curious fact worth noting is that the arsenic content of the mine water at Butte is extremely low (less than 0.0003%), although about one-quarter of the copper present in the ore is in the mineral enargite, which contains nearly 20% arsenic. At Rio Tinto, on the other hand, the amount of arsenic present in the ore is small, yet according to C. H. Jones* the copper-bearing solution contains 0.03% arsenic. This and the fact that pig iron instead of scrap steel is used for precipitation of the copper has a marked effect upon the process there employed, for both the arsenic and phosphorus come down in the final stages of the prócess, and form most undersirable constituents of the copper precipitate. It is not feasible, therefore, to reduce the copper content of the effluent solution so low as at Butte. At Rio Tinto, 140 lb. of pig iron is required to precipitate 100 lb. of copper, and at Butte about 150 lb. of scrap iron and steel is required.

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*Trans. Amer. Inst. Min. Eng. XXXV:6, (1905).

PRECIPITATION OF COPPER FROM MINE WATERS

(Discussion, February 24, 1912.)

The Editor:

Sir—A reference to the precipitation of copper from sulphate solution by the use of metallic iron, which does not appear to have come to the attention of your Rio Tinto correspondent, W. G. Nash, and which has considerable interest for students of the subject, is found in the writings of Aureolus Philippus Theophrastus Bombast, of Hohenheim, called 'Paracelsus the Great,' who was born about 1493 and died in 1541. In the 'Book concerning the Tincture of the Philosophers,' chapter VI, is found the following statement:

"This work is a wonderful one in the light of Nature, viz., that by the Magistery or the operation of the Spagyrist, a metal which formerly existed should perish and another be produced. This fact has rendered that same Aristotle, with his ill-founded philosophy, fatuous. For truly, when the rustics in Hungary cast iron at the proper season into a certain fountain, commonly called Zifferbrunnen, it is consumed into rust and when this is liquefied with a blast-fire, it soon exists as pure Venus [copper], and nevermore returns to iron. Similarly in the mountain commonly called Kutenberg they obtain a lixivium out of marcasites, in which iron is forthwith turned into Venus of a high grade, and more malleable than the other produced by nature."

This statement affords clear proof of the fact that this method of gaining copper was not merely an experiment, but was actually a process commercially resorted to for the purpose of procuring the metal. It is also one of the clearest and most definite statements of the reason why the alchemists believed in the transmutation of metals. If by merely immersing metallic iron in clear spring-water the iron disappears and copper is found in its place, what better evidence could there be? The paragraph cited above is found in the English translation by Arthur Edward White, Volume 1, page 28, published in London in 1894.

HORACE WINCHELL.

Minneapolis, February 8.

TREATING ZINC-COPPER ORES

(Discussion, April 12, 1913.)

The Editor:

Sir—It will, no doubt, be of interest to most of your readers to learn that a process has been developed for the successful treatment of ore containing too much zinc for the ordinary smelting practice, and not enough to be smelted for zinc in the old-style retort furnaces. Often such ore, usually sulphide, contains silver and gold with either copper or lead, or a small percentage of both these base metals. The presence of zinc makes it difficult and often too expensive to treat such ore for recovery of the other metals, whereas, the combined value of all the metals named, including the zinc. makes such ore very valuable. To illustrate this statement: Take an ore containing 50c. gold, 6 oz. in silver, 55 lb. in copper, and 400 lb. in zinc per ton; at the present gross values, the gold, silver, and copper alone would be $12.05 per ton, but the zinc makes the ore treatment expensive by any ordinary method, and the zinc itself is worth $20.40 per ton if 85% is recovered into a marketable product at present prices, making the total value $32.45 per ton.

The ore I refer to and similar ore elsewhere cannot be concentrated by comparatively simple wet concentration methods adopted in the Middle states, on account of its large content of sulphides and the fact that it contains so little mineral that can be washed out. Magnetic separation appealed to me for the concentration of such ore, but proved a failure at the Afterthought, chiefly on account of the peculiar close-grained fine crystalline structure of the ore. The use of acids to dissolve the metals in the ore never did appeal to me, for the following reasons: The waste of acid caused by the formation of salts, due more especially to the lime, aluminum, and iron contents of the ore; the mechanical difficulties of handling such a pasty mass; and the difficulty of precipitating a pure zinc from the other metals which are in solution with it, the zinc being the most difficult and last metal to be precipitated. To make sure on these points I spent considerable time with others experimenting, with the above results, and other objections equally as serious.

The flotation process, which has proved such a commercial success at Broken Hill, Australia, will make a 45 to 50% zinc concentrate from the ore on which experiments have been made by men interested in the flotation process, recovering 85% of the zinc in this form, after a preliminary treatment. The concentrate would then have to be shipped to the regular zinc smelters near cheap fuel to be recovered and smelted in retorts where about 85% of the zinc in the concentrate can be saved. After all this expense has been incurred for the recovery of about 72% of the original zinc, the question arises, what has become of the copper, silver, and gold? My informant, who made the flotation test, stated that about 60% of these remained in the tailing and about 40% followed the zinc concentrate. This splitting up of the copper, gold, and silver makes them practically worthless. If they all remained in the tailing it would pay to smelt them, or if they would all follow the zinc concentrate, then there would be some profit in sending the cinder mixed with ash from the coal used in the retorts to a lead or copper smelting plant. If cheap coal could be obtained close to large bodies of such ore as I have described, the practice adopted at Joplin, Missouri, and at one time in Canyon City, Colorado, might suggest itself, but even then the loss in silver would be excessive, and the zinc oxide produced is not generally of good quality. The cinder from such oxidizing furnaces containing the remaining silver, copper, and gold with some zinc would have to be smelted in a suitable furnace by ordinary methods.

Having investigated nearly every known zinc recovery process, I will briefly describe a process that, while not suitable for the treatment of all ores containing zinc, especially those containing the zinc in the form of a silicate or too much arsenic, is suitable for the treatment of ore containing zinc with or without copper, lead, silver, or gold, all these metals being saved if present, by partly de-sulphurizing the ore, removing the zinc by leaching with ammonia and carbon dioxide, and smelting the sweetened residue, which is only 50 to 60% of the weight of the original ore, in an ordinary smelting furnace. The reduction in weight would depend mostly on the zinc content. It has been found to be 45 to 50% on a 20% ore, recovery of the zinc being 90 to 95% from a 35% zinc ore; 85 to 90% from a 25% zinc ore; 80 to 85% from a 20% zinc ore; and from a 10.4% zinc ore containing 2.1% copper, 6.5% lead, the extraction was 73 to 80% of the zinc. (I believe the average zinc extraction from the original ore in the United States is only about 51%.) The ammonia and carbon dioxide are distilled off in order to precipitate the zinc as a basic zinc carbonate. This precipitate is calcined to a zinc oxide, which it is found convenient to make chemically pure. The ammonia and carbon dioxide are used over again continuously, with practically no loss, except a slight mechanical loss in the residue to be smelted, which is found to average 0.6 to 1.4 lb. ammonia per ton of iron. The cost of treatment is not excessive. The elimination of the zinc and reduction in weight reduces the cost of smelting to a minimum for the recovery of the other metals in a reverberatory matting furnace (and it may be possible, in some cases, to treat the residue by cyanidation). The expense of crushing, grinding, and roasting the ore will be about the same as preparing the ore for any leaching method treatment, or cyanidation. The preliminary treatment to eliminate the zinc leaves nothing objectionable in the residue for smelting.

The details of this process I have promised to write up and read before one of the leading mining engineers' societies with the consent of D. Mosher, who suggested this process to me in the first place, as described by Schnabel in his 'Metallurgy of Copper, Lead, Silver, and Gold,' Vol. 1, and F. L. Wilson, who experimented for me with this and other zinc recovery processes at the University of California for several months before graduating in 1912, and has since worked faithfully developing the process on a larger scale. Many new ideas and improvements suggested themselves, differentiating it from the process as described by Schnabel after it had been in use for several years at the Hoboken works near Antwerp, where they must have had fused products and arsenic to contend with.

S. E. BRETHERTON.

San Fancisco, March 31.

ANALYTIC WORK AT COPPER QUEEN SMELTER

(July 30, 1910.)

The analytic work of the Copper Queen smelter, at Douglas, Arizona, is done under the direction of Percy Butler. The facilities are superior, both as to the laboratory building and its equipment. A large separate structure is provided, which is divided into numerous rooms, protected against dust and draught, well ventilated, and superbly lighted. The routine samples for furnace control are here determined, and also the large number of samples of custom ores are assayed and analyzed.

In making assays a preliminary 1-10 assay ton charge on all unknown ores is run in order to work out the appropriate flux. The amount of nitre (KNO_3) required is determined by multiplying the weight of the button obtained by 5, substracting 22, and dividing the remainder by 4½. This gives the grams of nitre needed for the excess of sulphur, so that the resulting button shall weigh 22 gm. When the sulphur is high silica is added, and if the silver content is suspected to be high, borax is introduced to insure an acid slag. If much copper sulphide be present an excess of PbO is provided. Every ore is assayed in duplicate, two assays each being made of the original and of the duplicate sample. Unless three of these agree within the limits of experimental error new samples are taken and re-assayed. The charge in the crucible is always covered with a non-reducer, which consists of bicarbonate of soda 8 parts, potassium carbonate 8, and raw borax 4. Part of the above is also mixed with the charge. The reducer used consists of the same flux containing an addition of 4 parts of flour. A typical charge for oxidized ores is as follows:

Ore	½	A. T.
PbO	76	gm.
Reducer	8½	"
Non-reducer	17½	"

This is mixed and covered with 8 gm. of non-reducer. The oxide ores may be represented by the converter lining used, which shows on analysis:

	Per cent.
Silica	80 to 85
Iron (FeO)	3
Alumina	3
Lime (CaO)	1
Copper	±2

The sulphide ores have the following approximate composition:

	Per cent.
Silica	26
Fe (metallic)	28
Lime (CaO)	2
Sulphur	22 to 23
Alumina	6

A normal assay charge for these ores is:

	Grams.
PbO	100
Non-reducer	17½
Silica	4
Nitre	13
Cover of non-reducer	8

By using an excess of PbO (from 100 to 120 gm.), with nitre in correct amount to give the required button, enough copper can be driven into the slag, even if as much as 10% Cu be present, to give a button which on cupellation will yield a nice feathered litharge, reducing the absorption and keeping the volatilization very low. A comparatively soft cupel is used. Cupellation is begun at a low temperature, feathering from the start, and this continues until it is close to the button. The finish is made at a high temperature.

Dry-wood blocks are placed in the muffle-mouth to open the button after melting. The heat is carefully regulated throughout, oil being used as fuel.

The determination of gold and silver in the copper bullion is done as follows: one assay ton in duplicate is weighed into a 16-oz. beaker, and 20 c.c. of a solution of mercuric nitrate is added. This is made by weighing 20 gm. of mercury and dissolving in 8 c.c. of HNO_3; it is boiled until neutral, so that a small amount of basic nitrate of mercury will precipitate on the addition of water. This is made up to 2200 c.c. with water. Take 25 c.c. of this with a pipette, and cover the bullion with it. This amalgamates the bullion. Add 80 c.c. of strong H_2SO_4 and boil to fuming. The copper goes into the solution and the gold and silver remain undissolved. Take up the copper sulphates with water, and when partly in solution add hot water. To collect the gold and silver add a small portion of lead acetate solution, and before filtering add 1 drop of HCl + Aq, half and half, as a precaution. Filter through double paper, and scorify the paper with test-lead and borax, aiming at an 18-gm. button, which is then cupelled. The results are from 0.04 to 0.08 oz. per ton higher by this sulphuric acid method than are obtained with the old nitric acid method. The copper bullion samples are taken by saw-cuts from a sample pig poured from each converter charge. The saw-cuts are made by a horizontal band-saw, in opposite directions, extending a trifle over half-way through the bar, the cuts being a fraction of an inch apart.

From the blast-furnaces come two samples daily, namely, a slag and a matte sample. These are composites, that is, they represent the day's run on all furnaces. There is also a 24-hour sample of slag and matte, which is of large size, and is sent to the sampling works for reduction. The analysis of the slag and matte samples is finished and reported within 1½ hours, the substances determined being silica, iron (FeO), alumina, and lime (CaO). The caustic soda method is used for the alumina.

The slag analysis is made as follows: ½ gm., finely ground in an agate mortar, is weighed into a small casserole. It is just wetted with water, a pinch of $KClO_3$ added, and 1 to 1½ c.c. HCl dropped on while agitating the casserole and stirring with a glass rod. It will gelatinize in one minute. It is then evaporated to dryness. This is dissolved with HCl and water, and filtered through an ashless filter paper with quick suction. It is dried, ignited, and the SiO_2 weighed. The filtrate is treated with a slight excess of ammonia and 3 gm. ammonium chloride; the solution is boiled and filtered. The CaO is determined in this in the usual way. The precipitate of ferric and aluminum hydrate is washed by a small jet of water into the original beaker, and 1½ inch of stick caustic soda (Baker's C. P. by alcohol) is boiled in it for 3 minutes. The alumina goes into solution as sodium aluminate. It is filtered through a double filter to retain all the iron. The filtrate is acidified with HCl, and a slight excess of ammonia (determined by phenol-phthalein) with 3 gm. of ammonium chloride, are added; the solution is boiled five or six minutes, and then filtered through a double filter using one ashless paper (No. 589 S. & S.) and one ash paper No. 597. The precipitate on the filter is washed three or four times, then dried and ignited, and the Al_2O_3 weighed. The iron is dissolved through the paper with HCl and titrated with bichromate of potassium, after reducing with stannous chloride and taking up the excess of stannous chloride with mercuric chloride. The time required is 1 hour and 5 minutes.

IODIDE METHOD OF COPPER DETERMINATIONS

By L. D. McClure

(July 8, 1911.)

As my experience in using this method has been so satisfactory and the results obtained, during a period of several years, have been so uniformly exact, it is my desire to give, in detail, to those who may be interested, a simple and rapid working scheme which I have employed in my work where a large number of copper determinations are made daily from churn-drill and mine samples, at the Live Oak mine, Globe, Arizona. The scheme, with a little practice, will be found more rapid than the permanganate or electrolytic methods, and the results are exact within very narrow limits of experimental error.

I will endeavor to explain in a lucid and concise manner just how I carry out my daily work in the laboratory, so that no difficulty will be found in following and working out each successive step of the process, as it is employed in actual practice in handling a large number of samples each day. The Live Oak ore has a copper content ranging from 2 to 5%, in the form of disseminated chalcocite in silicious schist. In handling more complex ores and furnace products, modification of the process would be advisable.

Method of Standardizing.—Prepare a solution of sodium thiosulphate, containing 40 gm. of pure crystals to 2 litres of distilled water, and proceed as follows: accurately weigh three portions of 0.10 gm. each of pure copper foil into beakers of 300 c.c. capacity; add about 1 c.c. of strong nitric acid to each beaker; place upon the hot plate, and evaporate off the excess of nitric acid until almost neutral copper nitrate is obtained, which will be indicated when by tilting the beaker the solution will flow rather sluggishly over the bottom. The evaporation must not be carried beyond this point, as there is danger of producing some insoluble basic copper nitrate. Remove the beakers from the hot plate, dilute with about 25 c.c. of distilled water; stir in enough of a saturated solution of sodium carbonate to cause a slight precipitate; add 2 c.c. glacial acetic acid and then 3 or 4 grams of potassium iodide in crystals. By gently shaking the beaker the potassium iodide will dissolve almost instantly. Cuprous iodide will be precipitated and iodine, exactly proportional to the amount of copper present, will be liberated, and be held in solution in the excess of potassium iodide, imparting to the solution a reddish brown color, according to the reaction:

$$2Cu(C_2H_3O_2)_2 + 4KI = Cu_2I_2 + 4KC_2H_3O_2 + 2I$$

Proceed at once to titrate. Run the sodium thiosulphate solution into the beaker, constantly stirring until the iodine solution changes to a faint yellow color, and then add sufficient starch solution to produce a deep-blue coloration. Continue the titration cautiously, drop by drop, with constant stirring. Stop when the solution assumes a violet color and continue stirring for a few seconds. If the violet color does not rapidly change to a creamy white tint at this point, another drop from the burette will be required.

It will be found from the burette readings that about 20 c.c. of the thiosulphate solution has been required for each of the three beakers, or 1 c.c. equals 0.005 copper, approximately. If, however, there should appear a slight variation in the three readings, I use the average of the three readings to calculate my standard.

Treatment of Ores.—Weigh out 3 grams of low-grade ore, or proportionately less for ores of higher grade, into a 300-c.c. beaker. Add from dispensing burettes 10 c.c. strong nitric acid and 5 c.c. sulphuric acid. Put the beaker on the hot plate and boil nearly to dryness. Remove the beaker; cool somewhat; add 2½ c.c. sulphuric acid, which produces a very satisfactory degree of acidity for the copper precipitation; dilute with 25 c.c. distilled water; stir with a rubber-tipped glass rod to loosen any cake that may have formed on the bottom of the beaker. Place the beaker again on a warm part of the hot plate, and digest for about 10 minutes to bring the sulphates into perfect solution; then filter through a 15 cm. filter paper into a 150-c.c. beaker. Wash the insoluble residue on the filter twice with boiled water. Then pour filtrate back into the original beaker and rinse the filtrate beaker with a little water.

The filtrate should now measure about 75 c.c. and should fill the beaker to about one-fourth its capacity. Next place in the bottom of the beaker a piece of aluminum 3 in. long, 1 to 1½ in. wide, and bent at right angles at or near the middle, so that one-half lies in the bottom of the beaker and the top of the other half extends a little above the surface of the solution. Cover the beaker and boil for about 10 minutes, when the copper will be precipitated; the solution will be perfectly colorless; and the aluminum will appear clear and bright. During the last few minutes of boiling, the sulphuric acid attacks the aluminum vigorously, and the minute bubbles of hydrogen gas which are given off have a tendency to free the aluminum of all small adhering particles of copper.

Now take the beaker from the hot plate, and remove the aluminum from the solution with the forefinger and thumb of the left hand, at the same time playing a small jet of cold water upon it to cool and wash off any adhering particles of copper. Carefully decant the solution, leaving the copper on the bottom of the beaker, and wash twice by decantation with cold water. Dissolve the copper in about ½ c.c. of strong nitric acid, and place the beaker again upon the hot plate to evaporate the excess of nitric acid, and then proceed in the same manner as described for standardizing. The starch solution is made by rubbing 1 gm. of cornstarch into a smooth paste with a little distilled water, diluting with distilled water to 200 c.c., and boiling the solution for a minute. Cool before using. The paste may be kept for some time if a small quantity of lysol or other antiseptic be added to prevent the growth of mold.

THE DETERMINATION OF COPPER IN COPPER-BISMUTH ORES

By C. C. O'Loughlin

(August 20, 1910.)

The following method for assaying ores for copper that carry from 0.5 to 3% bismuth, is used at the Bogardus Testing Laboratories, at Seattle. It may be of interest to others, since no data on methods suited to these conditions are generally available.

At first the bismuth was precipitated with ammonium carbonate, making two separations, but this was not found satisfactory, as the separation was not complete. The filtrates were run electrolytically, but deposits were found to turn into a dark sponge, and drop off, even with different current density, because of bismuth still being retained in small amounts. The iodide method with the above separation, was found to give low and erratic results. It was then thought possible to precipitate the bismuth as the sub-nitrate, plating the copper direct in the filtrate; but the separation was not found suitable. The following method was finally adopted and found to be highly satisfactory.

Treat 0.5 to 1 gram of pulp with 8 c.c. HNO_3, digest at low heat until fumes cease, add 2 c.c. H_2SO_4 and evaporate to SO_3 fumes. When cool, wash down cover and sides of beaker, first add ammonia, and then acidify with HCl, adding a very slight excess. Keep the volume as low as possible. Have a 500-c.c. beaker ready, containing about 250 c.c. hot water, and a little ammonium chloride. Pour assay into this slowly with constant stirring and allow to settle one-half hour. Filter off the insoluble and the bismuth, which has been precipitated as the oxychloride, wash well, add 2 c.c. H_2SO_4 to the filtrate, and precipitate the copper with sodium thiosulphate. Filter, wash, burn precipitate, brush into a copper beaker, and dissolve with 5 c.c. HNO_3, heat until dissolved, dilute with water, add 3 c.c. H_2SO_4, and plate $N.D._{100}$ 0.10 amperes.

If preferred, the copper can be titrated with sodium thiosulphate by taking the solution nearly to dryness. After dissolving the copper precipitate HNO_3 neutralize with sodium carbonate, acidify with acetic acid, and titrate. Bismuth can be determined in the residue from first filtration by dissolving away from the silica and precipitation as oxychloride, burning and weighing as oxide.

ACCURATE 'SLOP-COPPERS'

By A. J. Sale

(January 27, 1912.)

The volume of work around large copper mines and mills, necessitating the employment of a system of assaying for mine or drill samples which is more rapid than the iodide or electrolytic methods, naturally forces the assayer to fall back on the so-called 'slop-coppers.' But the average chemist is apt to conceive the idea that slop-coppers means sloppy coppers, and to work accordingly. The 'slop' method, if proper precautions are observed, will give accurate results even on low-grade refractory ores.

The first consideration for accurate work is that, with ore assaying from 1 to 3%, the amount of copper contained in the ore is relatively so small that from a 1 gm. assay the solution will be so dilute at the time of titration that it will be difficult to obtain an accurate end-point. By starting with 2 gm. I consider that the large bulk of material handled in the assay is more than compensated for by the advantages obtained. The next important consideration is that some portion of the copper is usually closely associated with the iron pyrite, which is far more soluble in aqua regia than in nitric acid. The course of operations that I have found to give best results is to put 2 gm. of pulp (through a 100-mesh screen) into a casserole and boil nearly to dryness with 15 c.c. of aqua regia. Then add 10 c.c. of nitric acid and boil until it becomes fairly syrupy. Then add about ½ gm. of chlorate of potash and boil for five or six minutes. This will sometimes throw down a dark brown precipitate which is probably mostly manganese dioxide. I have found from a large number of results that the proper manipulation of chlorate will always prevent the yellow precipitate which sometimes forms when titrating, and causes difficulty in reading the end-point.

The assay is washed from the casserole into a small beaker with boiling water and allowed to cool until it will not 'spit' too violently when ammonia is added. The ammonia is put in, a little at a time with constant stirring, until the iron is completely precipitated. It is very poor practice to add ammonia directly in the casserole, as it is then difficult to see the precipitate, also to wash it out.

The assay is now filtered into a titrating flask and the precipitated iron washed with boiling water. The iron always contains some of the copper present and should be washed into a beaker and nitric acid added drop by drop until the precipitate is almost dissolved. It is usually better not to completely dissolve it, as this may again put the manganese into solution. Ammonia is now added until all of the iron is re-precipitated, and then the material is filtered through the same paper into the titration flask. The precipitate is washed with boiling water, and if the filter paper shows any signs of blue color the re-solution and re-precipitation is again repeated.

For standardizing, the best results are obtained when the amount of copper in the standard is approximately the same as in the assay; for instance, if the ore runs 3% the assay from 2 gm. will contain 60 mg. of copper, which should be approximated when weighing the standard. Also it is more accurate to add approximately the same amount of foreign elements to the standard as will occur in the ore and carry it through the same as if it were an assay. It is more rapid to find the exact weight of a piece of copper than to try to cut it to some exact weight, such as 40 or 50 mg., only, as before stated, it should be approximately of the same weight as the copper in the assay. If we start with 2 gm. of pulp, then each 20 mg. of copper is equivalent to 1%. It is much less confusing to divide the weight of the standard by 20 and record as an equivalent percentage than to keep its exact weight. For instance, if the weight is 44.6 mg., record as 2.32%. Two standards should be weighed and one titrated at the beginning of a set of assays and the other at the finish. In titrating, the solution should be at the temperature of the room and the same amount of time and agitation given to each. It is also better to stop at a light pink end-point than to take out all the color; the important thing is to have the same end-point for all assays and standards. To avoid errors in calculation, the results should be recorded in the form shown below:

No.	Burette reading.	Net reading.	Percentage copper.
2.76% standard	10.15	10.15	0.272 per c.c.
1	21.0	10.85	2.96
2	30.6	9.6	2.62
3	41.35	10.75	2.94
4	50.7	9.35	2.55
2.15% standard	58.55	7.85	0.274 per c.c.

The mean strength will be 0.273% copper per c.c. In rapid work there is possibility of an error of subtraction or multiplication; a quick check on the arithmetic is obtained by adding the final assays. The sum of these should equal their total net burette reading, times the mean strength of solution. In the above table the sum of the assays from No. 1 to No. 4 is 11.07%, and the total net burette reading is 50.7 - 10.15 = 40.55 c.c. Now, 40.55 × 0.273 = 11.07, which checks the calculations.

The water used in the analysis is an item of considerable importance. Most of the so-called 'distilled' water around a mine or mill is condensed steam, and frequently contains oil or boiler compounds which may seriously affect the results. The most satisfactory water that I have found, where pure distilled water is not obtainable, is secured by adding a little ammonia to ordinary spring or hydrant water and boiling until all of the excess of ammonia is gone. This, when decanted from the precipitated lime and other salts, will nearly always gives a good end-point. The final precaution for good results is the use of pure potassium cyanide. I have found from a number of experiments with partly decomposed cyanide that its strength is liable to vary with the part of the burette from which it is drawn, being stronger at the bottom of the burette than at the top, or the reverse. These precautions may seem to be trivial, but on refractory low-grade ores they are virtually essential, and if carefully followed, I think, will give practically as good results as the average iodide or electrolytic assay.

THE GREAT FALLS ELECTROLYTIC PLANT

By Willis T. Burns

*This is not a discussion of modern practice of electrolytic copper refining, but a record of a refinery that was among the pioneers in the field and that is today, and has been for 17 years, operating under conditions not to be found in any similar plant in the country. I refer to the current density, which is at least two-thirds higher than that regularly employed elsewhere, and to the fact that the anodes treated are the direct product of the converter rather than of the reverberatory furnace. I have no knowledge of any other refinery treating anodes of this character. The refinery was built at the smelting works of the Boston & Montana Consolidated Copper & Silver Mining Co. at the Black Eagle falls of the Missouri river, three miles from the city of Great Falls, Mont. Power for the operation of the reduction works and the electrolytic plant was obtained from water power developed at this point.

As originally designed and constructed the refinery consisted of the tank house, a brick building 171 ft. 6 in. by 108 ft., which contained 288 electrolytic-refining tanks of the following inside dimensions: 9 ft. 7 in. long, 2 ft. 4 in. wide, and 3 ft. 9 in. deep, built of Western pine. The sides and bottoms were of 3-in. lumber and the ends of 2-in. material. The tanks were tied at the ends with 3 by ½-in. iron straps held together by 1-in. rods in the usual way. The tanks were lined with ⅛-in. chemical sheet lead weighing 8 lb. per square foot. The tanks, which were supported in place by two 8 by 8 in. timbers which rested on stone piers 18 in. high, were insulated from the timbers by 4 by 4 by 1 in. glass insulators, eight under each tank. Had 5 to 6 ft. of head room, instead of 18 in., been provided under the tanks, subsequent operations would have been greatly facilitated. The tanks were arranged in 18 double rows, eight tanks deep with an aisle between each double row. The tanks as originally built were all on the same level, making it necessary to establish an individual system of circulation of the electrolyte for each tank. This was accomplished by providing two circulating tanks for each section of 96 tanks; these tanks were 10 ft. square by 8 ft. deep, lined with 12-lb. chemical sheet lead. One of these tanks was placed above the level of the refining tanks and one below, as shown in sketch. From the overhead tank the electrolyte was delivered, through an overhead system of lead pipes,

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*Excerpts from a paper presented at the Montana meeting of the A. I. M. E. August, 1913, reprinted by permission.

to each separate refining tank. The solution was discharged into the refining tank at one end, near the bottom of the tank, and was allowed to overflow through a 1-in. lead pipe entering the tank 3 in. from the top near the other end. This pipe discharged into an open launder leading to the lower circulating tank, from where it was raised to the overhead tanks by various devices to be described later. This system, which permitted only a periodical circulation of the solution, soon gave way to a more efficient system to be described further on. Heating the electrolyte was effected by passing live steam through 125 ft. of 1-in. 8-lb. lead pipe in each of the overhead circulating tanks.

A ¾ by 4 in. steel trolley track was suspended from the roof timbers over each row of eight tanks. The anodes and cathodes were removed in and out of the tanks by means of chain blocks suspended from two-wheel trolleys on this track.

The current was distributed by means of ¾ by 4 in. rolled-copper busbars attached to the sides of the tanks with wooden brackets. The tanks were all connected in series and the electrodes in multiple.

The original installation included a sulphate plant for the manufacture of bluestone and a boiler house containing one 100-hp. locomotive-type boiler to furnish steam for heating and boiling solutions.

The sulphate plant was well appointed for the manufacture of bluestone as described later. The capacity of this plant was 2,500,000 lb. of bluestone per year.

Fig. 60 is a sketch of the plant as it exists today. One division of 32 refining tanks has been added to the main tank room, 44 new electrolytic-refining tanks for the production of starting sheets have been housed in additions as have the 10 new insoluble-anode tanks shown in sketch. An addition has been built south of the tank room in which is installed the equipment for pumping and heating the electrolyte. An addition has been built on the east containing the scale house and a 7-ton traveling crane for handling the cathodes and scrap after weighing. The addition on the north side of the building contains store room, shops, and office. Fig. 61 is a flow sheet of the electrolytic plant. In the new building south of the bluestone building are the carpenter and lead-burning shops and the machinery used in trimming sheets. The four large round tanks situated farther south are for the storage of solutions. The original boiler house has been replaced by a brick building containing two 150-hp. Heine and one 100-hp. Scotch marine boiler from which steam for heating and boiling solutions is obtained.

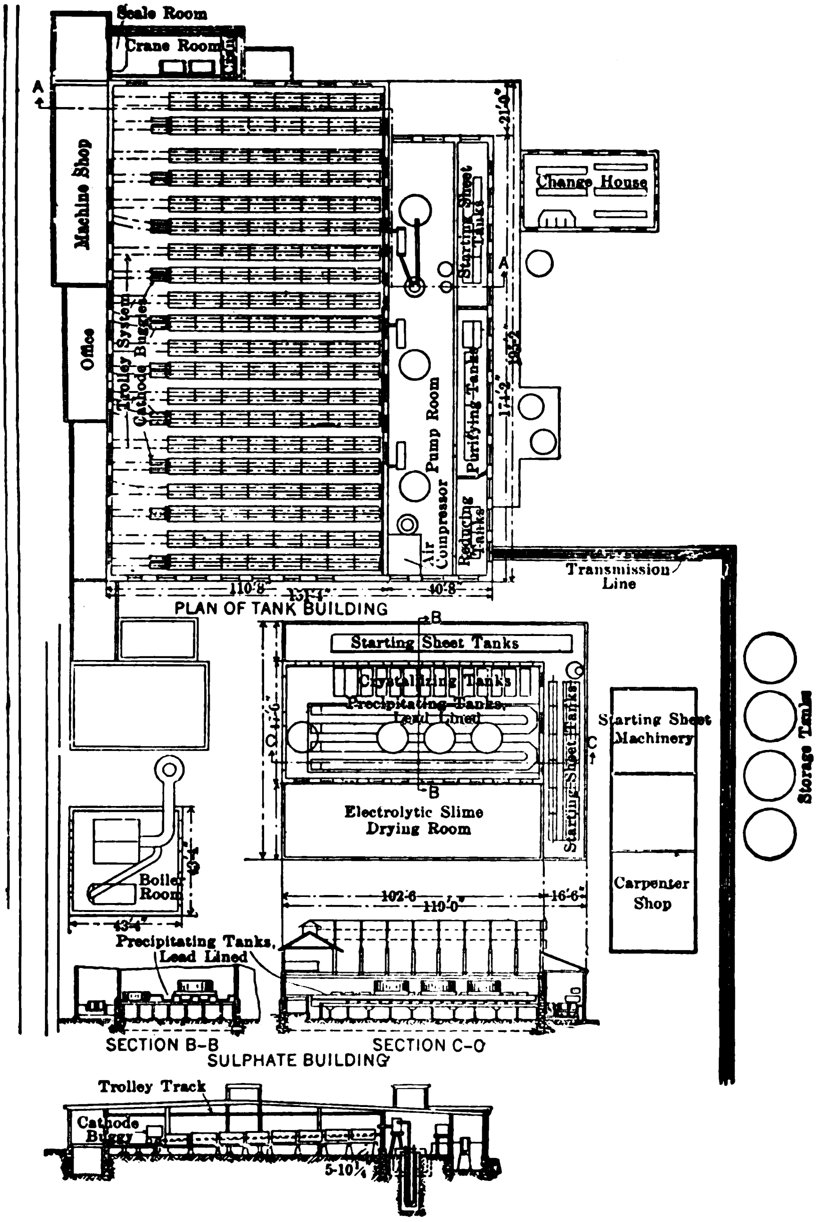

Fig. 60. Great Falls Electrolytic Plant as Remodeled.

As previously stated the refining tanks, as originally installed, were all on the same level and to obtain a circulation of the electrolyte it was necessary to deliver the electrolyte to each tank through a separate pipe line. This system was found unsatisfactory and the refining tanks were rearranged in a cascade of eight tanks with a drop of 2¾ in. between tanks as shown in Fig. 60. The electrolyte is now delivered to the upper tank of the cascade at the surface and is drawn off from a point 8 in. from the bottom at the lower end of the tank by means of the dam shown and is delivered to the next tank through a cast lead and antimony chute. From the last or lower tank of the cascade the electrolyte is returned to the pump room through a 2-in. 7-lb. lead pipe attached to the sides of the tanks.

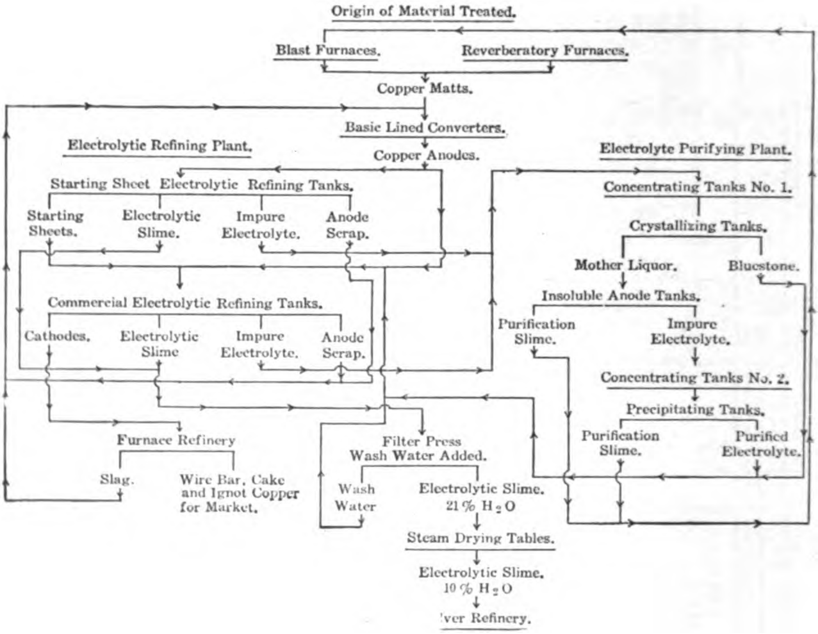

FIG. 61. FLOW-SHEET OF GREAT FALLS ELECTROLYTIC PLANT.

The original refining tanks were lined with ⅛-in., 8-lb. chemical sheet lead and the overflow consisted of a 1-in. lead pipe burned to the lining. Tanks constructed in this manner were found to last about 10 years in service. As they gave out they were replaced by tanks of the same dimensions lined with ⅛-in., 7-lb. lead containing 6 per cent antimony. The cast weirs of the same material were built into the ends of the tanks. Tanks thus constructed have now been in service for 11 years and do not show much deterioration. The antimonial lead is harder and has a less tendency to creep and buckle under extreme changes of temperature than is the case with the chemical lead. Practically all of our tanks are now lined with antimonial lead.

When the change from 4000 to 9000 amperes was made in 1896 it became necessary to increase the size of the tank busbars. The ¾ by 4 in. rolled-copper conductors were removed from the sides of the tanks and a taper bar 4½ by 2 inches in the centre and 1½ by 3 in. at the ends was laid on the top of the tanks. The bars were cast at the works from wire-bar copper. These bars are insulated from the tanks by means of ½ by 4 in. pine strips previously boiled in pine tar. These strips are renewed once in six months. Into the ends of the busbars ⅞ by 6 in. iron studs are screwed to receive the copper shunt used to cut four tanks out of circuit for removal of slime. The anode and cathode rods and the connection plates, used between the tanks, are also cast at our furnace refinery in copper molds.

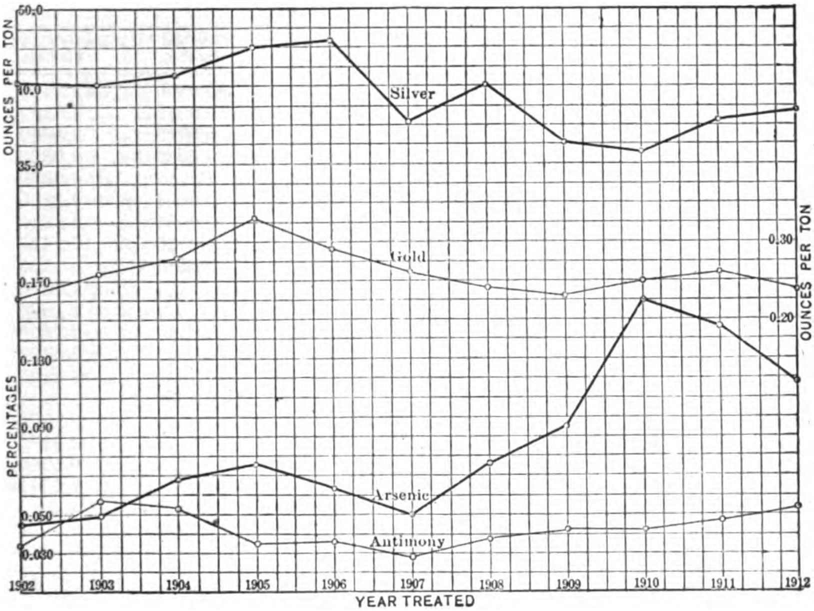

FIG. 62. CONVERTER ANODES.

As previously mentioned this refinery is the only plant of its kind treating converter-copper anodes. The general practice elsewhere is to transfer the molten copper from the converter to the reverberatory furnace. Here the necessary oxidizing and poling of the copper to bring it to the proper pitch for casting takes place. The copper is then cast into anodes by means of a suitable casting machine. This refining process has the effect of increasing the copper contents of the metal 0.3 to 0.4%, sulphur dioxide being the principal impurity eliminated. The resulting anode is much denser than the converter anode and the casting is free from the uneven surfaces which appear on the converter anode. The analyses of anodes are shown in Table I and Fig. 62.

TABLE I.—*Analyses of Anodes, Electrolyte, Wire Bar and Electrolytic Slime*

	Converter Anodes Per Cent.	Electrolyte Per Cent.	Wire Bar Per Cent.	Electrolyte Slime Per Cent.
Copper	99.1300	3.280	99.9500	43.3400
Arsenic	0.1183	0.500	0.0016	3.0300
Antimony	0.0534	0.041	0.0015	3.4600
Nickel	0.0420	0.377	0.0006	0.0800
Cobalt	0.0018	0.016	Trace	0.0060
Bismuth	0.0038	0.021	0.0004	0.1100
Iron	0.0110	0.600	0.0006	0.3640
Silver	0.1371	None	0.0030	17.1870
Gold	0.0008	None	Trace	0.1200
Selenium	0.0090	None	.....	1.2000
Tellurium	0.0170	None	.....	2.1000
Lead	0.0065	Trace	Trace	0.7600
Zinc	0.0035	0.418	0.0001	0.0900
Sulphur	0.2610	....	0.0025	13.2100
Oxygen	.....	....	0.0350	.....
Silicon	.....	....	.....	0.1770
Chlorine	.....	0.0040	.....	0.0260
Carbon	.....	....	.....	0.5900
Platinum	.....	....	.....	0.000166
Free sulphuric acid	.....	13.0300	.....	.....
Specific gravity	.....	1.220	.....	.....

Table II shows one of a number of comparisons, made at this plant, between converter and refined anodes. In conducting this test three divisions, of 32 refining tanks each, were employed. One-half of the tanks in each division were treating refined anodes while the other half contained converter anodes. By this arrangement the effect of the personal equation of the men in charge of the tanks was reduced to a minimum. It will be noted that while the ampere efficiency of deposit was 3.6% higher in the refined-anode tanks than in the tanks containing converter anodes the production per kilowatt hour was slightly higher in the converter-anode tanks. Subsequent experiments have shown that the refined anodes admit of closer spacing than do converter anodes, without the same reduction in ampere efficiency, the effect of which is to decrease the resistance and increase the production per kilowatt. The difference, however, was not found to be great.

The advantages in favor of the refined anodes, as they appear in Table II, are: First, higher grade of slime resulting in a saving in cost of slime refining; second, lower silver content of cathodes, all of the silver in the cathode representing a dead loss; third, the lower percentage of anode scrap. When reduced to dollars per ton of copper refined the sum of the savings that would result from the use of refined anodes, in a plant of this type, was found to be less than one-half the cost per ton of the reverberatory-furnace treatment of the copper. The cathodes produced from the two types of anodes differ but little in physical appearance.

In a crane-operated electrolytic refinery, where the anodes and

TABLE II.—*Comparison of Converter and Refined Anodes Cast in the Same Molds*

	Converter Anodes.	Refined Anodes.
Number of days covered by test	50	50
Number of refining tanks employed	48	48
Average Analyses of Anodes:		
Cu, per cent	98.91	99.27
As+Sb, per cent	0.072	0.071
Silver, oz. per ton	59.09	61.14
Gold, oz. per ton	0.200	0.219
Average Analyses of Electrolyte:		
Specific gravity	1.20	1.20
Copper, grams per liter	43.5	43.5
Free acid, grams per liter	160	160
As, grams per liter	11.97	11.97
Sb, grams per liter	0.49	0.49
Fe. grams per liter	10.09	10.09
Cl, grams per liter	0.045	0.045
Average Temperature of Electrolyte:		
Inlet of 8-tank cascade, C°	58	58
Outlet of 8-tank cascade, C°	54	54
Rate of circulation of electrolyte, gal. per min	6	6
Anodes per tank	20	20
Cathodes per tank	20	20
Average weight per new anode, lb	525	632
Average thickness per new anode, in	3	3
Distance, center of anode to center of cathode, in.	2.87	2.87
Active cathode surface per tank, sq. ft	252	252
Average amperes per tank	8,387	8,387
Average volts for 48 tanks	27.21	28.53
Average volts per tank	0.567	0.594
Average kilowatts for 48 tanks	228.2	239.3
Total copper deposited, lb	1,103,749	1,148,749
Average age of cathodes drawn	2½	2½
Average ampere efficiency of deposit, per cent	88.3	91.9
Average amperes per sq. ft. cathode surface	33.3	33.3
Average lb. copper deposited per kilowatt-hour	4.03	4.00
Average silver in cathodes, oz. per ton	1.25	0.95
Average As+Sb in cathodes, per cent	0.0043	0.0043
Average anode scrap, per cent	8.00	5.30
Analyses of silver slime:		
Cu, per cent	40.3	18.80
Ag, oz. per ton	6,755.00	14,079
Au, oz. per ton	18.34	38.45

cathodes are handled in tank units and not individually as here, difficulties would result if converter anodes were used because of the fact that converter anodes corrode less uniformly than refined anodes. Many of the converter anodes would become scrap before the date set for drawing the scrap from a tank and an equal number of pieces would not be scrap at the appointed time for drawing. The above comparison, together with other tests made at this plant, has demonstrated that refined anodes, while very desirable, would for this plant, be an expensive luxury.

Figures 63, 64, and 65 show the earlier form of anode used in this plant and the Morrow clip type[1] now used. The advantage of the clip type of anode is that the amount of inactive copper above the solution line is much less than is the case with the lug type. The average of anode scrap for the year 1912 was 5.9%; the average weight of the converter anode is 500 lb. The scrap resulting from each anode weighs about 30 lb.; this is resmelted in the converters. The clip type of anode also has the advantage, as compared with the lug type, of being suspended in such a manner as always to hang vertical in the tank. The standard converter anode measures 24.5 by 35.75 inches.

The clip or hanger is made from a ¼-in. round copper rod 60 ft. long rolled in a rod mill from a bar cast at the refining furnace. This rod is cut and bent into the loop form, in an automatic machine, and placed in the anode mold. Both the anodes and cathodes are supported in the tanks by cast copper rods as shown in sketch. The anodes are spaced 5.2 in. centre to centre. It will be noted that the anode is 3 in. thick at the top and 2.5 in. thick at the bottom. It is found that the anode corrodes more rapidly at the upper end where the current enters than at the lower end, that the wedge shape yields a lower percentage of scrap and that the anode retains its original shape for a longer time than is the case with an anode of uniform thickness.

The starting sheets are produced in 44 standard refining tanks divided into two circuits connected in parallel in the main circuit and are thus operating at one-half the current density of the commercial refining tanks. Each tank contains 21 cathode blanks and 22 anodes. The blanks are made of ¼-in. rolled copper 27.5 by 39.5 in. below the solution line, as shown in Fig. 66. The anodes used in these tanks are of the same type as the standard anode but larger, being 26.5 by 38⅝ inches.

The 12-hr. sheets are stripped at an average weight of 4 lb. each.

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[1] U. S. patents No. 621,121 and 631,471.

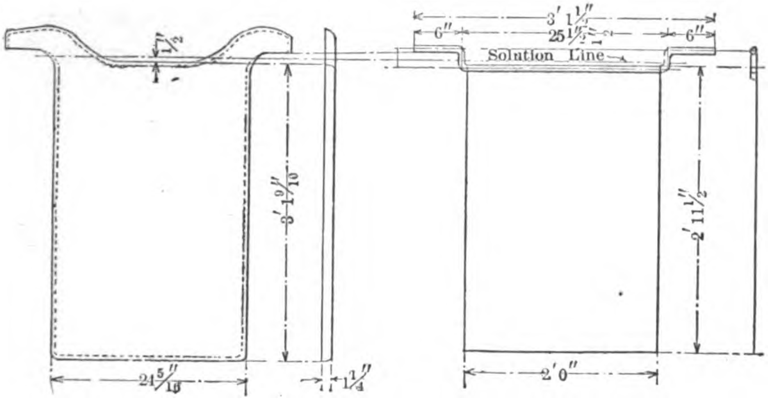

FIG. 63. ORIGINAL FORM OF ANODE; AND ORIGINAL FORM OF CATHODE.

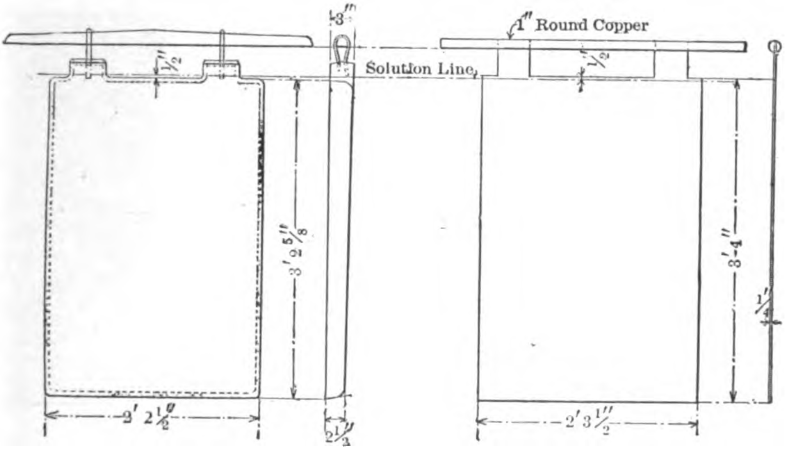

FIG. 64. STARTING SHEET ANODE, MORROW CLIP TYPE; AND ROLLED COPPER, STARTING SHEET BLANK.

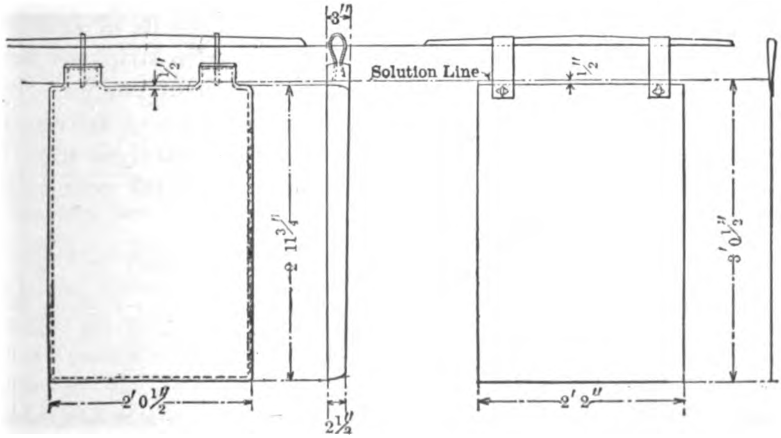

FIG. 65. STANDARD ANODE, MORROW CLIP TYPE; AND PRESENT FORM OF CATHODE, MORROW CLIP TYPE.

measuring 26 by 36.5 in. after trimming. The men employed in stripping the plates do their work directly over the tanks using a traveling bench suspended from the trolley track as shown in Fig. 66. This bench, on which the sheets are piled after stripping, is moved from tank to tank as the men advance. The blanks are covered with a thin coating of low-grade mineral oil to which 0.5 lb. of artificial graphite per gallon of oil is added. Grooved wooden strips are used on the edges of the plates to prevent deposition of copper. The strippers work independently and each man strips and delivers to the trimmer 360 sheets per 8-hr. shift. The electrolyte used in these tanks differs slightly from that of the main tank room. An average analysis is given in Table III.

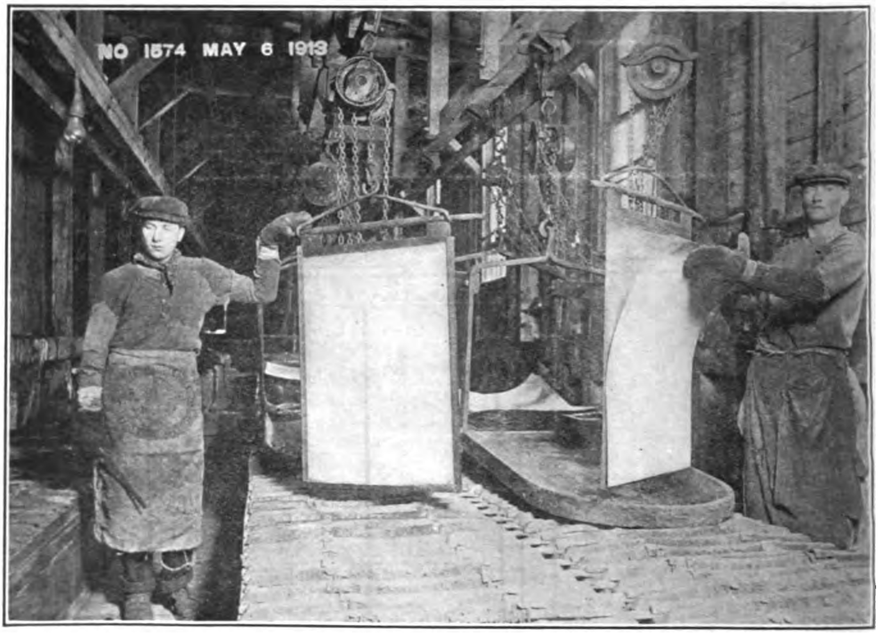

FIG. 66. STRIPPING STARTING SHEETS.

TABLE III.—*Starting-Sheet Tank Electrolyte*

Specific gravity	1.175
Free H_2SO_4, gm. per litre	120.0
Cu, gm. per litre	40.0
As, gm. per litre	5.0
Sb, gm. per litre	0.4
Fe, gm. per litre	4.5
[illegible] per litre	0.04

Table IV shows the effect of sulphuric acid in the electrolyte on the efficiency of deposition, the production per kilowatt-hour increasing as the acid increases. The sheets produced when the acid is about 120 gm. per litre are found to be smoother and tougher than when acid is higher. The starting sheets are trimmed on the four edges to prevent short circuits. Two clips or loops, cut from similar sheets, are then attached to each sheet by means of the Morrow clip machine[2].

TABLE IV.—*Effect of Free Sulphuric Acid on Starting-Sheet Production*

Converter anodes, 39.5 by 27.5 by 0.25 in. blanks, 3 in. anode; 2.5 in. center of anode to center of blank.

Circulating electrolyte at 4 gal. per minute.

Amperes.	Amperes per Sq. Ft.	Efficiency.	E. M. F. per Tank Volts.	Lb. Cu per Kilowatt-hour.	Average Weight per Sheet.	Electrolyte		
						Gal. per Liter Free H_2SO_4.	Cu.	Temperature C.°
4,580	16.2	86.5	0.475	4.75	5.86	74	42.6	52.5
4,490	15.9	86.5	0.462	4.84	5.78	74	42.6	52
4,453	15.7	85.2	0.462	4.80	5.61	70	40.1	55
4.450	15.7	83.6	0.475	4.55	5.50	69	39.8	49
4,390	15.5	90.3	0.482	4.84	5.87	69	40.1	50
4,403	15.6	90.2	0.496	4.71	5.87	73	38.4	47.5
4,290	15.2	88.2	0.462	4.94	5.60	80	38.9	51
4 397	15.5	91.6	0.470	5.04	5.93	82	38.8	52
4,538	16.0	90.3	0.476	4.90	6.06	88	38.8	51
4.461	15.8	91.9	0.450	5.28	6.06	89	39.9	52
4,408	15.6	92.5	0.426	5.63	6.03	110	40.4	52
4.506	16.0	92.5	0.452	5.29	6.17	109	38.9	53
4.496	15.9	91.7	0.424	5.55	6.10	116	40.6	53
4,512	16.0	90.3	0.426	5.50	6.02	126	40.5	50
4,550	16.1	89.7	0.420	5.53	6.04	128	40.4	48
4,430	15.7	90.7	0.400	5.86	5.94	129	42.6	48
4.435	15.7	84.7	0.385	5.69	5.56	128	42.0	48
4,420	15.6	87.3	0.365	6.09	5.62	126	42.4	50
4,445	15.7	91.8	0.357	6.23	6.03	124	43.6	48
4.453	15.7	85.1	0.360	6.13	5.60	132	42.1	46
4,404	15.6	87.0	0.342	6.55	5.66	148	40.8	49
4.450	15.7	88.6	0.350	6.62	5.82	149	42.3	46

NOTE: The sheets produced with between 116 and 126 g. per liter of acid were the toughest and best.

The starting sheets are now ready for the tanks. It will be noted that the cathodes hang ¾ in. lower in the tank than the anode and are 1.5 in. wider than the anode. When using anodes and cathodes of the same size a cathode very rough on the edges resulted; increasing the size has resulted in a cathode with comparatively smooth edges and an increase of 5% in ampere efficiency. When

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[2] U. S. Patent 600,498, Mar. 8, 1898.

the output of the generators is 2000 kw. or more the best results are obtained while drawing 2 and 3-day cathodes. With a lower generator output older cathodes are drawn. The generator output is governed by the head on the water wheels, which is dependent upon the stage of water in the river. To determine the most economical age of cathodes and number of electrodes per tank at which to operate, the labor, the production per kilowatt-hour, the silver lost in cathodes, and the impurities in the anodes and cathodes must be considered. Assuming the quality of the product as regards impurities to remain satisfactory under the different conditions the cost per ton remains as the only consideration. Table V is a summary of various comparisons made between different ages of cathodes with varying numbers of electrodes per tank.

Table V.—*Effect of Varying the Age of Cathodes and the Number of Electrodes per Tank*

Age of Cathodes. Days.	Electrodes per Tank.		Average Amperes.	Average Amperes per Sq. Ft.	Ampere Efficiency. Per Cent.	Cu per Kilowatt-hour. Lb.	Cathode Oz. Ag per Ton.	Analyses As + Sb Per Cent.
	*Anodes.	Cathodes.						
4	20	20	9,300	36.9	88.0	3.93	1.32	0.0036
2	20	20	8,808	35.0	90.85	3.72	0.83	0.0030
3	20	20	8,877	35.2	89.00	3.75	0.83	0.0032
2	21	21	9,035	34.1	90.90	3.84	0.89	0.0032
3	21	21	9,223	34.8	89.40	3.87	1.02	0.0029
2	22	22	9,071	32.6	90.50	4.02	0.89	0.0033
3	22	22	9,167	33.0	88.80	4.07	0.95	0.0030

*Converter anodes. Average analysis of anodes: Cu, 99.13; As, 0.127; Sb, 0.055 per cent.; Ag, 33.97 oz. per ton; Au, 0.22 oz.

The 320 commercial refining tanks are divided into three groups, two groups containing 96 tanks each and the third group 128 tanks. Each group is provided with a separate electrolyte and circulating system. Copper and acid determinations are made daily on samples of each electrolyte. As, Sb, Fe, and Cl determinations are made weekly. The two groups of starting-sheet tanks each have a separate electrolyte which is sampled and assayed in the same manner as the commercial-tanks electrolyte. The analysis of the electrolyte in the commercial-refining tanks is shown in Table I in terms of percentage in conformity with the analysis of the other materials shown in the table. The analysis of solutions is ordinarily reported in terms of grams per litre. With a current density of 34 amperes per square foot and a circulation of 6 gal. of electrolyte per minute the best results, while treating converter anodes, are obtained with an electrolytic carrying 40 gm. per litre of copper and 160 gm. per litre of free sulphuric acid, The circulation of the electrolyte is through a cascade of eight tanks as already described. In the pump room the electrolyte is delivered to the receiving tank, a round lead-lined tank 11 ft. in diameter and 4.5 ft. deep (see Fig. 67). In this tank are 150 ft. of 1-in. 8-lb. antimonial-lead pipe; live steam is supplied to these coils at a pressure of 65 lb. per square inch, by means of which the electrolyte is heated. From the receiving tank the electrolyte is raised to the discharge tank from which it is again conducted to the head tank of the cascade.

Various means of elevating the electrolyte have been employed in this plant. The first method used was the steam injector con-

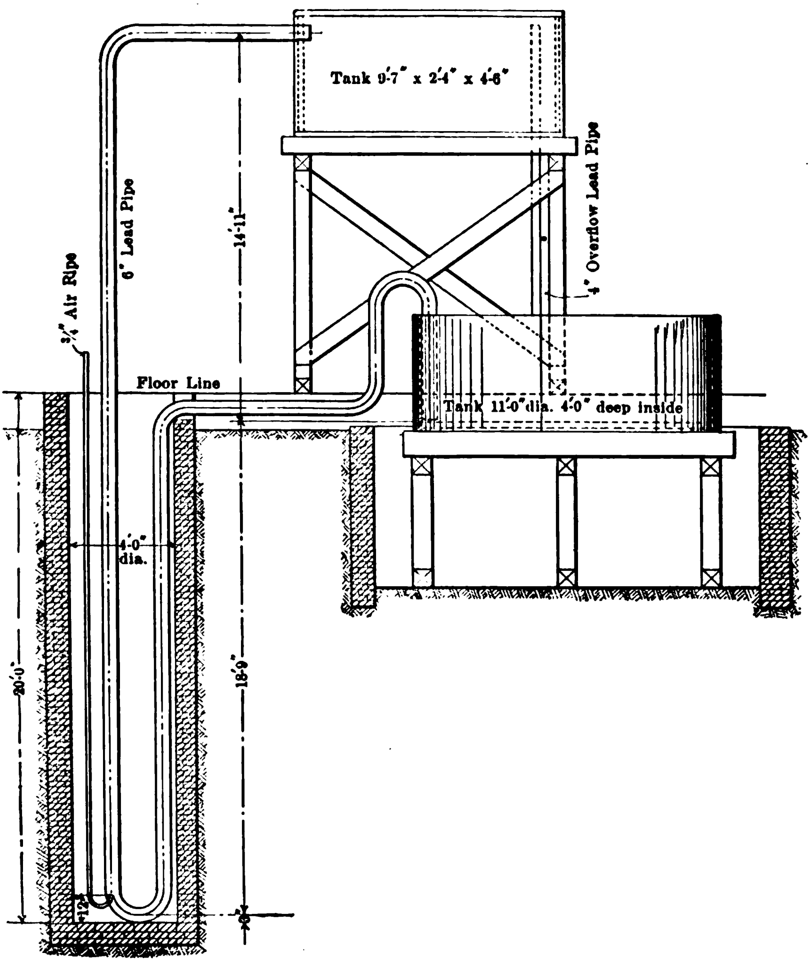


FIG. 67. ELEVATION OF SOLUTION AIR-LIFT PUMP.

TABLE VII.—*Effect of Speed of Circulation of Electrolyte on the Concentration of the Solution*

Current density 36.0 amperes per square foot, 4-day cathodes, converter anodes. Circulating Electrolyte at Rate of 6 gal. per minute.

Zone.	Specific Gravity.	Grams per Liter.						Copper per Kilowatt-hour. Lb.
		Acid.	Cu.	As.	Sb.	Fe.	Cl.	
1	1.213	174	38.4	7.5	0.56	2.95	0.044	
2	1.216	172	39.2	7.2	0.59	2.95	0.045	
3	1.216	171	40.2	7.1	0.59	2.95	0.043	
4	1.228	153	49.9	7.2	0.59	2.90	0.044	
Average	1.218	168	41.9	7.3	0.58	2.94	0.044	3.92
Circulating Electrolyte at the Rate of 4 gal. per minute.								
1	1.211	163	40.0	7.3	0.52	2.90	0.042	
2	1.214	161	41.1	7.3	0.55	2.92	0.040	
3	1.216	159	42.0	7.4	0.58	2.95	0.040	
4	1.263	139	67.8	7.4	0.55	2.90	0.040	
Average	1.226	156	47.7	7.4	0.55	2.92	0.040	3.85
Circulating Electrolyte at rate of 2 gal. per minute.								
1	1.210	167	37.8	7.6	0.52	2.90	0.042	
2	1.213	165	39.0	7.1	0.49	2.95	0.042	
3	1.217	162	41.0	7.6	0.51	2.95	0.042	
4	1.265	146	66.4	7.9	0.55	2.95	0.041	
Average	1.226	160	46.1	7.6	0.52	2.94	0.042	
Sampled after circulation had been entirely shut off for 7 hr. with current passing at the rate of 36 amperes per square foot.								
1	1.185	179	23.4					
2	1.210	164	38.0					
3	1.230	151	51.8					
4	1.255	138	65.5					
Average	1.220	158	44.7					

structed of lead and antimony. These pumps were of limited capacity and the dilution of the solution was excessive. They were followed by a system of lead-lined cast iron eggs in which the electrolyte was collected and blown to the overhead tank with compressed air. This system was very inefficient and gave way to a type of boiler-feed pump, the water end of which was of bronze. These soon became leaky from corrosion and were abandoned. A type of plunger pump with a lead-lined barrel and rubber valve and plunger was next tried. This pump was continued in service for several years with satisfactory results but was replaced 10 years ago by the air lift, which has continued to hold the field. The air used here is constructed of cast lead-antimony pipe 6 in. inside diameter, ¾-in. walls. The pipe is cast in 4-ft. lengths with flanged joints. A brick wall 20 ft. deep and 4 ft. diameter, made waterproof, with a cement lining, contains that portion of the lift that is below the floor level. Working against a head of 14 ft. 8 in., 160 gal. of electrolyte (1.22 specific gravity) is raised per minute with a consumption of 80 cu. ft. of free air compressed to 16 lb. per square inch (see Fig. 67).

The lift, while it is of low mechanical efficiency and sensitive to variations in the air pressure, is very satisfactory in its operation, having no moving parts and requiring but little attention. The loss in temperature of the electrolyte during its passage through the air lift from the receiving tank to the discharge tank averages 0.5°C. The compressed air for the operation of the lift is obtained from the converter-plant blowing engines which are operated by water power. A motor-driven compressor at the tank house supplies air in cases of emergency.

Circulation of the electrolyte is maintained at a speed of 6 gal. per minute, a lower circulation reducing the production per kilowatt-hour while a higher rate has the effect of disturbing too much the settlement of the slime and increasing the silver in cathodes. Table VII shows the effect on the concentration of the electrolyte of varying the speed of circulation. The samples which were taken in the regular refining tank and marked zones 1, 2, 3, and 4 were obtained in the following manner: A glass tube, closed at the top, was lowered to a point 6 in. below the surface of the electrolyte and a sample collected at this point and called zone 1. Zone 2 sample was taken in the same manner at a point 17 in. below the surface, No. 3, 28 in. below the surface, and No. 4, 39 in. below the surface. The column of electrolyte in the tanks was 40 in. high, there being 3 in. of slime in the tanks. These figures represent the averages of 104 samples taken during seven consecutive days from 48 refining tanks at 36 amperes per square foot. The samples were all taken between the centre pair of electrodes. These figures show a rather remarkable concentration of the electrolyte as regards copper and acid with negative results regarding impurities in solution and demonstrated that, while operating at 36 amperes per square foot, circulating the electrolyte at the rate of 6 gal. per minute, delivering the solution at the surface of the tank and discharging from a point near the bottom, fails to maintain the electrolyte in a homogeneous state, but is found to be the most economical speed at which to operate.

The arsenic and antimony are the impurities with which we are chiefly concerned in our refining operations, and as their effects on the conductivity of the copper are much alike our practice is to report them combined as arsenic plus antimony in the cathodes, making a separation on the average monthly sample to note the relations. These impurities in the electrolyte and in the anodes are determined separately. There is no silver in solution in the electrolyte and all of the silver present in the cathodes is deposited mechanically from the slime in suspension: the slime is entrapped on the rough surface of the cathodes, the silver contents increasing with the age and roughness of the cathode. The arsenic and antimony are deposited, both mechanically from the silver and electrolytically from the electrolyte. When refining converted anodes, carrying 0.12% As, at 34 amperes per square foot the arsenic in the electrolyte is not allowed to exceed 7 gm. per litre.

The results of an experiment with covered cathodes and converter anodes may be of interest. Three regular cathode starting sheets were placed in separate frames covered with 8-oz. duck in such a manner as to entirely enclose the sheets. These were substituted for three regular cathodes in a commercial refining tank, and after remaining 2.75 days they were removed, sampled, and assayed as follows:

Analysis of Electrolyte

Specific Gravity.	Grams per Liter					
	Acid.	Cu.	As.	Sb.	Fe.	Cl.
1.200	152	41.2	7.0	0.44	1.80	0.04

Analysis of Anodes

Per Cent Cu.	Oz. Ag per Ton.	Oz. Au per Ton.	Per Cent As.	Per Cent Sb.
99.15	39.0	0.25	0.211	0.04

Cathodes Produced

	Enclosed Cathode.	Regular Cathode.
Current density, amperes per sq. ft.	31.6	35.7
Age of cathodes, days	2.75	4.0
Ag, oz. per ton	0.2	1.2
As+Sb, per cent	0.0036	0.0042

The lower current density of the covered cathodes is due to the increased resistance offered by the covering. The canvas-covered cathodes were drawn at 2.75 days rather than four days, as the canvas was showing signs of failure. The results show that the vas covering was effective in preventing but little of the silver-ng slime in suspension from coming in contact with the cathodes, but that it had a lesser effect on the arsenic and antimony contents of the deposited copper.

The original method employed at these works of purifying the electrolyte was the manufacture of bluestone. An amount of electrolyte sufficient to maintain the working solutions at the required degree of purity was daily run off from the tank house to the sulphate plant. Here it was heated to the boiling point by means of steam coils and passed through open tanks containing shot copper into which compressed air was blown until all of the free acid was neutralized. After the necessary concentration by boiling, the solution was allowed to cool in an open chute where most of the copper crystallized out. The crystals from the chute were re-dissolved in water, concentrated, and run into the final crystallizing tanks. The crystals from these tanks were washed, dried, and packed in barrels for shipment. The mother liquor, after it had become too foul for further use, was run over scrap iron for the recovery of the small amount of copper remaining in solution. After passing over the iron the solution, which contained only impurities, was wasted. This was an efficient method of freeing the electrolyte of impurities and would probably still be employed had not the market for the bluestone failed during 1899. In 1897 2,300,000 lb. of bluestone was produced and shipped from our sulphate plant.

The electrolytic method was next employed for purifying the electrolyte. The process consisted in removing from the working solutions each day such an amount of electrolyte as was necessary to maintain the purity of the electrolyte. This solution was passed through four standard refining tanks containing lead anodes and copper or lead cathodes. These tanks were arranged in a cascade, the electrolyte flowing from one tank to another. The Cu, As, and Sb contents of the solution, as it left the last tank, depended on the speed of flow. With a circulation of 2 litres per minute but a trace of Cu, As, and Sb remained. The ampere efficiency of deposit, however, was extremely low at this rate of flow.

Table VIII shows the results obtained while circulating at 4 litres per minute. The percentages of elimination as shown in this table are high, while the ampere efficiency is very low, the two lower tanks in the cascade doing but little work. In subsequent experiments the most economical speed of flow was found to be 7 litres per minute with an elimination of 99% of the copper, 78% of the arsenic, and 91.1% of the antimony with a total ampere efficiency of about 50 per cent.

The increase in the iron content of the electrolyte between the

TABLE VIII.—*Removal of Copper, Arsenic and Antimony from Electrolyte in Insoluble Anode Tanks*

Circulation, four liters per minute. Lead anodes, copper cathodes, 9,000 amperes, 31.8 amperes per square foot.

	Grams. per Liter.					Volts per Tank.	Temperature, C.°
	Acid.	Cu.	Fe.	As.	Sb.		
Inlet tank No. 1...	144	37.060	6.242	3.200	0.463	...	**17**
Outlet tank No. 1..	184	7.376	6.813	2.240	0.260	2.22	**42**
Outlet tank No. 2..	194	0.504	7.364	0.400	0.061	2.25	**57**
Outlet tank No. 3..	208	0.088	7.701	0.056	0.038	2.25	**64**
Outlet tank No. 4..	216	0.048	7.915	0.028	0.028	2.25	**65**

inlet of the first and outlet of the last tank, as shown in Table VIII, is due to the concentration of the electrolyte by reason of the heat evolved by the electrolysis. These analyses are corrected on the basis of an unchanged volume of electrolyte in Table IX.

TABLE IX.—*Corrected Analyses*

	Grams per Liter.					Percentage Elimination of Original Amounts.			Ampere Efficiency, Per Cent.
	Acid.	Cu.	Fe	As.	Sb.	Cu.	As.	Sb.	
Inlet tank No. 1.........	144	37.060	6.242	3.200	0.4630	...	...	...	....
Outlet tank No. 1........	169	6.760	6.242	2.050	0.2380	81.8	35.9	48.7	**71.70**
Outlet tank No. 2........	165	0.427	6.242	0.339	0.0517	17.1	53.5	40.2	**19.50**
Outlet tank No. 3........	169	0.071	6.242	0.045	0.0308	0.9	9.2	4.7	**1.68**
Outlet tank No. 4........	170	0.038	6.242	0.022	0.0220	0.1	0.7	1.7	**0.15**
Totals and averages...	...	.....	....	....	.....	**99.9**	**99.3**	**95.3**	**23.26**

The process thus far described has removed only the Cu, As, and Sb from the solution, leaving the Fe, Ni, Bi, and Zn. To remove these the electrolyte, as it leaves the insoluble anode tanks, is run into a lead-lined tank 13 ft. in diameter and 4.5 ft. deep lined with 12-lb. chemical sheet lead and containing 600 ft. of 1-in. 8-lb. chemical lead pipe. In this tank it is concentrated to 55°B., decanted to an open tank 10 ft. long, 4 ft. wide, and 3 ft. deep, where it is allowed to stand for four days, during which time the salts of Fe, Ni, Bi, and Zn crystallize out, leaving a solution of approximately the following composition to be returned to the tank room: H_2SO_4, 1100; As, 1; Sb, 0.2; Fe, 1; Ni, 5.3; and Zn, 1.5 gm. per litre. The Cu, As, and Sb deposited in the insoluble-anode tanks are in the form of a black slime, some of the slime adhering to the cathode but the greater portion settling to the bottom of the tanks. An average assay of this slime is given in Table X.

TABLE X.—*Slime from Insoluble-Anode Tanks*

(Treating electrolyte direct from tank room.)

	%
Moisture	10.0
Cu	55.1
SiO_2	1.1
FeO	0.4
Al_2O_3	0.4
CaO	0.3
S	4.1
As	10.3
Sb	2.5
Ni	0.35
Zn	0.32
Ag, oz. per ton	3.4
Au, oz. per ton	0.02

This material was sent to the blast-furnace plant for treatment. The method of purifying the electrolyte now in use is a modification of the one just described and consists in running off daily from the tank house about 25,000 litres of solution, concentrating this, by boiling, to 48°B. From the boiling tank the concentrated solution is run to crystallizing tanks and allowed to stand for four days. At the end of this period 82% of the copper will have crystallized out. The mother liquor has the following analysis: Acid, 475; Cu, 17.4; As, 20.2; Sb, 1.1, and Fe, 15.2 gm. per litre. This is run to the insoluble-anode tanks where it is treated for the removal of Cu, As, and Sb as described above. Table XI gives an analysis of the slime obtained when using this method.

TABLE XI.—*Analysis of Insoluble-Anode Tank Slime*

(Treating mother liquor from crystallizing tanks.)

	%
Moisture	9.66
Cu	46.30
SiO_2	0.38
FeO	1.66
Al_2O_3	0.4
CaO	1.08
S	5.02
As	21.48
Sb	2.28
Ni	0.35
Zn	0.32
Ag, oz. per ton	3.61
Au, oz. per ton	0.03

After treatment in the insoluble-anode tanks the solution is either returned to the tank room or treated, by further concentration and crystallization as described above, for the removal of the Fe, Ni, and Zn. The analysis of slime from the second concentration and precipitation for the removal of Fe, Ni, and Zn is as follows: Moisture, 22.1; S, 16.6; Ni, 5.6; Cu, 1.3; Fe, 10.3; Zn, 2.7%. The advantage of crystallizing the copper from the solution before treatment in the insoluble-anode tanks is that the ampere efficiency of these tanks is greatly increased because of the concentrated solution treated and that the amount of slime produced is reduced by nearly one-half, as follows:

Production of Insoluble-Anode Tank Slime

	Wet slime per month Lb.	As per cent.
Treating electrolyte direct from tank room............	38,333	10.30
Removing 82% of copper before treating...............	21,600	21.48

The slime from the insoluble-anode tanks is sent to the blast-furnaces for treatment, the iron slime is washed for the removal of free acid and sent to the blast-furnace or wasted if free of copper. The arsenic liberated from the anodes in the refining tanks finds its way into the following products in the proportions shown: In cathodes, 1.22; in silver slimes, 17.19; and in purification slimes and cathodes, 81.59%. It is therefore necessary to remove from the electrolyte 81.59% of the arsenic liberated at the anode.

From a converter anode carrying 40 oz. silver and 0.24 oz. gold per ton, a slime carrying 43.3% Cu, 5000 oz. silver, and 34 oz. gold per ton is obtained: See Table I for complete analysis.

This slime is removed from the refining tanks once in 60 days in the following manner: A copper jumper, held by the iron studs provided, connects the two ends of tank bars on adjacent tanks and short circuits four tanks. The anodes and cathodes are removed and the slime, 5.5 in. having accumulated, is diluted with water and pumped by means of a bronze steam injector to a round lead-lined tank, 12 ft. diameter, 4 ft. deep, situated 200 ft. distant in the slime-sampling room. Before entering this tank the slime passes through a lead screen with 1/8-in. holes, 1/2-in. centres. In this tank the slime is washed with hot water to remove the free acid and soluble copper.

From this tank the slime is dropped into a cast copper, lead-lined egg from which it is forced by compressed air at 100 lb. pressure into a modified Bushnell filter press containing hard-copper plates and rings, 8-oz. duck being used for filter cloth. Thirteen cakes 26 in. in diameter and 1.25 in. thick are made at each charge; the wet weight is 865 lb. per charge at 21% moisture. From the filter press the slime is placed on steam-jacketed copper-drying tables where the moisture is reduced to 10%. The slime is then crushed with a roller on a cement floor and sampled in 22,000-lb. lots for shipment.

The slime is sampled in the following manner. The 22,000-lb. lot is shoveled into a conical pile, each shovelful being delivered upon the apex of the cone. To insure thorough mixing of the material this operation is repeated twice. The material is then split-shoveled four times; these operations reduce the sample to about 300 lb. From this point the material is quartered down with repeated crushing to about 130 oz., the laboratory sample. The slime is then packed in 10-oz. canvas sacks holding 140 lb. each. Four of these sacks are packed in paper-lined wooden boxes, in which shape the slime is shipped to an Eastern refinery for treatment. About 560,000 lb. of slime is produced and shipped per year.

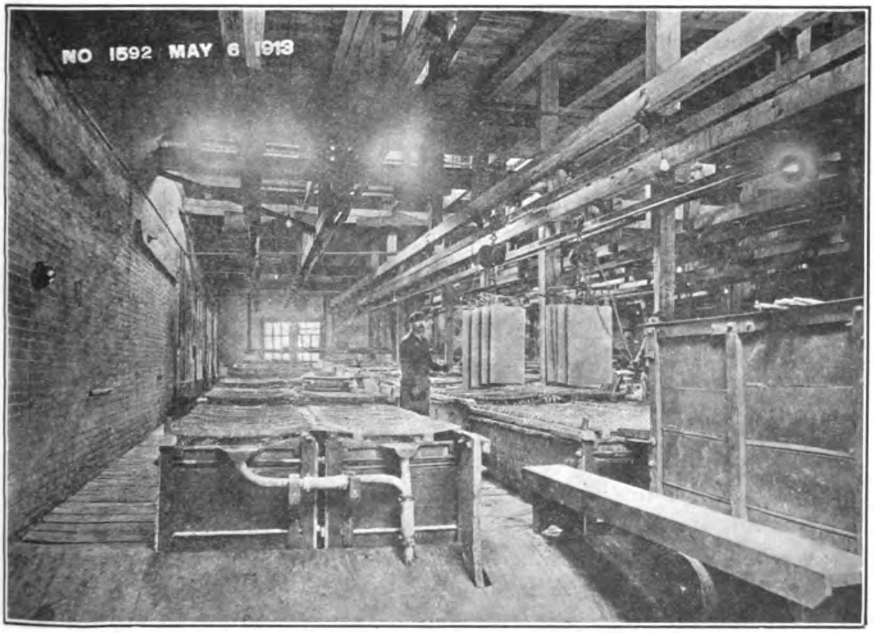


FIG. 68. DRAWING CATHODES.

The main tank room is divided into 10 divisions of 32 tanks each. Each division is taken care of by two men who are held responsible for these tanks. One inspector, locating and removing short circuits between anodes and cathodes, is employed for each two divisions. The inspector's work is done on night shift.

Sixteen tanks of 2-day cathodes, per division, or one-half of the tanks, are drawn daily, each tank containing 22 anodes and 22 cathodes. The day's starting sheets having been delivered to the different divisions by the night shift, the tank men begin drawing cathodes in the morning. Four cathodes are drawn at a time with a 1-ton Yale & Towne triplex chain block and multiple hook as shown in Fig. 68. The cathodes after being replaced with starting sheets are deposited in the lead-lined wheel carriages shown in Fig. 69; each carriage holds 66 sheets. The solution dripping from the cathodes is held in these carriages to be later drawn off through an outlet in the bottom. When the carriage is loaded it is moved by hand to the scales, situated as shown in Fig. 60. After weighing, the cathodes are lifted from the carriage by the traveling crane. The ends of the angle irons on which the cathode rods rest in the carriage are punched to receive the crane chain hooks by means of which the crane raises the load. After the cathodes are removed, the carriage is weighed to obtain the tare. The crane then raises the cathodes and loads them on to a 4-wheel truck for delivery to the furnaces.

After the cathodes are drawn the tank men inspect the anodes, marking such as are ready to be removed as scrap. The same carriages as were used for the cathodes are now spotted under the trolley track at the end of the tanks and the anode scrap is deposited in them. When all of the scrap has been removed in this manner the carriages are moved to the scales and weighed. The crane then lifts the scrap from the carriages, by means of the angle irons, and conveys it to a tank where the slime is washed from the surfaces with a stream of water. The scrap is then placed on trucks, by the crane, from which it is loaded into box cars for shipment to the converters. After the scrap has been all drawn from the tanks and the carriages removed, the tank men proceed to fill the vacancies, thus made in the tanks, with new anodes as shown in Fig. 68, one anode at a time being handled. As but four cathodes are removed from one tank at a time it is not necessary to cut the tanks out of circuit while drawing copper.

As compared with the traveling-crane system of handling the anodes and cathodes in tank units, the hand chain-block system of handling, as employed here, admittedly required more labor but is not without its advantages, which are as follows:

First, the chain-block system permits the use of converter anodes.

Second, the chain-block system, with aisles between the refining tanks, permits of starting sheets being straightened and placed in the tanks in such a manner that subsequent removal and straightening is not necessary.

Third, respacing of anodes and cathodes, from time to time, as the anodes corrode, to reduce the resistance of the tank, is unnecessary as the tank men properly space each separate anode and starting sheet as placed in the tank.

Fourth, there is a lower percentage of scrap and no scrap to be returned to the tanks.

Cash prizes are awarded to the tank men on the division making the best and second best records for the month. The division producing the highest tonnage of cathodes, after deducting the weight of the anode scrap, is considered as having the best record. This plan has been found very effective in stimulating rivalry between the different divisions and thereby improving the efficiency of operations.

Four insoluble-anode tanks to take care of the increase of copper in solution are provided. These tanks, as shown in Fig. 60, are situated in a separate room south of the pump house and, by means of a system of circulating pipes, can be run on any of the three electrolytes. It is not usually found necessary to operate more than two of these tanks at a time.

Four-day cathodes at 34 amperes per square foot are produced in these tanks. The product of these insoluble-anode tanks is of the same purity as the product of the soluble-anode tanks provided the rate of flow of the electrolyte is sufficiently high to insure ample copper in solution in the lower tanks. When the rate of flow is greatly reduced or altogether stopped the electrolyte soon becomes impoverished in copper, and arsenic is deposited out of the solution with the copper. The circulation is maintained in these tanks at the rate of 10 gal. per minute.

The following is a summary of the average amount of copper, silver, and gold locked up in the process while drawing 2-day cathodes and refining at the rate of 174,000 lb. copper per day.

Metals Locked up in Process

	Copper, lb.	Silver, oz.	Gold, oz.
In anodes	2,300,000	44,000	316
In slime	22,300	140,000	850
In cathodes	180,000	......	....
In solutions	95,000	......	....
Totals	2,597,300	184,000	1166

To arrive at the weight of copper in the refining tanks, for inventory purposes on the first of the year, a spring balance is attached to the chain-block hook and the anodes and cathodes are raised clear of their supports and weighed in solution. A factor, previously determined, representing the difference in weight of the average electrode, weighed in solution in this manner, and its actual weight is applied to the weights thus obtained. Weights obtained in this manner are found to check actual weights very closely.

The amount of slime in the bottom of the refining tanks at any time is calculated from the tank cleaning record, the number of days the slime has been accumulating and the amount of slime that is liberated at the anode per tank per day per ampere being known.

Current readings are taken and recorded hourly from a 10,000-ampere Weston station ammeter, the shunt of which is situated in the centre of the tank house. At the power house on each generator-switchboard panel is a 5000-ampere instrument of the same type. The averages of the hourly readings from these instruments are compared daily with the readings of the tank-house instrument to detect possible current leakage. A system of pressure wires connects the refining tanks and the generators with a voltmeter switchboard situated in the office at the electrolytic plant. Hourly readings of the voltages at generators, at the tank house, and at the different divisions and subdivisions of refining tanks are taken and recorded. In this manner abnormal conditions in any group of refining tanks or any change in resistance of the transmission line are promptly made known.

The daily report contains, on one sheet, the cathode production for the day and to date for the month, the number and weight of starting sheets used, the pounds of copper deposited per tank per day per ampere in the different sections, the percentage ampere efficiency of the total production, the amount and percentage of anode scrap for the day and to date for the month, the average amperes, volts, and kilowatts for the day and for the age of the cathodes drawn, the average weight per cathode drawn, the pounds of anodes, cathodes, and anode scrap on hand, the payroll for the day, and a digest of the electrolyte men's reports showing amount of solution treated in the purifying plant, character and speed of circulation of electrolyte, and other minor data.

The present practice is to draw 3-day cathodes on Mondays and Tuesdays, 2-day cathodes on the four days following, and none on Sundays, the generators running continuously during the seven days. Under the head of 22 anodes and 22 cathodes per tank in Table VI, the results obtained are shown in detail. The average daily production of refined copper is 174,000 pounds.

The refining-furnace plant, to be operated in connection with the electrolytic refinery, began producing anodes from converter copper in October, 1892. The original installation consisted of two coal-fired reverberatory furnaces. These furnaces were identical in design, having a hearth 15 ft. 6 in. by 10 ft. 6 in. with a firebox 4 by 4 ft. Each furnace was provided with a separate brick stack 65 ft. high, 28 by 28 in. inside. The capacity of these furnaces was 30,000 lb. of copper each. The furnace building was 80 by 128 ft. steel frame covered with corrugated iron. During 1893 and 1894 two additional furnaces of the same type and capacity were added. Two of the furnaces were employed in producing anodes from converter pig and anode scrap from the electrolytic plant and two were used to make wire bar, cake, and ingot from the cathodes.

All of the furnaces were dipped by hand, 9-in. ladles being used. The copper was cast in iron molds, the electrolytic copper being dumped in boshes containing water. The anodes were removed from the molds, by means of hooks, and allowed to cool in the air. Later these four furnaces were replaced by two larger furnaces of an estimated capacity of 50,000 lb. of copper each, subsequently increased to 100,000 lb. each. The hearths of these furnaces are 14 by 24 ft. and the fireboxes are 7 ft. long by 8 ft. wide. One of these furnaces is connected to one of the original brick stacks, described above, and the other is provided with a new brick stack 74 ft. high with a 34 by 34-in. flue. These two furnaces with a combined capacity of 200,000 lb. per day now handle the cathode output of the electrolytic refinery, the anodes coming direct from the converters.

The product of the refining furnaces is dipped into copper molds, made at the furnaces. These molds are hung in trunnions over cast iron boshes through which cold water is circulated. The mold is capsized, as soon as the metal is set, and the copper drops into the cooling water. Cooling in this manner prevents the formation of cupric oxide on the surface of the castings. The copper is conveyed from the furnaces to the molds by means of large ladles suspended by chains from a trolley running on a ¾ by 4-in. track suspended over the boshes. Hand-forged Welsh ladles of the following sizes and capacities are used: 14 in., capacity 200 lb. of copper; 16 in., 250 lb. of copper; and 19 in., 350 lb. of copper. The size of the ladle used is governed by the weight of the casting to be made.

The principal forms into which the copper is cast are as follows: Ingot, ingot bar, cake, wire bar, and a special form of round billet used in the manufacture of seamless copper tubing. The copper is shipped direct to the consumer. A 100,000-lb. charge is taken out of each furnace once in 24 hours. The usual practice of rabbling and poling the copper, for the oxidation of impurities and bringing of the metal to the proper pitch for casting, is carried on. The rabbling is effected by introducing compressed air at a pressure of 16 lb. per square inch into the molten bath by means of two ¾-in. iron pipes, the ends of the pipes being kept below the surface of the metal. About two hours is required for this operation. The reducing action is obtained by forcing the green pine poles into the bath and covering the surface of the metal with charcoal. About 28 poles, 8 in. at the butt and 4 in. at the top, and 35 bushels of charcoal are required per charge of 50 tons of copper.

The condition of the bath, as regards oxygen contents, is noted from time to time by the refiner as he examines the successive button samples taken from the bath in a small ladle. When the physical appearance of the button indicates that the 'tough-pitch' stage has been reached the poles are removed and the dipping operation is begun. As the dipping proceeds the refiner observes the set of the castings made and adds charcoal or logs of wood to maintain the oxygen at the proper point. The skimming of the furnace takes place just before the rabbling or oxidizing is started. The slag obtained is equivalent in weight to about 3.5% of the copper charged, varying with the amount of lime used. The following is the average analysis of slag produced at furnace No. 2 producing wire bar: Cu, 62.89%; SiO_2, 19.1%; FeO, 2.0%; Al_2O_3, 1.8%; CaO, 7.2%; MgO, 1.4%; S, 0.17%; As, 0.02%; Sb, 0.025%; Ag, 0.350 oz. per ton.

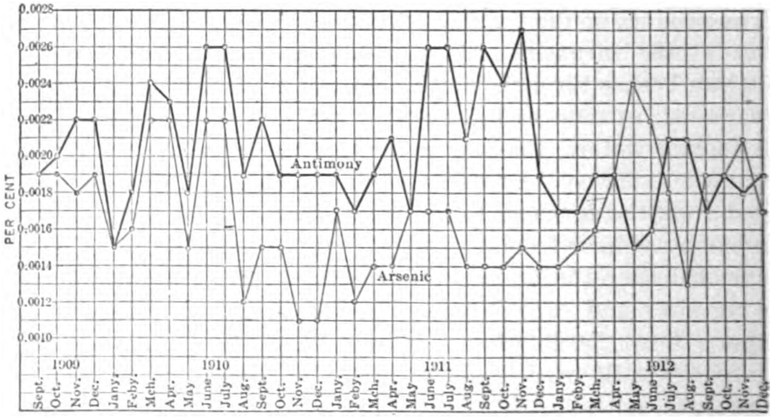


FIG. 69. RELATION OF ARSENIC TO ANTIMONY IN WIRE BAR.

The fuel used in these furnaces is known as Lochray lump coal, a bituminous coal mined at Tracy, Mont., 15 miles from these works. The analysis is as follows: Moisture, total, 7.01%; moisture, combined. 1.7%; volatile matter, 24.7%; fixed carbon, 48.0%; ash, 18.5%; sulphur, 3.5%; B.t.u. per pound of dry coal, 10,803. It will be noted that this coal is high in sulphur. To protect the copper as it melts from the sulphur dioxide gases, resulting from the combustion of the fuel, one-third of the cathodes going to make up wire bar charges are dipped in milk of lime before charging. As the copper melts, the lime forms a protective covering for the metal, hindering in a large measure the absorption of sulphur by the molten copper. This whitewashing process accounts for the lime and magnesia in the furnace slag. The average analysis of wire bar produced at these furnaces is shown in Table I, and the relation of arsenic to antimony in wire bar is shown in Fig. 69. The analysis of cake, ingot, and other furnace products, in which high electrical conductivity is not essential, will contain 0.02 to 0.05% less copper, due to the increase in oxygen content.

THE REFINING OF COPPER

(November 4, 1911.)

Eight plants handle the refining of upward of 1,000,000,000 lb. of copper in the United States annually, according to the *Boston News Bureau.* When all plans have been carried out, the Guggenheims, with their three refineries, will lead in capacity with a total of 480,000,000 lb. per year. At the moment there is being sold through their sales agency approximately 340,000,000 lb. per annum. The three Guggenheim refineries are owned by two of their companies. American Smelters Securities Co. owns the Baltimore plant, which at the time it was purchased had an annual capacity of 144,000,000 lb. Early in 1910 this had been increased to 240,000,000 lb. per year, and Daniel Guggenheim went on record some time ago as saying that its ultimate capacity would be 288,000,000 lb. The Smelters Securities company also owns the Tacoma smelter and refinery, which has a capacity of 24,000,000 lb. per year. The Perth Amboy refinery is owned by the American Smelting & Refining Co. It has a capacity of 168,000,000 lb. per annum. Thus for the three Guggenheim refineries an ultimate capacity of 480,000,000 lb. yearly should result when all improvements and additions have been completed.

By combining the capacities of the Raritan and Boston & Montana refineries the Amalgamated interests have a yearly total of 456,000,000 lb. in comparison with Guggenheims future total of 480,000,000 lb. The Laurel Hill refinery, owned by the Nichols Copper

Co., has long held first place with regard to capacity, or at the rate of 396,000,000 lb. per year. This place has been usurped by the Raritan refinery, formerly owned by the United Metals Selling Co., but now controlled and operated by the International Smelting & Refining Co., its capacity being 408,000,000 lb. per annum. The Anaconda company operates a refinery in Montana with a capacity of 48,000,000 lb. yearly.

In the following table increase in capacity of the various refineries in the past three years together with present annual capacity is shown:

		Monthly		
Plant.	Annual.	1911.	1910.	1909.
Raritan	408,000,000	34,000,000	27,000,000	24,000,000
Boston & Montana	48,000,000	4,000,000	4,000,000	4,000,000
Nichols	396,000,000	33,000,000	33,000,000	27,000,000
Guggenheims	*480,000,000	*40,000,000	25,000,000	12,000,000
U. S. Refining	180,000,000	15,000,000	14,000,000	10,500,000
Balbach	30,000,000	2,500,000	2,500,000	2,500,000
Total	1,542,000,000	128,500,000	105,500,000	80,000,000

*Ultimate capacity.

Selling and refining of copper has always gone, for the most part, hand in hand. The United Metals Selling Co. handles practically all of the Raritan and Boston & Montana output, which at full capacity would total 456,000,000 lb. The Raritan plant, under recently acquired contracts, has been treating the product of three companies, Miami, British Columbia, and South Utah, which it does not sell. All other copper refined by International Smelting & Refining Co. is marketed by the United Metals Selling Company.

The Nichols Copper does not maintain a selling agency, its product being sold by the American Metal Co. In addition to the Nichols output the American Metal Co. operates the Balbach refinery. The best customer of the Nichols has for years been Phelps, Dodge & Co., with about 135,000,000 lb. of copper at current rate of operations. Phelps, Dodge & Co. alone of the big selling agencies, does not own a refinery.

A close working alliance has existed for some years between the United States Smelting, Refining & Mining Co. and the L. Vogelstein & Co. selling agency whereby the latter markets the output of the former. The Calumet & Hecla Co. owns and operates a small refinery at Buffalo where a portion of its subsidiaries' product is being treated electrolytically. There also the silver-bearing native copper of the Quincy company has been electrolytically treated for some time past.

ELECTROLYTIC COPPER REFINERY

(July 16, 1910.)

The method of designing an electrolytic copper refinery may be illustrated by the following figures for a 12-ton plant. According to European practice one ampere-hour of current will deposit 1.183 grams of copper. One ounce equals 28.35 grams, and in a 12-ton plant the deposit of copper should be:

$$\frac{12 \times 2000 \text{ lb.}}{24 \text{ hours}} = 1000 \text{ lb. per hour.}$$

hence $\frac{1000 \times 16 \text{ oz.} \times 28.35 \text{ gm.}}{1.183} = 383.432$ ampere hours.

If the whole work has to be accomplished in one plant and plans for 90 vats in three series of 30 each be adopted, then the generator would have to furnish: $\frac{383.432}{90} = 4260$ amperes.

The voltage required per vat should not, in a well designed plant, exceed 0.6; but allowing for impurity of the anodes and other local conditions 0.8 will be a safe maximum. For 90 vats, then, $90 \times 0.8 = 72$ volts, will be needed, and $72 \times 4260 = 306{,}720$ watts $= 411$ horse-power to deposit 12 tons per 24 hours, or 34.23 hp. per ton as a maximum figure. But, as noted above, a voltage of 0.6 per vat should be ample under perfect conditions, and if this figure be adopted $90 \times 0.6 = 54$ volts will be needed and $54 \times 4260 = 230{,}040$ watts $= 308.3$ hp. to deposit 12 tons or 25.7 hp. per ton of copper deposited.

Suppose a current density of 12 amperes per square foot anode and cathode surface be figured, and for a generator maximum figures, 411 hp., be taken, with 10% added for loss in efficiency, the total will then be 452 hp. which, divided among three generators of 151 hp., calls for 150 hp. each. Assuming 12 amperes current density to be correct, $\frac{4260}{12} = 355$ sq. ft. anode surface will be needed in each vat. This may be taken as 360 sq. ft. in round numbers. Allowing a superficial area of 8 ft. per anode, 45 anodes $\frac{360}{8}$ will be needed in each vat. Figuring the size of the anodes to be 16 by 36 in., will just give 8 sq. ft. area for both sides. Anodes should be made about one inch thick, and should have a lug cast on each with a hole suitable for a hook. The width of a vat depends upon how many anodes are placed in it and connected on each single positive conductor bar. Suppose three to be arranged on a bar, and a 2-in. space be left between them, with 3 in. between the anodes and the sides of the tank, then the width of the tank will be 58 in. Dividing the 45 anodes by 3 gives 15 as the number of rows of anodes. A space of 2 in. should be allowed between the electrodes, and as there should always be one anode between two cathodes, it follows that there must be in this case 15 rows of anodes and 16 rows of cathodes. Fifteen rows of anodes, 1 inch thick equals 15 in.; 16 rows of cathodes, 1/8 in. thick equals 2 in., and 32 spaces of 2 in. equals 64 in. These added make the total length of a vat, 81 in. Allowing 10 in. space below the anodes will give a depth of 46 in. The size of the vat therefore should be 7 ft. long by 5 ft. wide by 3 ft. 10 in. deep.

At each series of vats the positive terminal of the generator is connected with two copper bars, well insulated, resting on the vat. Slightly elevated above these two bars are placed two negative copper bars, which at the end of the vat are bent and extended into the next vat as positive bars in line with the terminal bars in the first vat; and so on from vat to vat until the end of the series, where the last pair of bars is connected with the negative terminal of the generator. Iron cross bars from which the anodes are suspended rest on the positive conductor bars, and iron bars supporting the cathodes rest on the slightly elevated negative conductors. The vats are filled with the solution $H_2O + CuSO_4$, and it is important to have this solution circulate both rapidly and quietly in order not to disturb the mud collected. All the anodes and cathodes in a vat should be raised and lowered in a battery and transferred by help of a traveling crane or overhead crawl. The anodes may be cast in the copper converter plant in sizes as given above, transferred to the electrolytic refining plant, and placed in the vats. The cathodes should be of the purest commercial sheet-copper of the same sizes as the anodes. The electrolytic process transfers all the copper in the anodes, less the impurities, to the cathodes, and the copper accumulated on the cathodes varies but a small fraction from 100% pure. The plant should, by preference, be housed in a steel building on stone or on a concrete foundation.

ELECTROLYTIC COPPER REFINING AT THE NIKKO COPPER WORKS

By Tetsutaro Hasegawa

(October 12, 1912.)

The electrolytic refinery at Nikko was built in 1905 to refine the copper from the Ashio mine of the Furukawa Mining Co., and absorbed the older Honjyo plant at Tokyo. It is owned by the Furukawa company and serves to refine the blister copper produced at the various mines of that company. The growing demand in Japan for copper wire and other merchant forms of the refined metal has afforded a steady market, and the trade-mark of this plant is now well known throughout the Orient. The first plant was built upon

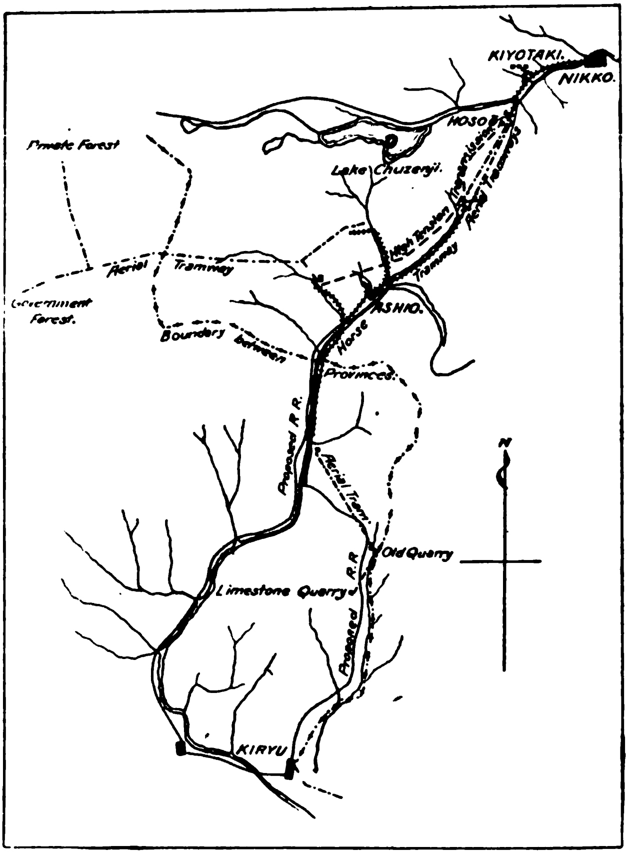


Fig. 70. Surface Map, Ashio Mine.

the series system, and a brief description of it appeared in the *Mining and Scientific Press* of October 14, 1914. (See p. 401.)

To meet the demand for increased output, an addition to the refinery has recently been constructed, of reinforced concrete after

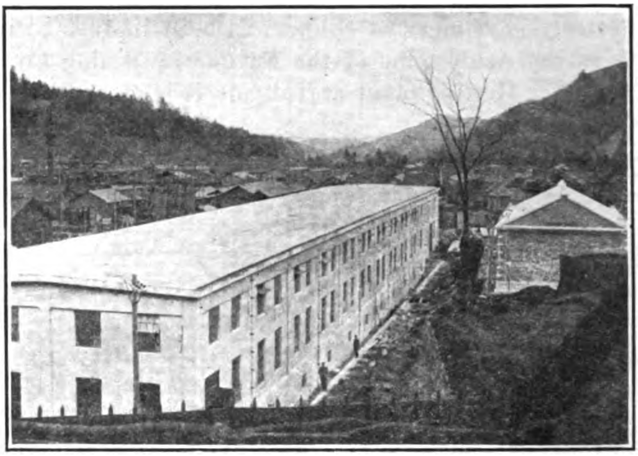


FIG. 71. ADDITION TO NIKKO ELECTROLYTIC REFINERY.

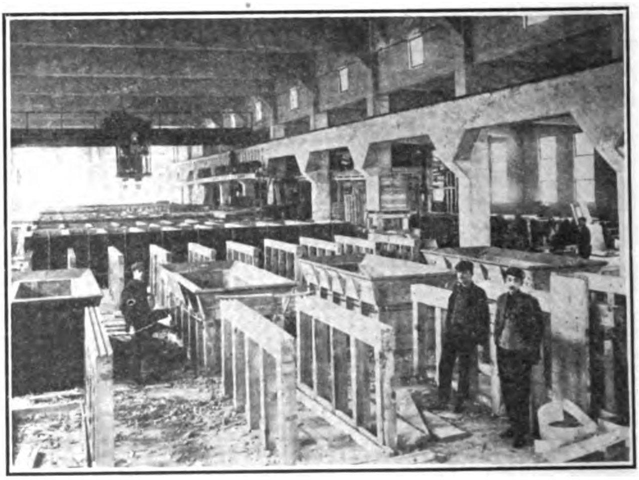


FIG. 72. INTERIOR OF ELECTROLYTIC REFINERY.

the Kahn system. This is shown in the accompanying illustrations, and contains 108 vats in 12 groups of 9 vats each. In this case the arrangement patented by Walker is used. The vats are made of wood and lined with 8-lb. sheet lead. A group of 9 vats is 9 ft. by

30 ft., and 4 ft. deep. The anodes used are 2 ft. 3 in. wide, 3 ft. long, and 1¾ in. thick, while the cathode strips are 2 ft. 4 in. wide, 3 ft. long, and 1/32 in. thick. A single vat will hold 18 anodes and 19 cathodes, spaced 4 in. centres.

At present the anodes are left in the vats about 21 days and the cathodes 7 days. The current density used is 20 amperes per square foot. The solution enters at the top of the vat and escapes at the bottom, the rate of flow being 100 cu. ft. per hour. The best results are obtained where the electrolyte is warmed to 40 or 50°C. The general arrangement of the plant can be seen in the illustration. The racks for the anodes are in the immediate foreground, just behind them are the tanks in which the anodes are washed, and to the rear of these are the racks for the cathodes. Behind these are the electrolyte vats, with the crane, of Whiting make, above them. The reservoir for electrolyte is seen to the right, in the prolongation of the line of the crane into the lower portion of the building, and the regenerators are seen in front. The annual output is expected to be 9,000,000 lb. of refined copper. The power-house is the small brick building, 54 by 36 ft., seen to the right in the illustration. Here a 450-hp. motor-generator set furnishes the current of 5000 amperes at 60 volts which is transmitted to the refinery.

WHO SELLS THE COPPER

(May 3, 1913.)

Agencies for selling copper usually handle the metal from several concerns. As the business is profitable, it is much sought, and changes are frequent. The most recent list, based upon pounds handled last year, and compiled by the *Boston News Bureau*, is given below:

UNITED METALS SELLING CO.

	Pounds.
Anaconda and North Butte	294,000,000
Lake Mines (Copper Range)	40,000,000
Arizona Copper Co.	38,670,000
Utah Consolidated Mining Co	8,000,000
Greene-Cananea	48,346,000
Giroux	3,768,000
Pennsylvania Salt Mfg. Co.	6,000,000
Imports	10,000,000
Total	448,784,000

American Smelting & Refining Co.

	Pounds.
Cerro de Pasco	48,000,000
Tacoma refinery and smelter product, Alaska, and miscellaneous	113,000,000
Utah Copper Co.	95,200,000
Nevada Consolidated	63,500,000
Tennessee Copper Co.	13,000,000
Ray Consolidated	35,400,000
Chino	28,700,000
Mason Valley	16,000,000
Ohio Copper	6,972,000
Total	419,772,000

American Metal Co.

Old Dominion Copper Co.		27,571,000
Shannon Copper Co.		16,500,000
Granby	22,630,000	
Imports, Ducktown, and miscellaneous	56,000,000	
		78,630,000
East Butte		14,765,000
Teziutlan Copper Co.		10,000,000
Torreon smelter		5,000,000
Total		152,466,000

Calumet & Hecla

Calumet & Hecla	65,490,000
Osceola	18,154,000
Allouez	5,627,000
Ahmeek	16,197,000
Superior	3,900,000
Tamarack	7,783,000
Isle Royale	7,968,000
Centennial	1,732,000
Victoria	1,200,000
Total	128,051,000

Phelps, Dodge & Co.

Copper Queen Smelter	123,276,000
Detroit Copper Co.	24,976,000
Calumet & Arizona	55,200,000
Total	203,452,000

L. Vogelstein & Co.

United States Smelting	22,500,000
Matte, smelter material, and imports	45,000,000
Orford Copper Co.	15,000,000
United Verde Copper Co.	33,000,000
Total	115,500,000

ADOLPH LEWISOHN & SONS

	Pounds.
Miami Copper Co.	32,814,000
Shattuck-Arizona (in 1913)	15,000,000
Total	47,814,000

BEER, SONDHEIMER & CO.

British Columbia	11,000,000
Cuba Copper	5,000,000
Miscellaneous	4,000,000
Total	20,000,000

E. P. EARLE

South Utah mines and smelters	2,000,000
Mexican imports	3,000,000
Japan imports	4,000,000
South American imports	4,000,000
Total	13,000,000

DIRECT TO TRADE

Quincy Mining Co.		20,250,000
Wolverine	9,120,000	
Mohawk	11,995,000	
		21,115,000
Franklin		2,450,000
Mass		1,400,000

W. PARSONS TODD

Winona	2,200,000
Lake Copper	800,000
Total	3,000,000

The leading metal concerns of Europe have representatives in New York who buy copper in large quantities direct from producers or selling agencies. Notwithstanding this fact the leading American producers and selling agencies have their own representatives in the various European markets. In London, C. S. Henry & Co. represents the Amalgamated Copper Co., Henry R. Merton & Co. the American Metal Co., while L. Vogelstein & Co., Phelps, Dodge & Co., and the American Smelting & Refining Co. also have men on the spot. Aron Hirsch & Sohn of Germany has close affiliations with L. Vogelstein & Co. and is represented throughout Europe; in addition the firm sells much Australian copper.

COPPER MARKETS OF THE WORLD

(October 5, 1912.)

The following is taken from reports from the cities mentioned, published in *Daily Consular and Trade Reports.*

The use of copper for manufacturing purposes in the London consular district constitutes but a small portion of the total consumption of the United Kingdom. The greater part of the copper is required in the Midlands and north of England, which contain such great industrial cities as Birmingham, Manchester, Newcastle, Sheffield, and Leeds. Liverpool, by reason of its situation, is the port which receives most of the copper imported into the United Kingdom from the United States. Some American copper is delivered at Swansea, but it forms a very small proportion of the total imports.

Liverpool is the principal port in the United Kingdom for the importation of copper ore and unwrought copper, taking about three-fourths of the former and nearly one-half of the latter. It may be accepted that all of the copper ore imported into Liverpool is smelted in this district, but diligent inquiry fails to obtain exact figures as to the local consumption of the unwrought copper. Neither manufacturers nor brokers will give definite information or even satisfactory estimates. The closest figures obtainable are that 70% of the imports of unwrought copper is used in manufacture in this district, 20% forwarded outside of the district, principally to Leeds and Birmingham, and about 10% re-exported to foreign countries. This re-export varies and was found to be one-third less in 1909 than in 1910, although the imports in 1909 were about double those of 1910.

Two of the manufacturers in the Liverpool district principally produce sheets, bars, and heavy plates for working into shape by coppersmiths, and known to the trade as 'copper bottoms.' Birmingham is the centre of manufacture of copper for kettles, pans, etc., and is the largest purchaser. Copper plates are also purchased by Egypt and India for use in the manufacture of utensils, trays, etc. A third manufacturer produces copper telegraph and telephone wire. There are no copper smelters in Liverpool itself, but this district is probably the greatest centre for smelting in England; Swansea, which previously led, being now second. Near Liverpool there are 7 firms operating 9 smelting works, and 4 have offices in this city.

The sale of copper ore is principally through brokers on commission, and not on the market. In the case of the unwrought copper, which is warehoused in Liverpool, the manufacturers purchase it on the London Copper Exchange, and the product of the smelters is purchased either in this way or through agents, and through dealers who buy large quantities to dispose of later. This copper is sold on what is known as 'standard copper' terms, which include various grades, the basis being 97% pure copper. If upon delivery the percentage should be below 97% and not less than 96%, the purchaser is entitled to a deduction accordingly, but if below 96% and not less than 93%, a double deduction is allowed. If under 93%, the copper can be refused. On the other hand, if over 97%, excess must be paid.

Birmingham, by variety of its industries, probably uses more copper for manufacturing purposes than any other city in the United Kingdom. There is a large and increasing consumption of copper, however, in London owing to the electrification of railways. Of the total imports of unwrought copper into the United Kingdom in the last three or four years, only about one-third was imported into London, but most of the re-exports were shipped from this district. The smelting and refining of copper and the making of articles of brass and copper have been important industries of Birmingham for many years. Not so very long ago nearly all of the copper or brass articles within the United Kingdom were of Birmingham origin, and this district properly claimed supremacy in the metal trades; but copper manufacture of various kinds has developed in other British cities, so that while Birmingham is still first, its leadership is by no means so unchallenged.

Several firms engaged in refining copper in Birmingham and vicinity are generally referred to as smelters, but as a matter of fact the copper that comes to them for refining has already been smelted and contains only traces of a few other metals. In the past, when it was customary for the copper ore to be brought to England, refining proper or smelting was really an important industry in Birmingham; but with the development of smelting and refining in the United States and in other copper producing countries, and because of heavy freight charges for carrying crude ore from the sea-coast to Birmingham, it became impossible for local smelters to compete with those more favorably situated on waterways, so what was left of the industry in England remained in the hands of refiners on the coast. Copper is used in the Birmingham district for the production of copper and brass sheets, tubes, and wire, for the manufacture of cartridge cases, bedsteads and bedstead mounts, fenders, brass founding, lamps, pins, church bells, gas and electric fixtures for plumbing, motor accessories, and many other articles in which copper is required. The output of these products is immense, and the quality usually excellent, the copper, brass, and light metal trades generally giving employment to thousands of people.

Estimates as to Birmingham's proportion of the copper required for manufacturing in the United Kingdom vary materially. Unfortunately, there is no way of ascertaining the truth, because there are no statistics of the raw copper that comes into Birmingham, while the same copper will be used over and over again in various degrees of manufacture. The six largest Birmingham firms with copper and brass rolling mills are said to consume at least 30,000 long tons of copper annually, while other mills and industries are thought to consume as much more. The quantity of copper from America and other copper-producing nations that comes to Birmingham or Great Britain fluctuates each year with trade requirements. Much depends upon the price at which the copper is offered and the supply available. At one time it may be more profitable for a copper manufacturer to buy Japanese or Chilean, Mexican, or other copper, while at another the American article may be more profitable to him because of the prices at which and the readiness at which it can be obtained at the moment. However, as American copper represents more than 50% of the production of the world, it necessarily stands pre-eminent in the market and must always command a considerable sale. Then the almost perfect purity, 99.9%, of the American cathodes, ingots, cakes, or wire bars that come to Great Britain gives American copper a high reputation with manufacturers, but what is regarded as the "independent attitude" assumed by the American companies that sell copper to England causes serious complaint on the part of those who are in the market for copper supplies. American copper companies are said to refuse to sell to England except on the American certified weight and not upon the weight of copper on arrival. As copper cathodes must be lifted and handled about eight times on the way to the consumer in England, and as they are rough and easily abraded, it can be seen that the percentage of loss in weight is considerable. Copper merchants generally, at the normal price of copper, calculate this loss as an increased cost to them of about 60c. per long ton.

Another objection made to American copper is that cathodes are frequently of too great weight. Sometimes a cathode will be 86 lb., which is not strongly objectionable, although even this must be cut before it can be used in manufacture; but American cathodes are said not infrequently to weigh 125 to 160 lb. each, when they become a serious nuisance and occasion many complaints because of the great difficulty in handling them. It is stated that reduction in the size of cathodes from the United States would popularize American copper considerably, while if all American copper were shipped in cakes, ingots, or wire bars of reasonable weight, and not in extremely heavy cathodes, the objections raised on account of the loss during shipment and as to difficulty of handling would be met.

It was suggested by one manufacturer that if there were representatives of American copper companies with offices in Birmingham who went after the business with great energy, the sales of American copper might be further increased, particularly when the demand is slack and prices comparatively low. At such a time the service of a representative or merchant who is in touch with all manufacturers, who knows their requirements and the quality and character of the copper that they need, would be invaluable and materially broaden the sale of the American product. However, there are representatives of the great copper merchants in Birmingham who appear to be ready and willing to meet most of the requirements of the market.

It would seem as if there should be a good market in Birmingham for copper and brass sheets and electric and telegraph wires of American origin if regular supplies were available, provided, of course, pains were taken to study the market and ascertain the exact kinds, quantities, and sizes that manufacturers of brass and copper goods require, the usual terms of sale, and conditions of delivery; but the effort to sell such products by correspondence and without the most careful investigation would be futile. Some organization such as that by which semi-finished steel and iron from the United States and Germany are sold in England would be wholly desirable.

At Havre 65 to 75 tons of copper per day is used in manufactures. practically all by one customer, La Société des Trefileries du Havre; the total consumption is between 20,000 and 25,000 tons annually. Havre's imports of copper, however, are nearly three times this tonnage, having amounted in 1911 to 74,977 tons, consisting of 2569 tons of ore, 68,370 tons of pigs and sheets, and 4038 tons of scrap and filings. The United States supplied 5680 tons of the pigs and sheets and 1510 tons of scrap and filings. Havre is the principal importing point for American stock and has practically a monopoly of the American copper trade with France. It must be understood, however, that this tonnage of copper received at Havre from America has heretofore not been traded in at Havre itself but at London. It is to handle the sales directly at Havre, and to save a present expense in London commissions, etc., amounting possibly to as much as 50c. per ton, that the Havre traders are endeavoring to create an independent exchange. Local bank conditions are quite adequate to such a market, and the hope is to buy direct from America hereafter, in so far as American importations into France are concerned. There is very little copper of any character exported from Havre. The total in 1911 was only 6634 tons to all countries, 5266 tons of which was copper wire manufactured in this city. The trade in scrap copper is insignificant.

The important element in the development of a direct American copper market at Havre will be the establishment of close relations between American and French shippers, without the intermediary of London or other traders. Steamship transportation is fairly adequate and cheap. British control of the Atlantic carrying trade and the necessity of making charters in London may possibly be used against the development of a direct market, but the consulate believes that American exporters will have no great difficulty on that score if they will make the proper connections at Havre. The transportation facilities of the port are excellent and are being enlarged to conform to the most modern practice. Freight connections with the United States are excellent. It is the expectation among French engineers that the French consumption of copper will continue to advance in fully the ratio shown for the past few years. Any artificially increased importations on account of the low prices prevailing in 1910 and 1911 will evidently be offset by greater consumption in manufacturing, the demand in France corresponding, as in other countries, to the expansion of industrial enterprises in the domestic field, and in the extensive enlargements of electric traction, transmission, and other systems which are heavy users. It is further to be pointed out that the Havre market is in direct touch with all parts of Europe and the United Kingdom, and that it already is headquarters for extensive direct dealings in coffee, cotton, rubber, woods, and other raw materials.

Russia, from January to May 1911, produced 454,526 poods (pood = 36.112 lb.) of copper, while the production for the same period of this year totaled 660,437 poods, an increase of 45.3%. This production for 4 months was distributed as follows: Urals, 350,664 poods; Caucasus, 198,906 poods; Siberia, 83,300 poods; smelters and refineries, 27,567 poods; total, 660,437 poods. It will be seen that the highest percentage falls to smelters and refineries, but the increased production of the Ural works is of much more substantial importance to Russia. Among the concerns of that region the largest gain in production is shown in the Kyshtim district, where the amount of copper produced rose from 56,675 poods in 1911 to 150,719 poods in 1912. The Akhtalsky works output advanced from 63,939 poods in 1911 to 89,230 poods in 1912. In the Bogoslov district 84,895 poods was produced during 1912, against 83,395 poods in 1911. The Spassky works increased its production from 48,767 poods in 1911 to 75,981 poods in 1912. The Caucasus Copper Co. (see *Mining and Scientific Press*, January 27, 1912) increased its output from 28,786 poods in the first four months of 1911 to 68,362 poods in the corresponding months of 1912.

TEMPERING COPPER AND BRONZE

(December 28, 1911.)

The production of bronze alloys in one operation and the hardening and tempering of copper are perpetual topics of popular discussion, so that the statements made by Walter Gowland at a recent lecture in London are of interest. Mr. Gowland, besides being a distinguished metallurgist and emeritus professor of the Royal School of Mines, is also an antiquary of note, as is evidenced by the fact that some years ago he was deputed to make the historical examination of Stonehenge. His researches into the history of metallurgy in all parts of the world are elaborate and learned. In his lecture he showed that during the early metal age, and even later, bronze was not obtained by melting metallic copper and tin together, but by the reduction of oxidized copper ores containing cassiterite, or of copper ores to which cassiterite ore was added. It is important that this fact should be recorded, seeing that some modern metallurgists allege that such a process is impossible, and that whenever tin and copper ores are smelted together the tin does not unite with the copper but passes into the slag. These allegations were evidently not founded on experiment, but on erroneous deductions from the methods of smelting in use at the present time.

In order to determine the possibility of producing bronze, Mr. Gowland constructed a simple furnace consisting merely of a hole in the ground. In this he treated a mixture of 15 lb. of malachite, containing 30% copper, and 10 lb. of cassiterite ore containing 20% tin, together with charcoal and limestone. The furnace cavity was filled with charcoal, which was also piled above it for 2 or 3 inches. When the charcoal was well alight, a layer of the charge was spread over its surface, then another layer of charcoal, and so alternately until a conical heap was formed. A gentle blast was then started through a 1-in. blast-pipe, and when the charge began to settle the blast was increased. On the conclusion of the reaction, the slag and remaining fuel was removed and the metal allowed to solidify.

An analysis of the metal showed 78% copper and 22% tin. A series of experiments were undertaken with varying proportions of the ore, and in every case copper-tin alloys were obtained. In the later bronze age, the varying qualities of alloys of different composition were well known, and the ancient metallurgists were obviously experts on the properties of alloys. The second point raised by Mr. Gowland referred to the supposed lost art of tempering bronze. In the old days the bronze castings for tools and weapons, were hammered at the cutting edges to produce the right degree of hardness and temper. No other method was ever employed, such as heat treatment. The same effect can be produced on modern bronzes, by hammering. and in fact an even greater hardness can be obtained. Thus the notion that the art of tempering bronze has been lost is a fallacy. Mr. Gowland might have added that soft copper tools can be hardened by heating them in contact with minerals containing arsenic or antimony, when a casehardening action is produced through the formation of a glass-hard copper-arsenic alloy. Analyses of some old 'tempered' copper tools have shown them to contain nickel and other metals; thus the supposed 'tempered copper' is really a hard bronze.

THE MASON VALLEY COPPER SMELTER

By Thomas T. Read

(August 31, 1912.)

That unexpected difficulties are usually met in the operation of a new plant, and that alterations are frequently necessary, is well known; so much so, indeed, that recently the correspondent of a well known mining journal, in describing one of the new mills erected at Porcupine, naively remarked that ample provision had been made for changes in the design of the plant. Not all designers make this provision, but this smelter, constructed to treat ores not previously successfully smelted, and which within six months of the time of its blowing in is successfully handling double the expected tonnage, with no delays or shut-downs and only minor changes in the design, is correspondingly unusual. The plant was designed by and erected under the supervision of Jules Labarthe, now its general manager. A. J. McNab is superintendent. To both these gentlemen I wish to express my appreciation of the courtesies extended me during a recent brief visit to the plant.

The Mason Valley smelter, owned and operated by the Mason Valley Mines Co., is at Thompson, Nevada, about two miles north-

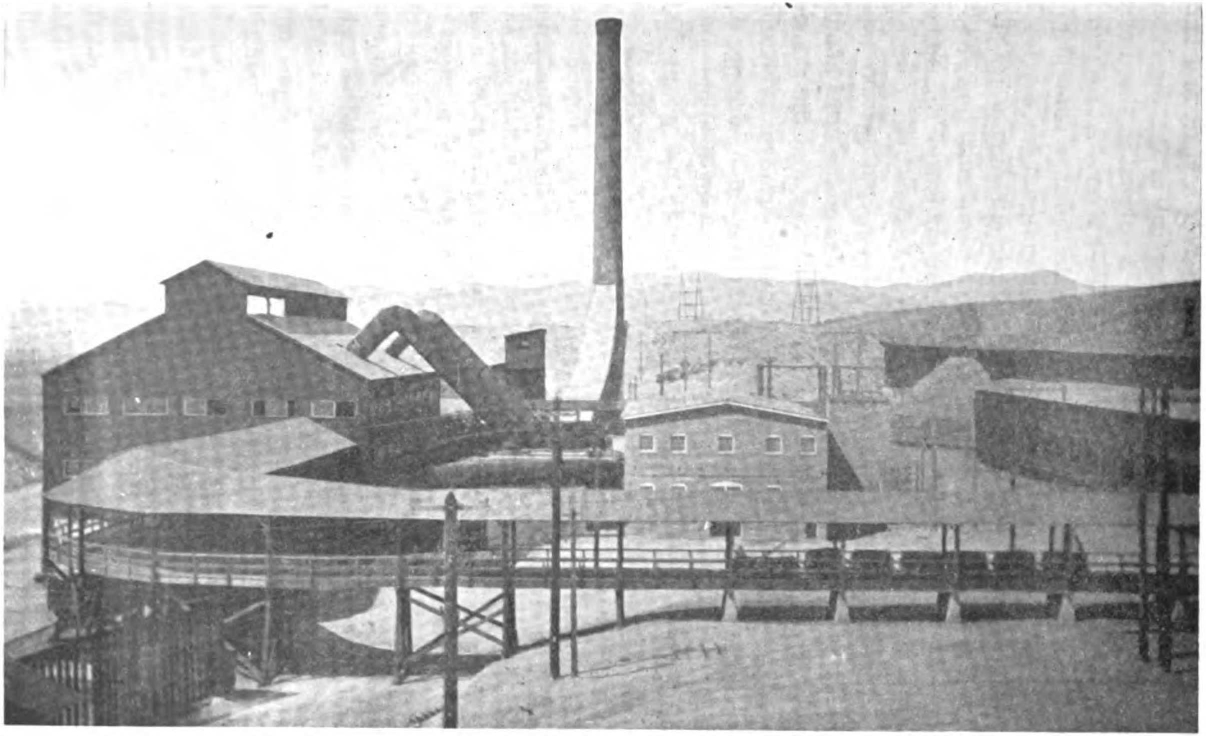
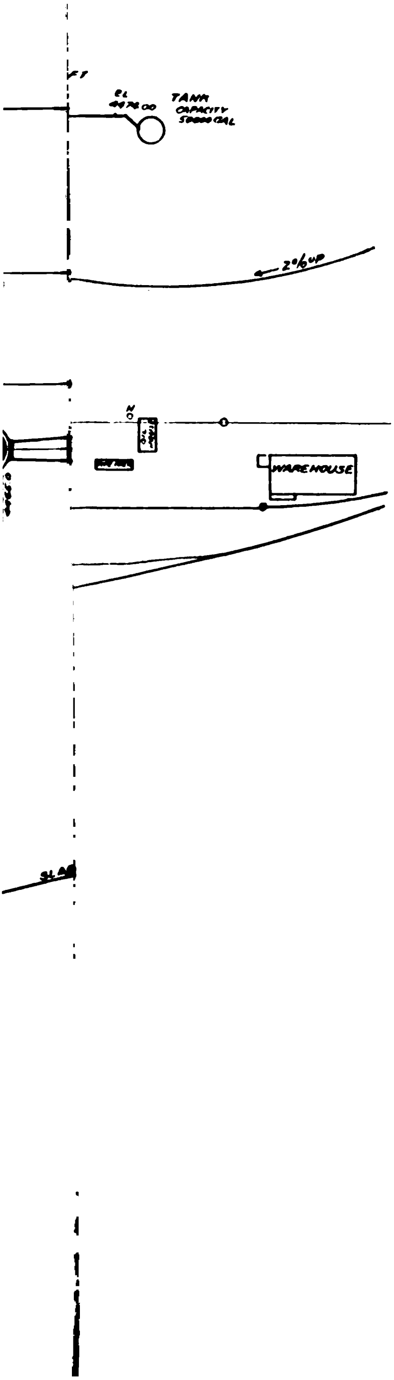


FIG. 73. GENERAL VIEW, MASON VALLEY SMELTER, THOMPSON, NEVADA.

west of Wabuska, on the Reno-Goldfield branch of the Southern Pacific railroad. The site was chosen after records of the direction of the prevailing wind had been kept for over a year, and is such that for 90% of the time the smoke is carried in the direction of Carson sink and for only 2% of the time is carried in the direction of the agricultural districts along the Walker river. At present this is an almost unnecessary refinement, for the sulphur content of the smelting mixture is so low that the escaping smoke carries very little SO_2, while arsenic and similar deleterious elements are absent. 'Smoke farmers' need but little cause for complaint, however, and the care and forethought exercised in the choice of the site were justified.

Ores.—The smelter is handling at present 750 to 800 tons of charge per day. Of this, the Nevada-Douglas contributes about 350 tons per day, the Mason Valley mine a little less, and the remainder is largely made up by shipments of copper ore from the smaller properties near these two, although about one car of ore per day is received from Luning district. A small amount of gold and silver ore is also received, and it is necessary to use about 25 tons per day of limestone as flux, as the Nevada-Douglas ore is rather silicious. The ore arrives at the smelter over the standard-gauge line of the Nevada Copper Belt railway, in 50-ton hopper-bottom cars, and is dropped into one of the two bins of the sampler. The general plan of the smelter is shown in the illustration. From these bins the ore is drawn directly into the throat of a No. 8, style K, Allis-Chalmers gyratory crusher, the crushed product being elevated to the top of the building by a manganese-steel chain-bucket elevator. This type of elevator was first used at the smelter at Trail, B. C., and is found to give longer service at a less cost. At the top of the building a Vezin sampler takes out one-tenth as a sample, the reject passing over a small grizzly, which screens out the fine sulphides and diverts them to a small bin on the rear side of the building. For reasons which will be explained later, this is not now in use, the grizzly being covered with a steel plate. The reject passes by a belt to the main 20-in. belt, which extends the length of the ore-bins and is provided with a tripper so adjusted that the ore may be discharged into any of the existing bins, or into the 10 new flat-bottomed bins, of a total capacity of 3600 tons, which are being built to serve as bedding bins for the shipment of small lots of custom ore, which are being received in increasing quantity.

The Sample.—The first cut of the sample goes to a 10 by 20-in. Blake crusher and then to a second sampler, which cuts out ¼. This goes to a second 10 by 20-in. Blake crusher and to a third sam-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pler, which cuts out 1/5; this goes to a set of 14 by 30-in. rolls, then to a fourth sampler, which cuts out 1/10; then to a set of 14 by 30-in. rolls for final crushing. A 50-lb. sample is thus obtained from a car of ore. The reject from the second, third, and fourth samples unite at a small bucket elevator and is raised to the main distributing belt, which deposits it in the ore-bins. The sampler has a capacity of 150 tons per hour.

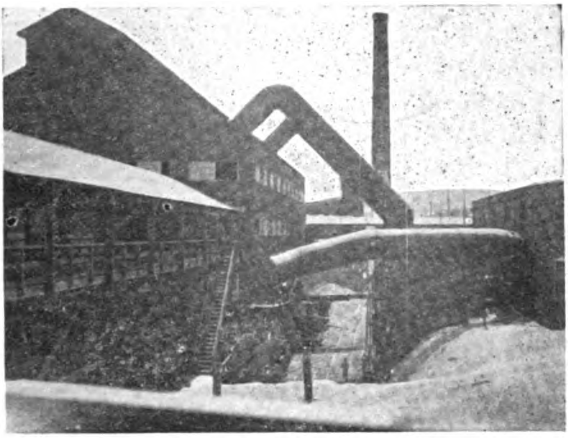

FIG. 75. BLAST-FURNACE, DUST-CHAMBER, AND STACK.

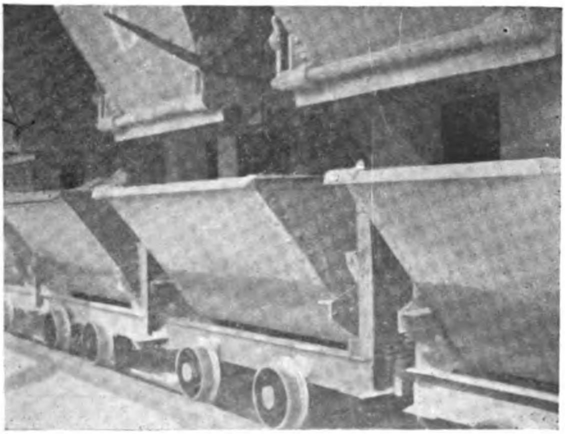

FIG. 76. WEIGHING-HOPPERS AND CHARGE-TRAIN.

Weighing the Charge.—The charging bins for ore and coke are 20

Howe weighing-hopper scales. The ore from the bedding bins will be drawn out upon a belt beneath them and delivered to the main elevator in the sampling mill, then discharged to a belt-conveyor and thence to charge-bins. Weighing the charge will thus be somewhat simplified, duplication of the equipment of weighing hoppers will be avoided, and an accurate sample of the bedded mixture will be obtained. The capacity of the sampler is amply large to perform this added duty. As shown in the plan, the bins and furnaces are connected by an elliptical track, and the charge is drawn from the hoppers into two-ton cars, of which six or seven, making up a train, are drawn by a four-ton Jeffrey MB-24 electric locomotive. These are shown in place beneath the hoppers in the accompanying illustration.

Blast-Furnaces.—There are two blast-furnaces, 42 in. by 25 ft. at the bottom, and 47 in. wide at the tuyeres. From the tuyeres to the charge-floor level is 15 ft., but the charge is kept 3 ft. below this, making the height of the smelting column 12 ft. The tuyeres are 4 in. diam., 25 on each side. The blast is admitted at 42-oz. pressure, and is furnished by two No. 11 Connelsville blowers. Each blower is driven by two 200-hp. A. C. motors, and is capable of furnishing 30,000 cu. ft. of free air per minute. Only one furnace is now in use; this furnace during the month of May smelted an average of 720 tons of ore and 80 tons of seconds and limestone per day. This corresponds to a smelting rate of 12 lb. per square foot per minute, considering the hearth area as the area at the tuyere level. Remembering that this is attained by the use of 9 to 10% of coke on the charge, the smelting rate here attained is the fastest of which I have any knowledge. High smelting speed has been attained in pyrite smelting, but in that case the bulky coke is not present to displace a much greater weight of smelting mixture. For a low sulphur, high-coke charge this must be either a record smelting rate, or very close to it, but I shall be glad to be corrected if I am in error in making this assumption. The rapid rate of smelting makes necessary the use of an increased amount of jacket-water, as might be expected.

Charging the Furnace.—The furnace is charged directly from the cars. Adjustable feed plates, as shown in Fig. 77, are provided, the angle of their inclination being capable of adjustment within the range 30 to 60°. By a little experiment the proper degree of inclination was found so that coke automatically falls in the position which gives the maximum smelting rate. After this was obtained it was found that the necessity for barring down crusts and punching tuyeres had almost totally disappeared. This, of course, contributes to the rapid smelting rate of the furnace. The operation of dropping a charge is quick and simple. A charge train usually consists of two cars of coke and four or five cars of ore. The furnace feeder draws up the charge doors as the train approaches, and as each car of coke is dumped the motor draws it slowly the length of the furnace to distribute its contents evenly. One car of ore is next dumped, and the other cars are then in position so that they may be at once dumped into the furnace. The operation of charging takes but 1½ to 1¾ minutes, and a round trip, loading and dumping the charge train takes but five to eight minutes. The smelting charge contains about 4% copper and 6% sulphur. Coke forms 9 to 10% of the smelting mixture.

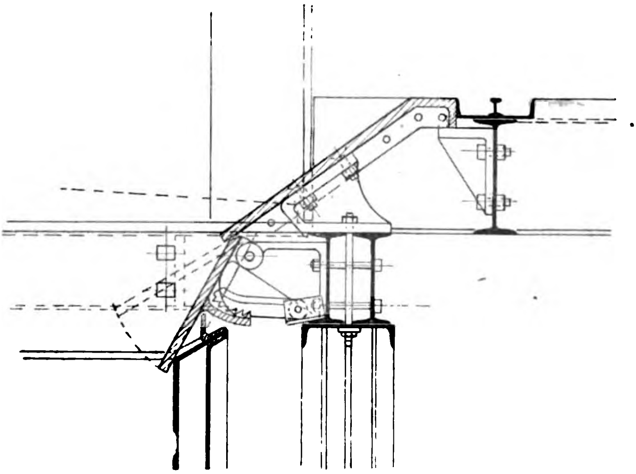

FIG. 77. ADJUSTABLE FEED PLATE.

Handling Slag.—The slag runs continuously through a tapping jacket of cast iron with pipe coil cast in. The spout is jacketed, the inner sheet being sheet copper. It is customary in many places to cast the spout from blister, but as only matte is made at this plant, sheet copper was employed and gives good satisfaction. The slag and matte go to two 5 by 12-ft. movable settlers. These were employed, as it was feared that trouble would be experienced from the formation of sows. The expected trouble did not materialize, and it is now planned to replace these by a stationary 8 by 18-ft. settler. The slag flows from the second settler into a 225 cu. ft. (21 ton) capacity steel slag pot, hauled by a 15-ton electric locomotive. In changing the pots a small pot pivoted on a jib-crane is swung under the slag spout while the full pot is being pushed out, and the empty one drawn into its place. The contents of the small pot are then emptied into the large one. The slag pot is pushed to the dump, and dumped by electric motor controlled by the motorman in cab of the locomotive; the skulls being sufficiently free of matte so that no attempt is made to remelt them. The motorman of the slag car also performs a similar service on the locomotive which draws the train of matte-cars, and operates the crane, without being overtaxed. The track on the dump is shifted from time to time, as required, by a gang of surface laborers, and the only persons regularly employed in dealing with the slag are the tapper and the motorman, the latter giving only a part of his time to it. The average composition of slag is:

SiO_2	FeO	CaO	Al_2O_3	MgO	Cu
42	26.8	22	7.3	1	0.38%

Handling Matte.—About 50 tons of 45% matte is made in a day, and it is necessary with the present settlers to tap at intervals of ½ to ¾ hour. The second settler is only tapped once each shift, yielding a few hundred pounds of matte. The method of handling this is the only instance in which any important change has been made in the original design of the plant. Three Kilker matte-cars, operating on a circular track were placed in the plant. The particular cars employed proved too small for this situation and are not now used. The method now employed is similar to that formerly used at the United States plant at Bingham Junction, Utah, except that the wire rope there employed for moving the train of matte-cars is replaced by a 4-ton electric locomotive. The matte molds are carried on light steel cars; the molds are of cast steel and have shown no sign of cracking. Each mold holds about 100 lb. of matte; and resembles a waffle-iron in pattern. The train will accommodate 50 molds; about 40 are in use. When the matte is tapped these are drawn back and forth below the matte-spout until all are completely filled. The operation of tapping takes 7 to 8 minutes. Then the cakes of matte are allowed to cool down to the point where it is safe to play a hose upon them, and are dumped into an ordinary 5-ton matte-pot, placed below the track. The cake in falling breaks up into eighty roughly cubical pieces weighing 12½ lb. each. The pot is dumped into a 50-ton railroad car, for shipment to the Garfield smelter. This handling is done by an ordinary 20-ton crane, with a 5-ton auxiliary. A certain amount of matte unavoidably spatters about; as this is cleaned up it is loaded into small dump cars which are hoisted by the crane to the charging floor. The building enclosing the blast-furnaces has been made large enough to contain converters, which will be erected when the daily output of matte reaches a tonnage large enough to permit reasonably continuous operation of a converter. The crane has been provided for this purpose, so the only construction necessary will be the addition of stands, shells, and blowing engine.

Flues and Flue-Dust.—The downcomers from the blast-furnace enter the brick expansion chamber, which has a cross-sectional area of 500 sq. ft., and is 200 ft. long. The flue beyond this point is 18 by 10 ft. and ascends the hill-side to the base of the stack at a point 40 ft. above the furnace level. The stack is 200 ft. high, 18 ft. diam. at the base, and 15 ft. at the top. On the side of the expansion chamber next the furnaces automatic discharge gates for the flue-dust are provided which permit it to escape to an automatic dust-conveyor that conveys the dust to the Dwight-Lloyd sintering machine. This is so placed that dust can be drawn by the conveyor from 50 ft. of the flue as well. It was planned to mix the flue-dust with fine raw sulphides before sintering, but delivery of the machine was delayed and the blast-furnace had been in operation for three months before the sintering machine was started. The flue-dust had thus accumulated to a point where it was necessary to handle it as rapidly as possible. The attempt was therefore made to sinter the dust without the admixture of any raw sulphides, and after a little experiment success was attained. The necessary heat is supplied by mixing with the dust ½ to 2% of coke-dust, prepared by grinding coke breeze in a clay-mill. Sufficient water is added to give a mixture containing 8% moisture, and since the aim is to secure the maximum sintering with the minimum of desulphurization the machine is run rapidly, treating 60 tons per day. The speed of travel of the pallets varies from 1 to 5 ft. per minute. Some trouble has been experienced from the tendency of the cast iron grates to crack, and it is planned to use malleable iron for the grates of the second unit, now under construction. The work of this sintering machine is of great interest, since it is the first instance of its use to treat flue-dust, unmixed with raw sulphides.

Fuel.—The coke used in the blast-furnaces is that of the Utah Fuel Co. The coke has 7% ash, and is somewhat stringy, but is a good burden carrier, and as it has to stand but little handling, it arrives at the smelter in good condition and works well. There is no steam plant at the smelter, all machinery being motor-driven, so there is no occasion for the use of fuel other than coke, beyond a small amount of coal in the blacksmith shop.

Water Supply.—There are two sources of water supply. Near the wagon-road between Wabuska and the smelter, a hot spring was first developed. This yields 40 gal. per minute at a temperature of 170°F.; the water is excellent and it was formerly cooled and used for domestic purposes. Later a well was put down at a point south of the smelter, about half way between it and the line of the Southern Pacific, and 90 ft. below the smelting level. This is a flowing well, 600 ft. deep, yielding 120 gal. per minute; another adjacent well yields 30 gal. per minute more, so an ample supply is available. The jacket water is cooled in a cooling-tower just below the general offices, to avoid the necessity of the 90-ft. lift on water that can be re-used.

Power Supply.—Electric power for the operation of the plant is purchased from the Truckee River General Electric Co. The current mainly comes from the new plant at Verdi, but there is another line connecting through Mason with the older line, so that the smelter has two independent supply-lines, and can draw current from any one of four power plants. Current is received at 60,000 volts, and is transformed to 440 or 220 volts for the A. C. motors. The cost of electric power is $70 per horse-power year.

Ore Supply.—The ore supply of the plant is rapidly increasing, as the low treatment rates accorded have served to stimulate a number of small shippers who are largely engaged in development work, while both the Nevada-Douglas and Mason Valley mines are prepared to increase their shipments. It is especially desirable to increase the tonnage of Mason Valley ore treated, as it is slightly basic, and if used in sufficient amounts the necessity for limestone flux is removed. Accordingly the No. 2 furnace, which has not yet been blown in, is being increased to 30 ft. in length; when it is ready to operate the No. 1 furnace will be blown out and increased to 35 ft. in length.

It is a pleasure to a smelterman to watch a furnace operate so smoothly as this one does, without constant necessity for barring down crusts. This is partly due to the good smelting mixture, perhaps partly due to that more or less fortuitous harmony of parts which makes one 6-cylinder motor car a much better hill-climber than another of identical pattern and make, but is largely due to its excellent design and management.

THE ASHIO COPPER SMELTER AND REFINERY

By Thomas T. Read

(October 14, 1911.)

[1]The Ashio mine, one of the principal objective points of the Japanese excursion of the American Institute of Mining Engineers, is of great interest, because it is the most productive copper mine in Japan, though closely pressed for the leading position by the Besshi mine and the Kosaka mine. The outputs of the three for 1910 are given as 11,843,406, 11,132,371, and 10,560,501 *kin*,[2] respectively. It is the largest of some fifteen copper mines owned by the Furukawa company, the Osaruzawa, the Furokura, and the Kune being the most important of the others. The company also owns three silver mines and seven coal mines, and occupies the dominant position in copper mining in Japan, which is the third most important copper producing country of the world. The mine, which is therefore of great relative importance, has further interest in the fact that it is practically privately owned. It was discovered in 1608, but its active development dates from 1877 when Mr. Ichibei Furukawa acquired it and worked it so successfully that he gradually acquired nearly all the

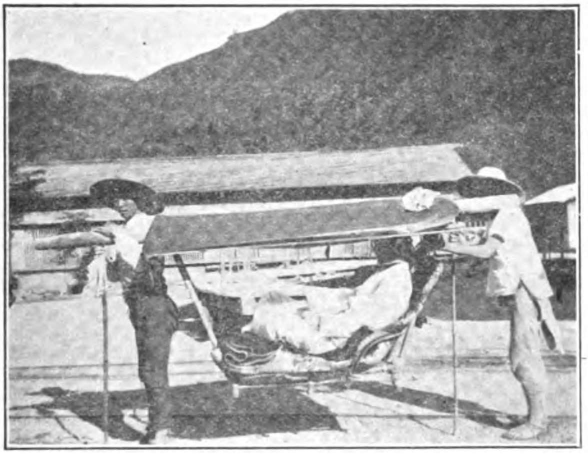

Fig. 78. Traveling to the Ashio Mine in a Kago.

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[1]Those portions of the original article which are descriptive of the mine have here been omitted.

[2]1 *kin* equals 1⅓ lb.; 1 *kwan* equals 8¼ lb.; 1 *yen* equals 49.8c. gold; 1 *sen* equals $^1/_{100}$ *yen*; 1 *shako* equals 11.93 inches.

more important copper and silver mines in the northeastern part of the main island (Hondo). These later passed into the possession of his son, Jun-Kiche Furukawa. It was my privilege to form the acquaintance of Mr. Toranusuke Furukawa, the present head of the company, some years since, and the following description is based upon a brief visit to the property during the summer of 1909, though I have supplemented my notes by the data given in 'Mining in Japan, Past and Present,' published by the Bureau of Mines of Japan. To those interested in the history of mining in Japan and the primitive methods employed in early times, this volume will prove a veritable mine of interest and information.

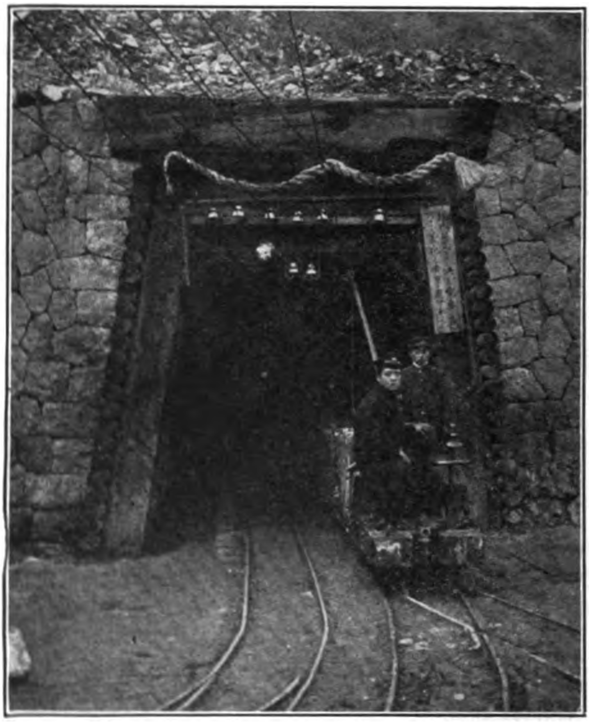

FIG. 79. MOUTH OF ADIT, HONZAN.

The Ashio copper mine is 100 miles north-northwest of Tokyo near the headwaters of the Watarose river, in the province of Shimotsuke. The mine can also be reached by the way of Kiryu, but the river navigable and at present supplies are brought to Nikko upon ay, transported to Hosoö in tram-cars drawn by bullocks, across the mountain to a surface tramway which takes them the remaining distance to the mine. The mountain is completely encircled by a tramway since Kotaki and Honzan are connected by a tunnel passing through what are now the upper workings of the mine. See p. 375 for map of Ashio.

The ore as mined contains from 1.5 to 9% copper, averaging about 4%; this is concentrated to about 12% copper. The mine ore is dumped on an 80-mm. grizzly, the oversize being sorted into rich ore, waste, and low-grade ore, which is spalled to pass the grizzly. The undersize goes to bins, whence it is drawn upon a conveyor-belt of Japanese cotton. This does not last long, but is enough cheaper to compensate for the shorter life. This delivers to a belt-conveyor which feeds a set of trommels, 40, 25, 17, 11, 7, 5, and 3 mm., respectively. The 40-mm. stuff is sorted by women into rich ore,

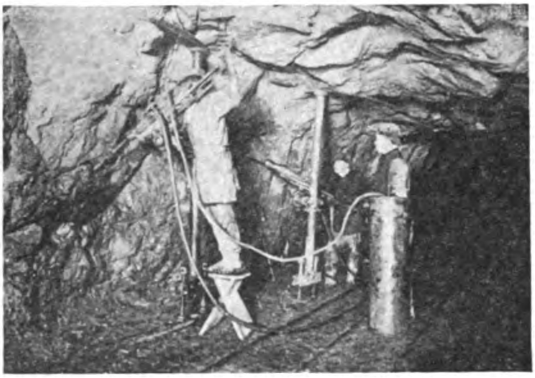

FIG. 80. LEYNER DRILLS AT WORK.

FIG. 81. ORE-DRESSING PLANT, KOTAKI.

poor ore, and waste. The rich ore is sent to the other dressing works, the poor ore is recrushed, first in jaw-crushers and then in rolls. The different sizes of finer material are jigged in 3-sieve jigs, the coarse middling being recrushed. The finest material is classified and treated on Overstrom and Wilfley tables; Cammett tables and Evans slime tables are employed at the other mills. The most notable feature of the milling practice, like the mining, is the low cost at which an involved series of operations, requiring much labor and personal attention, are carried on. The volume previously referred to states that the power used by the three mills is derived from eight motors, yielding a total of 318 hp., and two Pelton waterwheels. As this corresponds to little more than 0.6 hp. per ton-day it is remarkable, if true.

It would be interesting to trace the development of the present smelting plant from the *yama-shita* and *mabuki* first employed, especially as the latter presents a curious analogy wtih the production of blister copper in a converter, but considerations of space forbid. Until recently the fine ore was roasted in McDougall furnaces and many ingenious methods were tried for agglomerating the calcine, the most notable being a device of the superintendent whereby a shell was cast around each bucketful of calcine in an elevator. Blast-furnaces have always been used for smelting, and converters for bessemerizing the matte were adopted in 1893, the year after Manhés' American patents were issued. Since then all the types of converter except the Stalmann have been tried, and they are now using three 3-ton converters of the Great Falls type and two 5-ton stands of the Bisbee type. Liparite, mixed with clay, is used as a converter lining and as it probably has some such composition as 70 to 75% SiO_2 and 10 to 12% Al_2O_3, it gives a sticky slag. I did not learn whether this was employed because of the lack of pure silicious material, or because the low cost of labor for relining made it advisable to employ what is practically low-grade ore for converter lining. The composition of the matte for the day preceeding my visit was:

	%
Cu	33.5
S	25.0
Fe	36.6
Pb	1.44
Zn	1.20
As	0.72
Ag	0.03

No effort is made to save the lead and zinc.

The average product to be smelted has approximately this composition: Cu, 12%; Fe, 28%; S, 25%; SiO_2, 32%. The small proportion of lump ore goes directly to the blast-furnace. The fine is pot-roasted, according to a system devised by the resident engineer, in conical pots 8 ft. diam. at the top and 5 ft. at the false bottom, giving a depth of charge of 4½ ft. Nothing is added as a binder, the rate of oxidation being controlled so that the sulphides sinter to just the required degree during the roast. Each pot holds 7 tons and burns 12 hr., the combustion being started by the use of a small amount of coal and wood. The blast is furnished by No. 6 Root blowers. The operation of these pots seems to be entirely satisfactory.

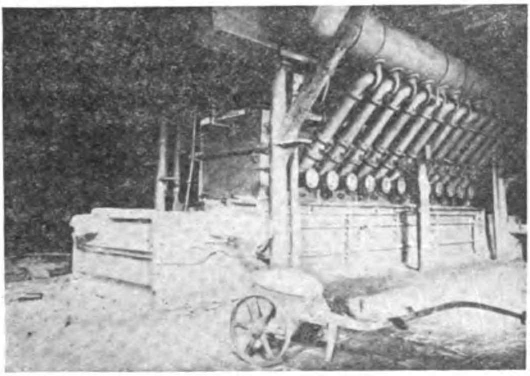

FIG. 82. BLAST-FURNACE, ASHIO.

FIG. 83. SETTLING PONDS.

Electric elevators lift the charge to the charging-floor. The relative proportions are: ore, 10 parts; limesone, 3 parts; converter slag, 2 parts. Smelting is done in 160 by 42-in. Allis-Chalmers water-jacketed matting-furnaces at the rate of about 2 tons per square foot of hearth per day, only a little over 6% coke being used, and a blast pressure of 22 oz. employed. The slag approximates 40% SiO_2, 30% FeO, 12½% CaO, and 7½% Al_2O_3, and carries 0.21% copper. It is necessary to convey it some distance up the

FIG. 84. SMELTING PLANT, ASHIO.

FIG. 85. HEADGATE, HOSOÖ POWER-HOUSE.

valley where there is space to dump it. I am told that it is now granulated by water as its flows from the forehearth. The blast-furnace gases pass through dust chambers to the main flue, which, as seen in the illustration, contains a series of chambers in which a rain of river-water is kept falling, much in the manner of a scrubber, the resulting sludge being thrown away. In spite of the care taken the escaping gases still do damage, because of the high SO_2

content, the dampness of the climate, and the narrowness of the valleys. Fortunately there is but little nearby agriculture and the damage claims are correspondingly small. The combination of smelter fume and the demand for timber and fuel has resulted in denuding the hills, and the latticework by which erosion is prevented can be seen in the background of the illustration. This, and the necessity for impounding the tailing from the ore dressing, and the neutralizing of the acid waters from the mine before allowing them to escape into the river, are a not inconsiderable addition to the cost of copper production, but a scrupulous performance of these duties is exacted by the Government in order to secure permission to continue working.

Over 7000 workmen are employed at the mine, about 3000 being employed underground. The electric power for the mine, mills, and smelter has been obtained from 10 power-stations on the Watarose and Daiya rivers, the turbines having a total of some 5000 hp. The expansion of the work calls for increased power, and the principal station, at Hosoö, has recently been doubled in capacity. The headgate of the penstock for this plant is shown in the illustration. The most improved types of direct-connected generators are employed at this plant, a visit to which is one of the principal features of the trip from Nikko to Ashio. The power is transmitted by high-tension lines to the points desired. As previously mentioned, supplies are brought in from Nikko and the blister copper is sent out by a wire-rope tramway from Hosoö. In addition to this there are several other aerial tramways, one of the most important connecting the mines with the forest reservations from which the supply of timber is derived. The mountain in which the mine is situated is girdled by surface tramway tracks, of which there are many miles.

The electrolytic refinery is at Kiyotaki, but a short distance up the Daiya river from Nikko (see Fig. 70). Here is refined not only the blister copper from Ashio, but from the other mines of the Furukawa company as well. The refinery has its own hydro-electric power-plants with 195 *shako* fall. The ingots from the mines are remelted in a reverberatory furnace, brought to proper pitch, and cast into plates, which are rolled into anode sheets and cut to the proper length. The general features of the plant are seen in the accompanying illustration. Three vats in series, each containing 40 plates, are placed *en cascade*, the electrolyte, after passing out of the lowest vat, is filtered through coke and pumped back to the uppermost tank. The volume previously referred to gives 234 as the total number of vats, but my own notes give double the number.

The electrodes are in series, the copper being dissolved from one side and plated on the other. When the operation is finished the electrodes are removed, scrubbed, melted down in a reverberatory furnace, brought to tough pitch, and cast into wire bars, cakes, and ingots. The making of copper wire is an important feature of the work, and is carried on with a high degree of skill. A Bates and Peard annealing furnace is employed. The finished product enjoys a high reputation in Japan, and the Furukawa trademark is widely known. The gold and silver slimes are cupelled in an English cupelling furnace. A great deal of work has been done on the problem of recovering the small amount of lead present in the blister, but I understand that no commercial results have yet been obtained in the experiments.

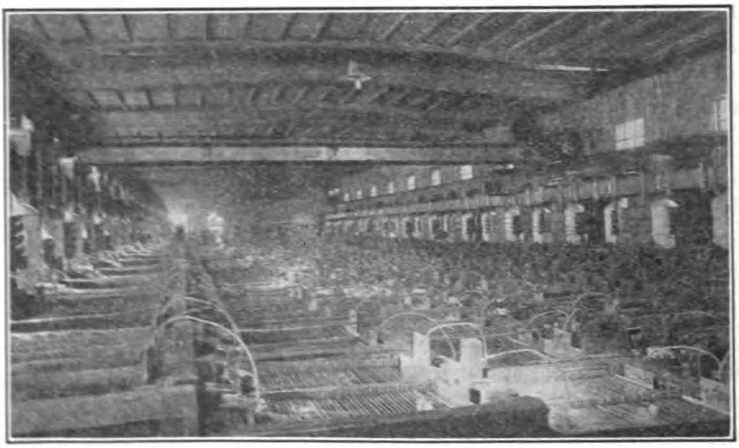
FIG. 86. TANK-ROOM, KIYOTAKI REFINERY.

The blister copper has a high tenor, the composition before refining being given as 99.7%. The yearly production of the works is stated at 5000 tons of copper wire, 2000 tons of copper ingots, and 500,000 oz. of refined silver and gold. The latter metals, of course, are chiefly derived from the bullion of other mines owned by the company. In concluding this description a brief reference should be made to the Kune mine in Totomi province, where there are enormous bodies of pyrite associated with chalcopyrite. Unfortunately the mine is distant from transportation facilities, nor will the Government permit the erection of a smelter, so that its development has been hindered.

IMPROVEMENTS AT ASHIO

By H. Foster Bain

(August 24, 1912.)

Among the mines visited by the American engineers who traveled through Japan last year was the Ashio, the leading copper producer of the Empire. This mine has already been described in the *Mining and Scientific Press* (October 14, 1911) by Thomas T. Read. I propose merely to present here supplementary data* gathered at the time of my visit or supplied since through the courtesy of the engineers of the Furukawa Mining Co., the owning corporation.

An interesting peculiarity of practice in the Ashio mines is that of neutralizing, as well as settling, the waste water. This process, which has in view the protection from damage of crops grown in the valley below the works, has now been so perfected as to result in saving enough copper to pay for the extra work. Details of the process have been published in the *Transactions* of the American Institute of Mining Engineers (February 1912), by J. W. Richards, and need not be repeated. Briefly, the sand and slime are settled as perfectly as possible in a first pool, metallic hydrates are carried down in a second and third, in a fourth by means of 10 to 20% milk of lime the neutralization and precipitation is advanced, and in a fifth it is completed. Finally in a sixth pool the water is run through a sand-filter before being sent back to the mill or allowed to escape. The water as it comes from the mills carries 0.00025% copper.

In the smelting and refining plants, as well as in the concentrating mills, there are many novel and interesting things to see. The ore is smelted in blast-furnaces with pot-roasting of the fine material and with the usual bessemer converters. Prior to the introduction of pot-roasting, which has become the common practice in copper smelting in Japan, the Furukawa engineers worked out a number of ingenious methods of handling the fine ore. Mr. Read has already mentioned the plan of making the ore into a ball and casting around it a thin shell of matte. On the whole, however, probably the most successful device is the briquetting machine invented by Sabro Watanabe, but four years out of school at Tokyo, and now an advanced student in Germany. Discarding the ordinary plunger and roll types of machines, he adapted the gravity stamp to the work. The moistened oil is fed to a horizontal revolving plate which carries

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*Only those portions of the original article which refer to smelting practice are here reproduced.

it successively below six stamps each weighing 120 lb. and dropping 10 in. The plate moves 1½ revolutions per minute and stops eight times in each revolution; once to permit filling the hole in the plate which constitutes the mold, six times under successive stamps to receive one blow from each, and once for a plunger to clear the mold. The briquettes are 4 in. diameter, cylindrical, and 4 in. high. The machine requires but one man and 2 hp. It prepares 35 tons of briquettes in 24 hr., and the cost is 30c. per ton.

Another interesting little trick in practice is a plan for promoting smooth running of the blast-furnaces by feeding coal, when needed, through the tuyeres. This was devised, with the assistance of T. Watanabe, by Kosahu Asano, now head of the smelting department and a graduate of the Massachusetts Institute of Technology. The device has been described with drawings in *Metallurgical and Chemical Engineering*, by J. W. Richards. It consists in placing in each tuyere double flap valves that can be opened at will, and using a cylinder of 4 in. diameter and 12 in. long with a movable piston to force coal, first placed in the cylinder, through the tuyere and into the furnace. By this means a source of heat and reducing gases may be placed at will at any critical point. In practical operation fuel equal to about 3% by weight of the ore is thus introduced into the furnaces.

One of the most interesting features of smelting at Ashio is the neutralization of the furnace fumes. As the average amount of sulphur in the ore as it goes to the roasting plant amounts to 25%, and the smelter is in a narrow valley in the midst of a rich farming country while the climate is exceptionally moist, it is extremely important to prevent damage from escaping SO_3 and SO_2 in the fume. The plan in operation was devised by K. Asano, and while not entirely successful, is susceptible by careful work of greatly reducing the danger of damage. The smelting plant includes three blast-furnaces, each 160 by 42 in. at the tuyeres, and 11 roasting pots. The gases from both roasting and smelting plants, after passing through a dust-chamber, are conducted through a chimney sloping up the side of the mountain and broken at four places by vertical steps 40 ft. high. These are 12 by 12 ft. in cross-section, and at the top of each is an iron plate containing 200 1-cm. holes through which rains down a solution of lime water. This reacts on the SO_3 and to some extent on the SO_2, probably forming sulphites. It requires 40 cu. ft. of water per minute and 300 lb. of burned lime per hour, to neutralize the gases, and they are reduced to a content of 0.5% SO_2. These purifying chambers are placed well up the hill so as to [illegible]ull of the gases as far as possible. No fan is used, but one will probably need to be added when, as is planned, additional washing chambers are built. Last fall when the plant was visited the gas coming from the chimney was nearly colorless, only a slight yellowish tinge, perhaps from sulphide of arsenic, being seen. Further progress with this interesting process will be watched with interest, but enough has perhaps been said to illustrate the fact that Japanese mines are not only hives of industry, but also of invention, and are well worthy of careful study by mining engineers and metallurgists.

THE SHISAKA SMELTER

By H. Foster Bain

(September 21, 1912.)

Among the great copper mines of the Far East is the Besshi, which is on Shikoku island in southern Japan. Travelers to and from the Orient pass within a few miles of the property as they go from Moji to Kobe through the wonderfully beautiful Inland sea. The mine is the property of the Sumitomo family by right of discovery and has been in their possession for more than 200 years. It was found in 1690 and, after being successfully worked by native methods through the intervening years, was opened according to modern plans in 1874 under the advice of a French engineer. It is credited with a total production of 475,250,000 lb. of copper to the close of 1910, and in 1911 yielded 15,145,070 lb., ranking thereby second only to the Ashio among Japanese copper mines. The mine has long been wholly under Japanese direction, and the family owning it is one of the most progressive in the Far East. Baron Kichizayemon Sumitomo, the present head, is a younger brother of Prince Tokudaiji and Marquis Sainoji, having been adopted into the family in 1892 at the age of 28. He resigned from the House of Peers in order to devote himself entirely to the business affairs of the family. Aside from the Besshi mine, the holdings include the smelter at Shisakajima, the Tadakuma colliery in Chikuzen, the Sumitomo Electric Wire & Cable Works, and the Japan Steel Mfg. Co. works at Osaka. On the occasion of the visit of the American engineers to Japan during 1911 the party enjoyed the hospitality of Baron Sumitomo at his villa near Osaka, and visited the smelting works at Shisakajima. Lack of time prevented a visit to the mine and rolling mills. The following description, therefore, so far as it relates to the mine, is based principally upon published data, generously supplemented by information given by Keijiro Nakamura, superintendent of the smelting works and an accomplished engineer familiar with the best mining practice both in America and Japan.

The ore from the mine is loaded on barges which are made into trains and hauled by tugs about ten miles to Shisakajima (Shisaka island), to which in 1904 the smelting works were removed in an effort to avoid danger from smelter fume. At the smelter about 600 tons of ore per day is handled, containing 30 to 33% S. With this is used 7% of gneiss as flux, and 4 to 5% coke.

The fine ore was formerly briquetted, but Mr. Nakamura has developed an alternative process which is now used. Noticing that some of the fine ore which had become wet while being transported, showed a tendency to 'ball up,' he experimented by allowing the ore mixed with varying amounts of moisture to roll down a slope. This led finally to the device shown in Fig. 87. It consists essentially of a conical wooden cylinder revolving on a horizontal axis. It is 8 ft. long, 3 ft. in diameter at the upper end and 5 ft. at the lower. Into the upper end is fed a mixture consisting of fine ore, 100 parts; flue-dust, 20; silicious flux, 40 to 50; water, 6 to 7%. The exact amount of water necessary was determined by experiment and seems to be important. The mixture as it rolls down to the delivery end agglomerates; the individual balls being half to three-quarters of an inch in diameter. The effect of the water seems to be purely mechanical, but it forms a bond sufficient to hold the mass together and permit successful pot-roasting, which is conducted in the pot shown in cross-section in Fig. 88. It also saves about 30% in the time necessary to conduct the operation. The pots used are essentially circular grates with shallow sides. made of steel and are set over firebrick-lined pits into

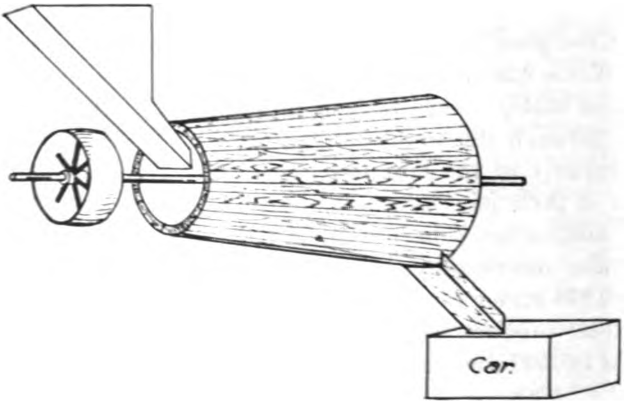

FIG. 87. NAKAMURA NODULIZING MACHINE.

which a blast is directed from a Root blower. They are covered by a heavy sheet iron hood luted at the sides with sand. In the top of the hood is a conical-bottom trap-door, similar to the top of an iron blast-furnace, through which additional material may be introduced during the roasting process. Below the trap are baffle-plates to scatter the charge. At the sides of the pit, above the grate, are outlets for the air and gases.

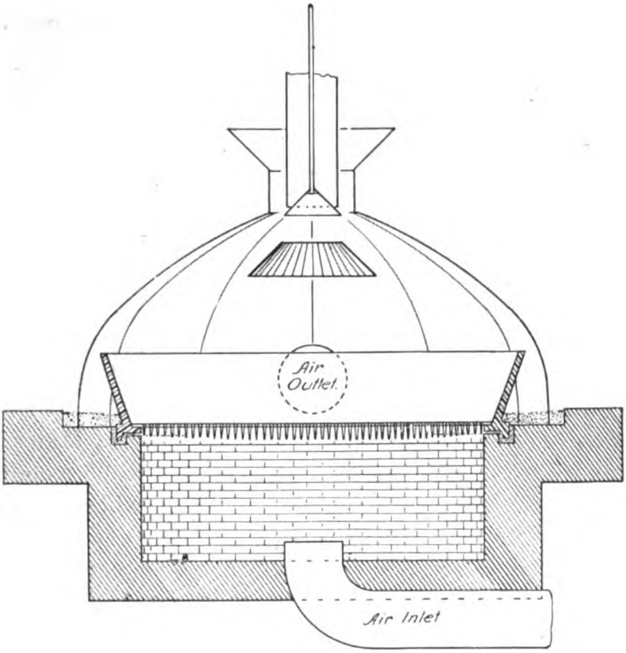

FIG. 88. NAKAMURA ROASTING POT.

The fine ore which is roasted in this device contains about 2½% Cu and 30% S. With it is mixed, as noted, the gneiss used for flux, and the flue-dust, which contains about 2% Cu and 10% S. The first material placed upon the grate is burning pyrite, which is spread about 1½ in. deep. Over this is placed a layer of hot dust from the pot last dumped, and then three tons of the fine-ore agglomerate. The pot is then covered, and the blast (about 40 mm. pressure) turned on. At intervals during the next four hours three additional charges of three tons each of ore are fed into the pot. In all, about 12½ tons is roasted in four hours. On completion of the roast the cover is lifted and the pot raised and turned, dumping the charge upon iron domes so as to break the sintered mass into convenient pieces for handling.

The roasted fine ore, with lump ore and coke, are fed to the first blast-furnace. This produces a matte containing 20% copper and a slag with 0.2% copper. The matte is smelted a second time, bringing it up to 30 or 35% copper and making a second slag assaying 0.3%. This matte is roasted in stalls and then smelted in a cupola furnace, bringing it up to 60 or 65% Cu. The slag from this furnace, assaying 0.8% Cu, is sent back to the first blast-furnace. The white metal is delivered to a reverberatory furnace, where a strong air-blast is blown over the top while a slag is rabbled off which is returned to the cupola. The product from this reverberatory is 98 to 99% copper. It is cooled in the air, no fresh water being available on the island, and finally melted, and poled in a refining furnace, from which it is cast into ingots. In all about 7000 metric tons per year is produced. Of this, about one-fourth is made into sheets and wire at the Sumitomo works at Osaka. The remainder is shipped in the form of ingots to Hamburg and London.

As in most Japanese copper mines, great pains are taken to purify the mine-water before running it to waste, and particularly to precipitate all copper. There are two cementation plants, and both scrap iron and coke are used to throw down the copper. About 180 tons of cement copper is saved per year. The precipitation plant is thus described in a report by the Bureau of Mines:

"The mine water from the third adit is sent to 90 wooden boxes, each 8 ft. in length, 3 ft. wide, and 1 ft. deep, which contain scrap iron; the slope being at the rate of 1 in 5.5. The average production of the precipitate is 35 tons per month which contains 50% Cu. The water which has passed through these boxes is conducted by launder to another plant about 1900 ft. below which is made up of 152 boxes each 12 ft. in length, 8 ft. in width, and 4 ft. 6 in. deep, sloping at 1 to 44. These boxes are arranged in four series, in each of which the water flows up and down. In this case scrap iron and coke are used as precipitants, consumption of the former being 23% of the precipitate produced. The precipitate taken is on the average 50 tons a month which contains 8% copper. The coke, in which the precipitate is absorbed, is produced to an amount of 200 tons per month which has 1% copper. All these products are sent to the smelting works at Shisakajima."

The cement copper from inside the mine, where boxes are placed at convenient points, averages 80% Cu; that from the outside boxes is of lower grade, owing to the presence of dust and iron oxide. From the precipitation plant the water is conducted by flume 9½ miles to the sea, so that there may be no possible chance of injury to crops growing below the mine.

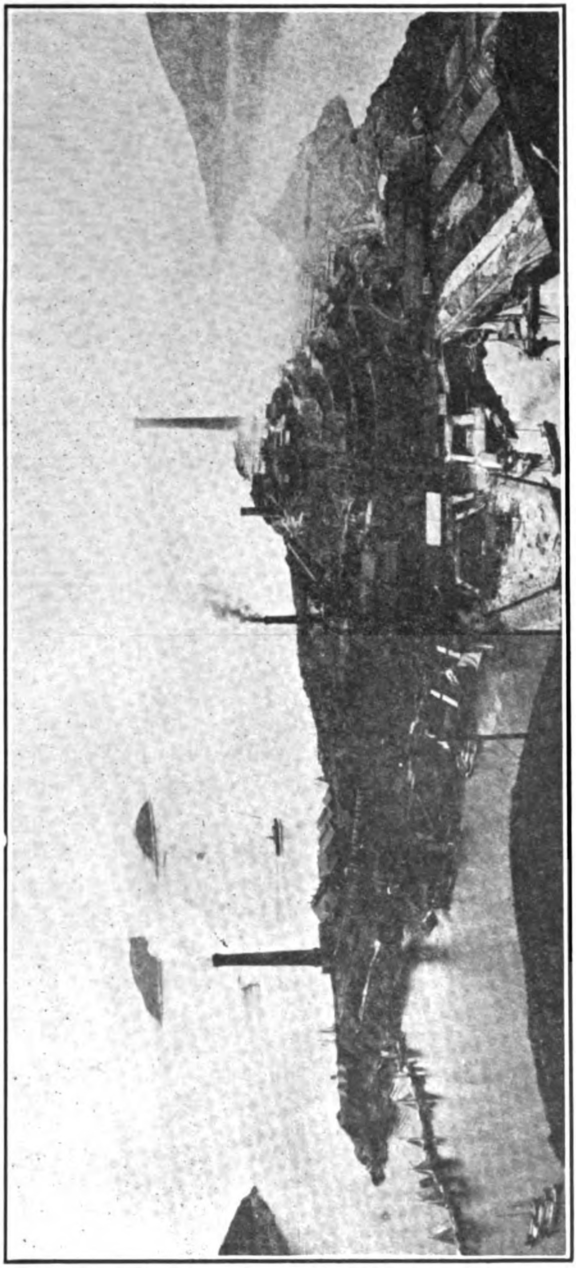

FIG. 89. SUMITOMO SMELTING WORKS, SHISAKAJIMA, JAPAN.

The Shisaka works are well and substantially built. They include coke-ovens from which the waste heat is utilized in boilers. and all modern devices and conveniences. The company provides homes, schools, and hospitals for its 4800 employees and brings fresh drinking water from the mainland. Wages are equivalent to 40 or 50c. per day, but it is to be remembered that houses, including water and baths, are rented to employees at a rate of 7½c. per month, and that rice, wood, and other supplies are sold at fixed rates less than their cost. With free medical and hospital service, here as at other works in Japan, mere statements of wages do not measure costs to the company or benefits to the men. The people look prosperous and happy. They were evidently well fed and cared for, and the company is known to be making an excellent profit. The American engineers steamed away from the hospitable harbor to the accompaniment of salutes and daylight fireworks, with the comfortable feeling of having seen good work well conducted.

THE CERRO DE PASCO SMELTING PLANT

By Spencer Bishop

(January 24, 1914.)

The smelter of the Cerro de Pasco company is situated at La Fundicion, Peru, a town having an elevation of 14,000 ft. Because of the altitude, the Company has a great many difficulties to fight against which are not seen or thought of by those who have not been there. There is snow, hail, or rain almost every day of the year. The average temperature is 44°F. The great majority of people going to that altitude are afflicted with *soroche* (mountain sickness), and the few who escape it have a great deal to be thankful for. The sickness is like a severe case of sea-sikness accompanied by sharp pains at the base of the brain and in the chest, lasting from two days to two weeks. The attack may occur each time one goes from a low to a high altitude. The fear of pneumonia is always present, as in any high altitude, but nature favors mankind, for germ diseases are not contracted as easily as in lower altitudes.

Callao is the port of entry, and also the terminus of the Peruvian Central railroad. Eight miles inland is Lima, the largest city in Peru. From here the ascent of the mountains commences. Leaving Lima at 7 a. m., the very gradual climb is begun. For an hour the roadbed is on one side of a valley which soon contracts to a ravine, just wide enough for the Rimac river and the roadbed. Now and again it widens so that small towns can find room for their mud huts. At 3 p. m., after winding up the narrow and deep cañon for 98 miles, Ticlio is reached, which is the highest station on the road, being 15,665 ft. above sea-level. A few hundred feet beyond the station the last tunnel is entered, where the elevation is 50 ft. higher than at Ticlio. Half way through the tunnel the grade starts downhill to Oroya, elevation 12,000 ft. above sea-level, which is reached at 5 p. m. and is 130 miles from Lima.

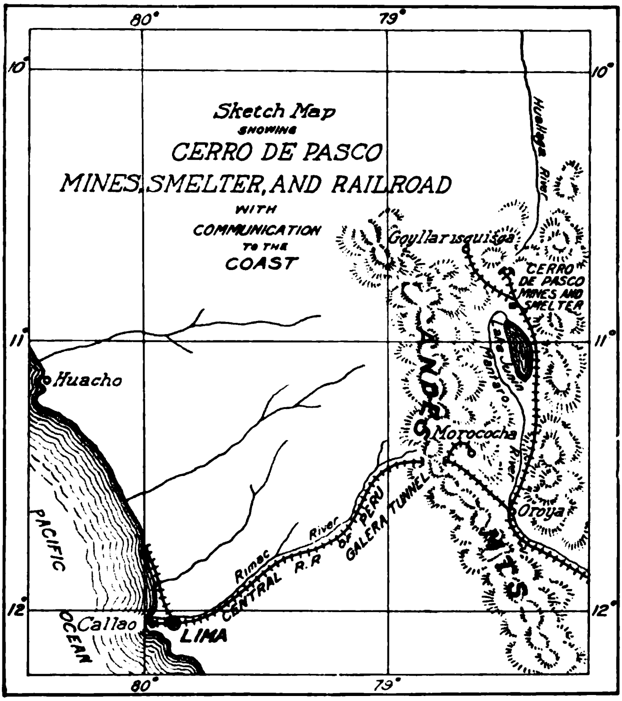

FIG. 90. MAP SHOWING SITUATION OF CERRO DE PASCO.

There are three long switch-backs, seventeen bridges, and sixteen tunnels on the road. At Oroya a change of cars is made for the smelter. This new line is owned by the Cerro de Pasco Mining Co., and runs from Oroya to the coal mines at Goyelarisquisga. There are two branches on the road, one of about six miles which runs to Cerro de Pasco, and the other about eleven miles long which runs to the other coal mine at Quishuarcaucha. The smelter at La Fundicion, 75 miles from Oroya, is reached about 9 p. m.; and eight miles beyond the smelter, Cerro de Pasco, where most of the copper mines are situated, is reached by 9:45 o'clock.

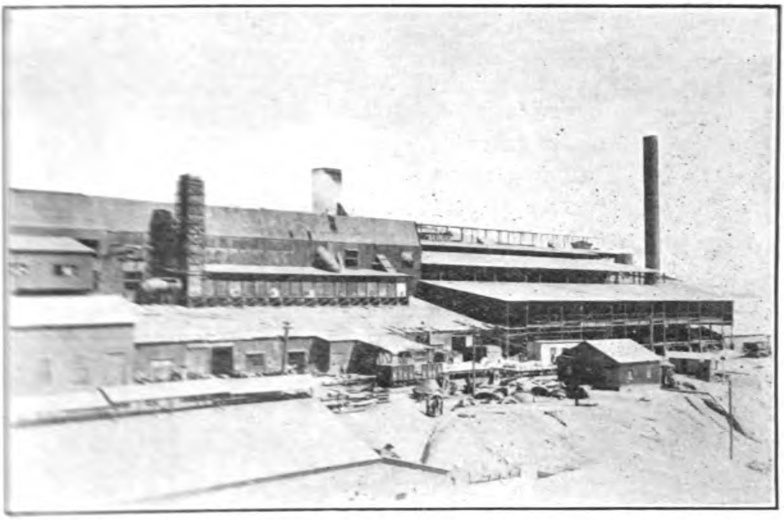

FIG. 91. EXTERIOR OF SMELTER.

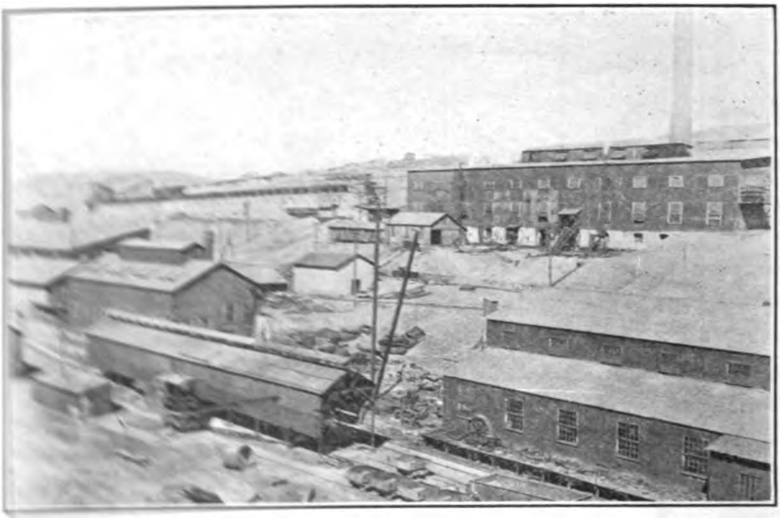

FIG. 92. SMELTING PLANT AND STACK.

[illegible] Pasco boasts of 25,000 inhabitants. It lies at an elevation [illegible] in a saddle of a mountain range, through which runs [illegible] Lima. The town is over a hundred years old. Various [illegible] have been given to the name. At a glance, 'A Pass [illegible]' appears correct, but inasmuch as *pasco* in the Indian [illegible]'s camping place,' more guessing is required for the true meaning. The Indians in the neighborhood seem to be ignorant of the origin of the name.

Great changes have taken place since the smelter was started, and problems have been solved that were at first thought to be impossible. From three 56 by 180-in. blast-furnaces and six 7-ft. Parrot-type converters, the plant has grown until it now consists of five blast-furnaces, two 10 ft. by 25 ft. 10 in. Pierce-Smith basic-lined converters, two 11-ft. spherical converters, and two of the old 7-ft. Parrot-type converters, five 19 by 60-ft. coal-fired reverberatory furnaces, fourteen 18-ft. six-hearth McDougall roasters, and ten Dwight-Lloyd sintering machines.

The blast-furnaces, converters, and new dust chamber have been remodeled. The lower parts of three furnaces have been left unaltered, the old brick tops were replaced by steel similar to those on the two completed furnaces, with the addition of a 9-ft. gooseneck instead of one of 6-ft. diameter. This larger flue proved to be exceptionally satisfactory. From constant use, these furnaces had opened out to 72 in. at the tuyeres, making the furnaces 72 by 180 in., although the two new ones are 48 by 177 in. With a blast of from 32 to 36 oz., these furnaces are smelting about double what they originally did.

The McDougall roasters have been installed as they were needed, there being now fourteen of them. A new flue is being built to increase the draft, so that the capacity will be greater than at present. Two of these furnaces are used to dry silica for the basic converters. The others roast ore for the reverberatories. Only three of the five reverberatory furnaces are in constant use. Each furnace smelts 60 tons of charge with 45 tons of coal. One of these furnaces has waste-heat boilers attached, and with the three 300-hp. boilers, 400 boiler horse-power is obtained. The steam is piped to the power-house. With the remodeling of the furnaces, more converters were necessary, and the original six Parrot-type were to be replaced by five 11-ft. spherical acid-lined converters. Before these had been installed, a Pierce-Smith basic converter was tried, with the result that the present battery comprises two 10-ft. by 25 ft. 10-in. Pierce-Smith, two 11-ft. spherical, and two 7-ft. Parrot converters, the last two being used only occasionally. At present a battery of ten Dwight-Lloyd sintering machines is being erected, the trial some months ago of one of these machines having proved their worth. With the large amount of fine ore the Company must handle, they will make a large saving.

At the power-house most of the engines are to be electrically driven by the English system of rope-drive. One blowing engine for the converters, one blast-furnace engine, and one air-compressor will remain on steam. These are to take care of the sudden extra loading that may occur. This will practically do away with the boiler-house, as the waste-heat boilers should deliver enough steam for this load. The electricity for this change and for similar work at the mines is to be obtained by water-power at Oroya, where 10,000 hp. will be generated by means of three Allis-Chalmers units which are being installed. The electricity will be sent to the smelter, 75 miles away, at 50,000 volts. There it will be reduced to 10,000 volts for the mines and 2200 volts for smelter use. At the mines 250 volts will be used for the motors. The system of rope-drive connecting the air-compressors which is used in the smelter power-house will also be used at the mines. The steam hoists will remain, but will be operated by air in place of steam.

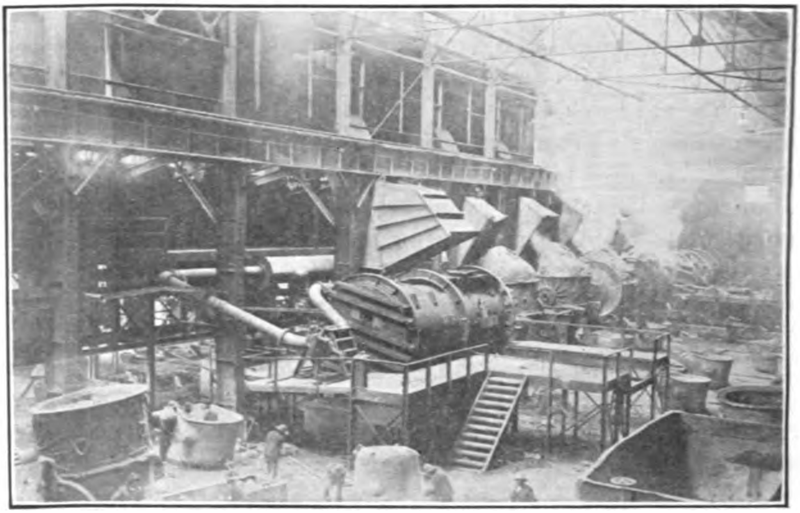

FIG. 93. CONVERTER-ROOM.

Quite a few foreigners are to be found among the employees, but the lower class of labor is picked from the natives. Unfortunately, these are very ignorant, small in stature, slow of action, superstitious, and not muscular. It takes several of them to do the work that the average white man can do. Most of the natives come from the agricultural districts and have no idea of mechanical work. Any excuse to lay off from work is welcomed, and the many religious holidays, together with 'pay-day,' afford the opportunity.

At La Fundicion the Company has a coal-washing plant, and seventy 12-ft. beehive coke ovens in which most of the Company's own coke is made. In addition to the group of mines at Cerro de Pasco, the Company is operating mines at Morococha, from which large quantities of high-grade ore is mined. Morococha is a few miles from Ticlio among very high mountains, and here most interesting geological formations can be seen.

The Company employs about 5000 men. Of this number, 3000 are at the smelter and the other 2000 in the different mines. At each camp a hospital for emergencies is maintained, and there is also a large, well equipped general hospital at Cerro de Pasco. A small fee is charged each employee toward the maintenance of the hospital.

HANDLING FLUE-DUST AT THE COPPER QUEEN SMELTER

By James Douglas

(December 13, 1913.)

*Experiments made at the Copper Queen smelter during 1909-10 showed that by reducing the velocity of the gases in the dust-chamber and preventing eddies, a large amount of the dust loss could be avoided. The shape of the dust-chamber was therefore altered, and its size increased. The gases from the converters were led through a large independent flue into a small subsidiary part of the dust-chamber at the base of the stack, and the finer material, as well as the dust, was extracted from the cupola charge and passed to the reverberatory furnace. A series of fresh experiments is about to be made to determine the efficiency of these conservative methods. But proof of their value is offered by the fact that the dust produced by the blast-furnaces has dropped from an average of 227 tons per day during the first half of 1912, when the reverberatory furnaces were not running, to an average of 116 tons per day since they started up.

The amount of coarse particles caught in the converter flue is two tons per day, but this is independent of the finer particles which settle in the brick header at the base of the stack. That recovered in the flue itself is very high grade, analyzing as follows: Au. 0.155 oz.; Ag, 15.4 oz.; Cu, 60.9%; SiO_2, 2.2%; Fe, 8%; S, 11.6 per cent.

A dust-chamber of 1200 sq. ft. cross-section and 132 ft. long was built, connected with the reverberatory furnaces, from the roof of

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*Reply to discussion of paper, 'The Copper Queen Mines and Works,' presented before the Institution of Mining and Metallurgy.

which are hung some 30,000 wires. From this chamber, during a period of thirteen months, only nine tons of dust has been recovered.

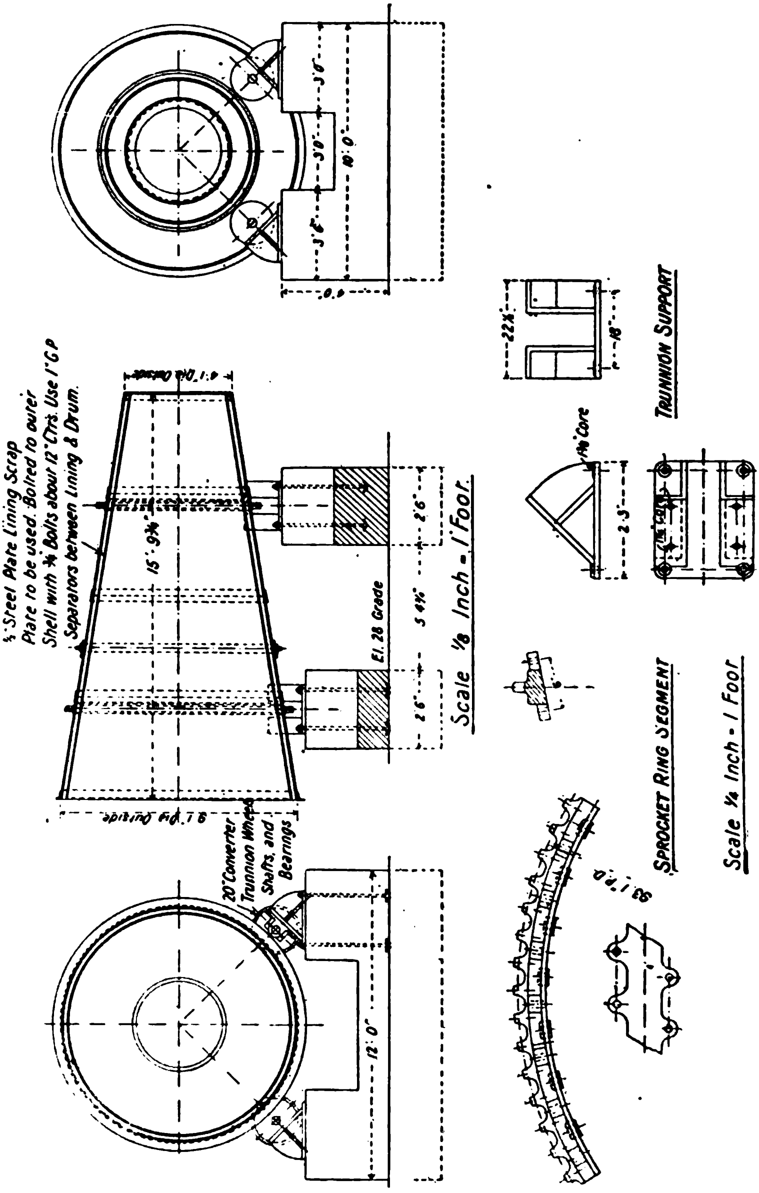

FIG. 94. MIXING DRUM FOR CONVERTER SLAG AND FLUE-DUST.

The dust in the chambers has not yet reached a point where the daily accumulation can be regularly drawn off. Most of it, however, probably will always settle in the boilers and flues which are

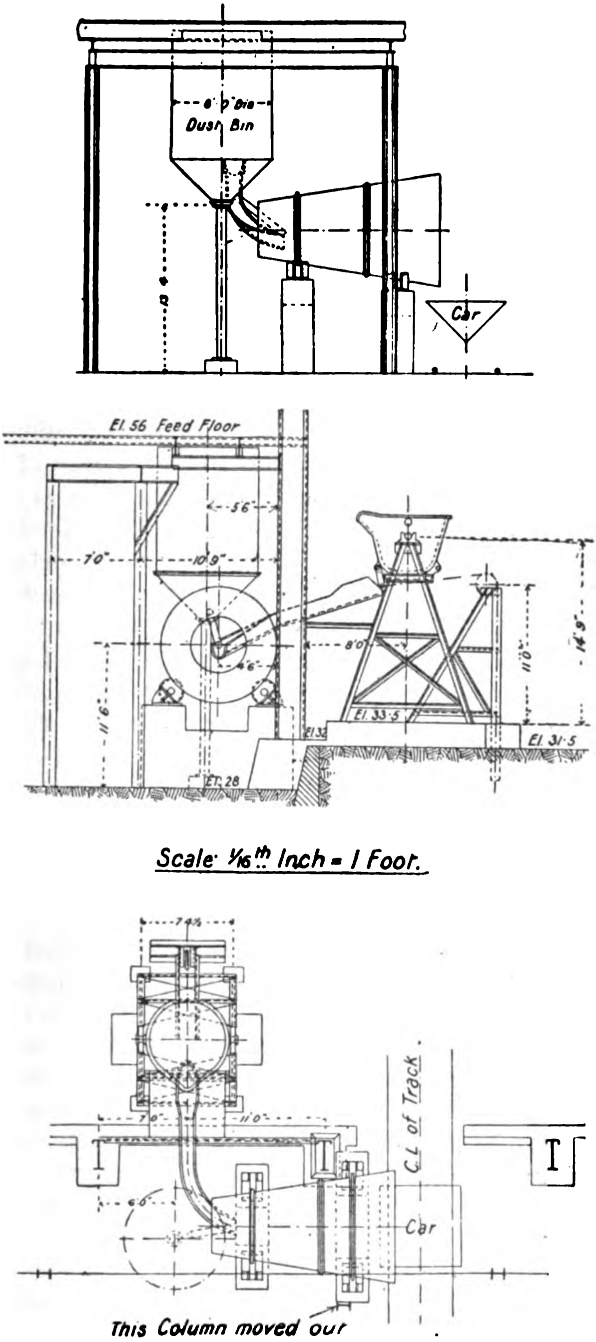

FIG. 95. ARRANGEMENT OF PLANT FOR MIXING CONVERTER SLAG AND FLUE-DUST.

interposed between the furnaces and the dust-chamber. The dust-chamber connected with the McDougall roaster plant is 1370 sq. ft. cross-section. It is 144 ft. long, and has some 42,000 wires hanging from the roof. To date, the dust recovered is 6% of the charge roasted.

Frank Rutherford has furnished the drawings and following description of the converter slag and flue-dust sintering apparatus and methods at Douglas:

In June 1909 a cylinder 4 ft. in diameter and 9 ft. long, lined with firebrick, was set up in an inclined position so that the material made would discharge from the end. This machine soon demonstrated that a good product for blast-furnaces could be made by pouring molten converter slag and flue-dust together into one end of it, but that the crust or collars made were very difficult to remove. Naturally, the conditions suggested that instead of a cylinder a truncated cone would overcome all the difficulties and still give a good product. The working of the cone completely satisfied expectations as to dislodging the crust, and was a delightful surprise as to the tonnage it could handle. The conveyor, as shown in the drawing, was an essential part of the plan.

This machine ran for a number of months and proved conclusively the practicability of the process. The machine, working three shifts, easily handled all the converter slag made, and from 60 to 70 tons per day of flue-dust. The best and most even product was made when the dust used was from 18 to 20% of the converter slag by weight, and 25% was found to be the practical limit at which good coarse product could be made unmixed with flue-dust. The maximum day's product, 24 hours run, was 535 tons, with an average of 325 tons per working day over a period of five months, which included time taken for all minor repairs done while the shifts were on. The power was supplied by a 50-hp. motor through a train of gears and chain drive, and gave eight revolutions of the drum per minute. This speed might, with advantage, be somewhat reduced, and the method of drive be improved upon. Both of these changes would have been made had the experiments been prolonged, but the reverberatory furnaces going into commission required all the flue-dust and brought the experiments to an end.

If the flue-dust contains much sulphur, the amount of sulphur dioxide gas liberated is very great, and provision for its removal must be made, and for collecting a small amount of fine dust. Trial runs made with matte instead of converter slag proved that, although the matte would take up much more flue-dust, the collars were much heavier than with the slag, and therefore harder to break. While working with converter slag and dust it was found that a small jet of water playing on the crust as it formed was sometimes quite a help, but with hot slag, free from white metal, the crust broke by its own weight, and therefore did not require the aid of a bar and hammer. Had the work been continued it would have been desirable to rebuild the machine, making it much heavier and increasing the diameter of the small end to about 6 ft. in order the better to dislodge the crust. Two of these cones have been installed in the new smelter of the Arizona Copper Co., Clifton, Arizona, primarily to mix fine silicious material with converter slag before going to the reverberatory furnaces, and thus obtain a better fluxing of this silicious material than could otherwise be obtained.

SMELTER CHARGES IN UTAH

(January 20, 1912.)

The bulk of the silver-lead ores shipped from Tintic district, Utah, to the smelters in Salt Lake valley pay a treatment charge of about $1.50 to $2.50 per ton, based on a certain content of lead, usually 10%. If the lead content is below 10% there is an additional charge of 8c. per unit, and if above 10% there is a credit of 5c. per unit. A charge of 12c. per unit is made on all silica, of which Tintic ores carry 60 to 80%. The shipper is debited with 25c. per unit for sulphur, 30c. per unit on all zinc over 8%, and 50c. for speiss, consisting mostly of arsenic and antimony. Iron is paid for at 10c. per unit, and in settling for the copper in lead ores the smelters first deduct 1.3% from the assay value, and about 5c. per pound from the market price on cathode copper; that is, copper in lead ore running less than 1.3% is not paid for. The producer is paid for 90% of the lead content, less 1¼ to 1½c. per pound deducted from the market price for transportation and refining of bullion. He is paid for 95% of the silver at New York prices. Based on the foregoing figures it is apparent that producers of Tintic ores pay a treatment charge of $8 to $10 per ton.

Most of the Park City operators pay the smelters a flat rate of $10 to $12 per ton, the transportation charge being $2 per ton on crude ore and concentrate. Settlements for lead, silver, and gold in the ore are the same as those applicable to Tintic ores. Copper in Park City silver-lead ores is paid for at 5c. per pound less than the New York price, after deducting the difference between the wet and dry assay. Zinc is an important factor in the lead-silver ores of Park City, and the entire product shipped is in the form of concentrate. Most of it is shipped to Colorado and Kansas plants. Apparently the zinc situation has undergone some changes within the past seven months, and some of the zinc producers complain of exacting conditions under which zincblende concentrate is now marketed. Such a product, sampling 40% zinc, 4 to 5% lead, and 8 to 10 oz. silver, with spelter at 5c. per pound in St. Louis, is sold f.o.b. Park City at $10 to $12 per ton. A penalty on lime is charged, and there is also a penalty if 50% of the concentrate runs finer than a certain mesh. There are no credits for iron, nor gold, of which there is a small amount. The only mine in Park City district the ores of which are not treated on the flat-rate basis is the Daly-Judge. Its lead-silver ore, both crude and concentrated, is being shipped to the international plant, the treatment charges being figured on the silica and iron basis.

A considerable tonnage of silver-lead-zinc ore is received at the Midvale mill of the United States S. R. & M. Co. In settlement for this the producer is paid $20 per ounce for 90% of the gold; he is paid market price for 80% of the silver, 80% of the lead, less 1½c. per pound; 60% of the iron at 6c. per unit, and 30% of the zinc content at 2c. per pound. He pays a treatment charge of $3 per ton, a roasting charge of $1.50 per ton, and is charged for 10% of the silica at 12c. per unit.

THE MINING AND SMELTING INDUSTRY OF COLORADO

By Franklin Guiterman

(November 26, 1910.)

*Abstract of an address read before the Denver Chamber of Commerce.

*In its recent call for the Annual Convention of the Mining Congress, the invitation said in part: "Within the last three years, it is safe to say, the total annual output of mineral, coal, and oil wealth in the United States has decreased not less than 25% and perhaps 33%." Presumably this startling declaration was founded on statistics furnished by the United States Government, and may in consequence be accepted as true. In Colorado the falling off in mineral production has been confined practically to gold, silver, and lead, especially silver and lead, and the decrease has been so startling as to lead this Chamber to appoint a committee whose chief function is to make a thorough inquiry into the causes for the decadence of our mining and smelting industries, and to suggest or their revival. I am advised that it is the custom of this to call for reports of progress from its several committees from time to time, and, as chairman of the committee on mines and mining, I have been requested to say something to you about the present status of the mining and smelting industries in Colorado, and to indicate in some measure the results which have followed our investigations to date. If your committee is not able at the present time to furnish the Chamber a specific report on the important matters which have engaged its attention, it may be said in extenuation that your board of directors was advised initially that the work which has been undertaken is one of magnitude and that the gathering of the necessary data would involve much time, detailed labor, and analysis. It is not unlikely even that failure to secure the desired and necessary information may result. At the present time it is too early to state what the outcome of our investigations will be. It may, however, be permissible to dwell briefly on some of the more important causes which have contributed to the decadence of mining in Colorado with the coincident enforced curtailment of the smelting industry, which is dependent thereon. It is most unfortunate for our state that the impairment of the mining and smelting industries has from time to time been made a political issue, and that in discussing these vital industries, some have not hesitated to misrepresent the facts, and thus mislead not only our own citizens, but also those from other states. It is unfortunate that the subject of ore-treatment charges and of ore schedules is complex; that the metallurgical processes are intricate and technical, and that the metal recoveries attendant on these processes are sometimes astoundingly variable and often uncertain in their outcome. Coincident with these technical considerations comes the application by smelting and milling companies of tariffs, or ore schedules, as they are termed, and as these schedules vary greatly with the character as well as the value of ore, it appears, superficially considered, that the rates of payment for metals in ores are not only complex and sometimes incongruous, but that they are framed with the specific underlying idea to confuse, if not to deceive, the ore seller. To him, however, who is familiar with the marketing of ores, a familiarity which comes with mining and commercial experience, there is nothing obscure or involved in the schedules; and for him it is a simple matter to resolve at once the terms of payment for his ore into a net value per ton. The more complex the ore, considered in its mineralogical character, the more complex apparently is the ore schedule on which the mill or smelter buys it. To illustrate:

The simplest tariffs are those applying to ores which are treated in a stamp-mill, where the ore is stamped, passed over amalgamated plates where the gold is extracted to as large an extent as possible, and the tailing treated on concentrating tables. In this case the millman charges the ore producer a fixed sum per ton for milling the ore, and turns over to him the gold amalgam and the concentrate. The amalgam is retorted and the gold sold to the Mint, while the concentrate is sold to the smelter on a simple schedule, because the product is simple in its mineralogical nature. The second case is that covering such ores as the Cripple Creek gold ores, in which a fixed sum per ounce is paid for the gold content and an increasing treatment charge per ton is applied as the ores increase in value. The last case is that involving the purchase of ores, complex in their mineralogical character, which may contain gold, silver, copper, lead, zinc, bismuth, antimony, and arsenic. Such ores in smelting involve the production of by-products which demand repeated metallurgical treatment. In these processes the recoveries of the metals, gold, silver, lead, and copper, are dependent on technical skill. The payment for recoverable metal content in such ores is governed by the ability of the smelter to effect certain savings which experience has shown can be attained, and the treatment charges of the ore schedule must be based not alone on the expense obtaining in the actual smelting operations, but on the technical considerations governing possible metal recoveries as well. Ores known as fluxing ores, and without which the latter could not be smelted, have the lowest treatment charges applied to them regardless of the smelting expense, while other ores which demand fluxing ores in smelting have to bear greater smelting charges. This is a natural as well as a commercial distinction, for the fluxing ores are, almost without exception, low grade, while the silicious ores are generally of higher value. Therefore, smelting schedules must apparently become more complex with increasing complexity of the ore to be treated. It is impossible to frame the schedules in a manner which will readily be understood by those inexperienced in ore production and ore rates, or unfamiliar with the mining and smelting industries.

COLORADO PRODUCTION, 1900-1909.

	1900.	1909.	Decline, %
Gold	1,400,000 oz.	1,061,000 oz.	24.21
Silver	20,300,000 oz.	8,900,000 oz.	56.16
Lead	82,137,000 lb.	32,360,000 lb.	60.60
Copper	No change.		
Zinc	Increasing in output.		
Total decrease in value	$50,314,000 to	$30,916,000	38.55

Notwithstanding the continued efforts on the part of milling and
companies to extend aid to the mining industry in Colorado.
has been a continued recession from the high level of

prosperity which it occupied a decade ago. The extraordinary decline in production for this period will be seen from the preceding comparison.

Colorado's ability to maintain its gold production even on the present level has been due alone to continued reductions in treatment charges, those on Cripple Creek ores having been reduced in the last ten years about $5 per ton on the average. The efforts of the smelting companies to check the decline in production of the silver-lead ores have been entirely unavailing, notwithstanding reductions in treatment charges aggregating on this class about $3 per ton. Not only have the heavy reductions in treatment charges not resulted in the maintenance of the former tonnage production, but with the steady decline in the mineral output, the American Smelting & Refining Co. has been obliged to put one-half of its Colorado plants out of operation and operates the remainder on curtailed capacity.

It is true that prospecting in Colorado has practically ceased for many years; that the orebodies which heretofore furnished ore for the smelters have been exhausted to a large extent, or are approaching that stage; that to a large degree mining properties once operated under company management are now worked by lessees, and that in most cases development work is being neglected; and finally, that in closed mines where orebodies have been exhausted, or only ore of too low grade is left, or where such physical disadvantages as the necessity for an ore concentration plant or a heavy pumping equipment has to be faced, the investment of new capital is demanded. Such capital, however, will not be forthcoming unless it is made clear to the investing public that Colorado still offers a profitable field for mining ventures when undertaken and guided by competent and conservative management. The desired end can only be accomplished by co-operation all along the line, between mining, milling, and smelting interests, railways, and by the press in inviting mining investors to come to Colorado with the assurance of square and fair dealing on the part of ore-purchasing concerns.

Colorado's metallurgical engineers, that is, those who have acquired their initial experience in this state, have indisputably stamped their individuality on metallurgical progress the world over. It was their efforts which have wrought success in the mineral fields of South Africa, New South Wales, and Australia. It is to them that the astounding and astonishingly successful copper milling and smelting methods in Montana and Utah are due. It is they who have placed the chlorination and cyanide practice of the Cripple Creek field on a higher plane of metallurgical achievement.

It is they who have created a practice of silver-lead smelting which is acknowledged as one to be patterned after in other states, and it is they who are now utilizing to the best and largest extent our present mineral resources and who may be trusted satisfactorily to take care of those which our pluck and energy may yet bring to light.

If indications point aright, there is reason to believe that our mining conditions will greatly improve in the near future. With the deep-drainage tunnel practically completed at Cripple Creek, an increasing tonnage may be expected to be produced in that district; with the discovery of important bodies of zinc carbonate at Leadville a new impetus will be given to enlarged development and operations in that district, which in truth furnishes the basis of our smelting industry; and with the renewed activities in those two districts and an advancing price of silver, we may confidently expect an awakening and quickening in all of our other mining camps.

COPPER THROUGH FIFTY YEARS

By James Douglas

(May 21, 1910.)

The copper industry of this country since the middle of the last century presents many points of interest. It has been subject to the usual vicissitudes of trade and finance. It has tried the experiment of high protection and of no protection at all. It was at one time controlled by an agreement of the large producers, but during the longer period has been subjected to unlimited and untrammeled competition. And yet it has prospered. In production the United States has grown from almost insignificance among the world's contributors to being the controlling influence of the world's market. For our report of our monthly output the copper market waits with bated breath.

In 1850 the rank of the world's copper producing countries was as shown in the table opposite.

In 1909 the position of most of these countries is strangely reversed.

The United States stands first with a production of 1,099,840,000 lb. or 58.23% of the world's production, and Great Britain's production has become a negligible quantity. Mexico does not appear in the old list, but today takes second rank. Australia has risen to third place, followed by Chile, which during the seventh decade of the last century rose to first place. Chile then declined through the exhaustion of rich mines, but is again coming to the front. During

World's Production of Copper for 1850.

Country.	Pounds.	Per cent of total.
Great Britain	29,648,000	26.8
Chile	25,000,000	22.9
Russia	12,900,000	11.8
Austrian Empire	6,660,000	6.0
Cuba	6,800,000	6.3
Australia and New Zealand	7,000,000	6.4
Scandinavia	4,000,000	3.7
German States	2,800,000	2.6
Rest of Europe	2,000,000	1.8
Turkey	2,000,000	1.8
Asia	6,000,000	5.4
South America (except Chile)	2,400,000	2.2
Africa	1,200,000	1.1
United States and Canada	1,300,000	1.2
Total	109,708,000	100.0

World's Production of Copper for 1909.

Country.	Pounds.	Per cent of total.
United States	1,099,840,000	58.23
Mexico	125,440,000	6.64
Spain and Portugal	116,480,000	6.16
Japan	115,280,000	5.56
Australasia	98,560,000	5.22
Chile	84,000,000	4.54
Canada	51,520,000	2.72
Russia	50,400,000	2.66
Peru	44,800,000	2.37
Germany	44,800,000	2.37
Norway	22,400,000	1.18
Cape Colony	15,680,000	0.83
Italy	7,840,000	0.41
Bolivia	6,160,000	0.32
Newfoundland	4,480,000	0.23
Sweden	3,360,000	0.17
Austria	2,800,000	0.14
Turkey	2,240,000	0.11
Argentina	1,120,000	0.06
Great Britain	1,120,000	0.06
Hungary	224,000	0.01
Algeria	224,000	0.1
Total	1,888,768,000	100.00

that period Venezuela became prominent mainly through one great mine, the New Quebrada, but another has never been found and the country has disappeared from the list. Peru, on the other hand, is in the ascendant. Africa's contribution during the half century has been from the coast mines of the Cape Copper Co. but may, now that the interior is being reached by rail, become a large factor. The rise or decline of a country's production, or even of a single large mine. may sensibly affect prices, as the volume of the world's copper production is comparatively small. This has been conspicuously shown by our own experience.

From 1857 to 1861, during which years our mines increased their output from 10,700,000 lb. a year to 16,800,000, copper was on the free list and the world's price virtually regulated ours. With the breaking out of the war the Morrill tariff act was passed, which taxed everything on which revenue could be raised and copper was stricken from the free list. The duty first imposed was 2c. per pound and from 1864 to 1869 2½c. was the rate. Our production increased from 16,800,000 lb. in 1861 to 28,000,000 in 1869. This rapid rise was due to the development of the Calumet & Hecla. The effect of the 2 and 2½c. duty was immediately felt in the higher price of copper here. In 1863 the price of Lake ingot in this country was 4c. above the price of Chile bars in England. Though this difference was not maintained, the average for a time was about 3 cents.

The tariff tendency, steadily upward till checked, was of course taken advantage of by the copper industries, and therefore they demanded more. The demand was acceded to and the duty on ingots was raised to 5c. in December, 1869. And at 5c. per pound it continued until 1882, except during 1872 and 1873, when it was reduced to 4½c. Meanwhile the annual production of the country had risen from 28,000,000 to 90,600,000 lb. and great changes had occurred, disturbing the balance of power, as it existed during the supremacy of the Lake mines. From the time when Calumet & Hecla commenced its phenomenal production, say in 1868, till 1881, the percentage of production by Michigan had never fallen below 80% of the country's total and had risen as high as 95.7 in 1872. During that period our production had begun to exceed our consumption. There was even a small balance of 102,821 lb. for export in 1864. In order to secure full advantage of the duty under these circumstances some combination by the copper companies had to be devised. The producers outside of Michigan up to that date might be neglected. The Lake companies alone were therefore included in the several plans which were either proposed or carried out. The *Mining and Scientific Press*, in its issue of July 20, 1872, said "The copper ring has contracted at a stipulated price for the product of the prominent Lake Superior mines for a certain length of time, reported all the way from six to twelve months." The scheme miscarried, for English holders of Lake copper who had bought large quantities of the metal the previous year, took advantage of the higher American price to unload their unused stock on the market, though the original buyers had pledged themselves not to re-export. The final attempt to create a copper trust commenced in 1879 and ended by decree of the court in 1885. The story is told in the 'Tariff Reform' Pamphlet Vol. III, No. 2, page 440.

"The combination was formed by the following mines: The Calumet & Hecla, Atlantic, Quincy, Central, Allouez, Franklin, Pewabic, and Huron; the Copper Falls and Conglomerate companies were admitted later. It was agreed that all the sales of copper should be made by the sales agent of the Calumet & Hecla. The aim was to keep up the price of copper in this country to the limit allowed by the tariff of 5c. per pound. The surplus produced over the consumption in the United States was to be exported and sold on the condition that the buyers would not re-import it. This plan continued till 1883, when the sales agent of the Calumet & Hecla died. Thereupon, written contracts were made by all the mines mentioned above providing that all the copper produced should be pooled and sold by the Calumet & Hecla, either in this country or abroad. Ten per cent of the receipts from the sales of copper was to be retained for use either to buy Lake copper in the markets when they should become glutted, or in case it was thought best to hold the copper for a considerable length to assist the weaker mines to continue their operations. This contract was drawn October 13, 1883, to continue in force one year. On February 14, 1884, new contracts were drawn to continue the arrangements till January 1, 1885, and the third and last contract was drawn up and signed August 20, 1884, to continue the pool one year longer, from January 1, 1885. These contracts gave the Calumet & Hecla complete control over all the copper produced in Michigan to sell when it pleased and at what price it pleased. In the last contract the following provision occurs: 'It is the purpose of the Calumet & Hecla Mining Co. to make a large sale of copper for export at the best price obtainable, from 12,000 to 17,000 tons.' The sale was made and the copper was to be delivered in three installments in November, December, and January, and the price agreed upon was an advance of £4 per ton (equal to 0.86c. per pound) above the average price of Chile bars for those months. The Quincy company objected to this and proceeded to sell its copper for export through certain brokers in New York, who succeeded in getting more than a cent per pound above the price obtained by the Calumet & Hecla.

"The Calumet & Hecla for itself and the other mines in the pool, applied for an injunction from the Supreme Court of New York to stop the independent sale of copper by the Quincy company. The

Sixth article of the complaint is as follows: 'Acting upon the faith of and in full compliance with said agreements, said Calumet & Hecla Mining Co. entered into an agreement and made a contract with certain European parties for the sale and delivery during the periods covered by said agreements of many thousand tons of copper. A part of the consideration of said last named contract with the said European parties was the agreement made by the said Calumet & Hecla Mining Co., acting for itself and for all the other companies above named, that no more copper should be sold by said companies for exportation during said period, and the European parties agreed on their part that none of said copper should be reimported into the United States.' In answer, the defendant, the Quincy Mining Co., declares: 'For an answer and defense herein the defendant alleges that the contract referred to in the plaintiff's complaint was executed by the several parties thereto for the purpose of carrying into effect agreements which they had made to combine with each other to prevent a free competition among themselves in the business of selling their production of copper and to enable the plaintiff, the Calumet & Hecla Mining Co., to control the amount of Lake Superior copper which should be put upon the market and also to control and keep up beyond what it would otherwise be, the price at which the same should be sold during the continuance of said contracts.' In January, 1885, Judge Lawrence rendered his decision refusing to grant the injunction on the ground that it was contrary to public policy.''

Thus ended the only copper combine made by all the large copper companies who were in active operation at the time of its original formation. But in the interval between 1880 and 1885 natural forces less easy to evade than legal statutes came into play. Arizona had entered the field as a large producer in 1881 and Montana had followed suit in 1882, and these Western companies contained elements more difficult to harmonize than had been found in the Michigan group of mining companies. The effect on production and price was startling:

Year.	Production, lb.	Per cent of increase over previous year.	U. S. price of Lake Superior ingots. Cents.	Foreign price of Chile bars 2c. under ingots. Cents.
1881	71,681,000	18.6	18⅛	13¼
1882	90,646,000	12.5	18½	14½
1883	115,526,000	27.4	15⅞	13¾
1884	144,946,000	25.5	13⅞	11⅞
1885	165,875,000	14.4	11⅛	9½
1886	156,735,000	5.5*	11	8¾

*Decrease.

The Lake combination was ineffective in sustaining prices beyond 1882, owing to the entrance of the production of these Western mines into the market of the world and the impossibility of incorporating them into any copper trust. The Arizona Copper Co., making bars which ranked as Chile bars and were worth about 2c. per pound under Lake ingot, might have combined with the Lake companies; for in those early days Montana made matte only, and it was all, or nearly all, shipped to Europe for treatment. An equitable agreement between the Michigan and the Western mines would therefore have been difficult, even if the personal element among the few men who controlled the copper trade had not erected a more unsurmountable barrier against co-operation than difficulties arising from discrepancies in values and divisions of quantities.

Soon after the appearance of the Western copper the law of supply and demand asserted itself. The enormous and rapidly increasing production of Montana, added to the notable but less phenomenal growth of Arizona's output, made the balance for export so large that the scramble for the protection of the home market became furious. As a consequence the duty soon became inoperative so far as the miner was concerned, and the price in this country fell to that of the world plus the cost of importation. This must inevitably happen with any protected article produced in excess of the home demand, unless some artificial method of making the duty effective is devised. The Anaconda output was in fact the principal factor in bringing about the drop in the world's price of copper in 1885, 1886, and 1887, which preceded the formation of the Secretan Syndicate. The fall in price was so grave as to tell on production which, as the statistics show, fell off 5.5% between 1885 and 1886. The decline in price was such that Lake Copper sold languidly at 10c. and the Copper Queen company made contracts for its bars at 8c. and at even a trifle under. The tariff gave the distressed industry no help, but M. Secretan came to its assistance, offering to buy the output of the mines for three years at stipulated figures. There was, however, no understanding or concurrent action among the American producers in their dealings with the French syndicate. Some producers refused to sell, and with each of those who did sell there was an independent negotiation, and all by no means received a uniform price. The price paid was on an average 5c. per pound higher than had ruled for the two previous years. The effect on production is indicated by a rise of 15.4% between 1886 and 1887, and of 25.17 during the following year. On the failure of the syndicate, production was so far arrested that the figures for 1889 showed, for the whole country, an increase of only half a million pounds. But this stagnation was only temporary. The syndicate stocks were rapidly absorbed or disposed of, and in 1890 the price of copper rose to 15c. and production increased 14.5% over that of 1889. But the terrible crisis of 1893 was at hand, and told disastrously on both production and prices.

In 1893 the situation changed. There was a decrease instead of an increase in production, and the average price for Lake ingot fell to 9.5c. Despite the staggering blow, the copper trade rapidly picked up courage, and in 1894 showed a 7½% increase over 1893 and a rise in price of 1¼c. This resiliency, when the whole country was prostrate, may have been in spite of, or may have been the result of, the self-reliance which the placing of copper on the free list by the McKinley bill may have engendered. The plea for punishing the copper companies was that they had conspired with a foreign syndicate to artificially and injuriously influence prices. In fact, the companies which did sell to M. Secretan did individually what has been done before and since, namely, sell their production ahead to the highest bidder, at a most tempting figure, which raised most of the blighted concerns from despair into easy circumstances. So little benefit was derived from the tariff that when the trade was threatened by the complete removal of tariff protection the companies did not send a lobby to Washington or even protest. For years protection had ceased to protect. Every large company had opened its European sales agency and had secured its own customers, and therefore a market had been found for the total American surplus, no matter what it might be, if only the world's price were accepted. The removal of the duty therefore made no difference, but the financial crisis of 1893 did. Yet it was chiefly in the dividends. The price continued low till 1898 when a wave of high price developed, followed in 1902 by another decline. This lasted for three years, when a second wave within a decade of excessive price set in, followed by a relapse. These serious vacillations in price doubtless create disturbance in trade and stimulate, even if they do not originate, speculative manipulation. The only remedy conceivable would be a vast combination which would include both the mining and manufacturing interests, wealthy enough to absorb the existing companies by tempting the weak, and so omnipotent that no opposition could be started without risk of being crushed. The Secretan schemes were wrecked by not placating the manufacturers of the world, who unearthed unsuspected stocks of old and new copper of surprising magnititude. This remedy would probably be again administered. But assuming that such a world-wide combination in restraint of trade could be brought about with the view of securing of price, two consequenees would follow. The price would inevitably be so high that the public would pay very dearly for the luxury of consistent uniformity in value. And as a result of high prices, reduction in demand would follow. The supply can be controlled, the demand cannot, unless under rigid public control under state socialism. Past history showed a tariff without a trust to be powerless to create uniformity. A copper trust can no more be framed today than it could in the past. What we need more than a trust at present in this country is a large increase in consumption to use up the surplus that has been created during the past dull period. In this connection there would appear to be a great field for increasing consumption, if our copper mills would see their way clear to roll standard sizes of copper sheets for architectural purposes, and sell them to the public for roofing and such work as cheaply as copper wire is being sold at present. In the copper mills of Europe the difference between the price of wire and copper sheets is very much less than it is in this country, and this matter is well worth the serious consideration of our copper manufacturers, if they desire to retain this branch of their business for any long period.

HOUSING CONDITIONS AT THE GARFIELD SMELTER

By L. S. Austin

(April 23, 1910.)

In settling upon a site for a smelting works regard is had not only for convenience to railroad lines and to water supply, but also to convenient places where the men may live. In the Salt Lake valley, Utah, however, the injury to the adjoining country from the escape of sulphurous acid fumes from the works has compelled the selection of sites for new plants, distant from settled places where the smoke can do but little damage, and where the smelting company can so control adjacent land that it shall not fear troubles from litigation. At the Washoe plant, Anaconda, Montana, the works are situated two miles above the town, and the company has provided a street-car system for transporting its 2000 employees to and from the works and their dwellings. At the new plant of the International Smelting & Refining Co. in Pine canyon, Utah, four miles from Tooele, trains take the men to their work, and Tooele itself, an old settled place, affords an attractive town in which to live. The works of the American Smelters Securities Co. at Garfield are 17 miles from Salt Lake City, the distance being made in 40 minutes. Taking the wait-ing time at both ends and for the time spent on street cars in Salt Lake City, as much as three hours per day may be consumed by employees in going to and from their work. Part of this difficulty has been overcome by the establishing of a town at Garfield, about one and one-half miles from the smelter. It has been planned to operate a street-car line for the employees, but this has not yet materialized. Consequently the advantages of Garfield are nearly nullified.

When the works were started much foreign labor was employed, and many of the men built shacks below the works where they lived under temporary, primitive, and unsanitary conditions. They were not well under control, there was an unruly element among them, and disputes occurred due in part to the free use of beer and liquor. It was found by the superintendent, W. H. Howard, that it was hard to retain proper labor and he felt that suitable provisions should be made to take care of these Greeks, Japanese, and Austrians. At Garfield, however, money had been spent on well built houses, all expenses of a townsite had been incurred, and no sufficient returns had appeared to be coming in on the money thus expended. Of course the reply to this would be, that having met all initial expense of water-supply, grading, stores, and public or semi-public buildings, all houses thereafter would be constructed at a profit, though, on the other hand, it would be a question whether the new buildings would be fully occupied unless the street-car line were put in.

Naturally, with such an experience, the smelter management was slow to acceed to the requests for the investment of more money to take care of men at the smelting works. The conditions, however, were adverse, and it was difficult to keep even the foreign laboring population, so Mr. Howard was finally instructed to put in comparatively simple accommodations. The new townsite or settlement now consists of 100 houses or cabins, each capable of accommodating six men, occupied by Greeks and Austrians; one large mess-house and dormitory used by the Japanese; quarters for the repair gang of the works; a store or supply house, and a butcher shop.

The 100 cabins, 14 by 28 ft. in size, are one-story frame structures divided by a central partition into two rooms. They are unplastered, with the woodwork showing inside, the walls being one board in thickness. They cost $130 each; one room being used for sleeping, the other for a kitchen and living room for the six men, who are charged $1 each per month, certainly a liberal return on the investment. After the first 75 houses were built the revenue from rentals was employed for the construction of new cabins.

The Japanese quarters consist of a house 120 ft. long, also of lumber, and costing about $2000. It consists of a central part approximately 33 by 50 ft. used as a dining room and kitchen. The wings on either side are for the sleeping quarters of the men. In these the company has provided two high single iron cots with spring mattresses. At the end of each dormitory is provided a wash sink and two cement shower baths. The Japanese are a scrupulously clean people and hence it is possible to accommodate them in these crowded quarters. This building is under the charge of a committee, chosen by the men themselves, who keep up a strict discipline. Offenders are reprimanded or sent away by the committee chosen by the men, or by the manager to whom the committee is responsible. Meals are served in the dining room, except in cases of accident or sickness, and at such hours as are convenient for changing shifts, each man returning his dishes to the kitchen. The food includes a good deal of rice, meat stew, tea, and some bread spread with jelly. Turns are taken by the men in cleaning their quarters, which operation includes a monthly disinfection. The clothes and other little properties belonging to the men are kept in lockers and trunks adjoining the cots. No gambling is allowed and everyone is in bed by 11 p. m. The men generally go to bed by 7 or 8 o'clock, having a light supper and rise early.

The high fence of the works adjoins the grounds on the south. Openings are made through it to privies set inside; but separated from the plant and these are used by all the men with some degree of privacy. In the six-man houses occupied by the Greeks the usual custom is to cook by some agreement among themselves, provisions being bought by the mess. They have cook stoves with tile chimneys and depend on these for warmth.

Finally, and this has constituted the success and popularity of the whole settlement, food and supplies are furnished to the men at cost. The company has a store carrying food-stuffs and clothing, and the domestic supplies called for by the men. A storekeeper takes care of this at a salary of $125 per month, and he has the assistance of a clerk. A man cannot overrun his credit at the works, so that collections are assured. The average price for supplies is at least 20% less than elsewhere. Bread costs 4c. per pound. Meat is slaughtered near by and sold from a butcher shop on the ground. A good deal of beer is sold at the rate of $2.10 per case. Of this the Greeks consume about one bottle daily, and the Austrians three. To maintain order, a mayor is employed, who also acts as sheriff. He enforces excellent discipline, and the settlement is peaceful. To keep the place clean a garbage cart goes around daily.

The Japanese are noted for their eagerness to learn, and the interest they take in their work. They are able to do considerable mechanical and repair work; though the work of the regular mechanics, brick layers, carpenters, and machinists is done by white labor. The Greeks take comparatively little interest in their work, but can be relied upon to carry it on steadily without objection to furnaces and danger. The Austrians, who are a large and vigorous people, are depended on for the heavier work.

ORGANIZATION OF SMELTING ENTERPRISES

By Herbert Lang

(April 19 and 26, 1913.)

The greater number of reduction plants are built by companies or individuals who are already engaged in mining, as a necessary adjunct to their mineral properties. Others are set up in order to enter upon the business of buying and reducing what are called custom ores, that is, ores which are bought and sold in the market. While in the former case policy dictates that the works be put as near as may be to the point of production, in the latter it is generally found expedient to place them in localities central to a large number of producing mines and convenient to good facilities of transportation, which cut a great figure in this kind of enterprise, enabling the most desirable ores to be secured for the processes. Whether the works comes under the one or the other head, the procedure which governs its inception is much the same as in organizing any sort of industrial undertaking. While there is no reason why such enterprises cannot be carried on by single individuals or by partnerships, they are usually maintained by incorporations, upon whose organization may profitably be expended a few moments' study. Such incorporations are commonly managed by sundry officers, whose duties vary but little throughout all mining countries.

At the head there is a board of directors—or, as known in some regions, a board of trustees—in whom reposes all the authority of the concern, and which they may, by vote, delegate to subordinates of their own selection. The directors are chosen at regular meetings of the stockholders and hold office as a rule for one year. Their duties vary somewhat with the laws of the country, and also with the degree of their accountability to the stockholders whom they represent. In America these duties are much less stringent than in England, for example, and may be and frequently are performed in a perfunctory and careless sort of way that at times works to the prejudice of the company. Quite as often, however, this apparent looseness and carelessness works to the advantage of the organization, as it devolves the important duties upon those better fitted for it. The English system, by placing heavy responsibilities and wide powers on the directors at home, puts a premium upon ignorant intermeddling; the American, by reposing confidence in able if sometimes dangerous employees, brings out the best or the worst there is in them, and bespeaks a great success or a great failure. The American system makes the best managers and the greatest engineers; the English the best accountants and routine men. The directors in America receive no compensation, as a rule, but on the formation of the company some shares of stock are usually divided among the earlier members as a sort of payment for their services. In other countries they receive so much for each sitting that they attend. The president—in England the chairman—of the board is elected from their number, and he, if he be charged with active duties, may receive a regular salary, although in this country it is often esteemed *infra dig* to accept such. In England the managing directors, generally two in number, who are delegated to perform the services which in this country are supposed to be performed by the president, are in the habit of receiving substantial sums therefor, by vote at the annual meetings of their companies. Business ability is chiefly sought in such places, but social and political celebrity are not without their influence. Technical knowledge is, of course, equally important as business ability, and it is often remarked that companies are successful in proportion as the managers possess among themselves a practical knowledge of the art in which they have embarked. Although it is often quite practicable to secure technical operatives of approved skill, it is still desirable that the business managers of the enterprise should understand quite fully the requirements of the work in order that the technical staff shall be properly supported in their plans and efforts.

Briefly, the principal functions of the board of directors are: to conduct, manage, and control the affairs of the company; to make rules and regulations for the guidance of the officers and managers of the corporation; to appoint and remove for cause all officers, agents, and employees, and fix their compensation. A majority of the board constitutes a quorum, which has full power to transact business. The officers are, in America, the president, vice-president, secretary, treasurer, and sometimes a manager or general manager. The two former must be members of the board, and sometimes the secretary and treasurer also must be. They severally hold office at the pleasure of the board. The secretary almost universally, and the treasurer quite frequently, is in receipt of a salary, usually quite moderate.

The president's duties are to preside at all meetings of the board and of the stockholders; to sign papers requiring the company's signature; to sign certificates of stock; to draw checks upon the company's funds; to supervise and control, subject to the board of directors, the subordinate officers of the company; and to make reports to the stockholders, at their meetings, of the condition of the company. Such reports are commonly made at the annual meetings of the stockholders, which are an important and in some respects an obligatory feature. The vice-president's duties are confined to serving in the presidential capacity at meetings when the president himself is absent.

The secretary has the custody of the company's seal, which he affixes to such papers and documents as may require it. He keeps a minute of the proceedings at the meetings of the board of directors. and of the stockholders. He keeps the accounts of the company, attends to all transfers of stock, and has charge of the books. He counter-signs the stock certificates as they are issued, as well as all drafts that may be officially signed by the president. He receives the money due to the corporation and deposits it with the treasurer. He makes out the regular statement of receipts and expenditures which accompanies the annual reports.

The treasurer is the custodian of the company's funds, and pays them out upon drafts drawn by the properly constituted officers. Such is the customary organization of the corporations that carry on the great metallurgical operations of the century.

It has become quite common of late years for industrial companies in America to place the immediate conduct of their affairs in the hands of a manager or general manager, whose duties resemble in a measure those otherwise given to the president while being of a more practical nature. The manager in such cases stands between the board of directors and the subordinate employees. and also between the company and its customers. All business dealings are in his hands, and the routine of buying and selling is particularly under his control. An increasing tendency is observed among such companies to erect in this manner a sort of one-man power, with the manager at the head, taking away a part of the duties and authority. on the one hand, of the president and board, and on the other of the superintendent of the works. The latter, who in addition to the direct conduct of the works often possesses considerable authority in related matters of business, now finds himself to a large extent confined to the technical management alone.

The superintendents of most of the more important reduction works are men of technical education and training, as befits their calling. It often happens, however, that whatever the need be of men of this sort, such places come to be filled and well filled by persons whose time has been spent and their experience gained rather in actual practice than in theoretical training. Some of the best superintendents of metallurgical works are persons who have never entered a school of chemistry, but have grown up as it were alongside a furnace or stamp battery, and graduated into responsible places by dint of observation rather than by the study of books. But, however many instances there may be such, the fact remains and must not be lost sight of, that the technical manager of a successful works must be a technical man, and also that the best education for a technical man is acquired, or at least begun, in a technical school. If we divest the superintendent of those duties which are purely of a business cast, such as hiring and discharging employees, signing pay checks, supervising account keeping, rendering reports of the routine work done, buying and issuing supplies, and making returns on ore consignments, in the most of which he is assisted by ordinary clerks and bookkeepers, the remainder relates closely to strictly metallurgical matters, among which are the calculating of the charges for the furnaces, the supervision of the various operations, and the minutiae of tests, assays, and analyses, all devoted to the perfect working of the plant. Thus a considerable amount of business skill is expected of the man in charge, as well as, in the ordinary case, much metallurgical skill of a practical sort. Since there are two sides to these qualifications, the business and the metallurgical, it is easy to understand why the positions of this sort are filled now by metallurgists who have risen, it may be, from the laboratory or the assay office, and now by business men who have graduated from the counter or the desk. The latter may make the best superintendents, the more especially if they have procured enough knowledge of chemistry to enable them to understand the theory of the operations under their charge. In most cases it is probably better to have a man who has a good business head, along with a little knowledge of science, than a truly scientific one who knows but little of the ways of the commercial world. It may happen that the man in charge of even important works may possess no knowledge whatever of metallurgy; in which case it is essential for him to make the best use of that of others. Given the proper kind of subordinates, he may utilize their knowledge to a great extent, and by the use of tact produce equally as good results as the most skilled operators. We can see, therefore, that in these places success is more likely to be achieved more frequently by shrewd management than by individual accomplishments.

Given a superintendent who is merely a business man, his necessary complement is a metallurgist who is competent to plan and conduct the regular operations of the establishment. This is a scientific person who has spent, it may be, years in acquiring a knowledge of the principles and practice of ore treatment. He is almost necessarily a school man, and his knowledge of natural science is of first importance to the concern. In small establishments he may be the only technical man about the place, combining the duties in himself of the assayer, the chemist, and the metallurgist. In addition to these he may even act as the superintendent, a position that he is likely in the course of time to be advanced to if he shows any aptitude for the management of affairs. The duties of the metallurgist are to plan and direct the actual operations of ore treatment, he being the technical man, *par excellence*, of the establishment. He is necessarily a skillful and expert chemist, who has specialized upon the lines followed in the works, and he has under his orders the staff of chemists and assayers, his chief business being to correlate the results which they have obtained and apply them to the conduct of the processes. Being regarded oftentimes as the brains of the institution, he exercises in some situations a greater influence upon the fortunes of the company than any one else, his technical skill sometimes surpassing in its effect the best business and administrative skill at the command of the corporation. This fact is especially true of those metallurgists who are connected with smelting works, who have in numerous cases built up the fortunes of their companies, while profiting considerably, perhaps, themselves. In such works the most marked of their duties consist in making up the charges, as it is termed; which means performing the calculations upon which the ingredients of the smelting mixture are arranged and weighed out. This, it is hardly necessary to say, demands an accurate and wide knowledge of the effects of the different components, as well as a deep working knowledge of the commercial aspects of the business, which vary from day to day and demand the closest attention that can be given.

It is difficult to discover the line of demarkation between the metallurgist and the chemist. Strictly speaking, the chemist discovers facts as to the composition of the bodies before him, while the metallurgist makes use of the facts to carry on the work. In reality, however, the two occupations are but one in many, perhaps most, cases. It is not a long step between making an analysis of an ore, and using the information thus acquired to calculate the charge of a furnace. Accordingly, many chemists perform the functions of metallurgists, and, on the other hand, many metallurgists do their own analytical work.

As might be surmised, it is only in the larger establishments where such a complete division of labor is necessary that both metallurgists and chemists are in control. In smaller and less pretentous ones there may be but a single person to combine all the functions of the metallurgist, the assayer, and the chemist. The difference between the assayer and the chemist is nominal. Originally the difference was that the former worked exclusively by fire methods, while the latter used indifferently wet and dry processes. Nowadays many wet methods, some of extreme accuracy and elegance, have been included in the assayer's curriculum, making his work almost identical with that of the chemist. Consequently the boundary which divided their crafts in years gone by has now become partly obliterated, and it seems that the division is becoming year by year more nominal. The practice in both assaying and chemical analysis has improved vastly within recent years, the increasing number of graduates of technical schools who have taken up such work having had an excellent effect upon the morale and personnel of those branches. Better methods have been introduced and better educated men have taken hold of the work. The useful effect of these schools extends to all branches of metallurgy; while everyone knows how much mining and especially mining engineering owes to the influx of the numerous well equipped young men whom the schools have turned out in increasing numbers within late years. The large majority probably of both assayers and chemists now active are from the technical schools, the most of them endeavoring to qualify themselves for higher work against a time when an opportunity may arise for advancement. Few assayers in particular look upon their calling as fixed or final.

The pay of the various classes of workers at a smelter varies greatly, being influenced by the custom of the country, the degree of proficiency shown, and by other contingencies. The manager of a large metallurgical works in America oftentimes receives as much as $10.000 per year, with instances where considerably more is paid. His pay is sometimes contingent upon the financial success of the enterprise, to which his services may contribute greatly. Considering the heavy responsibilities of the position, and the large amount of detailed knowledge demanded, such a salary cannot be considered unduly large, and is not greater than such a man would naturally earn in any other occupation demanding equal business skill. Superintendents subordinate to managers receive from $250 to $500 per month, according to circumstances. Those whose duties include business as well as technical management naturally get the most. It is difficult to estimate the pay of metallurgists, but it is probably about equal to that of superintendents, with the exception that there is a class of self-styled metallurgists who perform only the commoner functions of the office and receive hardly more than the pay of assayers. From this comparatively low plane we rise to that of the foremost men of the craft, who combine the duties and abilities of superintendents, or even of managers, with their own higher technical powers and are paid accordingly. Such persons, being supposedly capable of conducting the affairs of the concern in all respects, are expected in case of need to furnish the designs of new buildings and parts of the works and receive high pay therefor. Metallurgists who have attained proficiency in the practice of their art are sought in consultation by parties who have need of advice or assistance in the preparation of plans or the conduct of metallurgical work. They are then called metallurgical engineers, or more restrictedly, consulting metallurgists. This branch of effort, when it concerns the planning and construction of new or the rehabilitation of old plants, does not differ in character from that carried on by ordinary constructing engineers. Men of this sort, if well known, frequently receive very high compensation for such services, as high or higher than those engaged either in civil or mechanical engineering.

If of a reflective or inventive turn of mind, the metallurgist will find a great field of usefulness open to him in improving the apparatus and operations of his chosen branch. It is thought by those familiar with the growth and recent history of metallurgical engineering that there are few other pursuits that offer such strong incentive to the young and ambitious, and few professional means of livelihood that surpass it in the certainty and magnitude of its returns. The assayer, as a rule, expects to receive about $100 to $125 per month, if fairly proficient in his calling. Some are hired at much less, some at much more, depending mainly on the locality. In some regions, as at Butte, Montana, the custom has been to pay the assayer a daily wage of from $4 to $4.50 per day instead of the monthly allowance. Chemists are paid rather better, owing, no doubt, to the comparatively greater complexity of their work and the greater experience necessary to get reliable results. Probably $150 to $200 per month will cover the extreme range of their compensation.

In addition to the above groups of employees, who invariably receive their wage on a monthly or yearly basis, there are others who in the practical running of the plant are hardly less important, but, as they are paid wages and not salaries, are less regarded. They are the foremen, who take personal charge of the routine work, subject to orders of the superintendent, to whom they report. They are commonly two in number, the day and the night foremen, and are persons who have risen from the rank and file of the operatives, having been distinguished by their skill, intelligence, and devotion to their company's interests. As they stand between the ordinary workmen and their officials, they occupy positions of no mean importance in relation to the welfare and success of the company. It will be found that the wages paid to these men, while on the daily and not on the monthly basis, are often higher than those obtained by the assayers, who are commonly regarded as superior in standing. The reason undoubtedly is that the position of foreman requires a man of tact and good understanding and experience, while the assayers are much younger men, of better education but less experience, who are, as it were, undergoing a probation in the practical affairs of life, and who are prone to look upon their billet as a sort of stepping-stone to something better. The average wage of a good foreman is $5 per day, or $6 in the more important establishments. In a properly regulated concern, all orders to the men, instead of being given direct to them by the superintendent, manager, or other person in chief authority, are transmitted through the foreman, for he it is who is directly responsible for their proper execution. In this way misunderstandings are avoided and proper discipline is maintaned. In order to maintain the foreman's authority over the men, he is intrusted with hiring and discharging them, all applicants for such positions being referred to him from headquarters. The superintendent should be scrupulously careful to avoid any appearance of overruling the foreman and thus undermining his authority, careful also to avoid the impression that just complaints will not be heard.

Below the foreman come several individuals who in large works perform services somewhat aside from the usual run of laborers and receive rather higher pay. They are the samplers, weighers, timekeepers, etc. Their duties are variously performed according to the size of the works, it being found that the larger the establishment the more specialized the duties, until in very large concerns there is an almost complete division of labor, the result of which is that no individual is expected to perform any work at all outside of his narrow specialty. This peculiarity of large concerns is the opposite of that in small establishments where every man is expected to make himself generally useful, and where, in consequence, the inquiring student, bent upon perfecting himself in the whole body of the art of ore reduction, has an ampler opportunity of doing so.*

The subordinates are the office force, especially clerks, who are under the immediate control of the superintendent, and the workmen, skilled and unskilled, who carry on the labor of the plant. The latter are divided into classes according as they perform routine or occasional work about the place. The former class consists of the men who attend the furnace, including the tappers, the slag-wheelers (in case the barbarous and wasteful method of discarding the slag by manual power is followed), the feeders, the charge-wheelers (giving way to the drivers of horses and better methods of bringing up the charges), the day and night foremen, and generally a weigher on each shift. With these are two or three yardmen and an ore sampler, whose work goes on more or less steadily. The class of intermittent laborers contains principally roustabouts, an uninteresting and not too intelligent class, composed usually of foreigners whose business is to fetch and carry. They have the stowing away of the ore, the guiding of the useful but antiquated wheelbarrow, and other jobs for which machinery is inadequate. This is the class that gives the most trouble, works the cheapest but costs the most, and of which managers are always anxious to be rid. These individuals receive salaries and wages in the West about as follows:

Manager, on term contract, per year	$6,000—	$10,000
Superintendent, unlimited time, per month	250—	350
Metallurgist, usual contract, per month	150—	500
Assayer (per day $4 to $5), per month	100—	150
Chemist (per day $5 to $6), per month	150—	200
Sampler, per day		4

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*This feature should be borne well in mind by students and others wishing to familiarize themselves with the operations of general metallurgy. It is better for such to connect themselves with small establishments, where their opportunities for obtaining practical knowledge of many branches are much better than in large concerns, where the tendency is to confine employees to particular occupations. Entering the service of a small company which is about to begin operations in ore reduction, it is not difficult for the young assayer, chemist, surveyor, or clerk to obtain practice in nearly or quite every line of occupation about the works. With much profit to himself he may be in turn millman, tankman, weighmaster, slag rustler, furnace tender, or roaster-man, and acquire within a short space information and practice which will be of immense value in later life. Without this practical knowledge in the handiwork of metallurgy it is impossible to achieve the highest degree of usefulness in the various branches of the calling. There is no doubt that the most successful metallurgists are those who, in addition to a wide theoretical knowledge of their craft, have also a practical knowledge of how to do it; that is at least as valuable.

Foreman (day), per day	5—	6
Foreman (night), per day	4—	5
Tappers and feeders, 12-hour shift		3.50
Slagmen and charge wheelers, 12 hours	2.50—	3.00
Roustabouts, per day of 10 hours		2.00

The total number of men connected with the running of one furnace and performing their duties at the works is usually not less than 20, and if the carpenters and others whose exertions are frequently but not constantly employed be included, it may rise to 40. On the other hand, there are many small plants which need but a dozen or so of men, but these of the best class. The number employed will depend mainly on the arrangement of the machinery of the plant, its adaptation to the purposes intended, and the inventiveness and originality of the designer. In these respects there is a vast difference between plants and a corresponding difference in the cost and perfection of the work they do.

The consulting engineer, in name, at least, if not in principle, is the outgrowth of recent conditions. It has become quite the fashion with mining and metallurgical organizations that may desire to carry on their work in a more than ordinarily thorough style to employ consulting experts, much the same as one would employ an attorney or a physician. The consulting expert is a professional person who prescribes and gives advice. He has no authority, and is not supposed to participate in the management or conduct of any part of the business of his employers for the time being, but merely to observe conditions and plan and advise. The same qualifications which make the physician or the lawyer valuable to his clients must be possessed by the consulting engineer—namely, varied accomplishments, deep insight, and extended experience. It is not every new company that is circumstanced so as to avail itself fully of such qualifications.

In the first place, the tendency of beginners is to postpone the employment of the higher sort of professional assistance until a late date, under the erroneous impression that a small enterprise, and especially one in its earlier stages, has no need of particular skill or knowledge. This is a great mistake. If to this be added the natural feeling of dislike that many men have to association with persons of talent or attainment beyond their own, sufficient reason is found for the omission of many companies to employ skilled engineers in their work. There is something also in the desire of ambitious members of companies—stockholders, it may be—to distinguish themselves by conducting the work in person. No doubt the same is to be expected in any other pursuit; for human nature is much the same everywhere and under every condition. Should the new company contain no member experienced in mining or metallurgy, it will ordinarily dispense with high-class professional assistance, at least until mistakes have been made and recognized, and some loss of time and money incurred. Experience frequently induces them to place their affairs finally in some measure in the hands of qualified persons, among whom the consulting engineer may be one. Should he possess the requisite skill, integrity, and experience, together with enough force of character to impress his views upon his associates so that they feel impelled to carry them out, his engagement will likely be of great advantage.

According to the circumstances of the case, the engineer may have been hired either to give advice, to plan works and supervise their construction, or to do both. Or he may even have been asked to direct the running of the plant after it has been built. If his duties are solely to give advice, his designation properly is consulting engineer or consulting metallurgist; if to plan the works, he may be termed the designer; if to supervise the building of the plant, he is called the constructing or superintending engineer; and if to conduct it subsequently, he is rightfully called either the manager or superintendent, according to the nature and extent of his services. In the former case he resembles the attorney who confines himself to chamber practice. He has clients, perhaps several at one time, to whom he gives advice, but his time is not, unless by special arrangement, wholly at the disposal of any one of them, and he bases his charges upon the character and extent of the services rendered, or even, as do other professional men, perhaps upon the supposed wealth of his clients.

No one, not even the manager, has so many and favorable opportunities to benefit his company as the consulting engineer. Viewing the entire field of its activities in the light of his wider experience, he sees many possible improvements in equipment, practice, and methods. Doubtless the career of every such engineer contains instances where in the compass of a single letter of advice he was able to save his company the amount of salary for the year. A hint to a careful foreman about the conduct of the work, or a suggestion as to the saving of some hitherto unregarded by-product may often produce an almost invaluable effect upon the fortunes of the company. If mining and metallurgical companies knew how faulty even the best practice is, and how easy it is for the experienced engineer to suggest improvements, the employment of expert assistance would be the rule and not the exception. Consulting engineers might reasonably be paid according to the savings they are enabled to effect. Thus, if by some suggestion an economy of so many thousand dollars per year results, he might receive one-half or one-fourth of the sum for a stipulated number of years, or as long as his engagement lasts.

Metallurgical works should always be open to the visits and criticism of qualified persons, instead of being, as so often the case, shut against the outer world. The free exchange of ideas by the staffs of various works has done more for the prosperity of companies and for the advance of metallurgical science than any other cause which can be named. Those concerns which shut themselves up jealously, refusing the common courtesy of a visit of curiosity, are almost invariably behind the age in some department, perhaps in all, and manifest a disposition which is, to say the least, inharmonious with the spirit of the times. This narrow and illiberal spirit, which manifests itself oftenest in the behavior of those foreigners who have been received on our shores, reacts most injuriously upon the fortunes of those who practice such exclusiveness, and also on the progress of the arts which they make, as a rule, so much pretense of understanding.

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